EUROPEAN URANIUM
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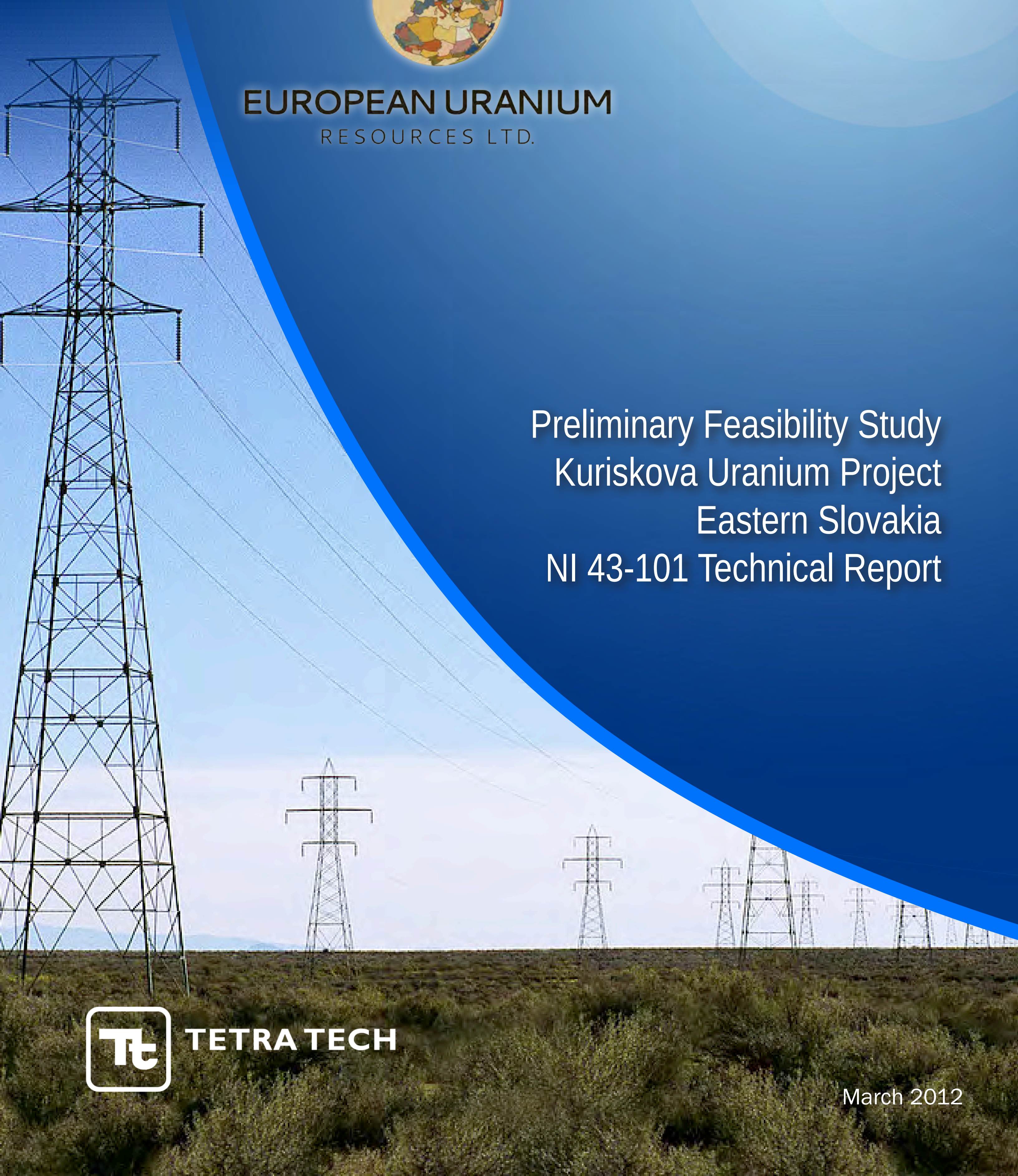
Preliminary Feasibility Study
Kuriskova Uranium Project
Eastern Slovakia
NI 43-101 Technical Report



TETRA TECH

March 2012

NI 43-101 Technical Report
Kuriskova Uranium Project

Eastern Slovakia

Prepared for:

EUROPEAN URANIUM
RESOURCES LTD.

15782 Marine Drive, Unit 1
White Rock, B.C. Canada V4B 1E6

Prepared by:

TETRA TECH

350 Indiana Street, Suite 500
Golden, Colorado 80401
(303) 217-5700
Tetra Tech Project No. 114-310990

Effective Date: March 13, 2012

Qualified Persons

"Original document dated, signed and sealed by Rex Clair Bryan, Ph.D"
Rex Clair Bryan, Ph.D

"Original document dated, signed and sealed by Richard W. Jolk, P.E, Ph.D"
Richard W. Jolk, P.E, Ph.D

"Original document dated, signed and sealed by Andrew P. Schissler, P.E, Ph.D"
Andrew P. Schissler, P.E, Ph.D

"Original document dated, signed and sealed by Landy Stinnett, P.E, A.S.A"
Landy Stinnett, P.E, A.S.A

"Original document dated, signed and sealed by Scott Voltura, P.E"
Scott Voltura, P.E.

TABLE OF CONTENTS

LIST OF TABLES

LIST OF FIGURES

LIST OF APPENDICES

LIST OF ACRONYMS

Acronym	Definition
AA	Atomic Absorbsion
ABA	acid-base accounting
Ag	silver
amsl	above mean sea level
AP	acid-generating potential
APOX	alkaline pressure oxidation
ASTM	American Society for Testing and Materials
Au	gold
bf-XRF	borate fusion bead and XRF
Bq/L	becquerels per liter
BWi	Bond Work Index
C	carbon
$CaCO_3$	calcium carbonate
CANMET	Canada Centre for Mineral and Energy Technology
CAPEX	capital cost estimate
CIM	Canadian institute of Mining, Metallurgy and Petroleum
cm	centimeters
CO	carbon monoxide

Coeff.	Coefficient
CSUP	Ceskoslovensky Uranovy Prumysl
Cu	copper
dia	Diameter
e U_3O_8	equivalent uranium assay data
EDA	Exploration Data Analysis
EDS	Energy Dispersiove Spectroscopy
EIA	Environmental Impact Assessment
EIS	Environmental Impact Statement
EPCM	engineering, procurement and construction management
EU	European Union
eU%	equivalent-uranium precent
FS	feasibility study
g	grams
g/L	grams per liter
Geostats	Geostats Pty Ltds
GOX	gaseous oxygen
GWe	gigawatt electrical
HG	high grade
Hg	mercury
HRI	Hazen Research, Inc.
IAEA	International Atomic Energy Agency
ICP	Inductively Coupled Plasma
IPPC	Integrated Pollution Prevention and Control
IRR	internal rate of return
IX	ion exchange
kg	kilograms
kg/t	kilograms per tonne
km	kilometer
km^2	square kilometers
kW	kilowatts
L/m	liters per minute
lbs	pounds
LOM	life-of-mine
LOX	liquid oxygen
m	meters
m^3/hr	cubic meters per hour
Mining Act	Law No. 44/1988 Coll
ML	Mining License
ml	milliliters
mm	millimeters
Mo	molybdenum
MoE SR	Ministry of Environment of Slovak Republic
MPa	Mega Pascals
MS	Microsoft

Na_2CO_3	sodium carbonate
$NaHCO_3$	sodium bicarbonate
NaOH	caustic soda
NI 43-101	Canadian National Instrument 43-101
NNP	net neutralizing potential
NO_2	nitrogen dioxide
NO_x	nitrogren oxides
NORM	Naturally Occurring Radioactive Material
NP	acid-neutralizing potential
NPV	net present value
NRC	U. S. Nuclear Regulatory Commission
O_3	ozone
OES	Optical Emission
OPEX	operating cost
PAH	Pincock, Allen & Holt
Pb	lead
PCO_2	partial pressure of CO_2
PES	Preliminary Environmental Study
PFS	Preliminary Feasibility Study
PM_{10}	particulate matter
POX	pressure oxidation
ppm	parts per million
pp-XRF	pressed pellet x-ray fluorescence spectrometer
psi	pounds per square inch
psia	pounds per square inch absolute
psig	pounds per square inch gauge
PUF	polyurethane foam
PWRE	polygonal wireframe resource estimate
QA/QC	Quality Assurance/Quality Control
QPs	qualified persons
RC	reverse circulation
RDi	Resource Development Inc.
RDQ	rock quality designation
RMR	rock mass rating
RO	reverse osmosis
Sb	antimony
SDU	sodium diuranate
SEDAR	System for Electronic Document Analysis and Retrieval
SG	specific gravity
SGUDS	State Geological Institute of Dionýz Štúr
SO_2	sulphur dioxide
SPT	Standard Penetration Test
SRM	standard reference material
SVOL	search volume

SWWB	site-wide water balance model
SX	solvent-extraction
t/m^3	tonnes per cubic meter
the Project	Kuriskova Uranium Project
tpd	tonnes per day
tpy	tonnes per year
U	uranium
U_3O_8	triuranium octoxide
UCS	Uniaxal Compressive Strength
UPF	underground process facility
Uranpres	Uranovy Prieskum
UTM	Universal Transverse Mercator
VSE	Východoslovenská energetika
WTP	water treatment plant
XRF	x-ray fluorescence spectrometer
Zn	zinc

1.0 SUMMARY

European Uranium Resources Ltd. (EUU) (formally Tournigan Energy, Ltd.) retained Tetra Tech, Inc. (Tetra Tech) to complete a Preliminary Feasibility Study (PFS) for the Kuriskova Uranium Project (Project) in Eastern Slovakia. The Project from which the results were used in the preparation of a Canadian National Instrument 43-101 (NI 43-101) Technical Report. The scope of work was to define the necessary elements of the project from startup, construction and commencement of operations through final closure and reclamation so as to estimate the value of the Project at a ±25 percent level of accuracy. All monetary units are in 2011 US$ and where necessary Euros (€) have been converted using a rate of 1.4 US$ per 1 €.

Since the press release of January 30, 2012, a non-material change to the net present value (NPV) has been identified and has been adjusted accordingly in this report.

1.1 Location and Access

The Project is located approximately 8 kilometers (km) northwest of the boundary of Kosice, a regional industrial and administrative city in east Slovakia. The Project property (Property) lies close to the main paved road No. 547 between Kosice and Spisska Nova Ves, and is readily accessible via a network of minor, un-surfaced roads, and four-wheel-drive trails that traverse the mineralized resource area.

1.2 Ownership

The official title to the deposit area is the Kosice I. The full title of the current exploration license issued to Ludovika Energy (EUU's wholly-owned Slovakia subsidiary) refers to "Cermel-Jahodna - U-Mo, Cu ores," and it was granted on March 21, 2005 by the Geology and Natural Resources Department at the Ministry of the Environment of the Slovak Republic. The Project license area amounts to 31.75 square kilometers (km^2). The initial period of validity of the license is four years. The license was extended for a second four-year term effective early April, 2009. In the future, this license can be extended or converted to a mining lease. The name and code of the region is Kosicky 8, and the name and code of the cadasters are Kosice I - 802, Kosice II - 803, and Kosice - Okolie - 806.

The conditions of the exploration license issued to EUU are enumerated in Law No. 44/1988 Coll (Mining Act) for protection and exploitation of the mineral wealth. A uranium royalty to the Slovak government is set at 10 percent of payable revenues, but can be lowered based on criteria presented in the Mining Act.

Tetra Tech is not aware of the terms of any royalties, back-in rights, or other agreements and encumbrances to which the Property is subject. Tetra Tech has relied on information provided by EUU personnel regarding property license status and believes all licenses to be in good standing, but Tetra Tech has not undertaken a title search.

1.3 Environmental and Permitting

The Environmental Impact Assessment (EIA) process under the Slovakian EIA Act (Act No. 127/1994 as amended most recently by Act No. 24/2006) will be the primary permitting driver and is anticipated to take 18 to 24 months to complete. A multi-agency regulatory process will be completed to obtain all required permits and approvals necessary to construct, operate, and ultimately close the Project. The permitting process in Slovakia is relatively complex and includes participation from the Regional Mining Bureau, Regional Construction Office, the

Slovakian environmental agencies, several other government agencies, companies, affected municipalities, and the public.

The Project area includes two Natura 2000 ecological protection areas. Natura 2000 is a network of areas designated by European Union (EU) member countries with the objective of protecting birds, biotopes, and other animal species and their habitat. To limit potential adverse effects to the overlapping Natura 2000 site, the Project includes minimization of surface disturbances. To this end, the Kuriskova deposit is accessed by means of a decline to the underground mine and process plant.

Baseline studies are being conducted with the primary goal of collecting and analyzing technically adequate data that will support the required permit applications and environmental documentation including an Environmental Impact Statement (EIS). Many of the baseline studies have been initiated and continue to advance as the Project moves forward. The primary study areas include:

- Water resources;
- Geochemical characterization;
- Water treatment;
- Ecology (flora and fauna);
- Meteorology, climatology, and air quality;
- Soils; and
- Radiological monitoring.

Reclamation will primarily occur at the end of the mine life with the exception of soil stockpiling and temporary stabilization that will be conducted during the initial site preparation. Post-mining land uses will include conversion of surface mine facilities to other feasible economic uses. Infrastructure and facilities that cannot be converted to a post-mining land use will be decommissioned, demolished, and reclaimed. The general approach will be to recontour, regrade, and scarify, where needed, placing topsoil and revegetate. The mine portal for mine access and shafts will also be sealed. Process plant equipment that cannot be salvaged will be cemented in place underground.

1.4 Geology

The main zone of the Kuriskova deposit occupies dilational zones along the geologic contact between the overlying competent andesitic metavolcanic unit and the underlying metasediments. Here, two styles of mineralization are present; firstly uranium mineralization associated with andesitic tuff/tuffite units at the base of the main andesite unit. The tuffs are phosphorous rich, and it appears that phosphorous has preferentially fixed the uranium minerals, resulting in localized high-grade zones of 1 to 5 percent uranium. Secondly, uranium mineralization hosted directly on the andesite/sediment contact, which is generally lower grade (0.1 to 0.5 percent uranium) and is regarded as a more horizontally shifted form of the tuff hosted zone described above.

Shearing along this contact has resulted in tectonic disturbance and poor ground conditions. Tectonic disturbances have also resulted in schistose foliation and slaty cleavage (giving poor ground conditions in some softer sedimentary units) and fault offsets, some of which disrupt the main deposit. Uranium mineralization hosted within hanging wall andesites are characterized by their presence as often discrete lenses associated with thin quartz-carbonate veins and hematite. Uranium grades within these zones are variable.

The overall dimensions of the main deposit established to date is approximately 750 by 550 meters (m) and about 2.5 m in average thickness. In some areas, the thickness is more than 10 m. As mentioned, there are also minor mineralized zones in the hanging wall of the main deposit, though their relationship to the main deposit is unresolved.

1.5 Mineralization

Uranium mineralization is stratabound with a "vein-like shape," the Main Zone deposit is hosted in sheared andesite tuff, and the mineralization of the overlying Hanging Wall zone is associated with stockwork veining in andesite flows.

The Main Zone has moderate to steep dips, an average thickness of 2 to 10 m, strike and dip extents of several hundred meters, and average grades range from 0.1 percent triuranium octoxide (U_3O_8) over 0.5 percent U_3O_8. The Main Zone North (Zone 1N) accounts for 63 percent of the total contained pounds of uranium in the deposit.

1.6 Exploration, Drilling, and Sampling

EUU continued to complete both infill and exploration core drilling on the Project during 2010 to 2011. During this program a total of 18 new drill holes were added to the database, a total length of approximately 4,548 m. Geologic logging, sampling, and assaying were completed using the programs, procedures, and methods in place and described in Sections 13.0 and 14.0 of this report.

1.7 Mineral Processing and Metallurgical Testing

Multiple metallurgical test programs since October 1993 have been performed; the most relevant of which have been performed by Hazen Research Inc. (HRI) between 2010 and 2012. These programs revealed the most viable means of recovering uranium and molybdenum is through the use of carbonate pressure leaching, followed by direct precipitation of uranium in the form of sodium diuranate (SDU) from the leach filtrate. The resulting SDU cake is washed, repulped, and the uranium is recovered as a hydrated uranium peroxide precipitate to improve product purity. Molybdenum is recovered via direct precipitation from the SDU filtrate as a molybdenum sulfide concentrate. Metallurgical testing revealed overall uranium and molybdenum recoveries of 92.0 and 86.8 percent, respectively, are achievable under the conditions selected.

1.8 Resource Estimates

Table 1.1 details the classified resources at the Project. Resources are stated at a 0.05 percent uranium cutoff grade, which is approximately 0.06 percent U_3O_8. The 0.05 percent uranium cutoff equates to approximately 1.18 pounds (lbs) U_3O_8 per tonne of insitu-mineralized material. At a uranium price of US$60 per pound (lb) U_3O_8, the cutoff grade equals an in situ value of approximately US$70/tonne; which is deemed by Tetra Tech to be sufficient to define a "reasonable potential for economic extraction;" a necessary condition for a resource statement. Tetra Tech cautions that it may become appropriate to use either a higher or lower cutoff grade to state resources, and that will only be determined from the mining scoping studies.

1.9 Recovery Methods

The underground processing facility for the Kuriskova deposit was developed to utilize conventional crushing and grinding processes for comminution of mined ore prior to leaching. Leaching of uranium and molybdenum from the ore will be achieved using a carbonate leach process and pressure oxidation of the ore. The ore will be oxidized at 200^{o}C and 100 pounds per square inch gauge (psig) oxygen partial pressure for two hours, achieving essentially 100 percent oxidation of all sulfides. Uranium extraction in leaching is estimated at 94 percent, and molybdenum extraction is estimated at 87 percent, based on test work at HRI.

Recovery of uranium from the leach solution will be achieved through precipitation with caustic soda and subsequent repulping and precipitation with hydrogen peroxide as a purification step. Overall uranium recovery is estimated to be 92 percent.

Recovery of molybdenum from the process solution will be achieved by pH adjustments of the solution prior to precipitation with sodium hydrosulfide. Overall molybdenum recovery is estimated to be 86.8 percent.

Uranium and molybdenum concentrates will be dewatered and packaged in barrels for transportation. Leached tailings from the plant will be combined with cement and deposited underground as paste backfill for the mine or placed in underground excavations.

1.10 Underground Mineral Reserves

The mineral reserves for the Project were developed by applying the relevant economic and design criteria to the resource model in order to define the economically extractable portions of the resource. The reserves were developed to meet Canadian National Instrument 43-101 (NI 43-101) standards. The NI 43-101 standards rely on the Canadian institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards on Mineral Resources and Mineral Reserves adopted by the CIM council.

The mineral reserve listed in Table 1.1 was generated from the indicated mineral resource after the application of the economic cutoff grade of 0.13 percent uranium, stope design, external dilution, and recovery parameters. The reserves have been shown to be economic, and Tetra Tech believes that they are reasonable for the statement of probable reserves.

Table 1.1. Kuriskova Mineral Reserves

Classification	Tonnes	Grade % U	Grade % Mo
Proven	0	N/A	N/A
Probable	2,528,000	0.346	0.046
Total	2,528,000	0.346	0.046

1.11 Mining Methods

The deposit is planned to be extracted by underground mining methods. The underground mine plan was designed around the steeply dipping mineralized zone, with an average thickness of 2.5 m and an approximate strike length of 800 m. Underhand cut and fill with paste backfill was chosen as the mining method after consideration was given to the geometry, rock mass strength, and the process plant feed tonnage requirements. The current plan is to access the mine by a 2.6 kilometer (km) decline, which will intersect a spiral ramp in the footwall of the deposit. Access drifts will be driven from the spiral ramp into the mineralized zone for production mining. Due to the low rock mass strength and Rock Quality Designation (RQD) of the mineralized zone drill and blast within the ore body may be difficult to achieve, so ore mining by road headed was chosen as the primary production method. Once mined, rock will transported to the process plant by 30 tonne underground haul trucks.

Development drifting will be accomplished by mechanized drill, blast, load, haul methods. Drifts sizes through the mine will be 5 x 5 m, and the decline will be 6 x 6 m. Development rock will be hauled to the surface by 30 tonne haul trucks where it will be crushed and screened. Total pre-production development time was estimated to be three years. Pre-production development included decline development, process plant excavation, process plant installation, ventilation development, and development to the ore body.

The total estimated underground mine life was estimated to be just more than 12 years of ore production. Daily output from the operation is expected to achieve 600 tonnes per day (tpd) with 350 working days per year for an annual total capacity of 210,000 ore tonnes.

1.12 Rock Mechanics

The rock mechanics analysis for Kuriskova centered on data collection and analysis which included drilling five geotechnical holes in 2011 with measured orientation, logging the core to establish rock mass rating (RMR) for mine design, and testing physical specimens for strength and other parameters. All information was considered in the mine design.

The results of the RMR are shown in Table 1.2.

Table 1.2. RMR Summary

Rock Unit (Top Down)	RMR Range	RMR Median	RMR / Type
Alluvium	Soil	Soil	Soil
Andesite Tuff	30/40	35	Upper IV, poor rock
Meta Tuff	30/45	38-40	Upper IV, poor rock
Schist	30/40	35	Upper IV, poor rock
Violet Schist	35/45	36-39	Upper IV, poor rock
Sandstone	35/50	42-44	Lower III, fair rock
Sandstone/Andesite	35/50	42-44	Lower III, fair rock

The RMR analysis revealed most of the lithology is in the 30 to 50 range on a scale of 0 to 100. Zero is rock with no strength and structure, and 100 is perfect rock. Kuriskova rock is classified as poor to lower fair rock. Based on case histories, this rock with this range of RMR requires ground control on cycle. For Kuriskova, this will be 2.5 m long tensionable resin bolts on a 1.5 m square pattern placed on cycle, wire mesh on 50 percent of all drifts, and 0.1 m shotcrete on 25 percent of all drifts.

The RMR assessment was supported by a physical testing program taking rock core specimens and utilizing certified laboratories in Slovakia and the United States. The results of the program are found in Table 1.3 below.

Table 1.3. Physical Testing Results

Test	Samples	Range (MPa*)	Mean (MPa)
Uniaxal Compressive Strength (UCS)	12	5.5-56.1	28.7
Brazilian Tensile Strength	10	1.2-8.4	3.3
Direct Shear	1	9.7	N/A
Elastic Modulus	2	11,511; 13,223	12,367

Measured specimen strength Kuriskova is higher than the rock mass due to jointing and fractures. The Kuriskova deposit has been subject to tectonic forces. As a result, the rock is severely fractured.
*MPa = Mega Pascals

The mining method for Kuriskova that best accommodates deposit shape, the fractured rock, and rock type is underhand cut and fill with structural paste backfill. The low rock strength has insufficient strength to stand more than 2 m span. Utilizing paste backfill will provide direct immediate support.

Paste backfill design was accomplished by composing various mixes of Kuriskova process plant tailings, water, cement, fly ash, and aggregate rock. The optimum mix to achieve a 3.4 MPa UCS target utilizes 60 percent tailings, 11 percent cement, and 29 percent water. Quarry rock was tested as not necessary for strength, and the fly ash added excessive alkalinity.

Paste backfill pump sizing was done by Putzmeister, a world leader in paste pumping. Based on 29 percent water content, an 85 millimeters (mm) mini-slump for rheology, and the mine layout, a pump was designed and estimated as having a capability to 20 cubic meters per hour (m^3/hr) at 100 bar. Operating pressures of the paste backfill are calculated to be in the 30 to 60 bar range at the pump outlet.

A survey was done for 33 Canadian mines using mine paste backfill of various types and various applications (Souza). This survey showed that 50 percent of the application was for ground control. The other 50 percent was for a combination of reducing mining costs, environmental protection, fire control, and ventilation. The types of mining methods where paste backfill was applied were 33 percent for forms of cut and fill, 67 percent for non-cut and fill.

1.13 Project Infrastructure

The surface facilities for the Project will be accessed using a service road into a perimeter security fenced area that will include the following structures:

- Administrative Building, which will also include the mine dry, sample preparation, assay, and environmental laboratories.
- Warehouse.
- Truck shop with five truck bays and a 10-ton overhead crane.
- Portal-mine entrance.
- A covered roof structure to park three man-trips and four boss buggies.
- An undercarriage washer and truck scale.
- Reagent storage.
- Molybdenum and uranium concentrate/products storage.

- Miscellaneous mining supplies storage.
- Potable water and fire water tanks and associated pumping systems.
- A new substation, generator sets for emergency service.
- A water treatment plant (WTP).
- A sewage treatment plant.
- A storm water retention pond.

Two additional sites on the surface include the exhaust ventilation shaft to include egress hoist and a second egress shaft and hoist.

1.14 Hydrology and Hydrogeology

Water affects the Kuriskova project in two ways. First, it affects the mining operations. The proposed underground mining method will intersect the groundwater, and water will report to the underground workings. Specifically, the rate of groundwater inflow anticipated to report into the underground working plays a role in the design of these facilities, the constructions methods, the infrastructure to handle this water, and the associated costs to construct and operate these facilities. Second, water is a natural component of the environment, and as such, how the water interacts with the mine must be considered.

In order to understand these issues, a program was designed to investigate the hydrology and hydrogeology of the site. These studies have been conducted by members of the State Geological Institute of Dionyz Stur (SGUDS), private consultants (such as HES-COMGEO), and staff from Tetra Tech. These studies analyzed published information on the geology, hydrology, and hydrogeology of the Project area. This included compiling climate, stream flow, springs, and groundwater well data. In addition to these studies, three wells were installed on the site that will provide data on groundwater levels and water quality.

Aquifer testing of hydrologic boreholes on the site, as well as published information, all confirms that the rocks in and adjacent to the site possess low hydraulic conductivity. The rocks are saturated, but tend to yield water slowly. Analytical models predict that on average, approximately 600 liters per minute (L/m) may be expected to flow into the working drifts. This rate represents a relatively small volume of water. Thus, the mine design is assumed to not require a separate, active dewatering system. Instead, the mine design assumes that underground seepage will be collected in underground sumps and mostly used in paste backfill production.

1.15 Market Studies

Fifteen countries depend on nuclear power for at least a quarter of their electricity. France is the leader at roughly 75 percent, followed by Slovakia at over 50 percent; Belgium, Ukraine, Lithuania, Hungary, Armenia, Sweden, Switzerland, Slovenia, Czech Republic, Bulgaria, and South Korea derive over one-third of their power requirements from nuclear generation. Japan, Germany, and Finland obtain more than one-quarter of their needs from nuclear; and the United States gets nearly 20 percent of its total through fissionable material. Presently there are 65 power reactors being constructed in 14 countries, to provide roughly 62 gigawatt electrical (GWe) of additional installed capacity. Uranium production to feed these units has increased substantially in the past decade. Total production throughout the world in 2003 was 35,200 tonnes; by 2010 this figure had risen to 53,700 tonnes, equating to a 4.3 percent per year compounded increase. Four companies accounted for 59 percent of world uranium production in 2010 (Cameco, Areva, KazAtomProm, and Rio Tinto); and the largest ten mines were responsible for 55 percent of the total. Thus, there is a notable concentration of supply reposing within a small number of entities.

Long-term averaging of prices has been used to assess behavior, and in this report the three-year and eight-year average projections are taken as reasonable bounds for future U_3O_8 prices. A single price is preferred, both for establishing a cutoff grade in the deposit, and to allow development of a simplified cash flow as part of the Project's economic analysis. In examining the underlying price data it is determined that the 10-year annual average is US$69.40/lb U_3O_8, whereas the three-year rolling average price from 2015 through 2027 is US$66.90/lb. In this report, a single, constant-dollar future price for U_3O_8 produced on site at Kuriskova is taken at US$68/lb.

1.16 Capital and Operating Costs

The initial capital cost estimate (CAPEX) for the Project is approximately US$225 million subject to qualifications, assumptions, and exclusions. The initial capital cost summary and distribution are shown in Table 1.4.

Table 1.4. Initial Capital Cost Summary

Item	US$ Millions
Direct Cost	
Underground Mine	$91.56
Process Plant	$28.37
Environmental/Reclamation	$1.03
Infrastructure	$23.18
Total Direct Cost	$144.14
Project Indirect Cost	$24.12
Other Owners Cost	$25.75
Total Indirect Cost	$49.87
Total Direct and Indirect Cost	$194.01
Contingency	$31.00
Total Initial Capital Cost	$225.01

Sustaining capital over life-of-mine (LOM) totals US$70.85 million. Table 1.5 shows a summary of the breakdown of costs.

Table 1.5. LOM Sustaining Capital Cost Summary

Area	US$ Millions
Underground Mine	$67.47
Process Plant	$.09
Infrastructure	$1.00
Environmental/Reclamation	$2.29
Total Sustaining Capital	$70.85

The LOM operating costs (OPEX) are estimated at US$201/tonne ore. Table 1.6 shows summary of the breakdown of unit operating costs.

Table 1.6. LOM Unit Operating Costs

Operating Costs	US$ / Tonne of Ore
Underground Mine	$86.51
Process Plant	$92.99
Infrastructure	$2.57
General & Administrative	$18.74
Total LOM Operating Cost	$200.81

1.17 Economic Analysis

Economic analysis of the Project was performed to assess the economic viability of constructing and operating the Project as designed. The economic analysis was based on the following factors:

- End of year discounting;
- Constant 2011 US dollars; and
- Stand-alone project.

The analysis was based on mine plans and production schedules derived from the most current resource estimates. Yearly LOM metal production averages approximately 786 tonnes of U_3O_8 as yellowcake and 84 tonnes of molybdenum as molybdenum sulfide over the 13 years of production. Details of the reserve calculations and production schedules are shown in Section 15.

A proforma cash flow statement projects potential revenues, transport costs and facility operating and capital costs to yield annual net cash flows which are then discounted to determine a project NPV. The cash flow excludes corporate income taxes, but includes the cost of all royalties and Local Community Support payments. The Base Case NPV, at 8 percent discount rate, and internal rate of return (IRR) are calculated to be US$276 million and 30.8 percent, respectively. Initial capital costs are US$225 million with a simple payback of 1.9 years. The highest sensitivity for both NPV and IRR is future uranium price. Changes to operating and initial capital costs had less of an effect on project NPV and IRR than uranium price. A detailed analysis of these values and other metrics are contained in the following sections of this report.

A Monte Carlo simulation suggest a worst-case situation wherein the Project returns an NPV at a discount rate of 8 percent (NPV8) of about US$202 million and a best-case scenario with an NPV8 of nearly US$319 million. It is noted that the single-point analysis resulted in an NPV of US$275 million, but under the conditions assumed in this exercise, the median value (50 percent above, and 50 percent below) is US$261 million. There is a 100 percent chance of achieving an NPV8 of US$202 million, but only a 20 percent probability of attaining or exceeding the base case US$276 million figure presented in the underlying cash flow analysis.

1.18 Project Opportunities

There are opportunities which may provide improvements and cost savings for the Kuriskova project including the following:

- EUU intends to conduct further step-out exploration drilling where the high-grade mineralization is open along strike and at depth;
- Additional geotechnical and hydrological studies are required to evaluate alternative mine designs, tailings placement, and mine accesses which may improve costs and schedules for construction and mine production.

Project improvements since the publication of the PEA in July 2009 include:

- Shortening of the preproduction construction period by one and one-half years to three years in the PFS from four and one-half years in the PEA;
- Increase in the indicated resources by 94 percent to 28.5 million lbs U_3O_8;
- Increase by 62 percent in the average uranium grade to the process plant from 0.252 percent U_3O_8 to 0.408 percent U_3O_8;
- Increase in the uranium recovery by 2 percent to 92 percent in the PFS from 90 percent in the PEA; and
- Lower LOM operating cost by 26 percent to US$22.98/lb U_3O_8.

2.0 INTRODUCTION

EUU retained Tetra Tech to complete a PFS for the Project. The Project is located approximately 8 km northwest of Kosice, the regional industrial and administrative center of eastern Slovakia. As part of this assignment, Tetra Tech has completed a Technical Report in accordance with NI 43-101 and Form 43-101F1.

2.1 Terms of Reference

This report has been prepared in accordance with the guidelines provided in the NI43-101 Standards of Disclosure for Mineral Projects dated July 2011.

Tetra Tech is not an associate or affiliate of EUU or of any associated company. Tetra Tech's fee for this technical report is not dependent in whole or part on any prior or future engagement or understanding resulting from the conclusions of this report. The fee is in accordance with standard industry fees for work of this nature.

2.2 Scope of Work

The scope of work conducted by Tetra Tech per the request of EUU was the development of a PFS for the Kuriskova Project that defines the necessary elements from construction and startup of the Project through final closure and reclamation at a ±25 percent level of accuracy. All monetary units are in 2011 US$ and where necessary Euros (€) have been converted using a rate of 1.4 US$ per 1 €.

Leading and working in coordination with its subcontractors, Tetra Tech developed the critical design parameters for the Project consisting of geology, mineral resources, mine plans, mineral reserves, metallurgical testing, process plant design, infrastructure, environmental requirements, site drainage, hydrogeology, permits, closure requirements, and capital and operating cost estimates, resulting in the overall economic evaluation of the Project.

2.3 Sources of Information

This report is based on data supplied by EUU, as well as previous technical reports by third parties. Tetra Tech has prepared this report exclusively for EUU. The information presented, opinions and conclusions stated, and estimates made are based on the following information:

- Source documents used for this report are summarized in the Section 27.0 of this report;
- Assumptions, conditions, and qualifications as set forth in the report;
- Data, reports, and opinions from prior owners and third-party entities; and
- Personal inspection and review.

Tetra Tech has not independently conducted any title or other searches, but has relied upon EUU and their legal firm of JuDr. Peter Kocicka of Banska Bystrica, Slovakia for information on the status of the claims, property title, agreements, permit status, and other pertinent conditions. In addition, Tetra Tech has not independently conducted any sampling, mining, processing, economic studies, permitting, or environmental studies on the Property.

2.4 Personal Inspections

The following qualified persons (QPs) conducted a personal inspection of the Kuriskova Property:

- Andrew Schissler, July 6 to July 13, 2011
- Rex Bryan, August 22 to August 24, 2011

Other contributors to the report have visited the site on several occasions (Table 2.1):

- Jim Donovan, October 9 to October 13, 2011
- Dwaine Edington, May 2011 and October 2011
- Larry McGonagle, August 22 to August 24, 2011
- Patsy Moran, November 16 and 17, 2010

2.5 Effective Date

The effective date of this report is March 13, 2012. This report is an update from an amended report issued by Tetra Tech on June 9, 2011 with an effective date of mineral resources statements of April 26, 2011.

2.6 Contributors to the Report

In addition to the QPs responsible for the technical report, a large number of contributors provided data and other information. A summary has been provided in Table 2.1.

Table 2.1. PFS Contributors

Discipline	Responsible Party	Subject Matter Expert
Metallurgy, Mineral Processing, and Recovery Methods	Tetra Tech	Richard Jolk, Ph.D., P.E. Cameron Wolf Alex Norgren
Mineral Resource Estimate	Tetra Tech	Rex Bryan, Ph.D. Geoff Elson
Underground Mineable Mineral Reserve and Mining	Tetra Tech	Andrew Schissler, Ph.D., P.E. Chris Schauffele
Site Service Facilities Infrastructure	Tetra Tech	Jim Donovan Scott Voltura, P.E.
Power Supply and Distribution	Tetra Tech	Jerry Harris, P.E.
Market Studies and Contract	Independent Consultant to Tetra Tech	Landy Stinnett, P.E., A.S.A.
Environmental and Permitting	Tetra Tech	Patsy Moran, Ph. D.
Hydrological Studies ,Water Balance, and Surface Water Infrastructure	Tetra Tech	Dwaine Edington, Ph.D. Aurora Bouchier
Mine Rock Management	Tetra Tech	Patsy Moran, Ph. D. Andrew Schissler, Ph.D., P.E.
Mine Closure Remediation and Reclamation	Tetra Tech	Patsy Moran, Ph. D.
Geotechnical Assessment	Tetra Tech	Andrew Schissler, Ph.D., P.E.
Economic Analysis	Tetra Tech	Richard Jolk, Ph.D., P.E. Cameron Wolf

3.0 RELIANCE ON OTHER EXPERTS

This report was prepared for EUU by the independent consulting firm of Tetra Tech. Other individuals have provided input to this report who technically would not be considered QPs under NI 43-101 guidelines, but who have the necessary qualifications and experience to provide input and opinions incorporated into the Report, include:

- Landy Stinnett, P.E., A.S.A., has been relied on for uranium market analysis and the Monte Carlo risk analysis
- Al Kuestermeyer, QP SME and AusIMM, provided the Uranium pricing used for the financial analysis

In addition, Tetra Tech has relied on Pincock, Allen & Holt (PAH) Consultants for information provided in previous reports, previous geology, previous models, and prior resource estimate. The source of this information is included in the NI 43-101 Technical Report on Preliminary Assessment – Kuriskova Uranium Project dated July 23, 2009.

Tetra Tech also relied on the United States consulting firm SRK Consultants Engineers and Scientists for information from their previous report regarding the Project titled NI 43-101 Technical Report on Resources Kuriskova Uranium Project, Eastern Slovakia dated April 16, 2009.

4.0 PROPERTY DESCRIPTION AND LOCATION

Material relevant to Sections 4.1 through 4.4 is detailed in the Technical Report (Section 2) prepared for Tournigan Energy Ltd. (Tournigan) by SRK Consulting, dated April 16, 2009 and is reproduced below with no material changes. Section 4.5 is has not been altered for this 43-101 update.

4.1 Location

The Slovak Republic is the eastern portion of what was once Czechoslovakia and has been an independent entity since 1993. The republic lies between Poland to the north, Austria and the Czech Republic to the west, and Hungary to the south (Figure 4.1). Ukraine adjoins the Slovak Republic at the far eastern tip of the Republic.

The Kuriskova (formerly known as Jahodna) Uranium Project (herein referred to as the Project) is located approximately 8 km northwest of the boundary of Kosice, a regional industrial and administrative city in east-central Slovakia. The Property lies close to the main paved road No. 547 between Kosice and Spisska Nova Ves and is readily accessible via a network of minor, un-surfaced road, and four-wheel drive trails that traverse the forested area.

4.2 Mineral Title in Slovakia

4.2.1 Concession Title

The official mineral title to the deposit area is called the Kosice I license. The full title of the current exploration license issued to Ludovika Energy (EUU's wholly-owned Slovakia subsidiary) refers to "Cermel-Jahodna - U, Mo, Cu ores," and it was granted on March 21, 2005 by the Geology and Natural Resources Department at the Ministry of the Environment of the Slovak Republic. The Project license area amounts to 31.75 km^2. The initial period of validity of the license is four years, which can be extended or converted to a mining lease. The license was extended for a second four-year term effective early April 2009. The name and code of the region is Kosicky 8, and the name and code of the counties are Kosice I - 802, Kosice II - 803, and Kosice - Okolie (vicinity) - 806.

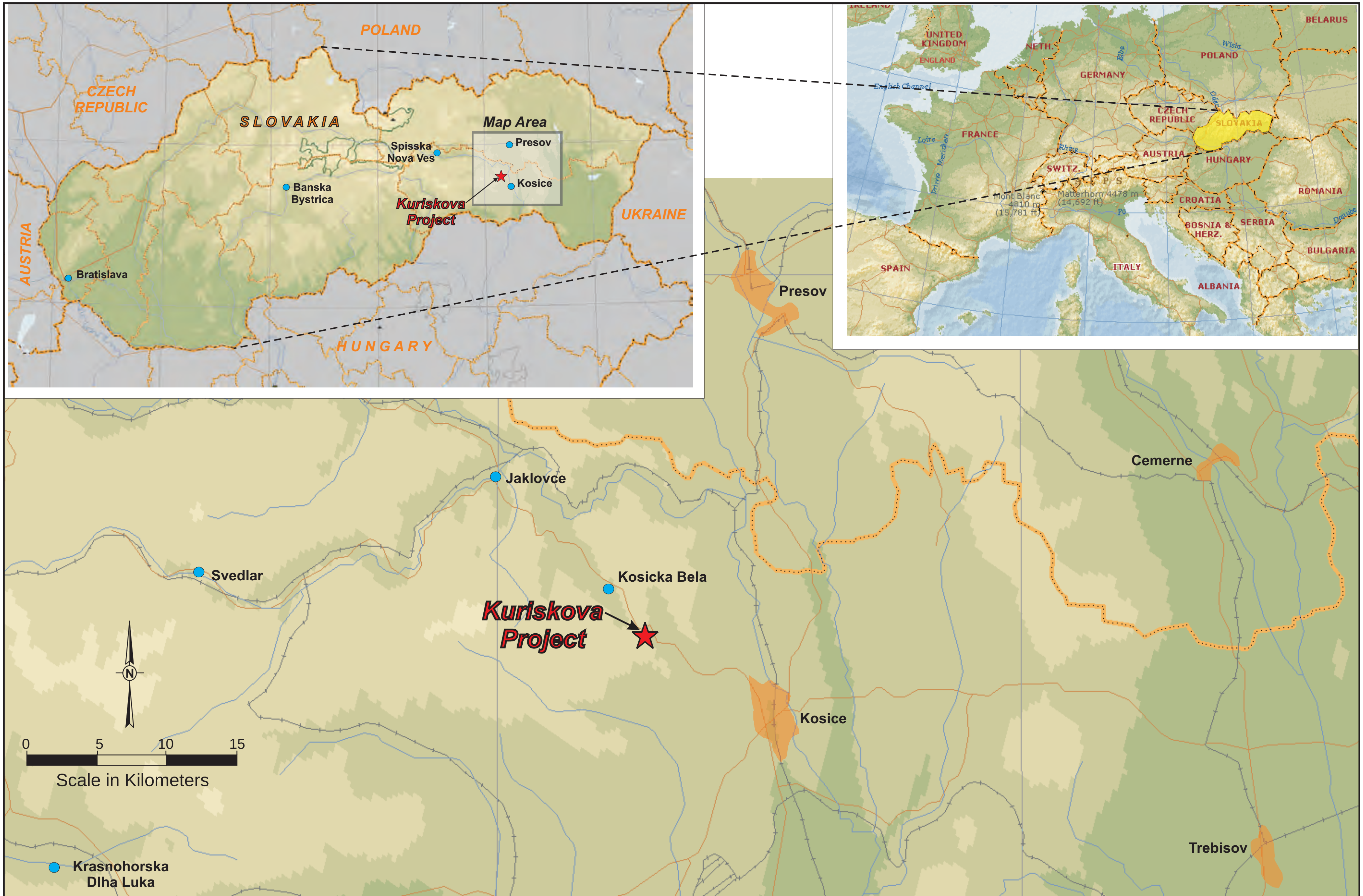
4.2.2 Acquisition and Maintenance of Mineral Rights

The limits to the Kuriskova exploration license are shown in Table 4.1. The Kuriskova deposit is approximately located at 48°45'50"North latitude and 21°09'14" East longitude.

The names and numbers of the cadastral areas are shown in Table 4.2. The costs to hold exploration license are:

- €99.58 per km^2 per year for the first four years;
- €199.16 per km^2 per year for the next four years;
- €331.93 per km^2 per year for the next two years; and
- €663.87 per km^2 per year for next years.

The total cost to a company to maintain a lease, €6,373.12 per year for the four years since exploration license was extended (April 2009), is dispensed by the government at 50 percent to an environmental fund and 50 percent to the towns and villages within the license area, as per the percentage of each village's lands within the license area (see Relative Distribution column in Table 4.2).



Issued by:	Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	***European Uranium Resources Ltd.***	Fig4.1.cdr	**Figure 4.1** **General Location Map of the Kuriskova Uranium Project**
	Project: Kuriskova Uranium Project	Project Number: 114-310990	
	Project Location: Slovak Republic, Europe	Date of Issue: 05/05/2010	

Table 4.1. Universal Transverse Mercator (UTM) Coordinates of Kuriskova License Area

Point No.	Easting	Northing
1	513,557	5,394,268
2	506,681	5,406,045
3	507,533	5,406,828
4	513,328	5,401,033
5	515,060	5,394,233

The "conditions" of the exploration license issued to EUU are enumerated in the Mining Act. The uranium royalty to the Slovak government is at 10 percent of payable revenues, but can be lowered based on criteria presented in the mining Act.

Tetra Tech is not aware of the terms of any royalties, back-in rights, or other agreements and encumbrances to which the Property is subject.

EUU represents that all conditions of the exploration license have been met and the license is in good standing.

4.3 Environmental Liabilities

No environmental liabilities have been identified by Tetra Tech that would materially impede the advancement of the Project to the next engineering study. EUU is responsible for surface disturbances associated with the exploration activities. These activities have been permitted and include financial assurance to cover the costs of reclamation and re-vegetation.

4.4 Permitting

On April 8, 2009, EUU received an extension of their Exploration License for an additional four years. The license encompasses the areas detailed in Table 4.2.

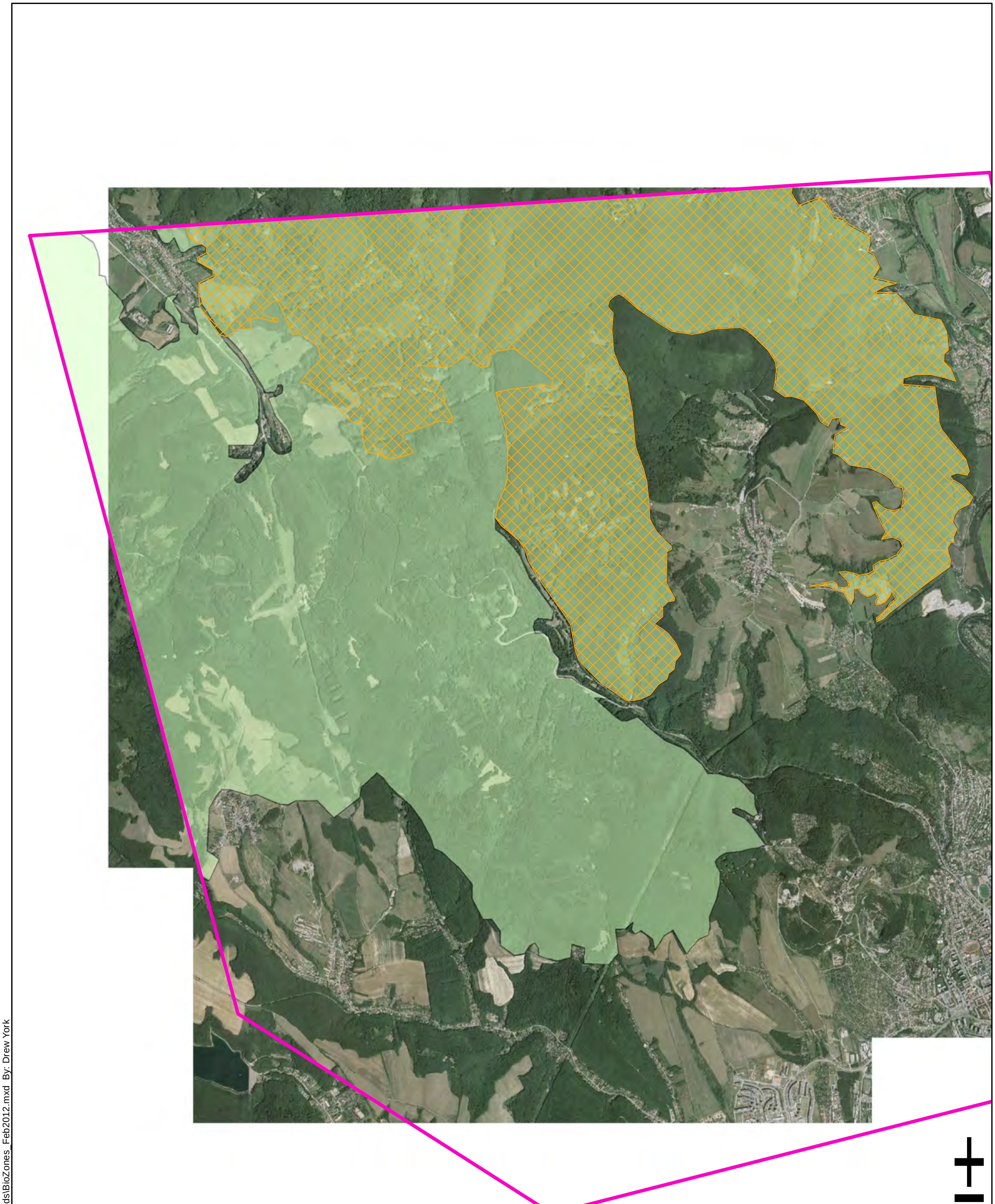
Table 4.2. Exploration License Areas

No.	Village Code	Village Name	Cadastral Area Code	Name of Cadastral Area	Component Fees		
					Relative Distribution %	Villages in €	Environmental Fund in €
1	598151	Kosice - Sever	827207	Cermel	51.59	1643.95	1643.94
2	598216	Kosice - Myslava	827428	Myslava	9.2	293.17	293.16
3	521159	Baska	802123	Baska	7.09	225.93	225.92
4	521574	Kosicka Bela	827606	Kosicka Bela	20.93	666.95	666.94
5	521752	Nizny Klatov	841129	Nizny Klatov	6.41	204.26	204.26
6	522210	Vysny Klatov	871516	Vysny Klatov	4.78	152.32	152.32

The EIA process under the Slovakian EIA Act (Act No. 127/1994 as amended most recently by Act No. 24/2006) will be the primary permitting driver and is anticipated to take 18 to 24 months to complete. A multi-agency regulatory process will be completed to obtain all required permits and approvals necessary to construct, operate and ultimately close the Project. The permitting process in Slovakia is relatively complex and includes participation from the Regional Mining Bureau, Regional Construction Office, the Slovakian environmental agencies, several other government agencies, companies, affected municipalities and the public.

The Project area includes two Natura 2000 ecological protection areas (Figure 4.2). Natura 2000 is a network of areas designated by EU member countries with the objective of protecting birds, biotopes, and other animal species and their habitat. To limit potential adverse effects to the overlapping Natura 2000 site, the Project includes minimization of surface disturbances. To this end, the Kuriskova deposit is accessed by means of a decline to the underground mine and process plant.

The presence of Natura 2000 areas does not preclude development activities. For example, active timbering and logging are conducted within the Natura 2000 area by the Kosice Timber Company. Development of the Kuriskova deposit with underground and minimal surface facilities is unlikely to result in impacts that would adversely affect the integrity of the Natura 2000 areas.



LEGEND:

ENVIRONMENTAL STUDY AREA

BIOTOPE ZONE

BIRD ZONE

SCALE IN METERS

0 1,000 2,000

SOURCES: (1)
ORTHOPHOTOMAP© GEODIS
SLOVAKIA, S.R.O., WWW.GEODIS.SK

NATURA 2000 BOUNDARY

A
REVISION

TETRA TECH
350 Indiana Street, Suite 500
Golden, Colorado 80401
(303) 217-5700 (303) 217-5705 fax

Project:	EUROPEAN URANIUM RESOURCES LTD.
Location:	TOURNIGAN, KURISKOVA
Project No.:	310990
Date:	FEB 2012

FIGURE 4.2

FEB 2012 T:\GIS\Tournigan\mxds\BioZones_Feb2012.mxd By: Drew York

5.0 ACCESSIBILITY, CLIMATE, ETC.

Material relevant to this section is detailed in the Technical Report (Section 3.0) prepared for Tournigan by SRK Consulting, dated April 16, 2009 and is reproduced below with no material changes.

5.1 Access

The Kuriskova deposit is located 300 m south of the main road No. 547 linking the city of Kosice and the town of Spisska Nova Ves; Kuriskova is within approximately 15 minutes access time from Kosice.

The hilly topography in the immediate area of the deposit varies from 500 m to 650 m above mean sea level (amsl), with total relief for the license area of several hundred meters. The topography is structurally controlled, with ridges and deeply incised canyons trending northwest-southeast. The Project area is approximately 8 km west-northwest of the city of Kosice and is easily accessed by a two-lane paved highway that passes through the Project area within a few hundred meters of the primary area of drilling.

5.2 Climate and Length of Operating Season

The climate of the Kuriskova area is a typical Central European climatic regime that is moderately cool and temperate, hosting cool summers and cold, cloudy, humid winters moderated by elevation. Most of the precipitation peaks in June and July. Winter snow cover usually lasts for three months. The Kosice region averages 612 mm of precipitation annually, with more than 30 mm precipitation of snow in January. Low temperatures average -3.9°C in January and highs reach 19.2°C in July. The climate is suitable for year-round mining operations. It is possible to drill year-round; however, drilling is typically curtailed from February through June due to the combination of muddy access conditions and Natura 2000 restrictions on surface activities that could infringe on bird fledgling areas.

5.3 Vegetation

The Project area is in the mountain eco-region zone of mature mixed woodland, dominated by two major species of deciduous vegetation: European beech (Fagussylvatica) and silver fir (Abiesalba) mixed with some conifers, chiefly Norway spruce (Pccea abies). Agriculture is restricted to the valleys and foothill areas and does not play an important economic role.

5.4 Local Resources and Infrastructure

5.4.1 Access Road and Transportation

The Slovak Republic is well served by a national transportation road and railroad network that connects Kosice with the major cities of Central and Eastern Europe. Major rail access is located in Kosice. Kosice hosts an international airport with connections to most of the major Central-European air transportation hubs.

The Jahodna ski resort is 1.8 km to the northwest of the deposit in the Volovec Hills and is a popular seasonal resort. Current or planned activities at the Project have not and are not expected to come close to or affect the Jahodna ski resort operations.

5.4.2 Power Supply

The Kosice region is served by the national electric grid. Slovakia produces 50 percent of its power needs from nuclear plants, and the reliability of the power supply is very good. The power system is operated by Slovakia's transmission system, Východoslovenská energetika (VSE) a.s. VSE is currently working towards Encrypt for Transmission Only (EFTO) certification within the European community.

Mining operations will require construction of a short spur transmission line from the main line near Kosice.

5.4.3 Water Supply

Potable water will be supplied by outside vendors. This will be trucked to the surface facility daily. Mine and process water will be derived from groundwater collected during underground operations.

5.4.4 Transportation Facilities

Slovakia is a land-locked nation. The nearest rail transport facilities from the Project area are in Kosice about 8 km, Margecany about 20 km, and Spisska Nova Ves about 60 km. From these locations, railroad distances to the nearest port cities are approximately 650 km northwest to Gdansk in Northern Poland on the Baltic Sea or approximately 900 km southwest to Thessaloniki, Greece. Additionally, the Slovakia road system provides excellent transportation connections to other European countries.

5.4.5 Buildings and Ancillary Facilities

The Project is in forested woodlands with no permanent building facilities.

5.4.6 Camp Site

The Project does not host a camp site, nor is one required. All drilling contractors and EUU staff are housed in either Kosice or Spisska Nova Ves, which can readily accommodate a potential mine work force.

5.4.7 Waste and Tailings Storage Areas

The Project is an exploration program. There are no mine workings or tailings storage area in the license area. Any studies to define such areas will be examined in subsequent engineering studies.

5.4.8 Manpower

The Slovak Republic and the neighboring countries have a history of exploration and mining and would be the source for experienced mining personnel. There are no uranium mines currently in production in Slovakia. A skilled labor force is available in Kosice, where a large steel mill facility is in operation. Kosice or nearby villages will easily accommodate a workforce of several hundred miners and families.

5.5 Physiography

The Kuriskova deposit is sited in undulating and hilly terrain, with ridges trending northwest-southeast. The ridges are surrounded by steeply incised streams with courses that parallel the ridges. Local relief is approximately 150 to 285 m in the area of drilling. Potential mine portal and decline access would likely be from lower elevations and on the perimeters of the Natura 2000 boundaries.

A small stream of intermittent flow drains northwesterly along the valley traversing the Kuriskova deposit, flowing into the Cermel Valley, which lies along the northeast side of the range. Another larger river (the Vrbica) is located approximately 1 km to the west, bounding the hills on the west side. The Vrbica and Cermel Rivers are tributaries of the Hornad River, which flows southwards past Kosice and ultimately into the Danube River.

5.6 Surface Rights

Surface ownership of lands at Kuriskova are held by the city of Kosice, under the administration of the Kosice Forest Autonomy. Conditions for drilling are defined in the Exploration License.

6.0 HISTORY

Material relevant to this section is detailed in the Section 4.0 of the Technical Report prepared for Tournigan by SRK Consulting, dated April 16, 2009 and is reproduced below with no material changes.

6.1 Ownership

During the years of Communist rule (1948 to1990), all exploration and mining ventures in Czechoslovakia were conducted by the state-controlled quasi-subsidiary companies of Uranovy Prieskum (Uranpres) and the Ceskoslovensky Uranovy Prumysl (CSUP). The Kuriskova uranium deposit was discovered in 1985, by CSUP and was drilled by Uranpres from 1985 to 1990. Following the break-up of the Communist state, the peaceful separation of the Czech and Slovak Republics in 1992, and the return to a free-market economic system, there has been minimum work undertaken on the Kuriskova deposit during the period of 1990 to 2005. EUU acquired the exclusive four-year lease on the Property in 2005.

6.2 Past Exploration and Development

The CSUP group discovered the Kuriskova uranium deposit in 1985. The deposit is virtually a blind target, with only rare outcrops exposed through the several meters of soil cover and arboreal growth. The exploration groups have flown a series of airborne radiometric surveys over the region, and identified a number of surface radiometric anomalies. Follow-up ground radiometric surveys were conducted followed by surface geological mapping and trenching. Weak uranium mineralization was discovered within Permian andesitic rocks of what was later determined to be the distal periphery of the mineralization. The soil cover was too thick for conventional trenching and pitting for geologic mapping and hand-held scintillometer follow-up. A systematic diamond drilling program was instituted by Uranpres to investigate the ground radiometric anomalies.

During the next five years, 53 diamond drill holes were drilled on the Property totaling 17,000 m. The depth of the target necessitated drill holes to 1,000 m in depth. The thin-walled drill pipe and pre-wireline drilling technology coupled with poor ground conditions resulted in continued drill-path deflection and poor recovery (overall average of 50 percent). Downhole radiometric logging was successfully used on all drill holes. The same system developed by CSUP was used for Kuriskova for correlation coefficients and factors derived from other uranium exploration projects in the region (Novoveska Huta) to convert the radiometric readings into equivalent uranium assay data (eU_3O_8). The implied continuity of mineralization was impacted by the poor core recovery.

The drilling program was terminated in 1990, and the last investigation of the Property ended in 1996 as state funding for exploration programs ceased.

6.3 Historic Mineral Resource and Reserve Estimates

Mr. Jozef Daniel, a geologist in the former Czechoslovakian uranium industry, undertook the first resource estimate of the Kuriskova uranium deposit in 1996. The resource estimation was constrained by Czechoslovakian state mining directives first issued in 1987 and revised in 1992. The estimation utilized a block model method using two different cutoff grades of 0.015 percent and 0.030 percent uranium (U). The resource estimations were limited to vein mineralization in the brecciated contact zone, while weaker stringer mineralization in the Hanging Wall Zone was assigned to the lower-confidence "prognostic" category. In 2005, Mr. Daniel updated the grade and tonnage calculations for Kremnica Gold Company, a precursor company to Ludovika Energy. A summary of the three historic resource iterations is not provided here as the iterations are not CIM-compliant resource classifications, have not been reviewed by a Qualified Person, cannot be reconciled with CIM classifications, and are not being used by EUU as current resources.

Mr. Daniel also made resource calculations for molybdenum, but the poor core recovery renders the estimate of little value. The molybdenum assays suggested uranium and molybdenum assays are not correlative and that molybdenum values showed an apparent increase in the distal margins of the deposit and into the hanging wall. Therefore, the molybdenum resource estimate produced by Mr. Daniel only serves to indicate molybdenum as a possible by-product or co-product to uranium mineralization. These early resource estimates are not CIM-compliant and are only presented as part of the historical recounting of the Property.

A.C.A. Howe made the third through eighth series of resource estimates during 2005 to 2007 for EUU (White et al., 2006, A, B; White and Pelham, 2006, 2007; White, 2007).

In their 2005 study, A.C.A. Howe utilized 13 of the original 53 Uranpres historical diamond drill holes and produced a non-CIM compliant resource estimate.

In the 2006 study, A.C.A. Howe used 13 of the historical drill holes, for which mineralization could be verified, and the first three of the new EUU diamond drill holes (White and Pelham, 2006; White et al., 2006). The 2006 study utilized a Micromine software-generated polygonal wireframe resource estimate (PWRE), a specific gravity (SG) of 2.72, and a cutoff of 0.03 percent uranium. The Inferred Resource estimate confirmed the nature and magnitude of Mr. Daniel's 2005 original resource estimate for the Kuriskova uranium deposit.

In their June 2007 study, A.C.A. Howe used 13 of the historical drill holes and 18 of the EUU diamond drill holes for their resource estimate to further define mineralized domains and sub-domains for modeling (White and Pelham, 2007). They defined the main strata-bound fractured contact zone, subdivided the hanging wall andesite into five mineralization sub-domains, and defined the sub-horizontal (#614) thrust fault and the transverse J-8 fault as separate mineralized domains (Table 6.1).

For the December 2007 report, A.C.A. Howe utilized the newly sub-divided domains to estimate a CIM-compliant Inferred Resource estimate (White, 2007). Using 20 of the completed EUU drill holes, they re-defined and removed the mineralized transverse faults from the model. In addition, they defined limited molybdenum and copper grades for the Main Zone using the recent EUU drillhole assays. However, since the data represent widespread sampling, A.C.A. Howe did not estimate molybdenum and copper resources for Kuriskova. Historically reported, CIM compliant resources estimated by A.C.A. Howe are presented in Table 6.1 and Table 6.2. The details of the resource modeling by A.C.A. Howe were not reviewed by SRK and are not presented here as current resource estimates. They are presented here for the historical record, as resource estimates were prepared by Qualified Persons within A.C.A. Howe and presented in NI 43-101 public documents.

In July 2008, SRK completed a resource estimate that was compliant with NI 43-101 and CIM standards, and that resource is presented in Section 15.0 of the 2009 SRK report. At EUU's request, and for comparison with the historical resources stated below in Table 6.1 and Table 6.2, the SRK resource estimate from 2008 for Kuriskova is presented here in Table 6.3 and Table 6.4, using the same parameters described in Section 15.0 of the 2009 SRK report, but at the same historical cutoff grade of 0.03 percent uranium.

Table 6.5 details the PAH resource estimate used in the July 23, 2009 Preliminary Assessment report completed for EUU and is located on the System for Electronic Document Analysis and Retrieval (SEDAR) website.

Table 6.1. Historical Inferred Resource Estimate by Mineralized Domains (A.C.A. Howe)

Report Cutoff	Domain	Category	Density (t/m^3)	Tonnes (Mt)	% U	% U_3O_8	% Mo	% Cu*	Mlbs U_3O_8
>0.03% U	Main Zone North	Inferred	2.63	2.170	0.487	0.575	0.115	0.073	27.50
>0.03% U	Main Zone South	Inferred	2.63	1.165	0.113	0.133	0.018	0.022	3.42
>0.03% U	HW Andesite	Inferred	2.66	0.782	0.128	0.151	-	-	2.60
>0.03% U	HW Andesite 1B	Inferred	2.66	0.006	0.090	0.107	-	-	0.01
>0.03% U	HW Andesite 2	Inferred	2.66	0.515	0.093	0.110	-	-	1.25
>0.03% U	HW Andesite 3	Inferred	2.66	0.027	0.068	0.080	-	-	0.05
>0.03% U	HW Andesite 4	Inferred	2.66	0.191	0.051	0.060	-	-	0.25
>0.03% U	HW Andesite 5A	Inferred	2.66	0.051	0.283	0.334	-	-	0.38
>0.03% U	HW Andesite 5B	Inferred	2.66	0.022	0.221	0.261	-	-	0.13
>0.03% U	HW Andesite 5C	Inferred	2.66	0.074	0.089	0.105	-	-	0.17
>0.03% U	Fault 614	Inferred	2.66	0.097	0.212	0.250	-	-	0.53
>0.03% U	All	Inferred	36.29						

*Cu = copper

Table 6.2. Historical Inferred Resource Estimate (A.C.A. Howe)

Study	Year	Description of Study	Tonnes	Grade (% U*)	Contained (lbs U*)
1	2006	Micromine PWRE 0.03% Cutoff	1,256,000	0.56	15,500,000
2	2007	Micromine PWRE 0.03% Cutoff	2,170,000	0.49	27,500,000

*Rounding by SRK

Table 6.3. SRK Historical In-Situ Resource at 0.03 Percent Uranium Cutoff

Classification	Cutoff	Model Zone	% U	Tonnes (K)	% U_3O_8	U_3O_8 lbs (K)
Inferred	0.03% U	All	0.209	5,765	0.247	31,337
Indicated	0.03% U	All	0.35	727	0.413	6,614

Note: See Section 15.0 (SRK Report) for current resources stated at a 0.05 percent uranium cutoff grade.

Table 6.4. SRK Historical In-Situ Resource at 0.03 Percent Uranium Cutoff

Classification by Area	Sub-Zone	Model Zone	% U	Tonnes (K)	% U_3O_8	U_3O_8 lbs (K)
Inferred						
Main	Zone 1N	1	0.394	1,918	0.464	19,620
	Up Main Zone	1.2	0.106	12	0.125	33
	Zone 1S	1.1	0.179	1,598	0.211	7,433
H.W. Andesite	Zone 2N	2	0.056	409	0.066	595
	Zone 3N	3	0.109	316	0.128	891
	Zone 4	4	0.104	265	0.123	719
	Zone 2S	2.1	0.052	629	0.061	846
	Zone 3S	3.1	0.075	617	0.088	1,200
Main Zone Total Inferred		1 + 1.1 + 1.2	0.295	3,528	0.348	27,087
H.W. Andesite Total Inferred			0.073	2,238	0.086	4,251
Total Inferred			0.209	5,765	0.247	31,337
Indicated						
Main	Zone 1N	1	0.366	633	0.432	6,033
	Up Main Zone	1.2	0.161	40	0.19	165
	Zone 1S	1.1	0.293	55	0.346	416
Main Zone Total Indicated		1 + 1.1 + 1.2	0.350	727	0.413	6,614
H.W. Andesite Total Indicated			0.000	0	0.000	0
Total Indicated			0.350	727	0.413	6,614

Note: See Section 17.0 for current resources stated at a 0.05 percent uranium cutoff grade.

Table 6.5. PAH Historical In-Situ Resource at 0.03 Percent Uranium Cutoff

Kuriskova In Situ Uranium Resources @ 0.05% U Cutoff (Feb. 2009)							Mo @ 0.05% U cutoff		
Classification by Area	Sub-Zone	Model Zone	% U	Tonnes (K)	% U_3O_8	U_3O_8 lbs (K)	% Mo	Tonne s (K)*	Mo lbs (k)
Inferred									
Main	Zone 1N	1	0.306	1,025	0.361	8,154	0.051	2,115	2,387
	Up Main Zone	1.2	0.112	11	0.132	32	0.030	46	30
	Zone 1S	1.1	0.162	1,543	0.191	6,499	0.014	1,586	496
H.W. Andesite	Zone 2N	2	0.067	235	0.079	406	0.005	230	28
	Zone 3N	3	0.127	250	0.149	824	0.010	250	56
	Zone 4	4	0.125	200	0.148	652	0.022	200	97
	Zone 2S	2.1	0.087	181	0.103	410	0.003	181	11
	Zone 3S	3.1	0.106	336	0.125	924	0.024	288	155
Main Zone Total Inferred		1+1.1+1.2	0.219	2,579	0.258	14,685	0.035	3,747	2,914
H.W. Andesite Total Inferred			0.103	1,201	0.121	3,216	0.014	1,149	347
Total Inferred			0.182	3780	0.215	17,901	0.030	4,897	3,261
Indicated									
Main	Zone 1N	1	0.495	1,090	0.584	14,027	—	—	—
	Up Main Zone	1.2	0.178	34	0.21	160	—	—	—
	Zone 1S	1.1	0.269	67.13	0.317	469	—	—	—
Main Zone Total Indicated		1+1.1+1.2	0.473	1,191.13	0.558	14,654	---	---	---
H.W. Andesite Total Indicated			0	0	0	0	0	0	0
Total Indicated			0.473	1,191.13	0.558	14,654	0	0	0

6.4 Historic Production

The Kuriskova uranium deposit is an exploration target. There has been no underground development work or production on the Property, only construction of temporary surface drill roads and drill sites.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Geological Setting

Material relevant to this section is detailed in the Technical Report prepared for Tournigan by SRK Consulting, dated April 16, 2009. Section 7.1 is summarized from Section 5.1 in the SRK report (2009). Sections 7.1.1 and 7.1.2 below are reproduced with no material changes from Sections 5.2 and 5.3 in the SRK report (2009).

7.1.1 Regional Geology

The Kuriskova uranium deposit is located in the Kojsovska Hola region of the Volovec Hills, which are part of the Western Carpathian Mountain Range. After the Alps, the Carpathians are the second-most prominent mountain chain in Europe, extending from Slovakia to the Ukraine and Romania in the east and to the Danube River between Romania and Serbia to the south. The Danube River separates the Alps and Carpathians near Bratislava, Slovakia.

The Carpathians form a semi-circular arc across Central and Eastern Europe, extending 1,500 km in length while ranging 12 to 500 km in width.

The Carpathians are the result of two great mountain-building events: the Variscan Orogeny of Late Paleozoic age (380 to 280 Mya) and the subsequent Alpine Orogeny of Paleogene age (200 to 150 Mya). A Lower Jurassic age of uranium mineralization at Kuriskova has been tentatively determined at 200 to 150 Mya (Tournigan, 2007).

The Variscan Orogeny resulted from the collision of the Laurasian and Gonwanda continents whose fusion formed the Pangea supercontinent. In North America, the deformation and magmatism associated with this event is locally known as the Alleghenian or Acadian Orogeny.

The Variscan orogeny was accompanied by extensive metamorphism and syn- to late-orogenic granitoid intrusions (Stussi, 1989; Dill, 1994). Variscan folding deformed a series of magmatic arc-related basins in the region; shallow marine sedimentation into these basins continued into the Permian under generally arid conditions.

Subduction-related high-potassium calc-alkalic rhyolitic volcanism with associated small S-type granitic intrusions was emplaced in the Gemericum tectonic unit in the Upper Permian from 280 to 250 Ma (Ivan et al., 2002). A later suite of post-Variscan/Early Alpine (extensional) S-type granites of Triassic age (250 to 235 Ma) are also recognized (Uher et al., 2002). The eastern continuum of the Variscan crustal deformation belt into Slovakia is masked by Alpine deformation.

The Alpine Orogeny resulted from continental collision of the northern Eurasian Plate with the Indian and African plates to the south. The Alpine cycle commenced in the late Permian with rifting and bimodal tholeiitic/rhyolitic volcanism, intrusion of anorogenic (A-type) magmatic rocks, and regional metamorphism.

Both the Variscan and Alpine orogenies were multi-cyclic, collisional events that generated regional metamorphism, calc- and sub-alkaline magmatism and unique I-, S-, and A-type granitic plutonic, extrusive lithologies, and associated mineralization.

In Slovakia, many base-metal deposits, including the extensive siderite-sulfide deposits of the Gemericum Domain, are interpreted to be of Variscan age having formed from circulating metamorphic fluids (Ebner et al., 1999; Radvanec et al., 2004). Late Variscan S-type granites intruded the Slovakian Gemeric units; these granites have been associated with small tin-tungsten, greisen-type mineralization, and polymetallic vein deposits.

The known Variscan uranium deposits in Slovakia share a common age, type of mineralization, and magmatic association with other base-metal, tin-tungsten, and uranium deposits throughout Western and Central Europe.

Mineralization types associated with Slovakian Tertiary volcanism are: epithermal veins [e.g., Kremnica Au-Ag-(Pb-Zn-Sb-Hg)[1]], stockwork and disseminated mineralization associated with stockwork-type intrusions, and porphyry-type or contact metasomatic (skarn) deposits related to granodiorite-diorite porphyries intruding Triassic carbonate rocks.

The formation of EUU's Novoveska Huta Uranium-Molybdenum-Copper Project (located approximately 60 km northwest of Kuriskova) has been attributed to the circulation of brines and meteoric fluids in an extensional environment alpine cycle.

7.1.2 Local Geology

7.1.2.1 Local Lithology

Soil and arboreal cover obscure most of the Kuriskova deposit. The lithological and structural setting is based on observations from rare outcrops, drill hole geology, cross sections, and projections of lithologic and mineralized units to the surface from drill hole data.

The mountain range at Kuriskova is composed of mesozonal metamorphic rocks known as the Gemericum tectonic unit of the Carpathian belt. Along the northern periphery of this tectonic unit, numerous uranium occurrences are contained in a nearly continuous zone of Permian rocks, 0.5 to 6.0 km wide and 80 km long. The Permian rocks are locally covered by Mesozoic carbonates (Figure 7.1).

The Permian rock units were deformed during the Alpine Orogeny into the system of folds and faults that produced the current complex structure of the deposit area as shown in Figure 7.2.

The Permian rock units, the Krompachy group, consist of the Knolske, Petrovohorske, and Novoveska strata and are composed of basal agglomerates (Muran) overlain by violet sandstones and slates (Markusovce). The overlying Petrova Hora group of strata forms the central part of the Permian sequence with a varied representation of volcanic, volcaniclastic, and sedimentary rocks. There is a component of basic, intermediate, and acid volcanic rocks, including the Huta volcanic complex (which hosts the Kuriskova deposit), and the Grun volcanic-sedimentary complex. The upper Permian Novoveske group consists of agglomerates, sandstones, slates, and evaporites. The earliest of the Novoveske rocks are of continental origin (fanglomerate, fluvial, limnic) and the latest are of lagoonal origin (evaporites). The total thickness of the Permian sequence is 500 to 2,500 m.

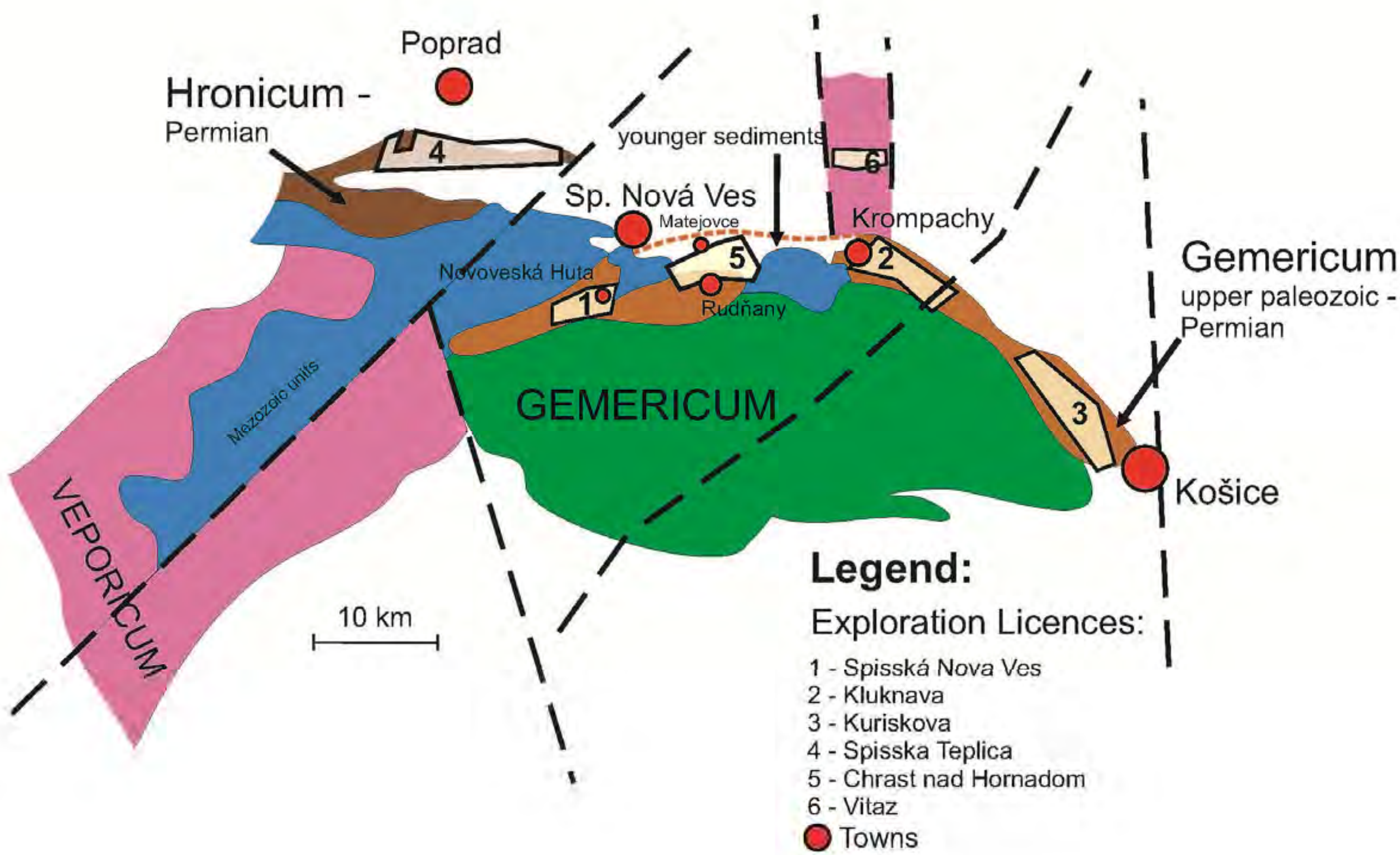
The rock units at Kuriskova are folded, strike northwesterly, dip variably to the southwest, and are strongly metamorphosed and fractured with prominent slaty cleavage. The deposit is transected by a series of post-mineral high-angle faults and several low-angle faults. The rocks range in color from light green through dark violet, dark grey, and black. Altered sections can be pinkish to green.

1 Au = gold; Ag =silver; Pb = lead; Zn = zinc; Sb = antimony; Hg = mercury

The footwall to the Kuriskova deposit is the Knolske Formation, which is a sequence of slates and quartzites of variable competence that are hundreds of meters in thickness. These metasediments are in apparent tectonnicized contact with the structurally overlying volcaniclastic sequence of the Petrovorske Formation. These intermediate fine-grained porphyritic volcanic rocks have an overall thickness of several hundred meters. The Main Zone of mineralization is hosted within the lowest 2 to 8 m of the fractured meta-tuffs and meta-andesite. The Hutniansky Complex (Huta volcanics), which forms the immediate hanging wall to the deposit, is a meta-andesite with a thickness of 20 to 50 m; it hosts discontinuous stringer-type uranium-molybdenum-copper mineralization and grades upward into a mixed volcaniclastic sequence.

Petrographic reports detailing the volcanic and volcaniclastic lithologies are briefly described in Ferenc and Mato (2006) and are included in the alteration Section 7.1.3.1.

Regional Geology



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TETRA TECH 350 Indiana Street, Suite 500 Golden, CO 80401 (303) 217-5700 (303) 217-5705 fax	***European Uranium Resources Ltd.***	Fig 7.1.jpeg	**Figure 7.1** **Permian Age Geological Units Showing EUU Licenses – Eastern Slovakia**
	Project: Kuriskova Uranium Project	Project Number: 114-310990	
	Project Location: Slovak Republic	Date of Issue: June 2011	

MAJOR TECTONIC UNITS OF THE ALPS, CARPATHIANS AND DINARIDES

S.M. Schmid, D. Bernoulli, B. Fügenschuh, L. Matenco, S. Schefer, R. Schuster, M. Tischler and K. Ustaszewski

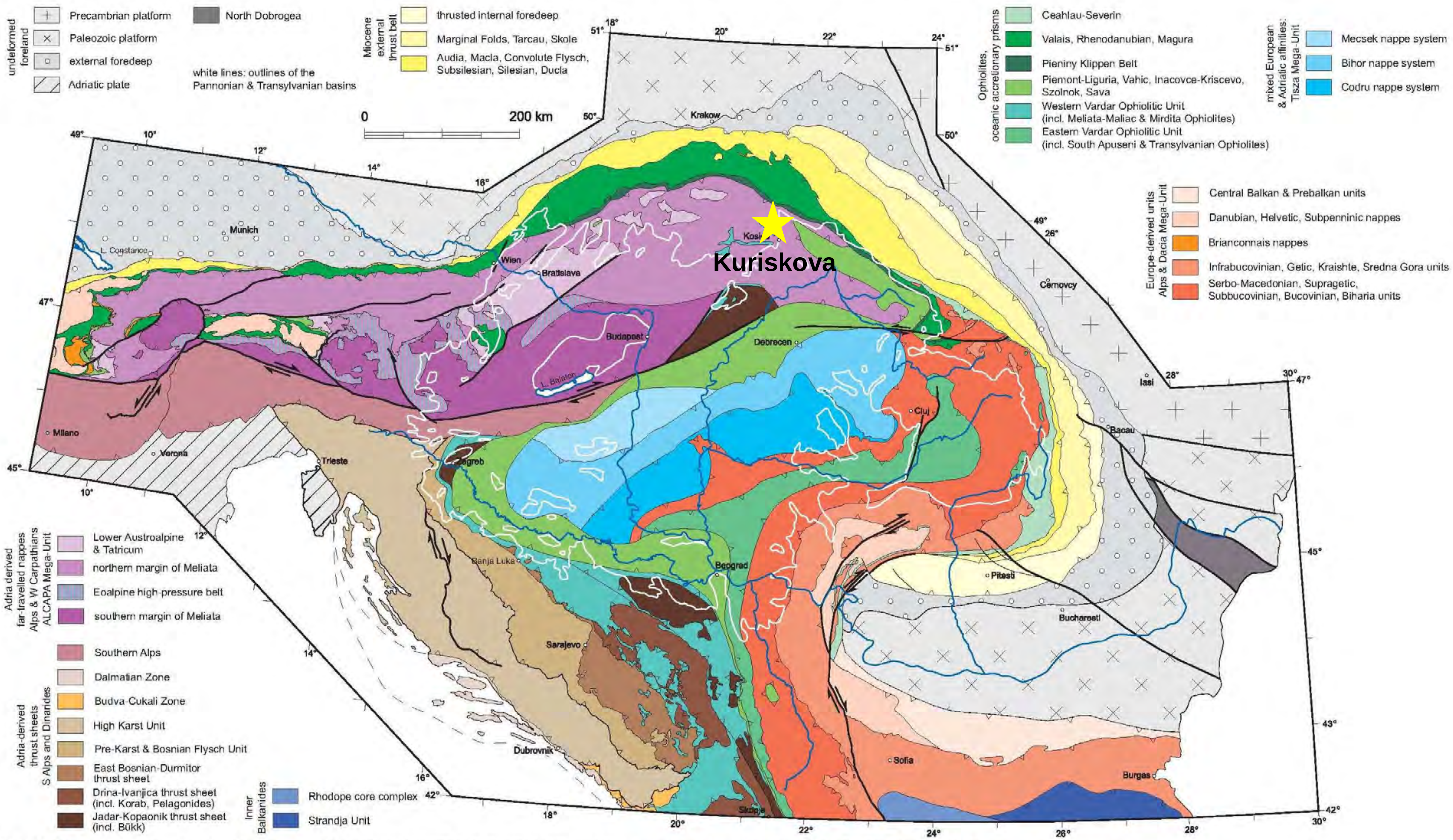

Plate 1: Major tectonic units of the Alps, Carpathians and Dinarides 1:5'000'000. Note that the locations of the cross-sections given in Fig. 3 and Plates 2 & 3 are found in Fig. 6.

S.M. Schmid et al.

Alps-Carpathians-Dinarides

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TETRA TECH 350 Indiana Street, Suite 500 Golden, CO 80401 (303) 217-5700 (303) 217-5705 fax	*European Uranium Resources Ltd.*	Fig 7.2.jpeg	**Figure 7.2** **Regional Geologic Map – Tectonic Map of the Carpathians**
	Project: Kuriskova Uranium Project	Project Number: 114-310990	
	Project Location: Slovak Republic	Date of Issue: June 2011	

7.1.3 Project Geology

Directly in the footwall of the Kuriskova deposit are the Markusovce quartzites and slates that belong to the basal Permian (Knolske group of strata). Typical of metasedimentary rocks are concretions, bedding laminae, laminae of Fe-carbonates, and bedding-parallel sedimentary breccias.

The metasedimentary units are in apparent sheared, bedding plane, fault contact with fine-grained tuffs that are often laminated. The tuffs are green-grey to dark grey in color, have an overall thickness of 1 to 10 m, form the basal member of the Petrova Hora group of strata. This tuff unit is the dominant mineralized lithology in the Kuriskova deposit. The tuffs change quickly into the overlying units, but in many cases the contact is gradational and rapidly changing, passing into touchstone-like (flinty, siliceous) andesite and to dacite and andesite with unclear spatial distributions. Due to regional metamorphism, the units are often referred to as metavolvanics. Uranium mineralization is present (mostly as disseminations) in the basal metatuffs and to a lesser extent in the overlying meta-andesites. The Main Zone of mineralization is dominantly in the metatuffs.

In the metavolcanic units above the Main Zone of mineralization, there is a 40 to 100 m thick unit of dark-green andesite and fine grained tuff that hosts the stockwork uranium and molybdenum mineralization known as the Hanging Wall (or Andesite) Zone. In these units, there are also distinct sedimentary laminations with dispersed pyritization, and in a few isolated cases, evidence of volcanic lapilli and bombs.

In the upper portions of the metavolcanic unit, there is a 3 to 20 m thick layer of violet-colored slates with concretions of sedimentary carbonate that transitions to green slate with pyrite impregnations. The slate represents a lacustrine, sedimentary environment deposited during quiescent times between periods of volcanic activity and is considered by EUU geologists to be the litho-stratigraphic equivalent of the upper intermediate layers of the Novoveska Huta Project mineralization 45 km northwest of the Kuriskova project area. The slate marks the transition from basic to intermediate volcanism to overlying acidic volcanism. This layer of lakebed sediments is thought to be the uppermost limit of the uranium mineralization having acted as an aquitard to hydrothermal fluids (Tournigan, 2008).

Overlying the Permian age metavolcanic and volcaniclastic rocks at Kuriskova are 1 to 5 m of Quaternary fluvial sediments and soil cover. A brief description of the Project geology, below, is provided by White (2007).

The main zone of the Kuriskova deposit occupies dilational zones along the geologic contact between the overlying competent andesitic metavolcanic unit and the underlying metasediments. Here, two styles of mineralization are present; firstly uranium mineralization associated with andesitic tuff/tuffite units at the base of the main andesite unit. The tuffs are phosphorous rich and it appears that phosphorous has preferentially fixed the uranium minerals, resulting in often high-grade zones (1 to 5 percent uranium). Secondly, uranium mineralization hosted directly on the andesite/sediment contact, which is lower grade (0.1 to 0.5° percent uranium) and is regarded as a more tectonised form of the tuff hosted zone described above.

Shearing along this contact has resulted in tectonic disturbance and poor ground conditions. Tectonic disturbances have also resulted in schistose foliation and slaty cleavage (giving poor ground conditions in some softer sedimentary units) and fault offsets, some of which disrupt the main deposit. Uranium mineralization hosted within hanging wall andesites are characterized by their presence as often discrete lenses associated with thin quartz-carbonate veins and hematite. Uranium grades within these zones are variable.

The overall dimensions of the main deposit established to date are some 650 x 550, and about 2.5 meters in average thickness. As mentioned, there are also minor mineralized zones in the hanging wall of the main deposit.

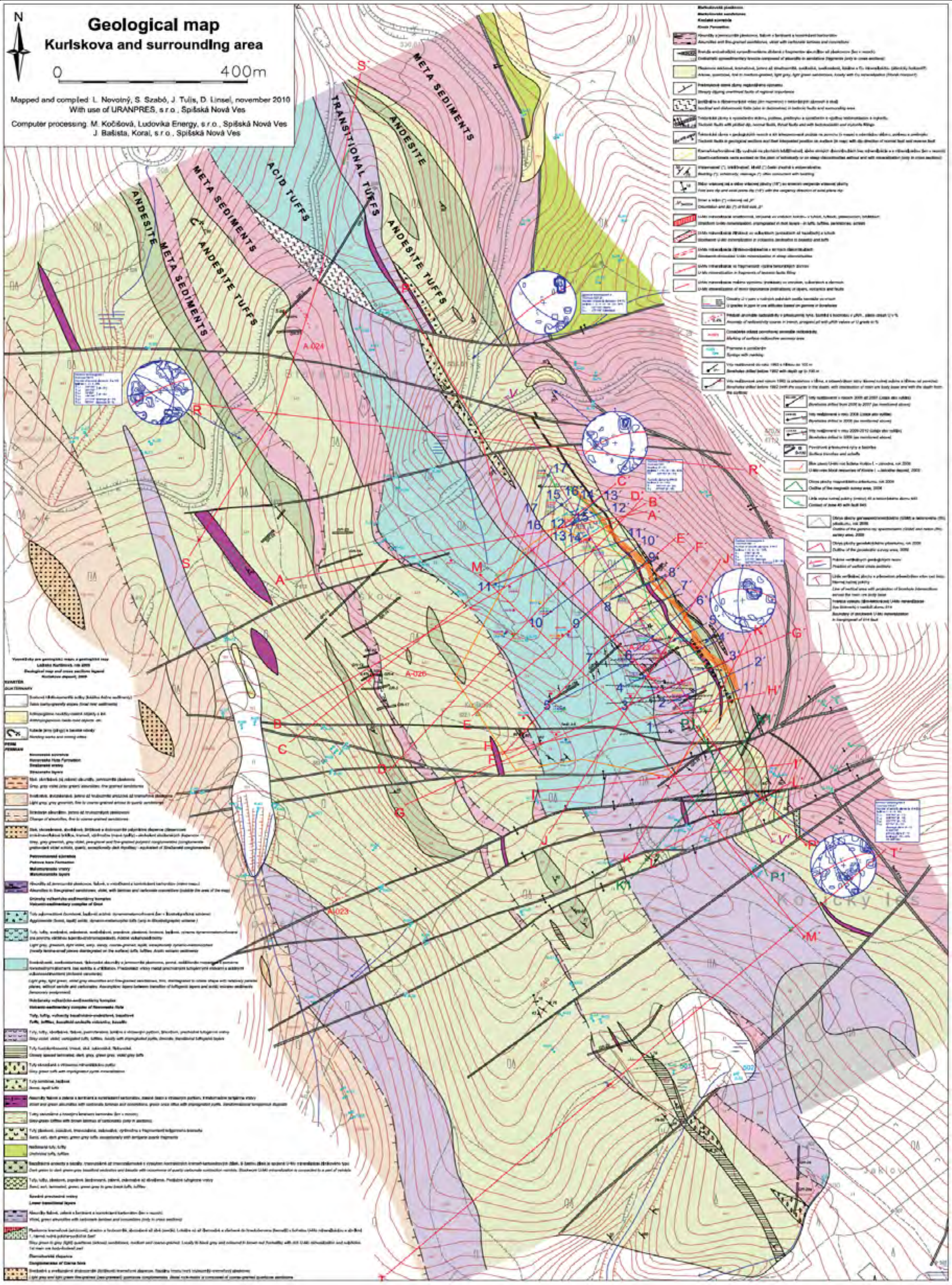
A surface geological map of the Kuriskova project area and cross section (F-F) through the deposit are given on Figure 7.3 and Figure 7.4, respectively. The cross section is looking to the northwest and is indicated on the geological map, Figure 7.3. These figures are used unmodified from Figures 5.3 and 5.4, respectively, in the SRK report (2009).

7.1.3.1 Alteration

Detailed studies of alteration paragenesis and distribution have not yet been attempted or formalized in the limited English translations of available project geology. The fine-grained andesites and tuffaceous andesites are variably sericitized and chloritized with both pervasive and veinlet alteration assemblages.

Phyllic assemblages of quartz-sericite-pyrite are noted, with plagioclase altered to sericite and quartz, and in places are overprinted with propylitic assemblages of chlorite-carbonate-hematite (Ferenc and Mato, 2006). Carbonate alteration (calcite, Fe-dolomite, siderite) is both pervasive and veinlet-controlled. Energy Dispersiove Spectroscopy (EDS) analyses indicate the presence of intermixed clay minerals, such as illite-mica mixture. Hematite tends to be pervasive throughout the matrix of mineralized intervals and as an overall impregnation of mylonitized wallrock. Older quartz-carbonate-sulfide veins are often cataclasized, and the rocks are microfractured passing into horse-tail fractures.

The tuffaceous andesites are finely laminated with ophitic-to-porphyritic textures and with a lepidoblastic to granoblastic groundmass. Feldspars (chiefly oligoclase and albite) are highly altered to quartz, sericite, carbonate, and chlorite assemblages. Chlorite and sericite aggregates form interstitial clusters in the groundmass. With tourmaline, chlorite clusters reach dimensions of 2.5 mm. Carbonates (calcite and siderite) form rhombs to 0.05 to 0.10 mm and aggregate to form veins to 10 mm thickness parallel to foliation with quartz as selvages.



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TETRA TECH 350 Indiana Street, Suite 500 Golden, CO 80401 (303) 217-5700 (303) 217-5705 fax	***European Uranium Resources Ltd.***	Fig 7.3.cdr	**Figure 7.3** **Surface Geological Map of the Kuriskova Uranium Project**
	Project: Kuriskova Uranium Project	Project Number: 114-310990	
	Project Location: Slovak Republic	Date of Issue: June 2011	

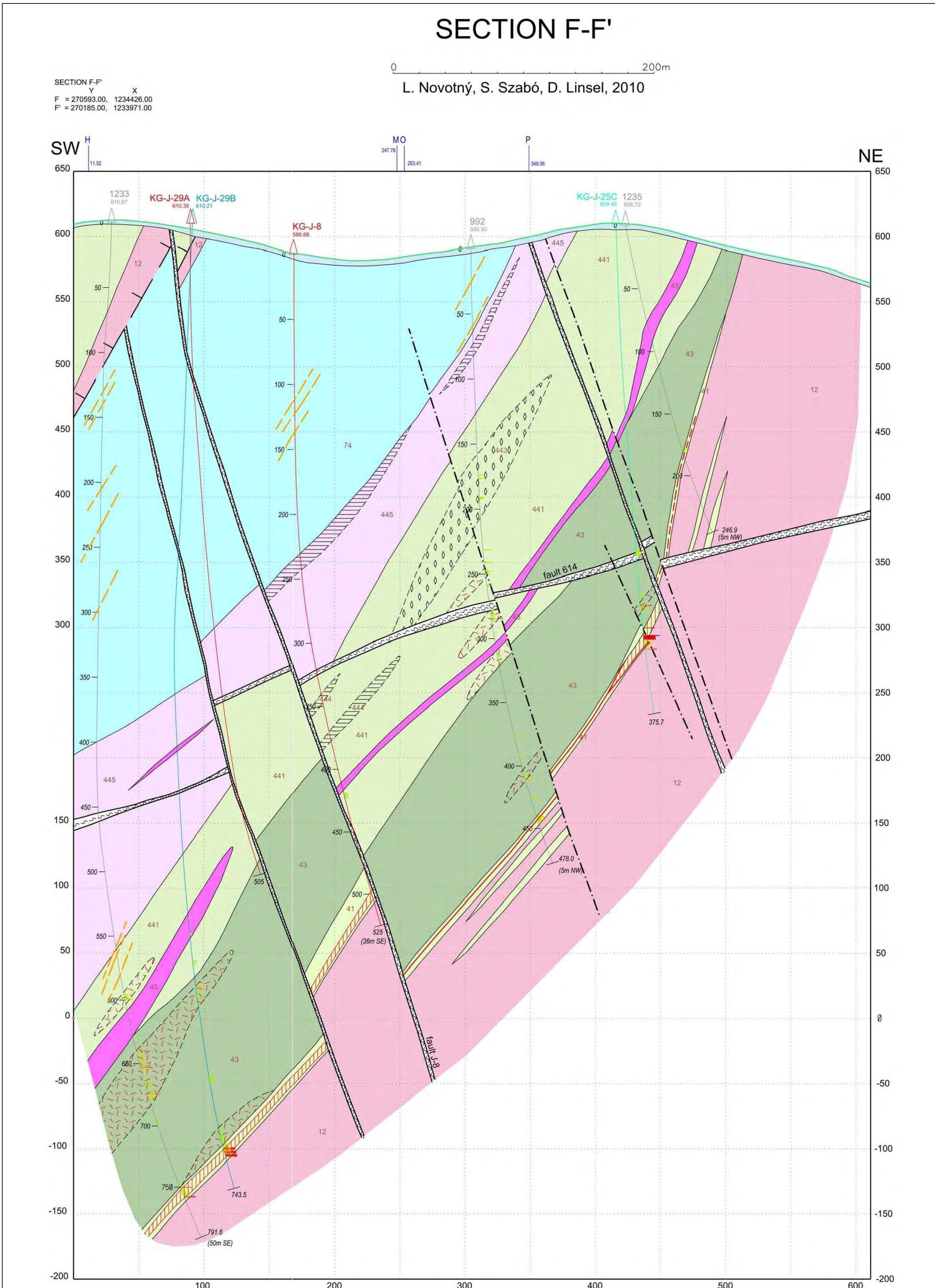

Figure 7.4
Cross Section F-F'

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	Project: Kuriskova Uranium Project	Project Number: 114-310990
	Project Location: Slovak Republic	Date of Issue: June 2011

Quartz, both in veins and groundmass, is coarse grained (to 0.5 mm). In some zones, matrix silicification is pervasive and intense. Pyrite and chalcopyrite are variable and ubiquitous with quartz veining and silicification. Quartz-siderite veins, 3 to 20 mm in thickness are associated with hematite in matrix and impart a characteristic violet hue to tuffs.

Accessory alteration minerals include tourmaline, concentrated both in selvages along carbonate veins and as fine disseminations in wallrock. Apatite forms prismatic to stubby grains to 0.2 mm in length. Zircon is common, and leucoxene less common.

The andesites are porphyritic with a pilotaxitic groundmass; altered andesites have a lepidoblastic texture. Albite and oligoclase phenocrysts are 0.4 to 0.6 mm with smaller feldspar strips to 0.03 to 0.10 mm oriented randomly. The feldspars are altered to sericite, and in matrix adjacent to quartz-carbonate veins, forming zebra-like parallel bands that can aggregate up to 1 m thick. Large pyrite grains are ubiquitous with alteration. Carbonates occur as veins and in nebulous envelopes in adjacent wallrock. Carbonate veins sequences are pronounced in proximity to major fault systems, and in higher concentrations tend to be associated with apatite, with grains to 0.1 mm. Quartz occurs as carbonate vein selvages and as irregularly distributed clusters in wallrock. Quartz crystal growth tends to be oriented perpendicular to the carbonate veins. Chlorite forms large flakes to 0.2 mm, often associated with sericite and carbonates, and in lesser veinlets cutting carbonate veins. Columnar grains of rutile and leucoxene to 0.04 mm and apatite to 0.03 mm are noted.

7.1.3.2 Structure

The Permian rocks in the deposit area trend northwest to southeast and are bounded by parallel, thrust faults. The Rakovec unit (Devonian age) displaces Permian units from southwest to the northeast, over the Cermel Group (Carboniferous age). The zone of Permian rocks are up to 2.5 km wide and are internally segmented by four faults into five tectonic blocks, numbered from northeast to southwest. Kuriskova is located in the second block that is 0.7 to 1.0 km wide.

The steeper bedding inclinations (60 to 70°) close to the surface and shallower inclinations (45°) at depths within an individual block indicate that these blocks represent parts of synclinal structures with the fold axes and layering trending northwest to southeast. They are segmented by parallel normal faults in that same direction. The andesite body in the deposit area and most probably mineralized bodies as well, are likely tectonically cutoff at depths of over 1.0 km below the surface. These longitudinal normal faults are the oldest in the Project area and are of the Alpine Orogeny age (Tournigan, 2008). The structural style is interpreted to be axial planar to regional folds and the precursors to thrust faulting resulting from nappe-like sheet folding (SRK interpretation, 2008).

The youngest fault in the area is a low angle fault known as the 614 Fault. It cuts stratigraphy and mineralization with a normal displacement of 50 to 100 m. The fault is evidenced in core by cataclastic deformation of the rock units and mineralization. The fault has an inclination of 20° to the southwest, and is 2 to 10 m wide.

There are west southwest to east northeast (and east to west) trending structures that have steep (60 to 80°) northerly dips and a normal displacement of 10 to 150 m. The J-8 Fault has an east-west orientation and a dip of 70 to 80° north; the range of displacement is approximately 1 to 20 m. These structures are considered the youngest and are probably Neogene in age (Tournigan, 2008).

A bedding plane parallel foliation is evident in the metatuffs or metasedimentary units and less well developed in the meta-andesite units. The foliations are considered related to regional folding and bedding plane slippage. There is significant bedding-parallel shearing at the metavolcanic-metasedimentary rock contact.

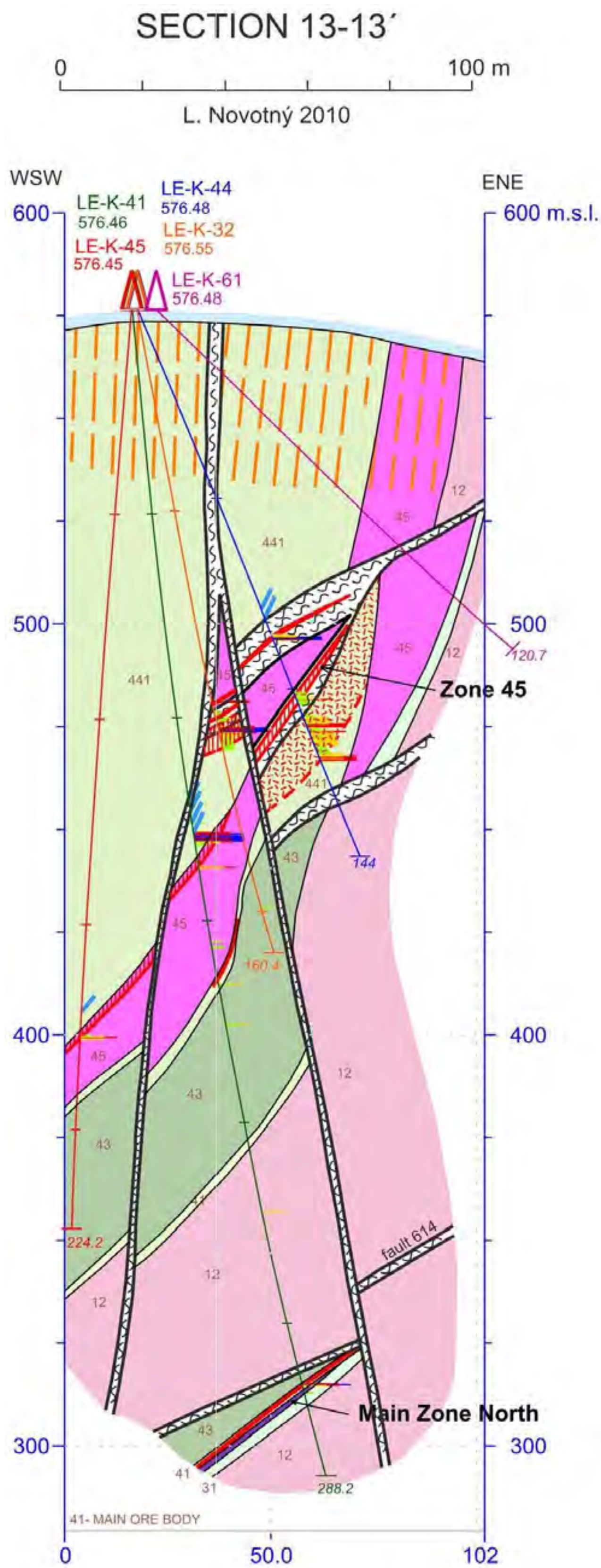
The uranium deposit has a northwest to southeast strike and a steep dip to the southwest. The upper part of the blanket-like mineralized body dips 60°, while the lowest explored parts of the body dip 45°. The deposit is cut by high-angle faults, such as the J-8 Fault, and by the enigmatic, low-angle 614 Fault. Some of the fracture zones accommodate very high grade uranium mineralization (greater than 6 percent uranium); however, the current interpretation is that all faulting is post-mineral and the uranium mineralization in the faults is a result of tectonic remobilization of uranium.

7.1.4 Changes Since July 23, 2009 Pincock, Allen & Holt Report

Since the Preliminary Assessment Report published on July 23, 2009 by PAH, EUU has developed the Zone 45 portion of the deposit into a significant new resource area. Zone 45 is the name given to a zone of high grade uranium (averaging 0.617 percent U_3O_8 in the Indicated portion of the resource) and molybdenum (averaging 0.425 percent molybdenum in the Indicated portion of the resource) mineralization that was discovered during EUU’s 2009 to 2010 drilling campaign. This drilling program was designed to test for extensions of the Main Zone and to test radiometric and radon-in-soil anomalies extending several hundred meters from the edge of the currently defined resource. Mineralization in Zone 45 is 1 to 2.5 m thick and as currently defined extends 220 m along strike and 120 m down dip. The mineralization remains open along strike.

Zone 45 occurs in an upper transitional layer of the Petrova Hora Formation (see litho-stratigraphic table). This unit is a layer of metasediments (siltstones and sandstones) intercalated with tuffaceous metavolcanic units. The cross section in Figure 7.5 illustrates the stratigraphic position of Zone 45 with respect to the Main Zone. Fault gouge is developed locally, indicating some component of tectonic movement during its emplacement. The host unit is locally carbonate rich and commonly contains what appears to be sedimentary pyrite. Uraninite and coffinite are the main uranium minerals.

Zone 45 appears to correlate stratigraphically with the relatively low grade Zone 4 (upper hanging wall) of the Kuriskova deposit, but is significantly higher grade. It is unclear at this time why Zone 45 is so much higher in grade than other zones identified to date in the hanging wall above the Main Zone mineralization. EUU’s immediate priority is to explore for extensions of Zone 45 to the northwest where radon-in-soil anomalies indicate possible extensions of the mineralization.



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TETRA TECH 350 Indiana Street, Suite 500 Golden, CO 80401 (303) 217-5700 (303) 217-5705 fax	Project: Kuriskova Uranium Project	Project Number: 114-310990	**Figure 7.5** **Zone 45 Cross Section 13-13'**
	Project Location: Slovak Republic	Date of Issue: June 2011	

7.2 Mineralization

Material relevant to this section is detailed in the Section 7.0 of the Technical Report prepared for Tournigan by SRK Consulting, dated April 16, 2009 and is reproduced below with no material changes.

7.2.1 Mineralized Zones

The uranium mineralization at the Kuriskova deposit is believed to be the result of mobilization and re-deposition of uranium and base metals in fractures controlled by multi-phase folding and possible thrusting. The various geometries of mineralized units defined to date reflect the rheological properties of the layered and folded volcanic and volcaniclastic lithologies.

The Main Zone uranium mineralization is a stratabound zone of mineralization following the once-horizontal contact between lower sandstones and shales and overlying andesites and volcaniclastics. Mineralization occurs in the fractured andesite tuffs immediately above the contact and extends into the hanging wall andesites for variable distances. Mineralization is fairly continuous, high grade, and varies in thickness from 2 to 8 m. The zone has been explored to date over 750 m of strike length and to 550 m depth. Both transverse and thrust faults have segmented the body into blocks with displacements of up to tens of meters. Mineralization along zones cut by thrust faults are enriched by later remobilization. In the hanging wall andesites, the uranium mineralization occurs in the form of stockwork veins and thin stringers that form irregular clusters. Stringers range from several millimeters to 10 to 15 centimeters (cm) wide. Grade tends to increase with increasing proximity to major faults and fracture zones (Ferenc and Mato, 2006).

The second mineralized zone is stockwork uranium mineralization that occurs in the approximate center of the hanging wall andesite unit, approximately 10 to 50 m stratigraphicly above the tabular Main Zone. The thickness of the zone is variable from 1 to 10 m (maximum of 20 m) that is roughly concordant with lithologic layering. The zone appears to occur in the rheological transition from competent andesite over schistose tuffaceous volcaniclastics and sediments. Faults segment the stratabound zone into blocks. The mineralization is lensoidal with thicknesses to 4.5 m, and generally hosts lower grade mineralization in contrast to Main Zone mineralization. The uranium mineralization occurs in irregular quartz-carbonate stringers with apertures of 1 to 5 mm (to 5 cm maximum). From a regional exploration perspective, the stockwork mineralization offers the potential of significant tonnage expansion, albeit of lower grade mineralization.

The third recognizable zone of uranium-molybdenum mineralization occurs within the tuffs and tuffaceous rocks overlying the andesite and volcaniclastic units. Mineralization is disseminated, very low grade and discontinuous, occurring 20 to 40 m above the andesite-tuffaceous contact.

The fourth type of mineralization is poorly defined by drilling to date, but is observed as fault or fracture-zone infilling along transverse faults above the andesite-tuff contact.

The majority of the Kuriskova uranium mineralization occurs in veins and disseminations that comprise a largely continuous +/- 2 m thick stratabound body along the meta-sedimentary-metavolcaniclastic contact the Main Zone. The Main Zone contains approximately 63 percent of the total contained U_3O_8 estimated for the deposit. The more lensoidal and discontinuous lower grade stringer-type uranium mineralization hosted within the meta-andesite stratigraphically above the Main Zone mineralization accounts for the remainder. Within tuffs, the uranium mineralization occurs as grains along fractures and in lesser quartz-carbonate-hematitephyllosilicate vein assemblages and on fracture surfaces. The detailed geometry of the vein systems is unknown as there has been no oriented drill core utilized on the Project to date.

Towards the northwest, the deposit boundary is gradational; towards the southeast the deposit is cut by a fault. The continuation of the deposit in the southeast direction has not been sufficiently resolved, and the deposit is open at depths below current drilling. Cross Section D-D' (Figure 7.6), shows the relationship of uranium mineralization to stratigraphy and geological structure in the Kuriskova deposit. The cross section is looking to the northwest and is indicated on the geological map, Figure 7.3. This figure is used unmodified from Figure 7.1 in the SRK report (2009).

7.2.2 Mineralogical Composition

The main uranium minerals of the Kuriskova deposit are uraninite (UO_2) and coffinite [U, Th[(SiO_4)1-x(C+H)4+x6]. There is a small amount of brannerite (U+4,Ca,Ce)(Ti,Fe+3)$_2$$O_6$) and orthobrannerite (U,U,Ti4O12$(OH)_2$). Orthobrannerite can form a solid-solution series with thorutite [(Th,U,Ca)Ti2$(O,OH)_6$]. Determinative mineralogical tests suggest Kuriskova orthobrannerite does not carry any thorium or cerium in the crystal structure.

In the Main Zone, uraninite is the most dominant uranium mineral, with lesser amounts of coffinite accompanied by abundant fine-grained molybdenite (MoS_2).

In the overlying stockwork mineralization in the hanging wall andesites, coffinite has a slight predominance over uraninite at the edges of silica-carbonate veins. Additionally, there is less molybdenite in the stockwork uranium mineralization, which also tends to have lower uranium grades than the Main Zone.

Minor copper mineralization is also present. Copper minerals are paragenetically younger than uranium minerals and often are found in association with coffinite. Minerals noted at the edges of silica-carbonate veins are tennantite [$(Cu,Fe)As_4S_{13}$] and chalcopyrite ($CuFeS_2$), along with very minor amounts of bornite (Cu_5FeS_4) and chalcocite (Cu_2S). Trace accessory minerals include covellite (CuS), gersdorffite (NiAsS), galena (PbS), and Cu-Pb-Sb sulfosalts. Pyrite is a very common sulfide mineral in association with uranium mineralization

The dating of the uraninite by means of electron micro-analyzer indicates the mineralization developed in several stages and it was polygenetic. The expected primary Permian age of mineralization was not identified. The oldest uraninite ages are 200 Ma; the youngest is 25 Ma. On the basis of age dating, mineralogy, and tectonic history, the uranium mineralization is interpreted to be derived from multiple tectonic and metamorphic processes with each process resulting in the remobilization of uranium to the current geological setting (Tournigan, 2008).

7.2.3 Relevant Geological Controls

As defined in Section 7.1.3 (Project Geology) and Section 8.0 (Deposit Type), the Main Zone mineralization is stratabound in a lower meta-tuff unit above the contact with underlying meta-sediments, likely due to the primary porosity and permeability of the unit as well as the tectonic (bedding parallel-shearing) induced permeability of the rocks. The limits to the tabular shape of the mineralization are defined in part by faulting; the controls to mineralization along strike and down dip are not fully understood. Internal to the Main Zone tabular body of mineralization there are local areas of high-grade (+1.0 percent) uranium mineralization; the controls on which are not yet fully understood due to an insufficient density of drilling information in some areas.

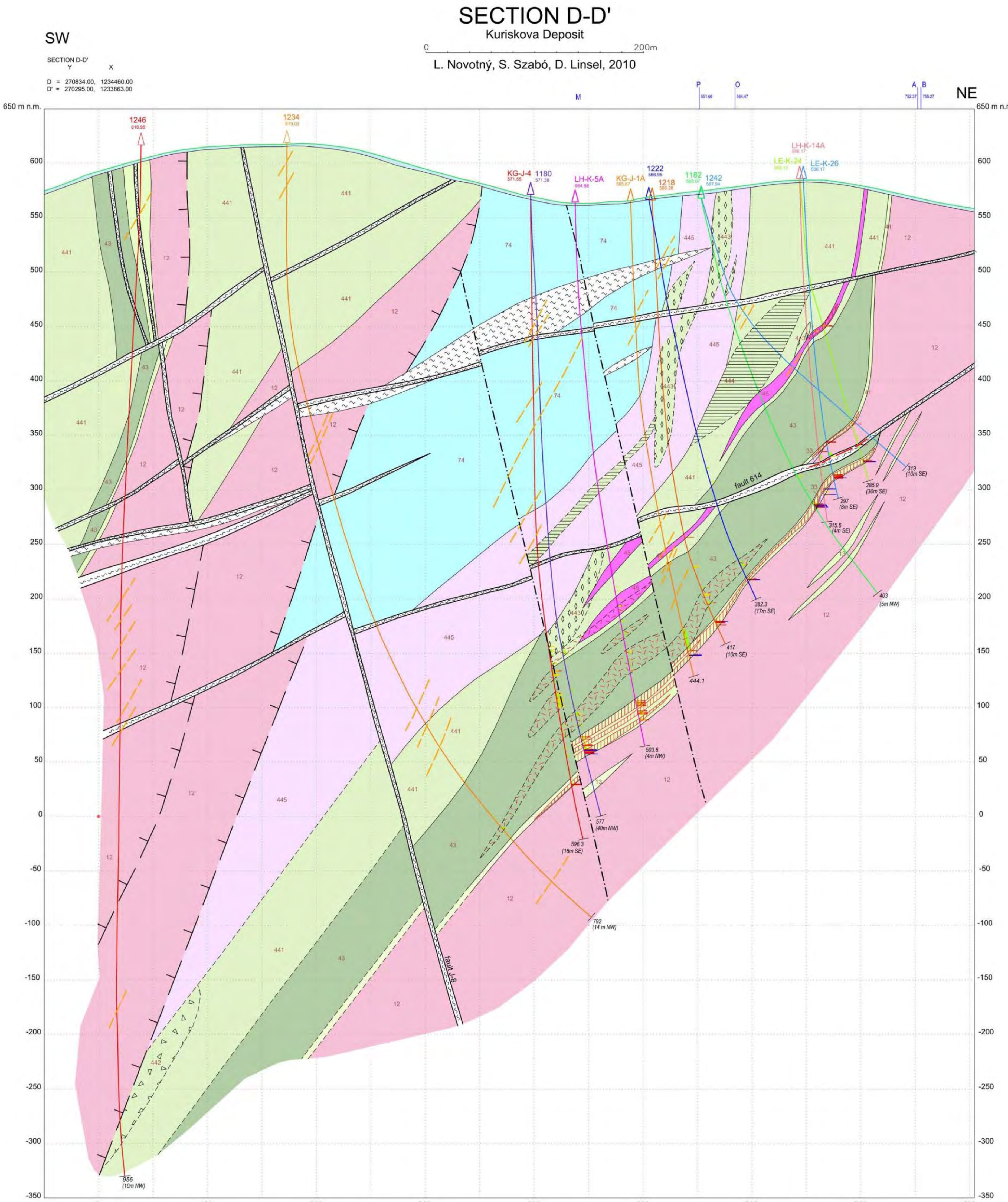

Figure 7.6
Cross Section D-D'

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	Project: Kuriskova Uranium Project	Project Number: 114-310990
	Project Location: Slovak Republic	Date of Issue: June 2011

7.2.4 Type, Character, and Distribution of Mineralization

Geochemically, mineralized horizons are enriched in copper, lead, cobalt, boron, and lanthanum.

Intervals of uranium-molybdenum mineralization are commonly cross-cut by veins of iron-dolomite and quartz, accompanied by chalcopyrite and tennantite. Molybdenite is suspected to be affiliated with uraninite clusters and stringers as minute inter-granular inclusions. Rare flakes to 0.02 mm are observed as disseminations within the wall rocks. Molybdenite concentrations in less mineralized uraniferous portions of the system tend to be low as well, and rarely do molybdenite assays commonly exceed 1 percent molybdenum.

Uraninite, coffinite, and orthobrannerite are the dominant Kuriskova uranium minerals. Uraninite occurs as fine grains within aggregates and irregular grains of sulfides, occurring most commonly as blackish selvages to quartz-carbonate-sulfide veins. The quartz-carbonate-sulfide veinlets and stringers are millimeters to tens of centimeters wide and locally form breccia matrix cement. The uraniferous grains are largely rimmed or enclosed by pyrite.

Uraninite grains are notably localized along the margins of small apatite stringers, which suggests fixation by phosphorous. Uraninite grains range from 1 micron in size to aggregates and microveinlets up to 25 microns.

Uraninite is observed to be distributed along quartz-carbonate-sulfide selvages and as 2 mm long, 10 to 60 micron wide micro-veinlets. It rims and replaces chalcopyrite and tennantite along grain and cluster borders. Distal from the veins, 5 micron grains can form aggregates and clusters to 30 by 80 microns and are often inter-grown with coffinite. Distribution in wallrocks away from veins is highly irregular.

Coffinite cross-cuts quartz-carbonate-sulfide veins and tends to be devoid of uraninite. It is observed as forming small stringers in pinkish altered rock, as disseminated 3 micron grains, and forming aggregates to 50 microns. Coffinite is sometimes affiliated with idiomorphic barite crystals. It can also form hair-like veinlets in wallrock cutting pyrite grains and aggregates. Coffinite within wall rocks away from veins is highly irregular in its distribution.

Orthobrannerite forms euhedral crystals to 10 microns and aggregates to 50 microns. It occurs sparingly with apatite grains to 20 microns and in wall rock in higher grade intervals with uraninite, coffinite, and molybdenite. Chalcopyrite and molybdenite are observed replacing orthobrannerite.

Based on Si values, wavelength- and energy-dispersive X-ray analysis (EDS scans) of uranium species, coffinite was probably derived by oxidation of uraninite. The absence of goethite and true hexavalent (uranium6) uranium minerals suggests no post-mineralization oxidation and, therefore, little involvement of circulating oxidizing meteoric waters. The two-fold population of daughter by-product lead serves to some degree to suggest the "older" uraninite (11.25 percent Pb) and "younger" coffinite (0.1 to 0.2 percent Pb) reflect an Upper Paleozoic and Alpine age, respectively. Alternatively, the bimodal lead population might also indicate a less coffinitized form of uraninite. Significant thorium concentrations have not been detected in any of the uranium minerals analyzed to date.

Pyrite commonly occurs as euhedral grains from 0.1 mm to 0.5 mm. Ilmenite ($FeTiO_3$) occurs as disseminations with grain dimensions of 10 to 50 microns.

Barite is also observed in quartz-carbonate veinlets as fine selvages, as aggregates to 0.15 x 0.60 mm, and as prismatic crystals to 0.2 x 1.0 mm. It can contain very fine grains of uraninite and is often encapsulated by chalcopyrite and quartz. It occurs invariably in hematite-impregnated wallrock and is replaced by uraninite and coffinite.

Alteration of the wallrock is limited to pervasive sericite, clay and very weak silicification, in addition to moderate to strong veinlet-associated and pervasive carbonate alteration.

Within the Main Zone, organic carbon content is virtually nil, and carbonate-affiliated (nonorganic) carbon values are low, varying from 0 to 6.02 percent carbon (C). Chloritization adjacent to veins is widespread and variable and is often associated with hematite and carbonate.

Fe-Ti oxides, primarily ilmenite, are concentrated along slip planes of foliation and quartz-chlorite lenses as selvages and tend to be affiliated with very weak uranium-molybdenum mineralization. Most occur as irregular grains to 5 microns with rare euhedral crystals to 20 microns and forming aggregate veinlets parallel to foliation. It also is found distal from veins in wall rock forming 5 micron inclusions in muscovite/sericitized matrix. Grains are elongated and parallel to foliation.

The overall paragenesis of the veins and stringers can be roughly summarized as such:

- Pyrite I + hematite;
- Quartz + pyrite II + uraninite and orthobrannerite as replacements of pyrite I, destruction of magnetite and Fe-Ti-(U) minerals; chloritization; and
- Carbonate; intense silicification, sericitization, carbonate, pyritization; introduction of base metal sulfides, coffinite alteration of uraninite, and barite.

Pyrite is ubiquitous with uranium mineralization, forming fine disseminations within wall rock and +1 mm veinlets within quartz-carbonate veins. It is also affiliated with uraninite and base-metal sulfides as selvages. Dimensions of irregular pyrite grains are ±50 microns to 1.0 by 1.5 mm. U-Ti grains are often enmeshed in pyrite veinlets. Pyrite occurs as euhedral grains to 10 microns and as anhedral grains along vein selvages to 50 microns.

Hematite is ubiquitous within wall rock as impregnations with aggregate dimensions to 0.1 mm. Uraninite can replace hematite.

Quartz is observed to occur both early and late in the paragenesis of the deposit. Quartz I is coarse-grained quartz forming polycrystalline aggregates to 150 microns and rimming carbonate veins. Quartz II is fine-grained (10 to 20 microns) and overgrows Quartz I grains. It occurs in bends of ptygmatic folds with a mylonitized fabric suggesting it represents recrystallization of Quartz I during metamorphism. Carbonate is also bimodal, with older carbonate I grains to 40 microns, replaced by younger carbonate II and cross-cut by fine-grained Quartz II veinlets.

EDS scans indicate that apatite is a component in carbonate and uraninite and coffinite lenses. In higher-grade intervals it is observed as thin veinlets to 2 mm length and as occasional isolated isometric grains to 0.2 mm. Sericite is seen in some veins as selvages to apatite veinlets and forming hair-like veinlets cutting apatite.

8.0 DEPOSIT TYPES

Material relevant to this section is detailed in Section 6.0 of the Technical Report prepared for Tournigan by SRK Consulting, dated April 16, 2009 and is reproduced below with no material changes.

The genesis of Kuriskova uranium deposit is not completely understood; however, it is suggested that the deposit is the result of secondary uranium derived from anomalously enriched volcanic or granitic bodies; the uranium mineralization being remobilized and precipitated in structurally-favorable units during the Variscan and early Alpine Orogenies. It is postulated that high heat flow through thinned crust, saline brine production, and thrusting and fracturing provided a permeability pathway into the meta-volcanic units and the mobilization mechanisms to accommodate hydrothermal fluid flow. The high phosphorous content of the meta-volcanic rocks may have been the fixation control on vein- and fracture-controlled uranium mineralization. The Kuriskova uranium deposit, therefore, is best described as an epigenetic vein-type uranium deposit; although, it may have had precursor sedimentary, volcanic, and/or hypogene origins.

Across central and eastern Europe a sequence of stratabound, thrust-bound, and granite-related uranium deposits developed during the Variscan and Alpine Orogenies and associated metallogenesis. The late Variscan/early Alpine and late Alpine uplift resulted in the formation of a set of unconformities or, in geomorphological terms, peneplains with which supergene and hypogene mineralization are associated. The time between the early and late Alpine generations of unconformity-related mineralization coincides with the period of maximum spreading during mid-Jurassic times. Re-mobilization along deep-seated fault zones during various periods of the Variscan and Alpine metallogenic cycles resulted in the supergene and hypogene deposits related to those unconformities. The contact of the basal meta-sedimentary rocks with the overlying meta-volcanic rocks at Kuriskova is one of those unconformities.

The derivation of the uranium from Variscan age devitrification of either tuffaceous material or weathering of granitic bodies, transport, precipitation, and fixation of uranium, molybdenum, and copper can be considered a source for the uranium in the Kuriskova deposit. The association of ilmenite and magnetite destruction and phosphorous and carbonate fixation are described from Kuriskova and other Slovakian volcanic fracture-hosted deposits (Rojkovic, 1997). The Slovakian geologic literature describes the geochemistry and origin of S-type granites of post-compressional Jurassic and Triassic ages and their rhyolitic extrusive equivalents (Uher et al., 2002; Petrik et al., 1994). In the Gemeicum and Veporicum units (basins), there are abundant descriptions of the predominance of glassy acid volcanic (rhyolite and dacite) tuffs as the major volcanic component (Broska, 2001; Broska et al., 2004; Ebner et al., 1999; Pal-Molnar et al., 2001; Rojkovic et al., 2005). Another possible deposit model for uranium mineralization in the Carpathians can be suggested as S-type granitic magmatism for source rocks with hydrothermal and/or metamorphic derived fluid movement and/or re-mobilization of uranium into fractured reducing host-rock environments in the Permian sediments and volcanic.

8.1 Geological Model

As an analogous deposit model, the Kuriskova deposit has been loosely linked to the Saddle Hills (Gurvanbulag) deposit in northeastern Mongolia. The Gurvanbulag uranium deposit is a shallowly-dipping, tabular deposit with strike and dip extents of more than 2.5 by 2 km, respectively. The mineralized horizon consists of two distinct domains adjacent to the hanging wall and footwall contacts of a barren, obsidian-bearing horizon within a dominantly felsic volcanic sequence. The mineralization appears to be predominantly stratigraphically controlled; however, vein-hosted mineralization is known to occur above and below the principal mineralized horizon. The Saddle Hills district is distinctive in terms of the presence of the laterally extensive volcanic obsidian horizon below rhyolites at Gurvanbulag and the laterally extensive uranium mineralization that is conformable to bedding in parts of this horizon.

The Kuriskova deposit takes the form of two zones of mineralization; the Main Zone and the Hanging Wall Zone.

The Main Zone is a thin stratabound (2 to 8 m thick) zone of fracture-controlled mineralization developed along the fractured or sheared/faulted meta-sediment-meta-volcanic contact with dimensions of at least 600 m along strike in a northwest-southeast direction, and explored depth of at least 530 m. The Main Zone mineralization does not crop out at surface, beginning at about 200 m below the surface. Weaker and stockwork-like vein mineralization is peripheral to the Main Zone of mineralization in the Hanging Wall Zone and was noted in sub-crop exposures during the original exploration.

The Main Zone of mineralization is stratabound and mostly hosted in the meta-tuffs and partly in the overlying meta-andesites; the immediate footwall rocks are the Markusovce sandstones of lower Permian age. The Main Zone mineralization dips to the southwest at 45° to 70°. While drilling data indicate the mineralization is continuous, grade and thickness varies considerably.

The Hanging Wall Zone is a quasi-stockwork zone of veins in meta-andesite that has an aggregate lower grade than the Main Zone. Lateral continuity of the Hanging Wall Zone is not well established.

There are other occurrences of stockwork mineralization in the meta-andesites that are not well defined by drilling or surface expressions and occur as shallow mineralization (70 to 200 m) in near vertical stockwork-like structures oriented transverse to foliation. Here mineralization is weak and typically five to 15 times background uranium values typical of surface exposures that initiated exploration in the mid-1970s.

9.0 EXPLORATION

Material relevant to this section is detailed in Section 8.0 of the Technical Report prepared for Tournigan by SRK Consulting, dated April 16, 2009 and is reproduced below with no material changes.

Early exploration began in the 1970s, as described in the History Section (Section 6.0). EUU's exploration began in 2005 and continues to present. EUU's exploration has consisted of airborne geophysical surveys and exploration core drilling.

9.1 Surveys and Investigations

Exploration of the Kuriskova deposit was initiated in 2005 as confirmatory diamond drilling of the historically delineated Main and Hanging Wall mineralized zones, followed by infill drilling to connect and extend uranium mineralization at depth and along strike. Descriptions of the drilling program and procedures are contained in Section 10.0 of this report.

EUU's efforts have been aided greatly by the utilization of a local geological staff that has both uranium exploration experience and knowledge and experience specific to Kuriskova.

9.2 Procedures and Parameters

EUU has conducted extensive regional surveys of Permian volcaniclastics along strike from Kuriskova in the Gemericum and Veporicum Units (former basins), as well as follow-up surveys of historical radiometric anomalies first noted by the Czechoslovakian state exploration entities in the 1980s. EUU contracted McPhar Geophysical, a well-known geophysical contracting group of Canada, which flew approximately 1,450 km^2 of airborne radiometric surveys in 2007. Total kilometers flown in the survey were in excess of 16,250 line-km. The airborne geophysical survey consisted of magnetics and spectral radiometrics (potassium, thorium, and uranium). Figure 9.1 illustrates the location and extent of the survey for the area around the Project.

Details of the survey equipment, airborne procedures, and data processing are not available to SRK; nor are they relevant to the current project activities, which at this point are focused on the Kuriskova resource.

EUU's local geological staff have completed data verification and compilations of the historical drilling at Kuriskova, have provided geological interpretations and oversight on the drilling program, and are responsible for the drill hole database, Quality Assurance/Quality Control (QA/QC) monitoring and compliance procedures, and development of the 3-D geological wireframes of mineralization envelopes.

In SRK's opinion, as in their 2009 report, the exploration drilling efforts at Kuriskova by EUU are appropriate techniques that have verified and added to the historical database for the Project. SRK believes the data are sufficient to support current resource estimates. Tetra Tech affirms these findings and believes current drill programs are appropriate for this level of study and adequately provide data necessary for this resource estimate update.

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TETRA TECH 350 Indiana Street, Suite 500 Golden, CO 80401 (303) 217-5700 (303) 217-5705 fax	***European Uranium Resources Ltd.***	Fig 9.1.cdr	**Figure 9.1** **Location of the** **Kuriskova Survey Area**
	Project: Kuriskova Uranium Project	Project Number: 114-310990	
	Project Location: Slovak Republic	Date of Issue: June 2011	

10.0 DRILLING

Material relevant to this section is detailed in the Technical Report prepared for Tournigan by SRK Consulting, dated April 16, 2009 and is reproduced below with no material changes. Sections 10.1 and 10.2 detail information from the previous resource estimate update, and Section 10.3 details drilling for this resource estimate update.

10.1 Exploration Drilling Prior to 2009

10.1.1 Historical Drilling

Historical drilling programs conducted by CSUP from 1985 to 1990 at the Kuriskova uranium deposit have poorly documented procedures. It is known that 53 core holes were drilled with non-wireline, thin-walled single tube diamond drilling equipment (White, 2007). The thin-walled tubing was easily deflected, resulting in pronounced deviation from the projected drill path and consequently poor drill targeting. The highly broken volcaniclastic meta-andesites possess a slaty cleavage and fracturing, which resulted in poor core recovery. Overall core recovery was estimated at approximately 50 percent. Large areas of the deposit were left untested due to the poor drilling equipment utilized.

Only 27 of the historical drill holes were judged sufficiently verifiable, and the radiometric gamma logs provide the only reliable uranium analysis [equivalent-uranium percent (eU%)] for these holes. The data for these 27 holes have been included in the current drill hole database and are used for resource estimation. Gamma radiometric data have been compared with assay data to confirm the gamma-only data from these drill holes are acceptable for use in resource estimation. Since 2007, EUU has instituted a comprehensive QA/QC sampling program for core sample uranium assays, which constitute the majority of data used for current resource estimation.

10.1.2 EUU Drilling Program 2005 to 2008

In 2005 to 2006, EUU drilled 18 core drill holes, totaling 7,595 m, both for verification of uranium-molybdenum mineralization and for in-fill exploration drilling on the Kuriskova deposit. Historical drill hole #1218 was twinned to confirm the average grade and thickness of reported uranium mineralization, using downhole radiometric gamma logging and chemical assays. In-fill and step-out exploration drilling were conducted to test the numerous gaps in historically defined uranium lenses and envelopes and to extend mineralization along strike.

In 2007 and 2008 (through June 2008), an additional 38 core holes, totaling 12,712 m, have been completed and assayed and included in the database. From June 30, 2008 through December 2008, 23 additional infill core holes were completed, totaling 9,267 m. The drill hole database was updated on February 23, 2009. Table 10.1, below, summarizes drilling activity on the Kuriskova deposit from 1990 to 2008. This table is used unmodified from Table 9.1 in the SRK report (2009), which states that the data are current to February 23, 2009.

Table 10.1. Uranium Project Drilling Activity 1990 to 2008

Year	Drill Holes No.	Count- 0.5m Composited >0.00% U	Count- 0.5m Composited >0.03% U	Mean Grade % U
1990	27	209	193	0.242
2005	3	33	28	0.514
2006	15	102	80	0.267
1990-2006	75	344	301	0.274
2007	27	736	472	0.238
2007-2008	8	215	215	0.201
2008 in-fill	23	636	430	0.324
ALL		1,931	1,418	0.269

10.1.3 White (2007) Describes the Drill Activities

The drilling program was contracted to Geotechnic Consulting of Bratislava. Each hole was first drilled with PQ bits to approximately 100 m using a Hanjin diamond core drill rig. Then the drilling continued using the HQ with switching to a smaller diameter using the NQ, if necessary. Prospector II diamond core drill rig is used if it is possible to reach the final depth, otherwise the HQ and NQ should be used. The drilling contractors use double-barreled drill pipe to maximize core recovery and provide better directional control of the drill path.

At the completion of each hole, the hole is probed using a downhole instrument that measures gamma-ray emissions as counts per second, downhole orientation data (dip and azimuth), as well as other parameters including resistivity and self-potential. Downhole logging is undertaken and equivalent uranium content is calculated from gamma log counts according to a standard method whereby measurements begin at a point half that of background, to the peak of the anomaly and then recording counts per second every 10 cm. Average counts per second are determined for a mineralized interval by dividing by the number of measurement intervals within the total anomalous interval. The downhole probe is calibrated several times with geochemically derived uranium. The eU% values are calculated from downhole gamma readings using a complex differential equation utilizing a symmetric inversion filter. Base inputs into the equation include absorption in drilling mud, diameter of hole, absolute density of wall rock, diameter of the probe, length of detector, measurements at each point and a conversion factor. SRK has not examined in detail the gamma logging procedures, as the eU% values are not used as the primary assay in the database; analysis from core samples are the basis for the drill hole database.

In view of the difficult drilling conditions (i.e., steeply dipping bedding and cleavage planes), the drilling speed is reduced in order to improve the core recovery (average daily meterage achieved was 23 m/day). Additionally, an organic polymer (premix-type, made in France) is mixed with water and used throughout the drilling program. These precautions help to maintain an adequate standard of core recovery throughout the program (i.e., greater than 90 percent recovery overall or almost 100 percent in the fresh rock).

EUU has documented downhole surveying procedures used at Kuriskova (Tournigan, June 2008b). Downhole deviation surveys were done by Russian built IK-2 and UMI-30 electrical resistance inclinometers, performed at various times by Uranpres (drilling company) and also by Koral s.r.o. (geophysical contractor). In 2006, the drilling contractor also used a Tropari survey as a QA/QC check. Surveying was performed at 10 m intervals at the completion of each hole.

The surveying results were good; however, to achieve better accuracy and to match industry standard multi-shot equipment, EUU bought an EZ Trac downhole survey tool in August 2007, made by Reflex of Sweden. The drilling contractor was trained to use the EZ Trac tool by an external consultant, and all drilling since September 6, 2007 has been surveyed with this tool. Surveying is carried out as single-shot measurements at 15 m intervals, while the hole is drilled and as a complete hole multi-shot survey at the completion of the hole drilling. The single shot data provide a check against the multi-shot survey, which is considered the final survey, and as a backup in case the hole is lost prior to completion. The EZ Trac tool is a magnetic instrument that is used in an open hole or is extended out the end of drill rods. It provides downhole azimuth and dip information that is gathered both digitally and manually as a backup check.

Mr. Jozef Cisovsky, geologist in charge of database and QA/QC at Ludovika Energy S.R.O in Slovakia, a wholly owned subsidiary of EUU, carries out QA/QC on downhole survey data at EUU's offices in Spisska Nova Ves, where hard copy back-up survey reports are kept along with the digital files. During the 2007 drilling program, four holes were surveyed using EZ Trac, as well as the Russian instrument, as data verification; no significant differences were noted in the results.

As the EZ Trac system is a magnetic tool and the meta-andesite have the potential for magnetic variations, magnetic references (magnetic base station) are set up prior to drilling and the information compared with magnetic information from the Slovakian national geophysical data center. The reference data are compared with the EZ Trac measurement data for variations in the earth's magnetic field. The data are also examined for large variations in azimuth or dip over short intervals, typically due to movement in the instrument during measurements, and data are adjusted accordingly. All data modifications are recorded and stored with original data for archival purposes.

EUU's drill collar locations are surveyed in the field before and after drilling, and drill holes are marked in the field with a steel pipe cemented in the top of the drill hole (Figure 10.1). A surveyor uses SOKKIA Power Set 4000 theodolite precision surveying equipment to establish drill hole collar coordinates. A registered independent contract surveyor (Prachnar Marek) provides the surveying and individual drill hole survey reports. Surveyed drill holes are tied to established benchmarks in the area, and surveyed from two different stations to verify measurement and avoid errors. Accuracy is to Slovakian Class 3 survey standards; 6 cm in the x and y direction. EUU has documented the surveying protocols in use (standard operating procedures and reference information; Tournigan, June 2008a).

10.1.4 Results to June 2008

The EUU drilling program has been successful for verification of mineralized uranium thicknesses and grade, as first defined by the historical drilling. The drilling confirms the thin blanket-like geometry of the Main Zone and is variable in dip from 60° near the top of the section to 45° towards the deepest drilled portions of the body. Targeted infill drilling during the second half of 2008 was successful in intercepting higher grade mineralization, as planned, demonstrating continuity and predictability of mineralization.

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TETRA TECH 350 Indiana Street, Suite 500 Golden, CO 80401 (303) 217-5700 (303) 217-5705 fax	***European Uranium Resources Ltd.***	Fig 10.1.cdr	**Figure 10.1** **Drillhole Collars in the Field**
	Project: Kuriskova Uranium Project	Project Number: 114-310990	
	Project Location: Slovak Republic	Date of Issue: June 2011	

SRK concludes the drilling methods employed by EUU since 2005 are sufficient to derive satisfactory core samples for analysis and gamma-probe analysis for eU% analytical checks. A 3D picture of the drill hole pattern is shown in Figure 14.11. The hole deviations shown are as expected for most holes, given the depths and the geology.

The poor core recovery of pre-2005 drilling and the presence of strong fracturing and shearing in the host meta-tuffs of the Main Zone of mineralization, suggests poor ground conditions might exist for the Main Zone mineralization. SRK recommends geotechnical investigations of the current core and/or oriented drill core dedicated for geotechnical work. Appendix B shows the Kuriskova Drill Hole Collar Data 1990 through June 2008 provided by SRK (2009).

Shown in Appendix B are significant mineralized intervals from 1990 to 2008.

Drill holes are oriented to cross-cut the tabular mineralized Main Zone; however, intercepts are not true width measurements of mineralized intervals. This is accounted for in the generation of the wire-framed mineralized boundaries.

10.2 EUU 2009 to 2012 Exploration Drilling

The 2009 to 2010 drill program focused on the confirmation of possible extensions of high grade mineralization at the northern edge of the Main Zone North and step out holes to possibly increase pounds in the resource. Targeted drilling was successful in intercepting higher-grade mineralization, as expected, demonstrating continuity and predictability of mineralization. Drill results confirmed the high grade continuity in north and discovered high grade mineralization to the northeast of current resources called Zone 45. EUU also drilled holes in central area of Main Zone North with aim to upgrade category to indicated resource. The two main successes of this drill program are described below.

- High grade mineralization in the northern areas confirmed continuity high grade intercepts of 2008 to 2009 drilling.
- Discovery of Zone 45 in north east extension. The nine holes drilled in this area resulted in high grade mineralization in inter-formation schist in hanging wall. This confirmed radon anomaly. Radon anomalies demonstrate the continuity of this zone further to the northeast, which will be targeted in future drilling programs.

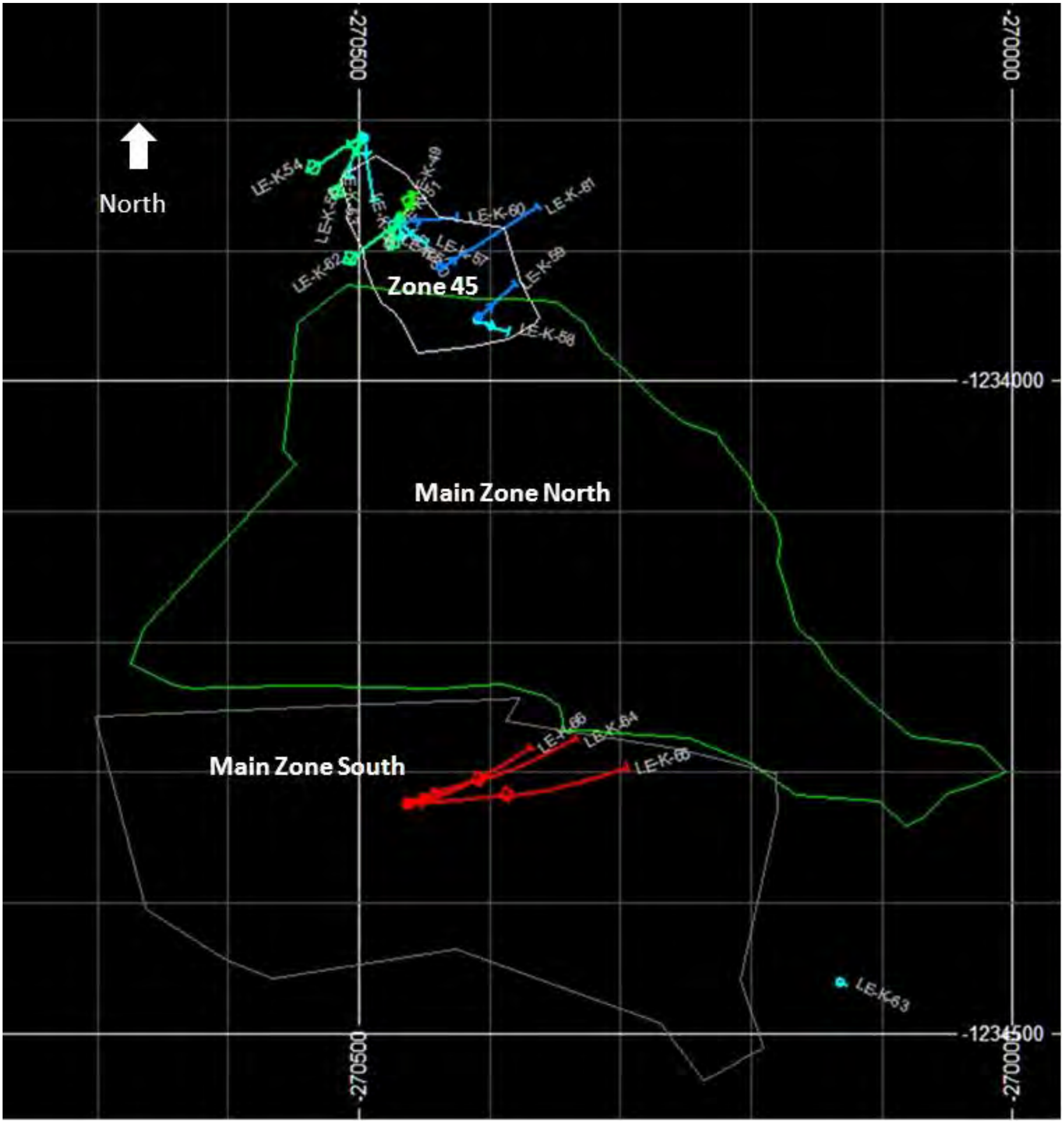
Drilling started in September 2009 and continued until March 2010. EUU drilled 28 core holes totaling 7,548 m of drilling. Two holes were lost due to technical reasons. Appendix B details information on the 2009 to 2010 drill hole program. Appendix B details the significant intercepts from the 2009 to 2010 drill hole program.

10.3 EUU 2010 to 2011 Exploration Drilling

The 2010 to 2011 drill program focused on step out holes in the western extension of Zone 45 and infill drilling in main body of Zone 45. Targeted drilling was successful in intercepting mineralization and demonstrating continuity and predictability of mineralization. EUU also drilled three infill drill holes in Main Zone South area and one drill hole near historic drill hole 1226 to confirm the geology of historic drill hole 1226.

Figure 10.2 illustrates holes drilled in 2010 and 2011. The drilling started in August 2010 and continued until March 2011. EUU drilled 18 core holes totaling 4,548 m. A summary of drilling to date including 2010 to 2011 drilling is given in Appendix B.

Appendix B details information on the 2010 to 2011 drilling. Appendix B details the significant intercepts from the 2010 to 2011 drilling.



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TETRA TECH 350 Indiana Street, Suite 500 Golden, CO 80401 (303) 217-5700 (303) 217-5705 fax	*European Uranium Resources Ltd.*	Fig 10.2.jpeg	**Figure 10.2** **2010-2011 Drill Hole Location Map**
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	Project Location: Slovak Republic	Date of Issue: June 2011	

11.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

The details of sample preparation and analysis from 2005 to 2009 can be found in the technical report prepared for Tournigan by SRK Consulting dated April 16, 2009 and the technical report prepared by Tetra Tech in May 2010. This section summarizes the sample preparation and analysis in different years and discusses the assessment of sample quality and combining data from 2005 to 2011 drilling.

Of the 27 historical gamma-only drill holes deemed useable for resource estimation, nine of the drill holes have table-format data only as gamma-determined eU% data at 0.1 meter intervals; one drill hole has a graphic log and tabulated eU% determinations from the graphic log, and 17 drill holes have graphic gamma logs only. Those graphical gamma logs were digitized to generate 0.1 m eU% data, and the values compared to the values estimated directly from the graphical logs.

11.1 Sample Preparation and Analysis

11.1.1 Drilling 2005 to 2007

White (2007) describes the analytical procedures used in the EUU drilling program from 2005 to 2006:

The samples from the first two drill holes (KG-J-1 and KG-J-2), totaling 26 core samples, were securely air freighted to OMAC laboratories Ltd. in Ireland for analysis. The samples were dried at 850°C, jaw crushed to minus 2 mm, and the total amount of crushed material was milled using LM2 mill to minus 100 µm. Because the mineralized interval from the third drill hole (KG-J-1a) was high grade (over 6 percent uranium for the entire interval), it was unable to be assayed at the OMAC laboratory. Accordingly, it was sent to Ecochem, a laboratory in the Czech Republic (owned by ALS Chemex). There they undertook a spectra-photometric determination of uranium (with an Inductively Coupled Plasma (ICP) determination of other elements). The final determination of uranium grade was by the David-Gray-Eberle titrimetric method.

Core samples from the remaining holes drilled as part of the program were securely dispatched to an ALS Chemex sample preparation laboratory in Pitea, Sweden (in the case of non-Naturally Occurring Radioactive Material (non-NORM) samples) and to the ALS Chemex laboratory in Vancouver (in the case of NORM samples). Non-NORM samples were crushed, pulverized, and the resultant pulps were shipped to the ALS Chemex laboratory in North Vancouver, Canada, for geochemical analysis.

The ALS Chemex sample preparation facility in Sweden is also fully accredited and sample preparation is clearly defined and monitored. Here, core material is crushed to minus 2mm and undergoes ring and puck pulverizing, such that +85 percent of the material passes through a 75 micron screen. The resultant pulps are then dispatched to Canada where they are again screened so that +80 percent of the sample passes through a 75 micron screen. Prepared samples were analyzed for 45 element suite using MA/ES procedure (ME-MS61U), which involves digestion of 0.2 grams (g) of sample in the mixture of nitric, hydrofluoric, hydrochloric, and perchloric acids, bringing solution to dryness and re-dissolving salts in 10 milliliters (ml) of 10 percent aqua regia solution followed by reading using Inductively Coupled Plasma Optical Emission (ICP-OES) spectrometer. The samples were also analyzed for gold using Au4 procedure that involves fusion of 50 g of sample with lead collection, cupellation, dissolving resulting prill in aqua regia, and Atomic Absorbsion (AA) analysis. Samples with greater than

10,000 parts per million (ppm) uranium (1.0 percent uranium) were analyzed using Fusion x-ray fluorescence spectrometer (XRF) (UXRF10).

Between July 2007 and October 2009, the samples were routed to SGS' Lakefield facility in Lakefield, Canada, for primary analysis and to the secondary lab, Actlabs in Toronto, Canada. The sample bags containing sawed mineralized drill core were shipped directly from Slovakia to Toronto where the laboratory crushed, split, and pulverized the core on site before analysis. This step allows the consolidation and retention of all rejects and pulp material at one central site. After sample preparation, SGS sent the samples to Actlabs for renumbering and insertion of quality control samples and the re-numbered samples with QC inserts are sent back to SGS. In addition, samples are measured for SG (density determinations) by three different methods (wet, wax seal, and pycnometer).

EUU decided to proceed using only water method for 2007 and 2008 because of following reasons: practical problems expressed by lab to do wax on the sample (SG during SGS as a preparation laboratory was done on all samples). Results of 155 samples showed excellent correlation between these two methods.

Part of the pulped sample was prepared for multi-element ICP analysis for 52 elements, using a three-acid digestion. Another portion of the pulp was used to prepare a pressed-pellet sample for XRF analysis for uranium and molybdenum determinations. Earlier samples from the 2005 to 2007 drilling campaign, utilized a fusion bead preparation for XRF analysis. Standard samples that were statistically analyzed showed tighter value clusters for the pressed pellet XRF samples over fusion bead preparation, both in ±5 percent and 3a plots. Anomalously high samples and samples for random cross checking were sent to Actlabs for further processing and determination.

There is no change in EUU's sample analysis procedure for the samples analyzed after 2007.

11.1.2 Testing Laboratories

For the 2005 to 2007 drilling program, accreditation of the analytical laboratories, meet full CIM and EU specifications. ALS Chemex's North Vancouver and Ecochem Laboratories in the Czech Republic (now wholly merged with ALS) hold ISO 9001:2000 registration. The North Vancouver laboratory has also received ISO 17025 accreditation from the Standards Council of Canada under CAN-P-1579 Guidelines for Accreditation of Mineral Analysis Testing Laboratories. The ALS Chemex sample preparation facility in Sweden is also fully accredited and sample preparation is clearly defined and monitored (http://www.alsglobal.com/mineral/DivisionProfile.aspx).

Beginning in July 2007 to October 2009, Kuriskova drilling samples were sent to SGS Lakefield laboratory in Lakefield, Canada as the primary laboratory, with Actlabs in Toronto providing a secondary function. Both SGS-Lakefield and Actlabs are reputable commercial labs using industry-standard analytical methods. The procedure implemented by EUU in 2007 worked very well, but total turnaround time was 10 to 12 weeks. Close follow-up was required to monitor each step and coordinate between Actlab and SGS. Another major drawback was sending half core to SGS incurring high cost of transportation, storage at SGS, and shipping back to Ludovika Energy in Slovakia. Shipping half core as radioactive material was also an issue with many legal formalities and documentation.

To overcome these issues, in 2009, EUU identified the following changes to carry out sample preparation and assaying as per EUU QA/QC standard operating procedures. These were mainly changes in laboratories without changes in EUU procedures and not compromising with controls implemented by EUU in 2007.

- Sample preparation: EL laboratory, Spisska Nova Ves, Slovakia. EL laboratory is certified laboratory having ISO/IEC 17025:2005 accreditation.
- QC Inserts: Directly by Ludovika Energy staff.
- Assay: ALS, Seville, Spain
- Check Assay: State Geological Institute of Dionýz Štúr (SGUDS) laboratory, Spisska Nova Ves, Slovakia. Certified laboratory lab having ISO/IEC 17025:2005 accreditation

EUU audited EL and SGUDS laboratories in Slovakia by auditing their procedures to ensure they met the required standards and give confidence to EUU for switching to these laboratories. A summary of QA/QC on EL and SGUDS Laboratory is below. The objective of this test was to ensure proper sample preparation and to check contaminations at EL Laboratory and assay procedure at SGUDS laboratory.

QA/QC steps undertaken to conduct audit of EL and SGUDS laboratories:

- Ten one-quarter core samples selected from 2007 and 2008 drilling program.
- After each of above mentioned samples, a field blank was inserted.
- Samples were submitted for sample preparation to EL laboratory.
- In the presence of Ludovika Energy, duplicates were created from coarse reject material and pulp reject material.
- QC inserts (standards, pulp blanks and duplicates created in point 4 were inserted into sample stream).
- Samples were submitted to SGUDS for analysis.

Table 11.1 details the QA/QC program utilized by EUU to check the new laboratories.

Table 11.1. 2009 to 2010 QA/QC Program Samples To Check New Laboratories

QC SAMPLES	No. of Samples
Check Samples (1/4 Core)	10
Field Blanks	10
Duplicates from Coarse Reject Material	3
Duplicates from Pulp Reject Material	3
Standards	4
Pulp Blanks	1
Grind Check (150, 106 µm)	1
Total	32 Samples

The quality assessment of these 32 control samples is detailed in Section 13.6 of Technical report prepared for Tournigan by Tetra Tech in May 2010.

Based on the results of check assays carried out by SGUDS laboratory in Slovakia during the 2009 to 2010 drilling program, in August 2010, the primary assay laboratory was changed from

ALS in Spain to SGUDS laboratory in Slovakia. Since August 2010, ALS has been used as secondary laboratory.

Between 2008 and 2010, EUU sent 428 samples of core coarse rejects to Energy Labs in Casper, Wyoming, USA for closed can radiometric analysis to examine the state of equilibrium of Kuriskova mineralization. Energy Labs, is a certified commercial analytical laboratory that has provided service to the uranium industry since 1952. See Section 12.1 for a discussion of closed can radiometric analysis and the results.

11.2 Quality Control Samples

11.2.1 2005 to 2006 Drilling Program

Standard laboratory QC procedures were applied to the sample analysis at the ALS Laboratory in North Vancouver; 10 percent of samples analyzed were duplicates, blanks, and reference materials inserted into the sample stream.

Geochemical analysis was monitored via the use of internal control standards that were then compared to certified Canada Centre for Mineral and Energy Technology (CANMET) and Geostats Pty Ltd's (Geostats) standard reference material. As part of data verification, A.C.A. Howe received all unmodified assay information relating to EUU's sample analysis, reviewed the laboratory QA/QC procedures, and found the QA/QC data to be satisfactory.

11.2.2 2007 to 2011 Drilling Program

As a result of an earlier external audit, in August 2007, EUU instituted a rigorous QA/QC program, under EUU's control, that is summarized in their project reference manual (Tournigan Energy, 2007). The reference manual enumerates the sampling steps, chain-of-custody (sample management), QA/QC procedures performed, and reporting procedures. Once the samples were delivered to the laboratories, a dedicated EUU geologist tracked the samples and consolidated and reported all assay data completed as it was received.

Once initial analyses were completed, random samples were sent from the primary laboratory to secondary laboratory for check assays, to establish precision (repeatability), and analytical bias. Additionally, coarse sample rejects were chosen at random and sent to the secondary laboratory for preparation and analysis and to check the accuracy and repeatability of the original sample preparation. A further check on the primary laboratory precision was conducted by renumbering pulps and re-submitting from the secondary to primary laboratory for analysis. The primary and secondary laboratories used between 2007 to 2009, 2009 to 2010, and 2010 to 2011 are described in Section 11.1.2 Testing Laboratories. EUU monitored QA by plotting and analyzing the data, as received, and activated re-assay of sample batches that did not meet pre-determined standards. Table 11.2 and Table 11.3 are from SRK's 2009 technical report and show a typical sample log sheet produced by EUU staff and a summary of the objectives of the QA/QC program established by EUU, respectively. The log sheet is utilized to date by EUU.Figure 11.1 through Figure 11.3 are updated flow charts and graphically depict sample preparation, analytical procedures, and QA/QC procedures respectively used by EUU since August 2010.

Once the data have been received, the assay package undergoes a thorough statistical evaluation as per the Project reference manual by a EUU geologist dedicated to the QA/QC program. Both sample blanks and standard reference material are compared, and any analytical "breaks" are noted. The Kuriskova Project utilizes commercially prepared standard reference material (SRM) purchased from CANMET. Uranium SRM samples (BL2, BL2a, and BL3) and one each for copper and molybdenum (HV2) are used. Field blanks are prepared from Kuriskova core having assay values below detection limits and of similar lithologies. The sample field blanks are used for checks on sample preparation and to check for contamination during the crushing and pulverizing stages.

Analytical values for blanks, duplicates, and re-analysis are plotted on an error plot graphs and scatter plot graphs, with 10 percent warning lines and 20 percent action lines drawn parallel to the sample plot line on the error plot. Any samples that fall between the 10 percent and 20 percent lines are subject to investigation and re-analysis if they fall outside the 20 percent line. Standard sample analyses are plotted on line graphs with ±5 percent limits and ±3 standard deviation limits. Blanks are plotted on line graphs.

The rigorous QA/QC measures used by EUU from the inception of their drill program are judged by SRK in 2009 to be excellent and suitable to maintain strict controls on the Kuriskova sample stream. The 2 percent to 5 percent occurrence of analytical "breaks" is well within mining industry standards and is acceptable for resource estimation. Tetra Tech affirms the findings of SRK in 2009 and finds the implementation of quality control samples by EUU to date, to be excellent and in line with common standard practices.

11.3 Bulk Density Measurements

A total of 4,970 SG laboratory measurements to determine rock bulk density were done on samples submitted for analysis. The current data suggest an average value of 2.75 for all mineralized zones and is sufficient for scoping level studies going forward. As of 2009, a good correlation between methods used by SGS on core samples exists; as described in Section 11.1.1 the SG was determined with and without wax coating prior to standard wet method of measurements.

TABLE 11.2: SAMPLE LOG SHEET

EUROPEAN URANIUM RESOURCES LTD. - KURISKOVA URANIUM PROJECT

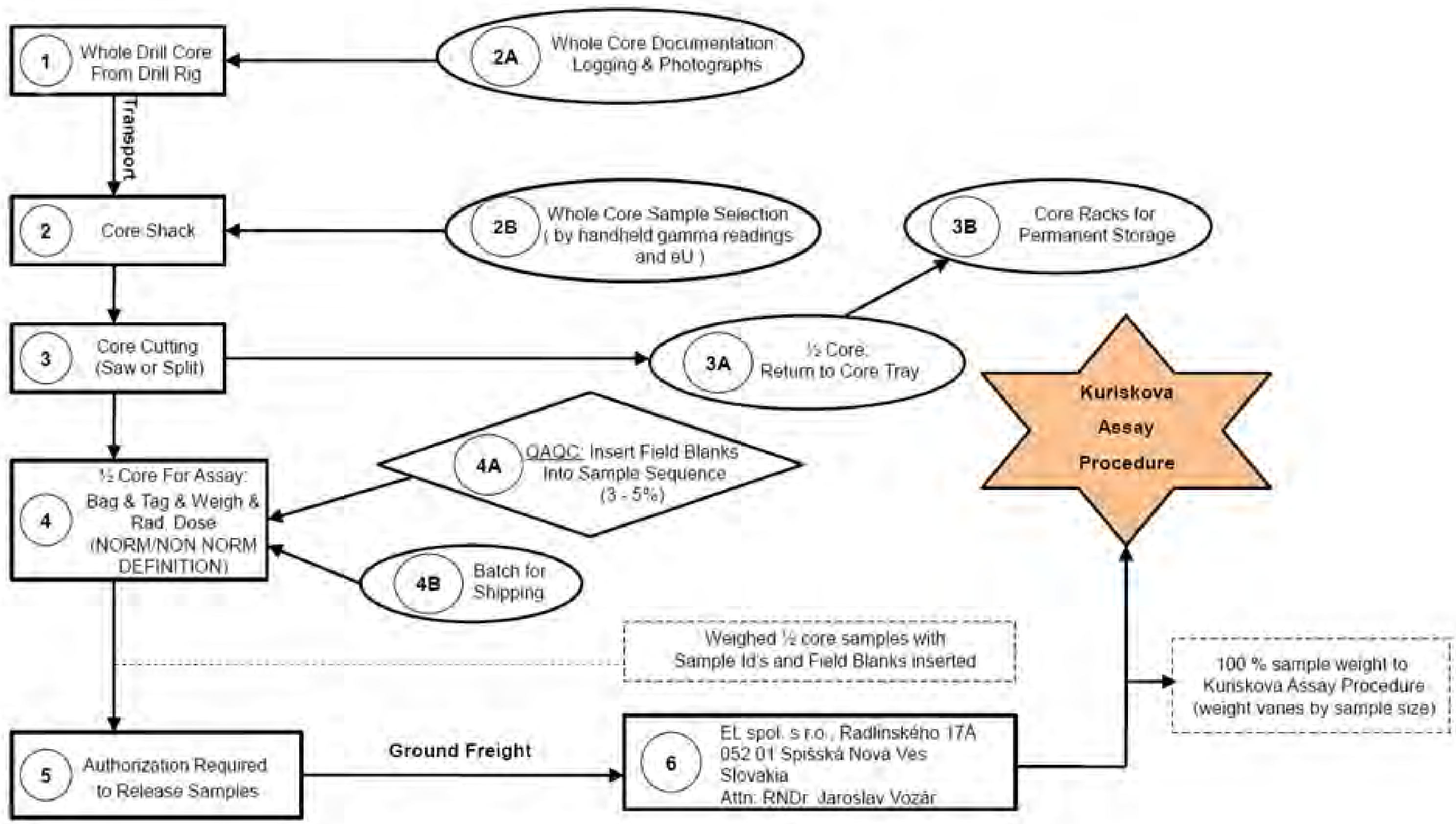
Tournigan 2008, SRK 2009

1	Unique	Site_ID	Hole	No. of sample	From (m)	To (m)	Length (m)	Weight /kg/	Recovery (%)	No. of Core Box	Gamma activity From (imp/s)	Gamma activity To (imp/s)	Rock code	Comments	Diameter	Calculated-weight	uSv/hr	No.of Drum	Actual Weight Minus Calc. Weight (kg)	Difference
2	1	43784	LH-K-6A	1	200.5	201	0.5	1.940	95		190	210	44	tuff	HQ	2.04725	0.19	183	-0.107	-6%
3	2	43785	LH-K-6A	2	201	201.5	0.5	2.205	95		250	280	44	tuff	HQ	2.04725	0.25	183	0.158	7%
4	3	43786	LH-K-6A	3	201.5	202	0.5	2.169	90		230	270	44	tuff	HQ	1.9395	0.23	183	0.230	11%
5	4	43787	LH-K-6A	4	271.4	271.9	0.5	1.894	90		190	230	44	tuff	HQ	1.9395	0.21	183	-0.045	-2%
6	5	43788	LH-K-6A	5	271.9	272.5	0.6	2.606	90		210	240	44	tuff	HQ	2.3274	0.20	183	0.279	11%
7	6	43789	LH-K-6A	6	272.5	273	0.5	1.938	90		280	310	44	tuff	HQ	1.9395	0.45	183	-0.001	0%
8	7	43790	LH-K-6A	7	273	274	1	3.671	90		260	300	44	tuff	HQ	3.879	0.25	183	-0.208	-6%
9	8	43791	LH-K-6A	8	294	294.7	0.7	2.597	95		200	300	44	tuff	HQ	2.86615	0.22	183	-0.269	-10%
10	9	43792	LH-K-6A	9	294.7	295.1	0.4	1.485	90		270	320	44	tuff	HQ	1.5516	0.29	183	-0.067	-4%
11	10	43793	LH-K-6A	10	295.1	295.5	0.4	1.272	80		500	600	44	zrudnený tuff	HQ	1.3792	0.6	183	-0.107	-8%
12	11	43794	LH-K-6A	Field blank	Field blank	Field blank	Field blank	0.55								Check Formula		183	#VALUE!	#VALUE!
13	12	43795	LH-K-6A	11	295.5	296.5	1	3.495	95		280	320	44	tuff	HQ	4.0945	0.3	183	-0.600	-17%
14	13	43796	LH-K-6A	12	296.5	297.5	1	4.167	95		280	320	44	tuff	HQ	4.0945	0.22	183	0.072	2%
15	14	43797	LH-K-6A	13	297.5	298.2	0.7	2.408	85		280	320	44	tuff	HQ	2.56445	0.21	183	-0.156	-6%
16	15	43798	LH-K-6A	14	298.2	298.6	0.4	1.578	80		200	240	44	tuff	HQ	1.3792	0.21	183	0.199	13%
17	16	43799	LH-K-6A	15	298.6	299	0.4	1.428	80		300	700	912	tektonika s rudou	HQ	1.3792	0.5	183	0.049	3%
18	17	43800	LH-K-6A	16	299	300	1	3.654	95		200	250	44	tuff	HQ	4.0945	0.23	183	-0.441	-12%
19	18	43801	LH-K-6A	17	300	301	1	3.666	80		200	250	44	tuff	HQ	3.448	0.24	183	0.218	6%
20	19	43802	LH-K-6A	18	301	302	1	3.752	80		200	250	44	tuff	HQ	3.448	0.24	183	0.304	8%
21	20	43803	LH-K-6A	19	302	303	1	3.381	95		190	220	44	tuff	HQ	4.0945	0.20	183	-0.714	-21%
22	21	43804	LH-K-6A	20	303	303.3	0.3	1.251	90		180	200	44	tuff	HQ	1.1637	0.21	183	0.087	7%
23	22	43805	LH-K-6A	21	303.3	304	0.7	2.910	95		180	200	44	tuff	HQ	2.86615	0.49	183	0.044	2%
24	23	43806	LH-K-6A	22	323.5	324.4	0.9	3.851	100		180	230	45	jemnozrné sedimenty	HQ	3.879	0.20	183	-0.028	-1%
25	24	43807	LH-K-6A	23	324.4	324.9	0.5	2.271	100		220	260	911	mylonit s kremeňom	HQ	2.155	0.21	183	0.116	5%
26	25	43808	LH-K-6A	24	324.9	325.6	0.7	2.310	95		260	400	45	jemnozrné sedimenty s kremeňom	HQ	2.86615	0.38	183	-0.556	-24%
27	26	43809	LH-K-6A	25	325.6	326	0.4	1.326	95		230	260	45	jemnozrné sedimenty s kremeňom	HQ	1.6378	0.25	183	-0.312	-24%
28	27	43810	LH-K-6A	26	326	326.3	0.3	1.204	100		260	300	45	pyritizovane sedimenty	HQ	1.293	0.21	183	-0.089	-7%
29	28	43811	LH-K-6A	27	326.3	327.5	1.2	4.601	95		220	250	45	jemnozrné sedimenty s karbonátmi	HQ	4.9134	0.20	183	-0.312	-7%
30	29	43812	LH-K-6A	28	327.5	328	0.5	2.474	95		220	250	45	jemnozrné sedimenty s karbonátmi	HQ	2.04725	0.23	183	0.427	17%
31	30	43813	LH-K-6A	29	328	328.9	0.9	3.696	95		220	250	45	jemnozrné sedimenty s pyritom	HQ	3.68505	0.46	183	0.011	0%
32	31	43814	LH-K-6A	30	328.9	329.5	0.6	2.532	100		290	350	45	jemnozrné sedimenty	HQ	2.586	0.27	183	-0.054	-2%
33	32	43815	LH-K-6A	31	329.5	330.5	1	4.348	100		230	280	45	jemnozrné sedimenty	HQ	4.31	0.20	183	0.038	1%
34	33	43816	LH-K-6A	32	371	372	1	3.660	95		240	300	43	andezity	HQ	4.0945	0.27	183	-0.435	-12%
35	34	43817	LH-K-6A	33	372	372.5	0.5	1.947	100		240	300	43	andezity	HQ	2.155	0.23	183	-0.208	-11%
36	35	43818	LH-K-6A	34	372.5	373	0.5	2.123	100		240	300	43	andezity	HQ	2.155	0.23	183	-0.032	-2%
37	36	43819	LH-K-6A	35	373	373.4	0.4	1.485	95		500	1100	43	zrudnené andezity	HQ	1.6378	0.8	183	-0.153	-10%
38	37	43820	LH-K-6A	36	373.4	374	0.6	2.293	95		200	230	43	andezity	HQ	2.4567	0.22	183	-0.164	-7%
39	38	43821	LH-K-6A	37	382.6	383	0.4	1.732	100		160	180	43	andezity	HQ	1.724	0.18	183	0.008	0%
40	39	43822	LH-K-6A	38	383	384	1	3.887	95		180	230	43	andezity	HQ	4.0945	0.19	183	-0.208	-5%
41	40	43823	LH-K-6A	39	384	384.3	0.3	1.019	90		230	260	43	andezity	HQ	1.1637	0.18	183	-0.145	-14%
42	41	43824	LH-K-6A	40	384.3	384.8	0.5	2.121	100		180	220	43	andezity	HQ	2.155	0.20	183	-0.034	-2%
43	42	43825	LH-K-6A	41	384.8	385.2	0.4	1.530	95		300	340	43	andezity	HQ	1.6378	0.35	183	-0.108	-7%
44	43	43826	LH-K-6A	42	385.2	386	0.8	3.510	100		180	200	43	andezity	HQ	3.448	0.17	183	0.062	2%
45	44	43827	LH-K-6A	43	401	402	1	4.329	100		200	240	43	andezity	HQ	4.31	0.19	183	0.019	0%
46	45	43828	LH-K-6A	44	402	402.7	0.7	3.163	100		240	330	43	andezity	HQ	3.017	0.20	183	0.146	5%
47	46	43829	LH-K-6A	45	402.7	403.3	0.6	1.900	90		250	270	43	andezity	HQ	2.3274	0.25	183	-0.427	-22%
48	47	43830	LH-K-6A	46	403.3	404.3	1	3.424	90		260	320	43	andezity	HQ	3.879	0.30	183	-0.455	-13%
49	48	43831	LH-K-6A	47	404.3	405	0.7	2.990	100		200	250	43	andezity	HQ	3.017	0.22	183	-0.027	-1%
50	49	43832	LH-K-6A	48	405	406	1	4.106	95		200	250	43	andezity	HQ	4.0945	0.19	184	0.011	0%
51	50	43833	LH-K-6A	49	406	407	1	4.259	95		180	230	43	andezity	HQ	4.0945	0.19	184	0.165	4%
52	51	43834	LH-K-6A	50	407	407.8	0.8	3.141	95		180	260	43	andezity	HQ	3.2756	0.17	184	-0.135	-4%
53	52	43835	LH-K-6A	51	407.8	408.3	0.5	2.025	100		200	240	43	andezity	HQ	2.155	0.17	184	-0.130	-6%
54	53	43836	LH-K-6A	52	408.3	408.8	0.5	2.037	100		270	320	43	andezity	HQ	2.155	0.19	184	-0.118	-6%
55	54	43837	LH-K-6A	53	408.8	409.4	0.6	2.167	100		200	250	43	andezity	HQ	2.586	0.22	184	-0.419	-19%
56	55	43838	LH-K-6A	54	409.4	409.7	0.3	1.124	90		400	650	43	zrudnené andezity	HQ	1.1637	0.7	184	-0.040	-4%
57	56	43839	LH-K-6A	55	409.7	410.2	0.5	1.870	95		200	270	43	andezity	HQ	2.04725	0.21	184	-0.177	-9%
58	57	43840	LH-K-6A	56	410.2	411	0.8	3.150	95		270	300	43	andezity	HQ	3.2756	0.25	184	-0.126	-4%
59	58	43841	LH-K-6A	57	411	411.7	0.7	3.027	95		220	300	43	andezity	HQ	2.86615	0.19	184	0.161	5%

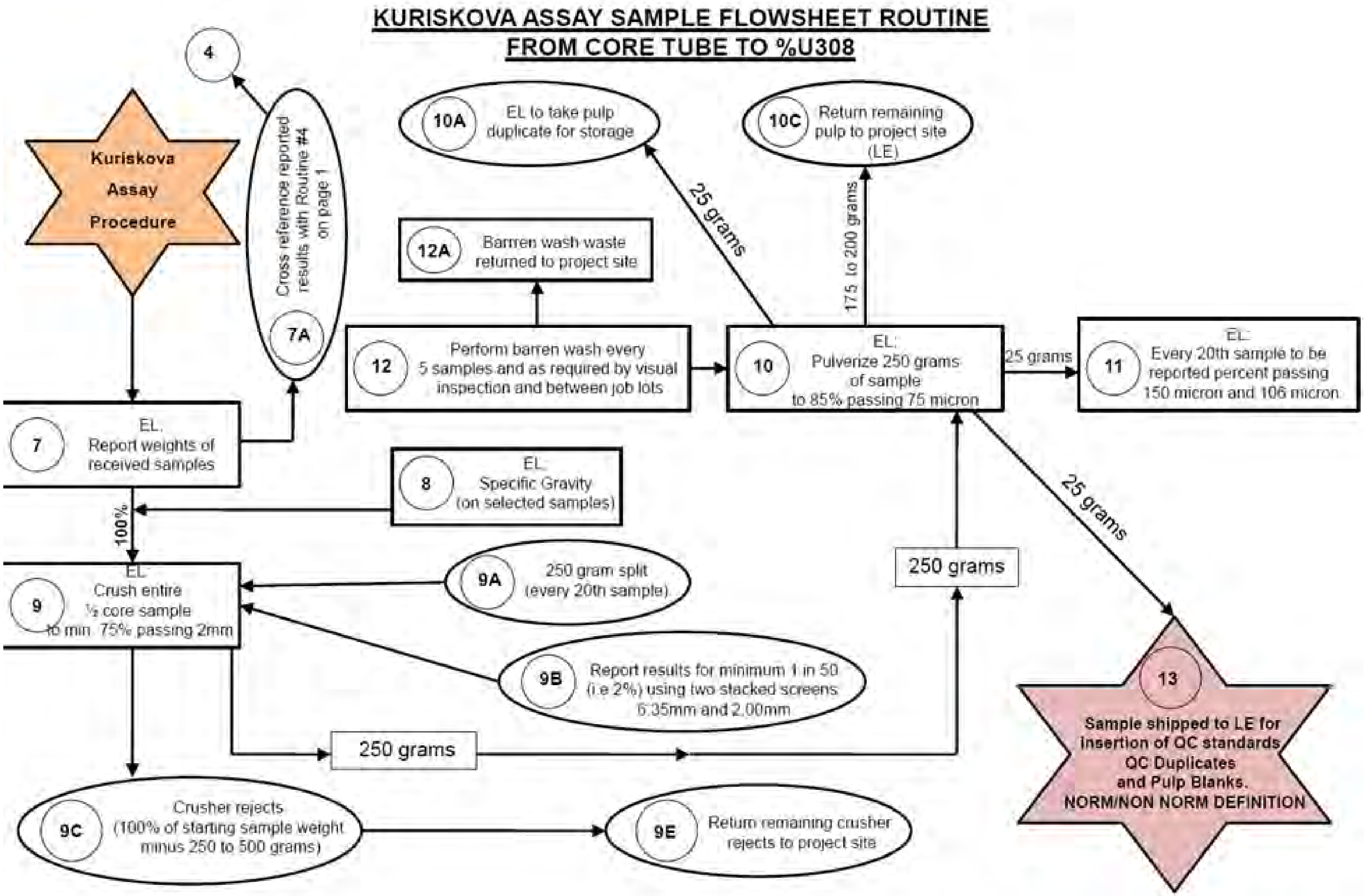
Table 11.3. Summary and Objectives of the QA/QC Program

Control Type	Objective	Area of Concern Frequency	
SRM	To check accuracy and possible sample mix-ups	Analytical	3 - 5%
Pulp Duplicate	Accuracy and precision of assays	Analytical and preparation	3 - 5%
Check Assay	Precision and bias of assays	Analytical and preparation	3 - 5%
Field Blank	To check contamination	Sample preparation	2 - 3%
Check Samples	To check sample preparation and precision	Preparation sample protocol	2%
Grinding Check	Pulp homogeneity	Sample preparation	2%

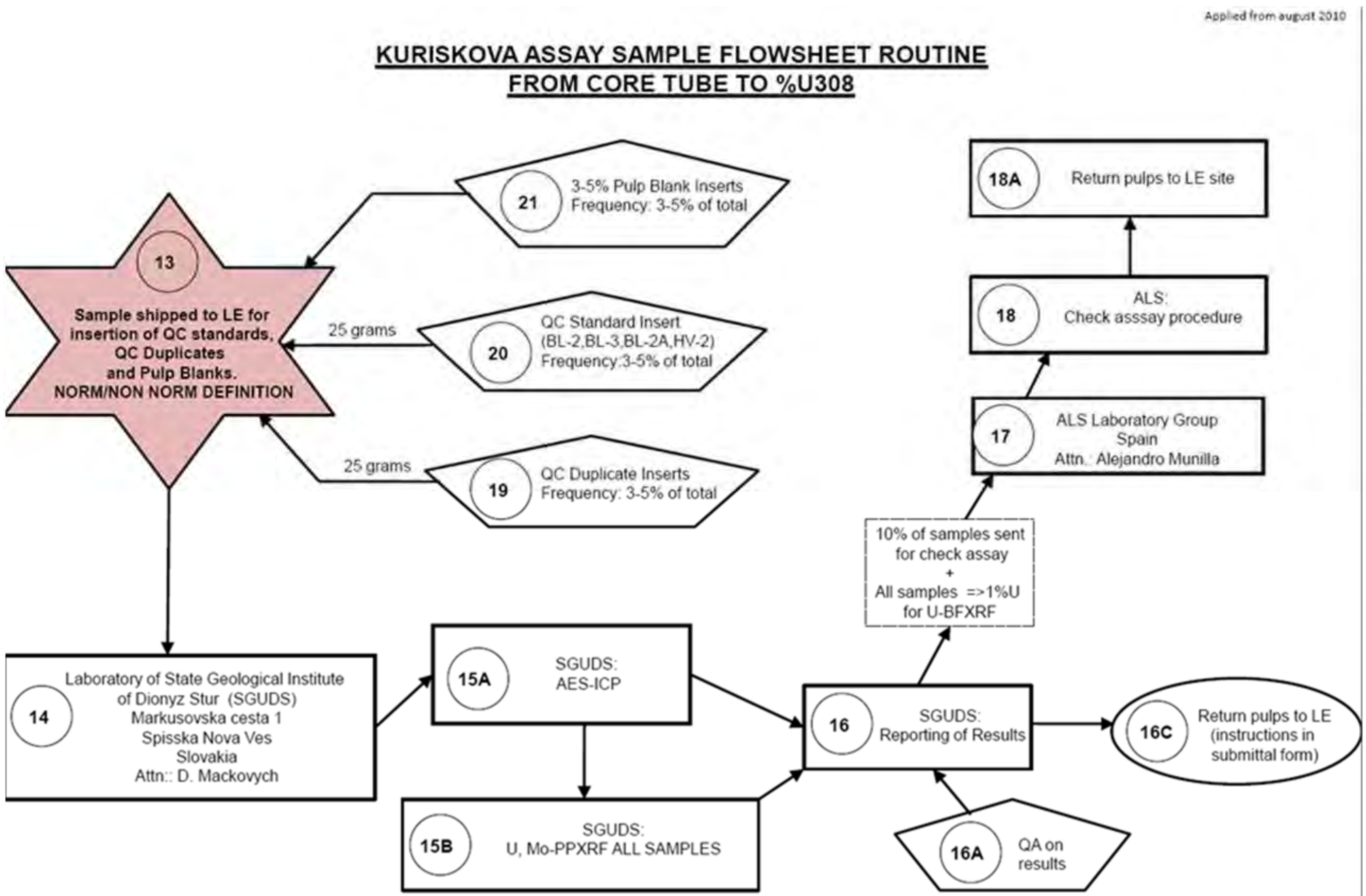
KURISKOVA ASSAY SAMPLE FLOWSHEET ROUTINE FROM CORE TUBE TO %U308



Issued by:	/Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, CO 80401 (303) 217-5700 (303) 217-5705 fax	***European Uranium Resources Ltd.***	Fig 11.1jpeg	**Figure 11.1** **Sample Flow Chart – Sample Preparation and Shipment (2011)**
	Project: Kuriskova Uranium Project	Project Number: 114-310990	
	Project Location: Slovak Republic	Date of Issue: June 2011	



Issued by: TETRA TECH, 350 Indiana Street, Suite 500, Golden, CO 80401, (303) 217-5700 (303) 217-5705 fax	Prepared for: *European Uranium Resources Ltd.*	File Name: Fig 11.2.jpeg	**Figure 11.2**
	Project: Kuriskova Uranium Project	Project Number: 114-310990	**Sample Flow Chart – Primary Analytical Procedures (2011)**
	Project Location: Slovak Republic	Date of Issue: June 2011	



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TETRA TECH 350 Indiana Street, Suite 500 Golden, CO 80401 (303) 217-5700 (303) 217-5705 fax	***European Uranium Resources Ltd.***	Fig 11.3.jpeg	**Figure 11.3** **Sample Flow Chart – QA/QC Procedures**
	Project: Kuriskova Uranium Project	Project Number: 114-310990	
	Project Location: Slovak Republic	Date of Issue: June 2011	

11.4 Results, Interpretations, and Conclusions (All Quality Samples from 2005 to 2011 drilling)

EUU has documented the results of analysis for standards, duplicates, and blanks for each batch; a summary of the data is presented here. Table 11.4 shows the quantity of control samples and their percentage to total number of samples.

Table 11.4. Details of Number Of Different Type of Control Samples

QC Samples	No. of Samples	% of Total Sample	Total Sample (Excluding QC samples)
Standard (U)	308	3.80%	8112
Standard (Mo, Cu)	190	2.34%	8112
Flied Blank	270	3.33%	8112
Pulp Blank	91	1.12%	8112
Duplicate	287	3.54%	8112
Check Assay for U	620	7.64%	8112
Check Assay for Mo	552	6.80%	8112

11.4.1 Standard Reference Material

EUU has done a sufficient amount of analyses by multiple method, ICP, pressed pellet-XRF (pp-XRF), and borate fusion bead and XRF (bf-XRF) to determine that pp-XRF provides the most accurate and precise analysis for uranium at Kuriskova uranium deposit. Figure 11.4 to Figure 11.6 show the primary laboratory results of analysis of certified reference material (standard) using the pressed pellet sample preparation and XRF analysis for uranium. The results show less than 5 percent deviation from the known value and a tighter spread of values than received from the preparation by either bf-XRF or ICP. EUU has elected to use pp-XRF as the primary analytical method for all assays going forward.

Figure 11.4 to Figure 11.6 show line graphs for SRM BL2, BL3, and BL2a by pp-XRF. Through the review of the aforementioned figures provided and additional analysis, EUU has decided that pp-XRF results are the best analysis to be used. In 2008, SGS laboratory reviewed the results and agrees with the conclusion. EUU has analyzed 308 uranium and 190 molybdenum standards from 8,112 total samples during drilling from 2007 to 2011.

11.4.2 Duplicates

EUU analyzed 287 duplicate samples from a total of 8,112 samples between 2007 to 2011 drilling and the re-assay program of prior drilling. Re-assaying program of prior drilling (2005 to 2006) was done to ensure prior drilling qualifies EUU QA/QC protocols implemented in 2007. EUU's analytical results for duplicates samples are shown Figure 11.7 and Figure 11.8.

11.4.3 Check Assays

EUU analyzed 620 check assays for uranium since 2007 drilling program. Check assays analyses were carried out by Actlabs until 2009, SGUDS (Slovakia) from 2009 to 2010, and ALS (Spain) from 2010 to 2011. In addition, EUU carried out check assay analyses for all the 2005 drilling samples (primary assay by Echochem laboratory, Czech Republic) and all of the 2006 drilling program samples (primary assay by ALS Chemex). Figure 11.9 and Figure 11.10 show the results of check assay samples.

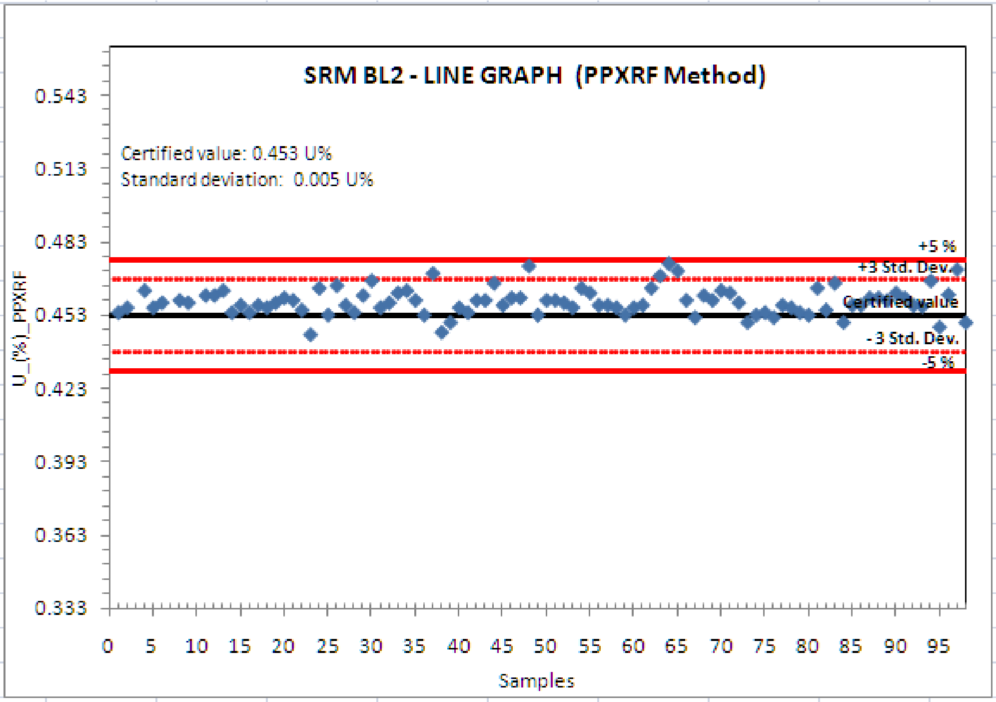


Figure 11.4. Analyses of SRM BL2 (0.453 Percent Uranium) by pp-XRF

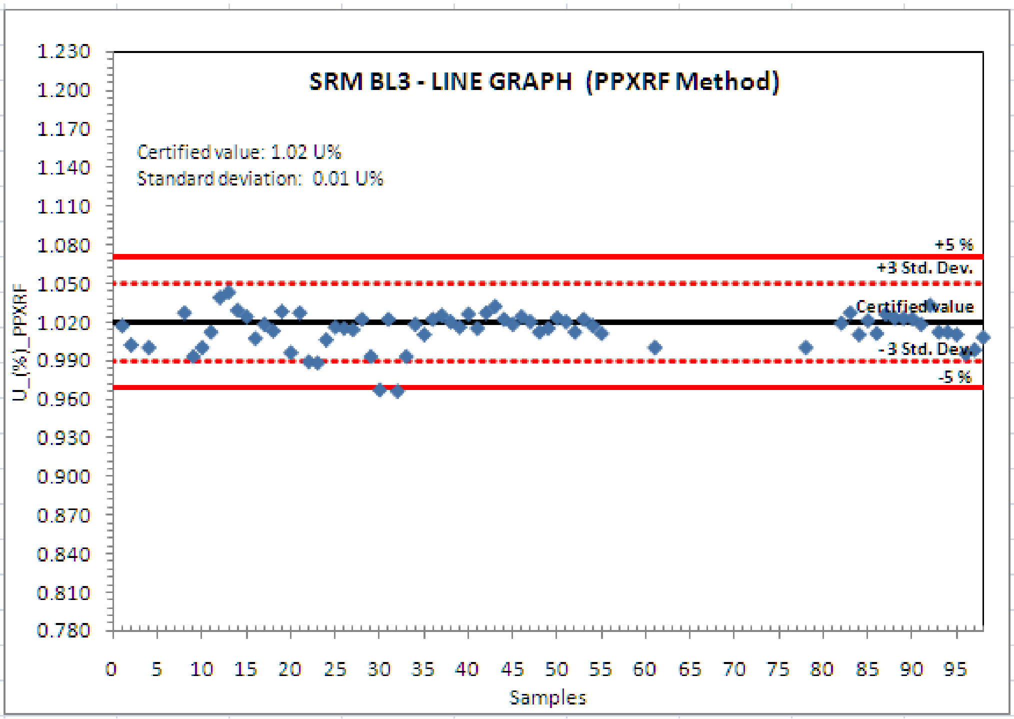


Figure 11.5. Analyses of SRM BL3 (1.02 Percent Uranium) by pp-XRF

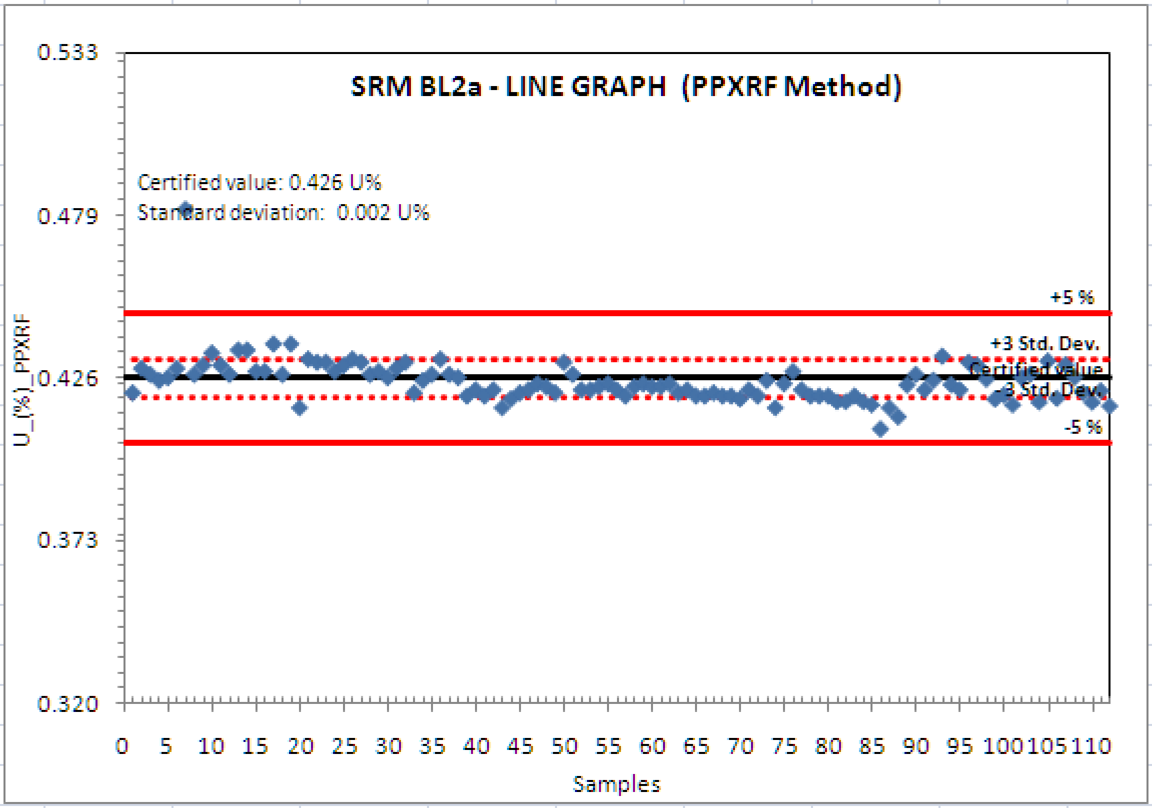


Figure 11.6. Analyses of SRM BL2a (0.426 Percent Uranium) by pp-XRF

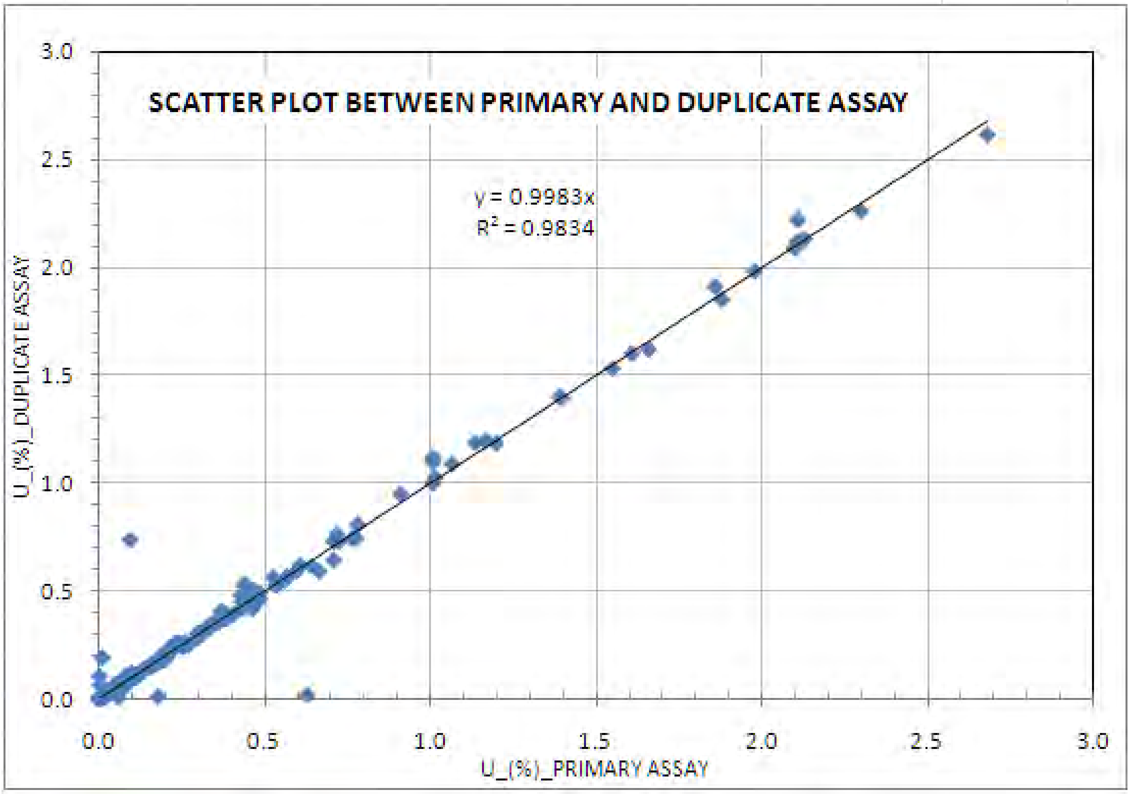


Figure 11.7. Scatter Plot of Duplicate Samples Analyses

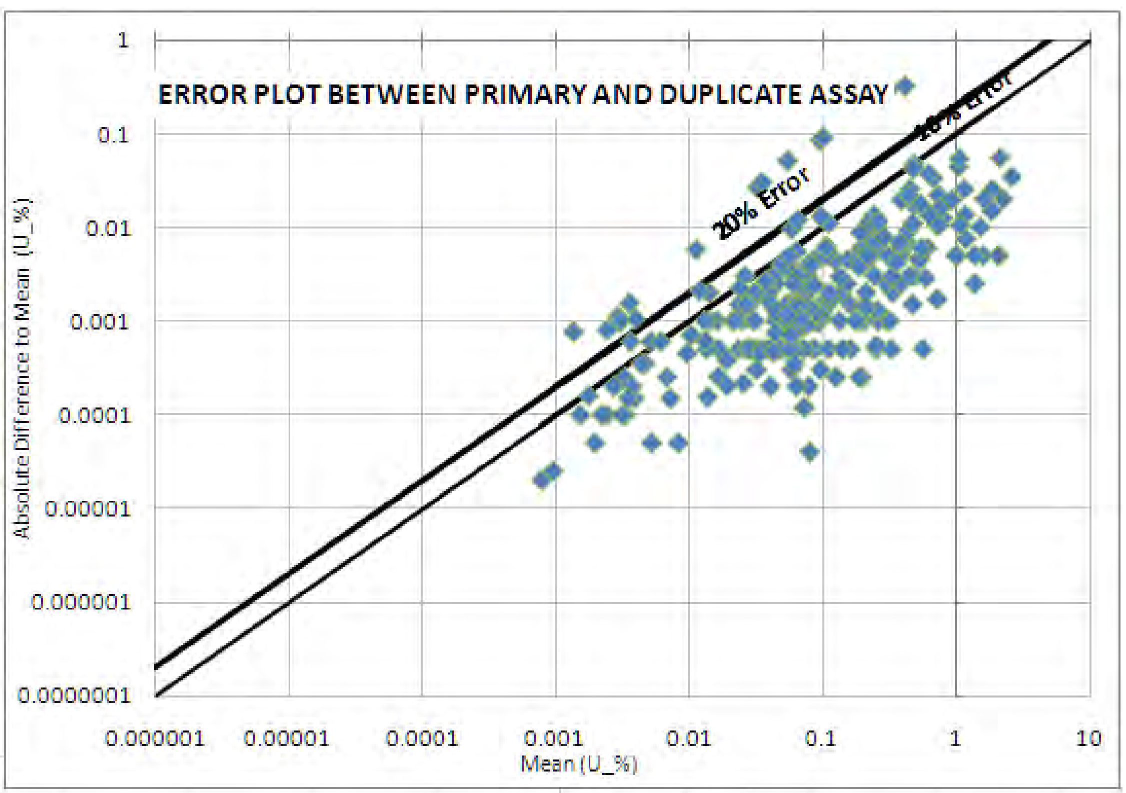


Figure 11.8. Error Plot of Duplicate Samples Analyses

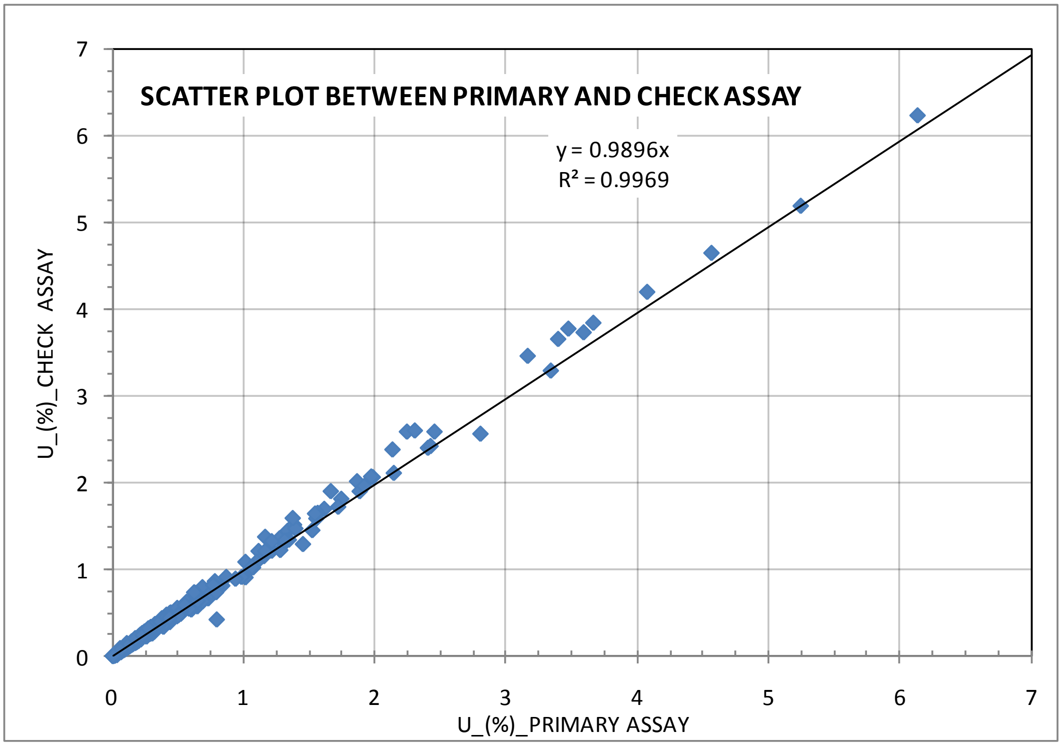


Figure 11.9. Scatter Plot of Check Assays

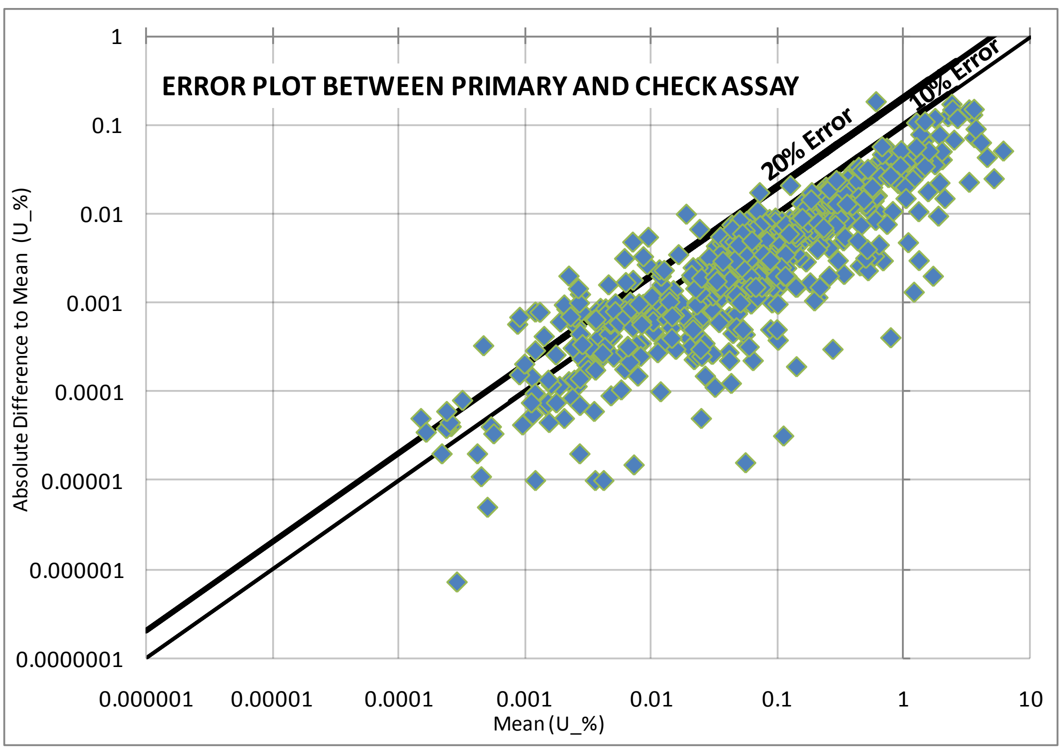


Figure 11.10. Error Plot of Check Assays

11.4.4 Blanks

To generate field blank samples, EUU uses half core from non-mineralized rock. Achieving true blank assay is a challenge, and the result is a very low grade material, but can sometimes be higher than expected in the 5 to 10 ppm grade range. Field blanks used at Kuriskova are derived from andesite from EUU's Kremnica Gold Project, located elsewhere in Slovakia, and from non-mineralized core from Kuriskova deposit. Blank analytical results are shown in Figure 11.11.

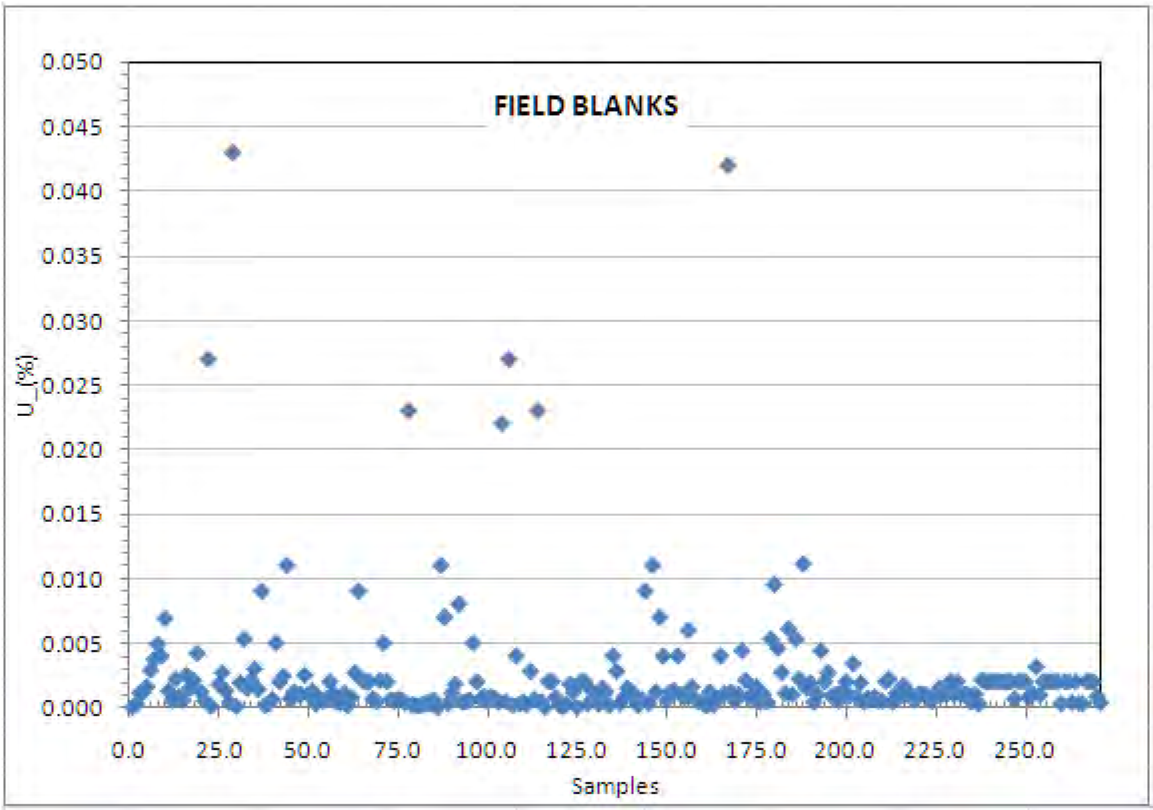


Figure 11.11. Plot of Blank Sample Analyses

EUU has also performed QA/QC on the sample preparation process to ensure laboratory grinding procedures meet the laboratory specification, as shown in Figure 11.12. This was done for all crushing and grinding size fractions with satisfactory results.

Grind size checks were done to examine the homogeneity of pulp prepared by the laboratory and was performed for one in 50 samples with 6.35 and 2 mm screens (75 percent of the weight of sample should pass through the specified screen), and one in 20 samples were checked with a 150 and a 106 micron screen (90 percent of the weight of the sample should pass through the specified screen).

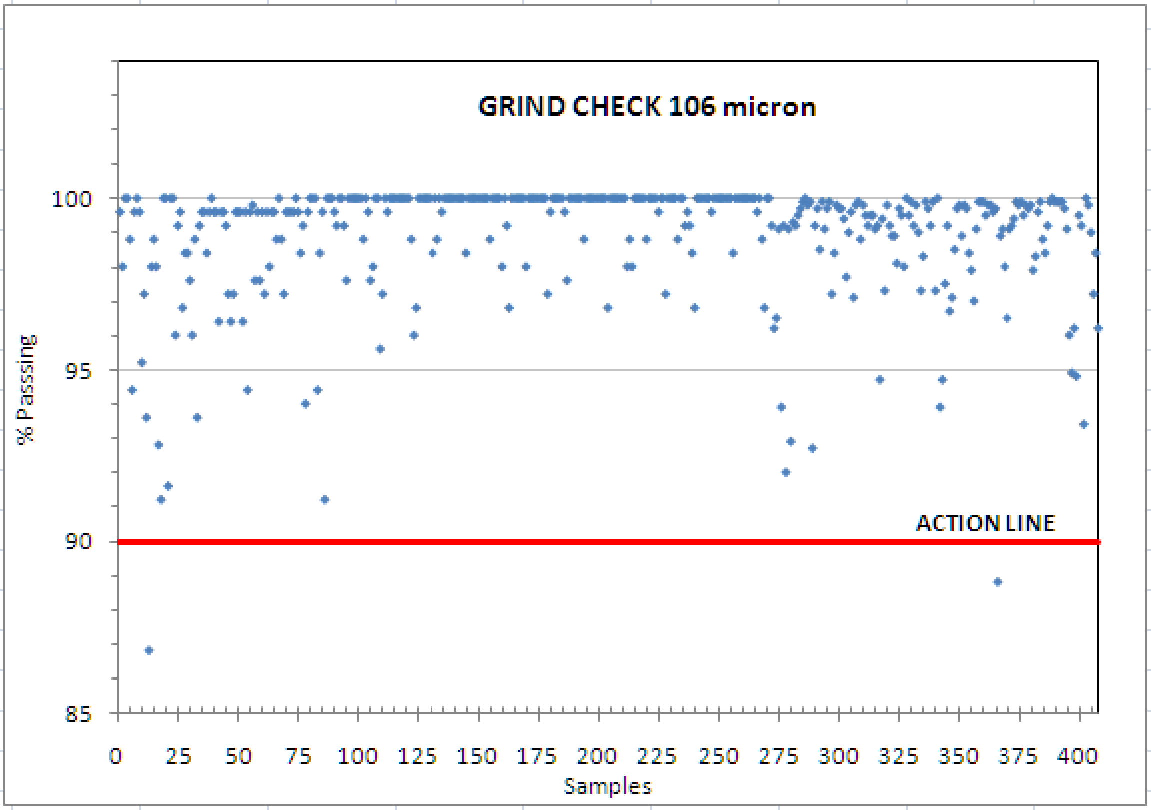


Figure 11.12. Line Plot of Grind Size Checks

Minimal issues were noted, and the QA program in place allowed for re-assay of those sample batches for which unacceptable results were noted in QC samples. This occurred two times since December 2007, and two complete sample batch submissions were re-analyzed

EUU has documented QA/QC reports, communication with laboratories as action taken, and all the relevant QA/QC data for each batch since 2007. Sample tracking and quality assessments on control samples have been carried out by a dedicated geologist at Ludovika in charge of QA/QC.

SRK in 2008 and 2009 concluded that EUU has addressed analytical procedures and determined the best analytical method for use at Kuriskova. In addition, the QA/QC procedures in place have verified the 2005 to 2007 assay data and are providing an effective means to generate the best possible assay database for the Project. SRK deems the QA/QC procedures to be very good and sufficient to support the Project database.

11.5 Tetra Tech Review and Comments

Of 8,112 samples analyzed from 2007 to 2011 EUU has implemented an additional 2,318 QA/QC samples, which account for 28.6 percent of the original amount of samples tested. The use of 28.6 percent QA/QC samples is well within standard industry practice for QA/QC programs and most importantly is an adequate population size to facilitate meaningful data review. Tetra Tech finds EUU’s QA/QC analysis through defined “action lines” to be satisfactory and effective for identifying necessary re-testing. Tetra Tech concludes EUU’s QA/QC procedures to be prudent and a comprehensive system for review of analytical laboratory results.

12.0 DATA VERIFICATION

In 2007, EUU initiated detailed data verification of historic hole gamma data and checking of equilibrium between eU% by gamma and percent uranium by chemical assay. Data capturing and data verification are done by a team of three staff in Slovakia under supervision of Mr. Ravi Sharma, Resource Manager for EUU. EUU also reviews and documents gamma logging and calibration procedures. The details of data verification work carried out re documented for an audit trail. Before each resource model is created, EUU carries out detailed verification of data, including closed can analysis for equilibrium analysis on new drilling data and verification of assay methods by comparing QA/QC results of different analytical methods. EUU has a detailed data verification and documentation procedure in place.

Data verification by EUU consists of:

- Double entry of data for eU percent from historical drill hole files.
- Confirmation of drilling results from historical to current and from year to year.
- Equilibrium measurements.
- Correspondence of multiple assay methods for percent uranium.
- The rigorous QC program as described in Section 11.2 of this report.
- Verification of the consistency of formulas and processes used during calculation of eU% for historical holes.
- Each data capture from historic holes was manually checked for input/output error and verification of data from historical records in archive.

For the historical drill holes that have gamma-only eU% data, EUU has verified the gamma log conversions data. EUU has re-digitized the graphical logs, re-output a data table of 0.1 m eU% data, and compared that data with the original 0.1 m interval eU% data, essentially a double entry check of the data. For the 17 historical gamma-only holes that have only graphical logs, all logs were digitized and output as 0.1 m data tables, then re-plotted graphically, and checked against the original plots. EUU has randomly selected six drill holes and re-run the software to calculate eU% data and confirmed the output with the original data values. Three holes were re-logged, and the results were compared to verify the accuracy of the logging.

EUU drilling campaigns have essentially replicated the mineralization described by historical work, as shown by similar mean grades for drilling campaigns.

A state of equilibrium, or the ratio of chemical uranium to radiometric uranium (U/eU) for the same sample volume, is best done on core or reverse circulation (RC) samples. A common method is called "closed can" radiometric analysis, where a sample is allowed to equilibrate in a canister for approximately three to four times the half-life of radon gas, and the radiometric eU, therefore, is back-calculated from the radon measurements and compared to an ICP or XRF analysis for the sample. The state of equilibrium or disequilibrium is not an issue for a resource model based on chemical assays (XRF or ICP), but is important when the data are eU determinations from gamma logs. Nonetheless, to use any of the EUU gamma-log determined eU% data in future deposit modeling requires this data verification. The state of equilibrium was investigated by EUU by sending 428 samples of coarse reject material to Energy Labs in Wyoming for closed can radiometric analysis. Comparison of U_3O_8 (ICP) and eU_3O_8 (closed can gamma) are shown for the samples in Figure 12.1 and Figure 12.2 and indicate a relative state of equilibrium exists (no significant bias high or low for eU). The Scatter Plot between U_3O_8 and

eU_3O_8 (Figure 12.2) indicates a slight (6 percent) low bias of radiometric analysis compared to chemical (ICP) analysis; however, this is within an acceptable range for a relatively small sample population, analyzed across a broad grade range. The apparent low bias of radiometric analysis is not particularly relevant since only about 27 holes out of 151 of the data in the resource model are radiometric analysis. Figure 12.3 and Figure 12.4 show the cumulative frequency plot of all assays globally, irrespective of deposit sub-zone, for the deposit. It shows a break in the curve at 0.04 percent uranium, which represents the lower end of a primary grade range of 0.04 percent to 0.37 percent and confirms the use of 0.03 percent uranium to define the mineralized envelopes (wireframe shapes) is reasonable.

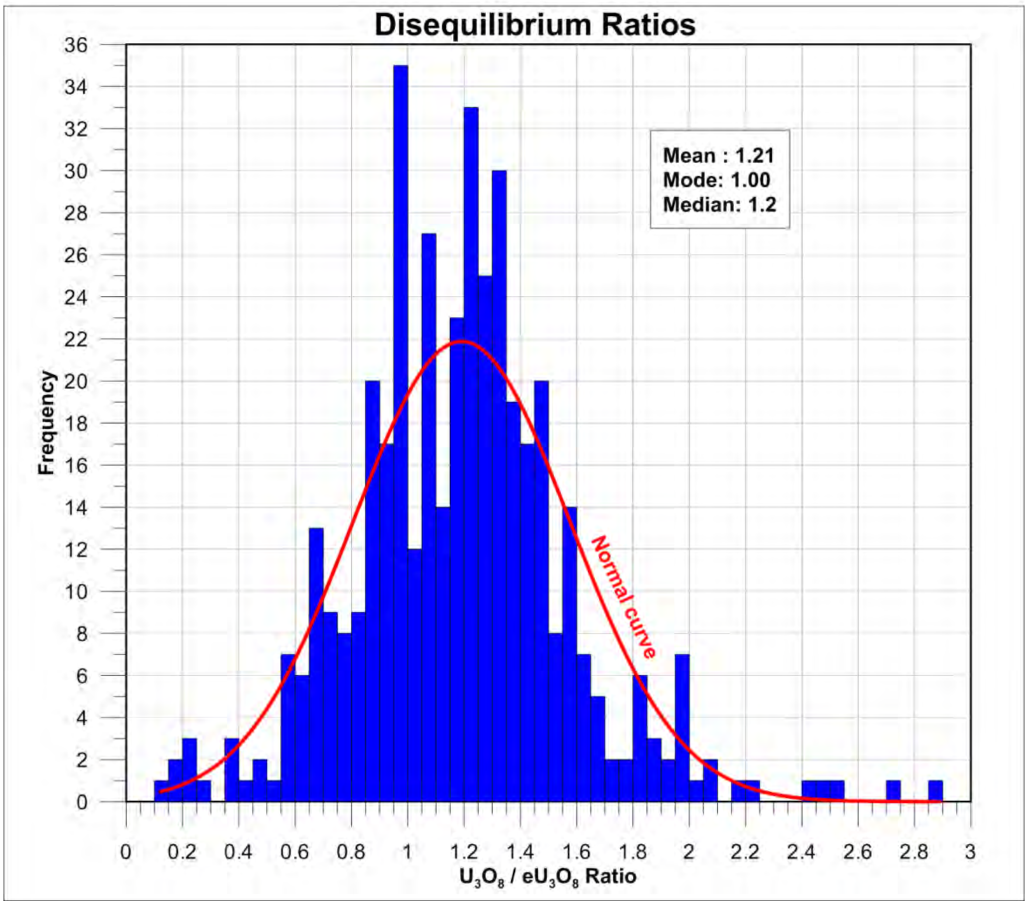


Figure 12.1. Disequilibrium Analysis (Tournigan, April 2011)

From the above histogram samples 47025 and 47175 were excluded in order to avoid relatively larger plot for two values with U_3O_8 / e U_3O_8 ratio greater than three.

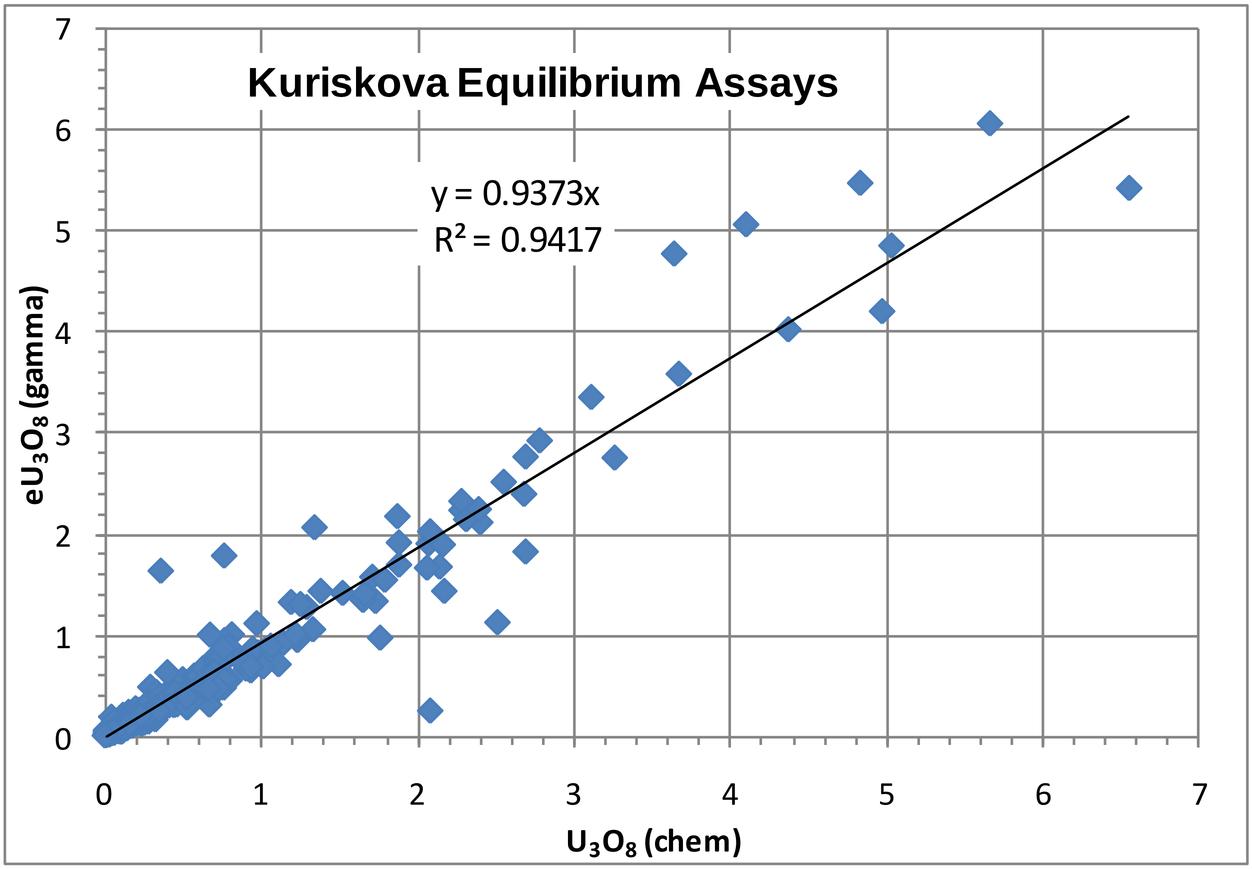
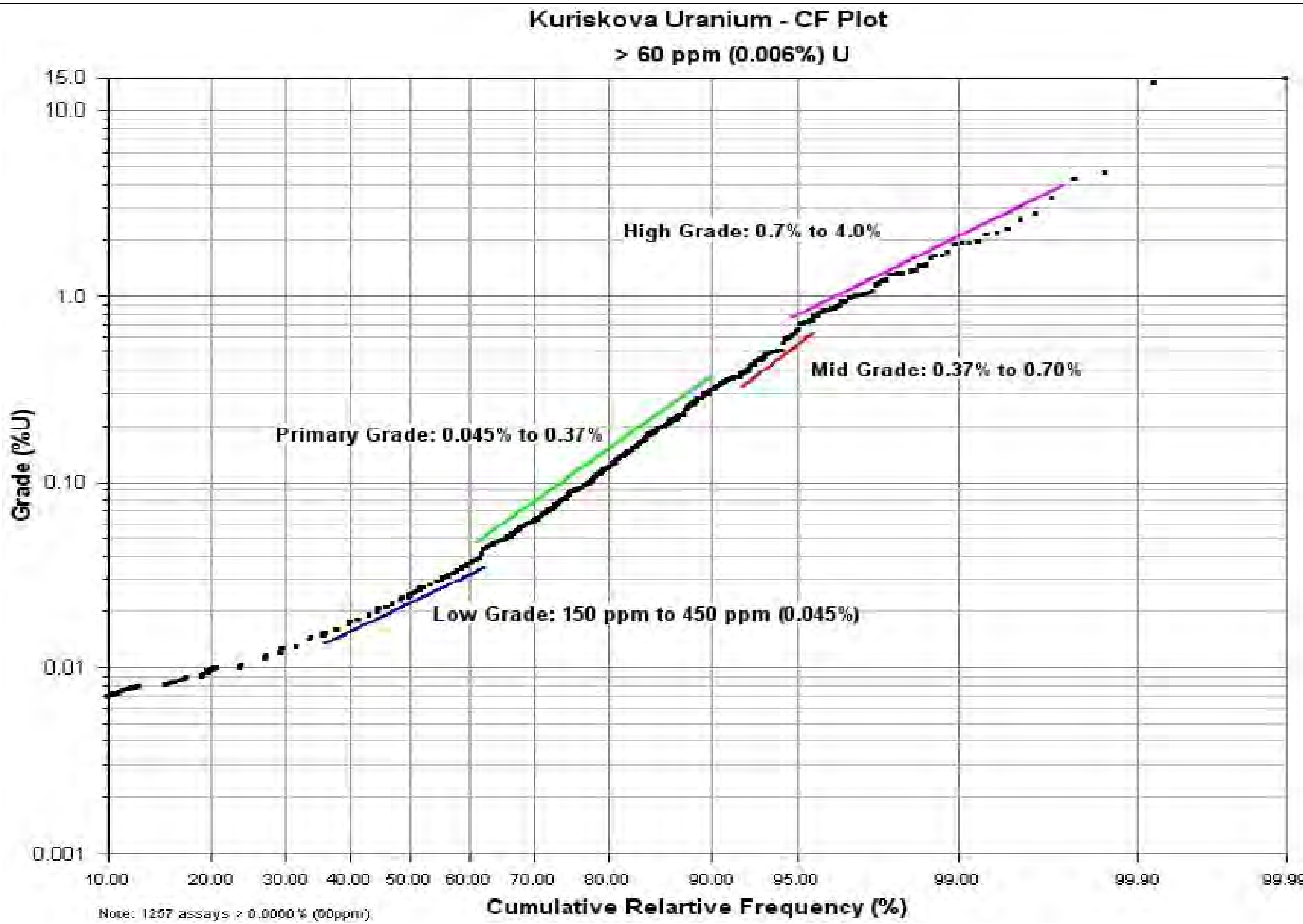
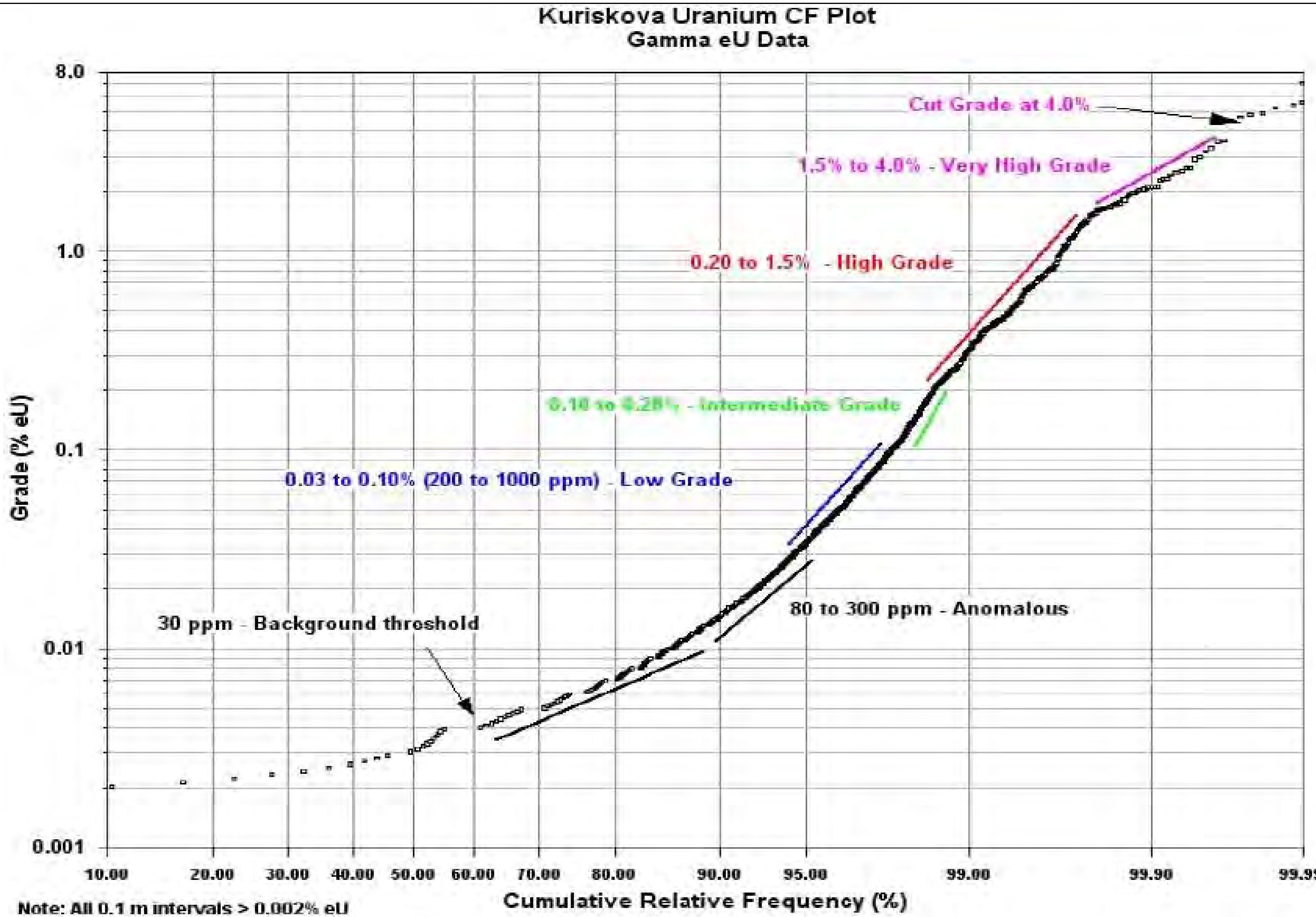


Figure 12.2. Scatter Plot Between U_3O_8 and eU_3O_8



Issued by:	Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, CO 80401 (303) 217-5700 (303) 217-5705 fax	***European Uranium Resources Ltd.***	Fig 12.3.jpeg	**Figure 12.3** **Cumulative Frequency Plot of U% Assay Data Greater Than 0.006% (SRK 2008)**
	Project: Kuriskova Uranium Project	Project Number: 114-310990	
	Project Location: Slovak Republic	Date of Issue: June 2011	

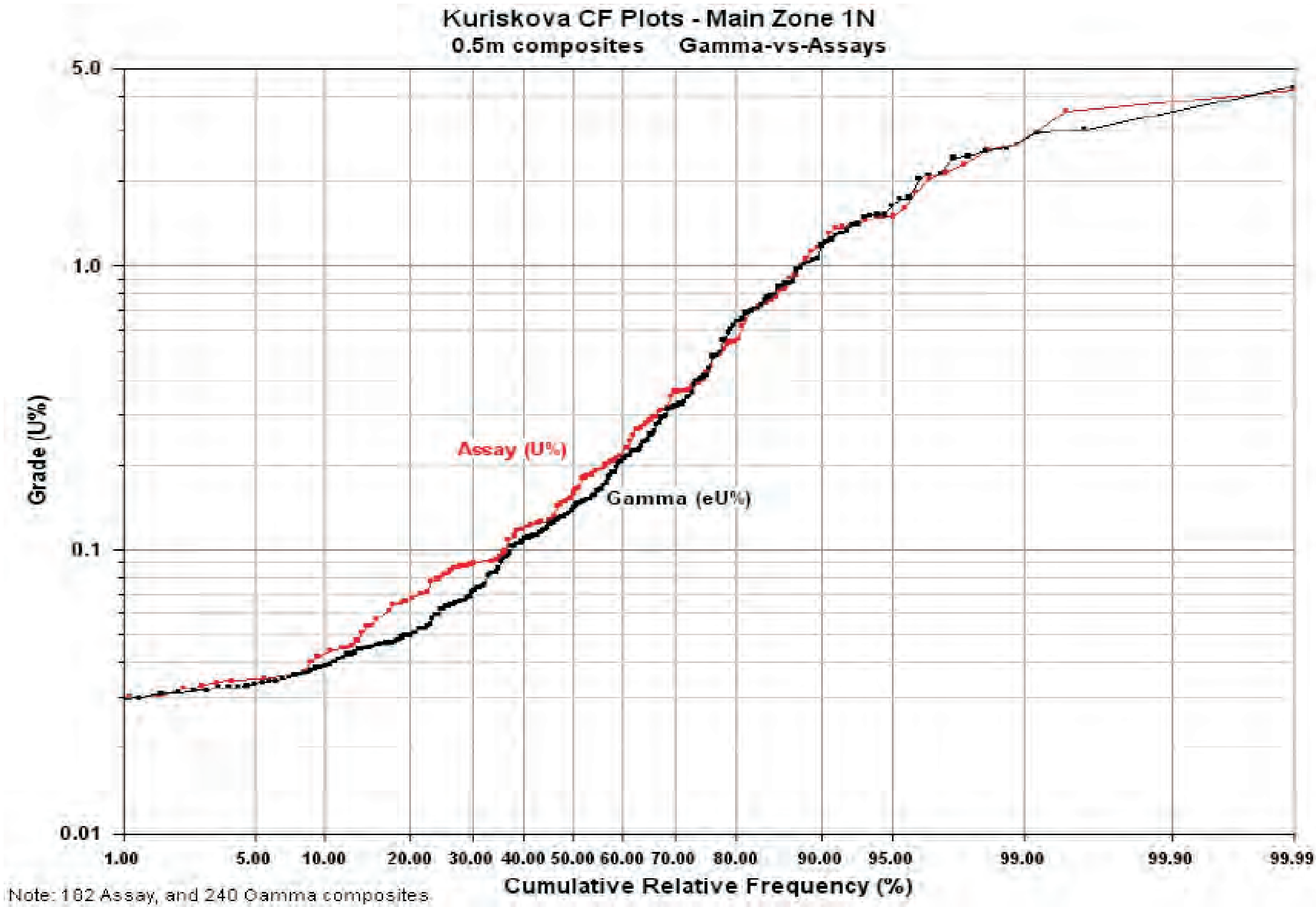


Issued by:	Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, CO 80401 (303) 217-5700 (303) 217-5705 fax	***European Uranium Resources Ltd.***	Fig 12.4.jpeg	**Figure 12.4**
	Project: Kuriskova Uranium Project	Project Number: 114-310990	**Cumulative Frequency Plot eU% Gamma Data**
	Project Location: Slovak Republic	Date of Issue: June 2011	**Greater Than 0.002% (SRK 2008/2009)**

The following are noted for Figure 12.5:

- Both Cumulative Frequency (CF) curves are similar in slope and nearly overlay each other, suggesting the two data sets are representing the same grade distribution of data, within a reasonable margin of error for a minimal dataset (approximately 200 composites);
- The gamma CF curve is smoother in part due to the larger volume support of gamma readings; hence, the grade smoothing effects of gamma probes relative to assays;
- There is a slight understating of grade by gamma relative to assays in the grade ranges of 0.04 percent to 0.35 percent uranium; and
- The gamma data and the assay data are separately representing essentially the same volume of rock (with only minor differences).
- Three important conclusions can be made from the gamma and assay data:
- The gamma data provides data verification for the assay data, or an independent confirmation that the assay data are representative.
- The historical gamma-only drill holes that are included in the drill hole database are considered acceptable data, if not somewhat conservative.
- Therefore, the mix of gamma and assay eU% data is acceptable for the work thus far at Kuriskova.

The detail data verification by SRK in 2008 to 2009 is detailed in the Technical Report prepared for Tournigan by SRK Consulting dated April 16, 2009.



Note: U% and eU% composites greater than 0.03% are used for comparison

(SRK, June 2008, April 2009)

Issued by: TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	Prepared for: *European Uranium Resources Ltd.*	File Name: Fig12.5.cdr	**Figure 12.5** **Kuriskova NORTH - Gamma Versus Assay**
	Project: Kuriskova Uranium Project	Project Number: 114-310990	
	Project Location: Slovak Republic, Europe	Date of Issue: 05/05/2010	

12.1 Composite Data Verification

Additional statistical analysis confirms that 0.5 m is a reasonable composite interval to use for resource estimation, as shown in Figure 12.1 and Figure 12.2.

The database is, by necessity, a mix of both gamma eU% and assay percent uranium data; 1990 drill holes used are gamma-only drill holes and are necessary to include in the resource database. In addition, there are eight drill holes from 2008 in the database for which assays are not yet complete. For the 2008 drill holes, gamma eU% data were used. SRK does not recommend mixing gamma derived eU data with assay derived uranium data in a resource database; therefore, SRK examined the relationship of gamma to assay data. However, this cannot be done on a hole-to-hole basis, as there are no true twin holes, and twin hole data analysis has limitations. An interval-to-interval comparison of percent uranium and eU% within holes where both values are available is problematic not only because the "from-to" intervals are different, but more importantly the geometric support of the samples differ considerably. Gamma eU% values are derived from instruments (downhole probes) that measure orders of magnitude larger volumes of material than can be measured by XRF or ICP for the samples derived from half core. For Kuriskova, the best method of comparison is to examine the grade distributions of each within the Main Zone (Zone 1 North), where the bulk (63 percent) of the total resource is located.

The gamma data are 0.5 m composites from within the gamma wireframe for the Main Zone 1 North area only. The assay data are the 0.5 m composites from within the assay wireframe for Main Zone 1 North area; noting that two wireframes were constructed independently, one for gamma and one for assays. A CF plot of the grade distributions are shown in Figure 12.5, and the basic statistics are shown in Table 12.1.

Table 12.1. Comparison of Assay and Gamma Composite Data – Main Zone 1 North

Zone North	Assay Composites	Gamma Composites
Number	182	240
Mean value	0.416	0.404
Std Dev.	0.63	0.62
C.V.	1.51	1.54

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction and Historical Metallurgical Development

Multiple metallurgical testwork programs commencing in October 1993 have yielded substantial information about the Kuriskova deposit including insight into the physical and chemical properties of the contained uranium and molybdenum mineralization and its response to various metallurgical extraction and recovery techniques. The types of testwork performed on this material since 1993 includes comminution tests, flotation tests, thickening and filtration tests, acid and carbonate leach tests, and tests performed using ion exchange (IX) and direct precipitation to evaluate uranium and molybdenum recovery techniques.

Testwork was first performed by the MEGA Laboratory in the Czech Republic in October 1993. Work included flotation, acid leaching, carbonate leaching, and precipitation of molybdenum with sodium sulfide. The work was conducted to professional standards; however, the information generated in the area of carbonate leaching has been superseded by results on more concise carbonate leach testwork performed at Hazen Research (HRI).

Resource Development Inc. (RDi) performed tests on Kuriskova material reported in November 2009. Testwork included mineralogical examination, grind studies, flotation tests, and acid and carbonate leaching of whole ore and flotation tailings. Again, the information generated in the area of carbonate leaching has been superseded by results on more concise carbonate leach testwork performed at HRI.

HRI performed three sets of tests, each of which contributed substantially to understanding the Kuriskova ores and metallurgical parameters associated therewith. Ultimately, the HRI testwork results formed the criteria basis upon which the process design for the 600 tpd underground process facility (UPF) were developed. The select process combines comminution, carbonate leaching, and direct sodium diuranate (SDU) precipitation in conjunction with a releach and reprecipitation of a uranium peroxide yellowcake concentrate and a molybdenum-sulfide concentrate. Tetra Tech considers the Kuriskova composite samples for the HRI test programs to be representative for the Kuriskova PFS-level study.

Commodas Ultrasort performed one set of radiometric sorting tests in March 2011 on 237 Kuriskova mineral samples selected by HRI. It was determined that radiometric sorting would not be compatible due to the amount of fines generated by using a road header as the primary mining tool.

13.2 Composite Sample Preparation

The first metallurgical testing by MEGA in October 1993 used three composite samples, named Samples 16, 17, and 18, were obtained from two drill holes. The intervals and hole locations used to create each composite are detailed within the same report. Assays of each composite are included below in Table 13.1.

Table 13.1. MEGA Composite Sample Grade

Element	Sample No.		
	16	17	18
U, wt%	1.87	0.075	0.311
Mo, wt%	0.45	0.0066	0.086

RDi performed hydrometallurgical analysis in 2009 on samples provided by EUU. The tests utilized three composite samples taken from drill holes of the main deposit. The assays associated with each composite are included below in Table 13.2.

Table 13.2. RDi Composite Sample Grade

Element	Composite No.		
	1	2	3
U, wt%	2.18	0.5	0.2
Mo, wt%	0.454	0.228	0.043
S, wt%	2.18	1.11	0.46

The specific locations of the relevant drill holes, as well as the intervals used to make up each composite, are detailed within the same report. These samples were not used for any further testing.

Metallurgical testing was also performed on mineral samples received by HRI on May 21, 2010 provided by EUU. These samples were pre-identified as Met Composite 1, Met Composite 2, and Met Composite 3. Assays of the composite samples used in the Hazen testwork below in Table 13.3.

Table 13.3. Composites Used in the Hazen Metallurgical Testing

Composite No.	1	2	3
U_3O_8 %	0.163	0.422	0.628
Mo %	0.020	0.074	0.435
Sulfide %	0.47	0.85	1.48
Carbonate %	13.6	13.2	3.86
Major Uranium Minerals	Uraninite	Uraninite	Uraninite, Coffinite
Sulfide Minerals	Molybdenite/Pyrite	Molybdenite/Pyrite	Molybdenite/Pyrite
Major Gangue Minerals	Quartz/Dolomite	Quartz/Dolomite	Quartz

Samples of Composite 1 and Composite 2 were used for both metallurgical testing and sample characterization. Composite 2 represents a sulfur rich mineral with high levels of uranium and molybdenum, while Composite 1 represents a sulfur rich mineral with low uranium and molybdenum contents. Composite 3 was used only for sample characterization as it was deemed unrepresentative of the overall deposit for purposes of evaluating comminution, leaching, and/or precipitation parameters.

HRI received 22 additional drill core samples in August 2010. These individual samples were composited, and SMC Testing was performed to obtain the JKSimMet grinding parameters.

13.3 Key Metallurgical Results

Results from the HRI metallurgical test programs were used as the design basis for the Kuriskova process plant indicated that conventional processing methods could be used to obtain high extractions of uranium and molybdenum at a grind size of 200 mesh and alkaline pressure oxidation (APOX or POX) leaching (Hazen, 2011).

Testwork results indicate that the Kuriskova deposit responds well to carbonate leaching, particularly APOX leaching. The testwork showed optimum uranium extraction occurs at 200°C at 100 pounds per square inch (psia) oxygen overpressure within a two-hour retention time. It is estimated a sodium carbonate and bicarbonate addition of 69 and 23 grams per liter (g/L) in slurry consisting of 40 percent solids will be adequate for favorable leach extractions, although additional testwork is recommended to optimize these values. The tests indicate that leach extractions of 94 percent uranium and 87 percent molybdenum can be achieved at these conditions.

Precipitation testwork showed that direct SDU precipitation recoveries derived through caustic precipitation were higher than other methods, such as IX. Precipitation recoveries of 96 percent were achieved in the testwork, producing a high grade SDU cake. Although no specific re-precipitation testwork was performed, the technology associated with re-processing of SDU in a low pH extraction process is well known and is accepted for use in the process circuit. Subsequent re-precipitation of uranium as uranium peroxide will produce a higher purity product.

Additional precipitation testwork indicated that direct precipitation of molybdenum from the SDU filtrate is the best method of recovery when compared to methods, such as IX. Recoveries of 99 percent of molybdenum from the filtrate were achieved in the testwork with a corresponding grade of 10.6 percent molybdenum. Additional testwork, including the evaluation of alternative molybdenum extraction methods and examination of molybdenum recovery techniques, is recommended in subsequent studies.

Based on the HRI test work, it is estimated that overall uranium and molybdenum recoveries of 92 percent and 86.8 percent, respectively, should be achievable. Results of the HRI testwork led to development of the design criteria for the Kuriskova process plant shown in Table 13.4.

Table 13.4. Major Process Design Criteria

Selected Pressure Leaching Parameters	Units	
Leach Feed Solids Pulp Density	wt%	40
Leach Feed Grind Size, P_{80}	micron	75
Leach Feed Grind Size, P_{80}	mesh	200
Leach Temperature	°C	200
Leach Retention Time	hrs	2
Oxygen Overpressure	psia	100
Sodium Carbonate Dose	g/L	69
Leach Feed Addition Ratio	kg/t	94
Sodium Bicarbonate Dose	g/L	23
Leach Feed Addition Ratio	kg/t	31
Uranium Leached	%	94
Molybdenum Leached	%	87
Sulfur Oxidation	%	100

13.4 Comminution

Results from HRI comminution tests show that the Kuriskova ores are of medium hardness for Bond Ball Mill Work Index's and SMC Parameters as presented in Table 13.5 and Table 13.6.

Table 13.5. Bond Work Index (BW_i)

Sample	Feed F_{80} micron	Product P_{80} micron	BW_i kWh/t
Composite 1	1,452	111	12.8
Composite 2	1,443	110	13.2

Table 13.6. Summary of SMC Parameters

SG	A	b	A x b	DW_i, kWh/m^3	DW_i, %	M_{ia}, kWh/t	M_{ih}, kWh/t	M_{ic}, kWh/t	t_a
2.77	49.6	0.85	42.2	6.61	62	18.8	13.8	7.2	0.39

Subsequent leach tests revealed uranium extraction appears to be at a maximum at a grind size of P_{80} of 300 mesh under POX conditions. While 300 mesh produces slightly better extractions of uranium than at 200 mesh, the increase in power requirements as well as the capital cost of larger equipment required to produce a 300 mesh product outweigh any benefit to doing so. As such, a target grind size of P_{80} of 200 mesh was selected for the design criteria.

13.5 Thickening

HRI completed an investigation of the settling characteristics of the POX leach slurry was , the results of which can be seen in Table 13.7.

Table 13.7. Summary of Thickening Characteristics

Settling Time (hrs)	Feed Solids (%)	Terminal Pump Solids (%)	Calculated Thickener Unit Area (m^2/tpd)	Initial Settling Rate (m/h)	Flocculant Dose (g/t)
23	14.6	42.4	0.15	1.52	46
23	13.3	39.7	0.17	1.51	34

These unit areas are comparable with those encountered in other uranium alkaline leach operations. It is recommended that more detailed investigation of settling characteristics for both the ground and leached slurries be performed in future testwork.

13.6 Ore Leaching

Both atmospheric and POX leaching processes were tested by HRI. Based on the test results, the POX leaching route was selected as the preferred method for the Kuriskova process plant as described in the subsequent sections.

13.6.1 Atmospheric Alkaline Leaching

HRI test work showed that the Kuriskova ore to be amenable to a carbonate leach. Significant uranium extraction occurs under atmospheric pressure provided adequate amounts of sodium carbonate and sodium bicarbonate are present. While the initial tests indicated that 90 percent and greater uranium extractions could be achieved under atmospheric conditions, the finer grind size and retention time requirements in order to achieve this recovery are impractical.

Subsequent tests focused on duplicating the intended operating conditions and opportunities for leaching expected to occur in the proposed mill circuit and slurry conditioning tank. Results indicate that approximately 10 percent of the uranium would leach at atmospheric temperature and pressure prior to subjecting the ore to the autoclave with equivalent reagent additions. Given the costs associated with increased equipment size, operating costs associated with finer grinding, and the lesser recovery of molybdenum, atmospheric leaching was not explored further as the primary means of recovery.

13.6.2 Carbonate Pressure Oxidation Leaching

Significant testwork occurred over the course of HRI's programs focusing on the optimization of POX alkaline leach parameters. The testwork demonstrated that 94 percent of the uranium and 87 percent of the molybdenum can be successfully extracted from the feed mineralization into the leach solution under these conditions.

Preliminary testwork issued in July 2011 investigated the parameters necessary to achieve optimum uranium extraction under POX conditions, with the secondary objective of optimizing molybdenum extraction. Leach parameters investigated during the testwork included grind size, leach temperature, retention time, oxygen overpressure, and sodium carbonate reagent levels. These tests did not include the introduction of any sodium bicarbonate. Later HRI testwork issued in December 2011 investigated the reagent addition requirements including the addition of sodium bicarbonate, as well as examining recoveries under milder operating conditions.

13.6.3 Feed Grind Size

As detailed above in Section 13.4 Comminution section, a grind size of 200 mesh was selected. The leach characteristics of uranium are not significantly different at 200 mesh than at 300 mesh for POX conditions, thus any subsequent tests utilizing 300 mesh material are still reasonably analogous to those obtained at the coarser grind.

13.6.4 Temperature

Design criteria for target leach temperature were chosen to be 200°C as this temperature was determined to maximum uranium extraction. This test also indicates that the kinetic effects observed in the atmospheric leach tests were sufficiently overcome such that high extractions of both uranium and molybdenum can be achieved within comparatively short retention times. Maximum molybdenum recoveries were achieved at 210°C. The slight increase in molybdenum extraction is insufficient to warrant increasing POX temperature given the corresponding drop in uranium extraction, thus 200°C was selected as the design temperature.

13.6.5 Retention Time

The HRI July testwork results indicated uranium extraction continues to increase up to a maximum within two hours; thus, two hours was selected for the design criteria. These tests also reveal sulfur oxidation occurs very rapidly.

13.6.6 Oxygen Overpressure

The July 2011 testwork examined two different overpressures at two different temperatures, namely 200°C and 210°C. Given an optimum temperature of 200°C has been thoroughly demonstrated to be superior to 210°C on multiple points of consideration, only the results of the trials performed at 200°C are shown in Table 13.8.

Table 13.8. Oxygen Overpressure Versus Extraction and Oxidation

O_2 Overpressure	% U Extraction		% Mo Extraction		% S Oxidation	
psia	Comp 1	Comp 2	Comp 1	Comp 2	Comp 1	Comp 2
75	88.6	85.6	84.1	86.8	95.4	89.8
100	93.5	93.9	85.1	89.9	99.5	96

At 200°C after two hours; 69 grams per liter of sodium carbonate, no bicarbonate addition, and 20 percent solids pulp density

Oxygen demand in the autoclave is driven both by uranium and sulfur content. Table 13.8 shows that increasing oxygen levels increases uranium and molybdenum extractions, as well as sulfur oxidation. The higher uranium leach levels exhibited by Composite 2, representing high grade ore, justifies the choice of 100 psia O_2 overpressure for the design criteria.

13.6.7 Sodium Carbonate/Bicarbonate Addition

Ultimately, the purpose of carbonate and bicarbonate addition is twofold. First, to provide sufficient additions to meet the stoichiometric demands required for successful leaching. Second is to keep the pH levels of the autoclave leach solution between 9 and 10.5, as any pH levels higher than this will result in the re-precipitation of uranium as SDU. Optimized addition levels and dosages of carbonate will vary over the life in response to changes in the mineral content. HRI July 2011 testwork examined the effect of sodium carbonate addition levels on the extraction of uranium and molybdenum. Results suggest an optimum addition level of 69 g/L of sodium carbonate at 20 percent pulp solids density, corresponding to an addition ratio of 277 kilograms per tonne (kg/t) ore. These results are higher than the 40 g/L often encountered in many similar uranium operations. This may be explained by the lack of sodium bicarbonate addition, as the tests relied on the high sulfur content to provide the necessary bicarbonate for the uranium leach reaction.

Later testwork issued in December 2011 re-examined the reagent addition requirements. Unlike the July tests, the December tests included the addition of sodium bicarbonate. These tests revealed that lower addition ratios than used in the July report can be used; however, there is still a minimum threshold driven by the uranium and sulfur content in the ore. Differences between the July and December leach test results are non-trivial in that direct comparisons of results may not be drawn. However, the combined effects of the December testwork parameters produced only slightly reduced recoveries compared to the July testwork; thus, the results are still relevant and may be used to conservatively estimate recoveries.

The tests demonstrate relatively high uranium extractions ranging from 88 percent to 96 percent at reagent addition ratios of 60 kg/t and 34.5 kg/t of sodium carbonate and sodium bicarbonate respectively with atmospheric leaching prior to POX. These trials produced an average uranium extraction of 92.3 percent. The inclusion of atmospheric leaching is representative of the intended mill process, as the ore will spend up to 16 hours in a conditioning tank in which it will be agitated with atmospheric air prior to the autoclave.

Optimal reagent addition levels have not been determined at this time. It has been assumed for costing purposes that 69 g of sodium carbonate (Na_2CO_3) and 23 g of sodium carbonate ($NaHCO_3$) will be needed per liter in a pulp consisting of 40 percent solids. This reagent concentration corresponds to a leach feed ratio of 94 kg/t and 31 kg/t of sodium carbonate and bicarbonate respectively.

Further optimization of the sodium carbonate and bicarbonate addition levels is recommended in future test work.

13.7 Filtration

Under the proposed process flow, the POX leach liquor would be separated from the leach residue via the use of a horizontal belt vacuum filtration unit. No testwork has been conducted yet that characterizes the filtration characteristics of the leach residue; however, this method has been used successfully for other uranium operations. Although filtration has not been characterized, the POX filtrate was filtered in each trial with no mention that filtration was particularly problematic. Characterization of filtration properties will be studied in later testwork.

13.8 Uranium Precipitation Reactions and Mechanisms

13.8.1 Uranium Precipitation and Recovery

The initial July 2011 testwork evaluated multiple avenues for the recovery and precipitation of uranium from the POX leach liquor. It was determined that the best method would be to directly precipitate uranium for the POX liquor as sodium diuranate, wash the resulting cake, repulp and releach the SDU cake at an adjusted pH, and reprecipitate the uranium as uranium peroxide. While the method of repulping the SDU cake has not been tested to date, it is believed that high precipitation recoveries of 99 percent of the uranium in the leach liquor are possible with recirculation of seed for precipitation. This would result in a predicted overall uranium recovery for the mill circuit of 92 percent.

13.8.2 Recovery of Uranium via Strong-Base IX

One of the methods examined in HRI's July 2011 testwork was the use of strong-base IX to capture the uranium from the carbonate leach solution. These tests performed poorly, resulting in low loading values of 14 g uranium/L, the use of which would have resulted in large column volume requirements. Carbonate and sulfate species in the leach liquor are believed to be the cause of poor IX loading. By comparison, an acceptable loading capacity is considered to be 50 g uranium/L or more for practical use. IX as a method for recovery was eliminated from consideration when compared to the results produced by other methods.

13.8.3 Direct Precipitation of SDU from POX Filtrate

Subsequent HRI testwork in January 2012 revealed 96 percent of the uranium in the pregnant solution could be directly precipitated as SDU with the addition of 4 g/L excess caustic soda (NaOH). This precipitation extent will likely be higher during operations where a targeted addition of 5 g/L caustic soda is used in conjunction with recycling a portion of the SDU precipitate back into the precipitation circuit as a seed for the reaction. The combined effects of both modifications could boost SDU precipitation to 99 percent. It was observed that SDU precipitation does not result in appreciable levels of co-precipitation of molybdenum, thus SDU precipitation serves as an effective method of separating the two products. Precipitation resulted in the production of SDU cake containing 63.4 percent uranium which meets the standards required for uranium content in a yellow cake product. Aside from sodium, which is to be expected in an SDU cake, phosphorous was the only constituent in the SDU cake that exceeded the penalty threshold for a yellow cake product.

13.8.4 Direct Precipitation of Uranium Peroxide from Pregnant Leach Liquor

Direct precipitation of uranium peroxide from the pregnant leach liquor was also evaluated as a means of recovering uranium in the form of Uranyl Peroxide (UO_4). The resulting precipitate is hydrated uranium peroxide. Initial testwork by HRI in July 2011 focused primarily on precipitating uranium peroxide from IX eluate solutions; although, a single direct precipitation trial was also performed. The direct precipitation trial demonstrated an 89 percent uranium recovery. As the resulting yellow cake product was mixed with the precipitates from IX testwork prior to analysis, correlations to the grade of the direct precipitate were not possible.

The January 2012 testwork also evaluated the direct precipitation of uranium peroxide from the pregnant leach liquor. Similar to the earlier testwork, there was slight occurrence for co-precipitation of molybdenum. Precipitation efficiencies were high, achieving a maximum value of 99.8 percent uranium precipitation, indicating that virtually all uranium reporting to peroxide precipitation from the SDU precipitation circuit can be recovered.

13.8.5 Uranium Peroxide Precipitation from a Re-Leached SDU Cake

No testwork has been performed to date to evaluate the precipitation of uranium peroxide from a re-pulped SDU cake; however, the promising recoveries obtained in peroxide precipitation tests paired with the high grade of the SDU precipitate produced suggests doing so would not be problematic. In the absence of relevant testwork to date, and using Merritt as a guideline, it is assumed that 1.5 times the stoichiometric addition of hydrogen peroxide will be required.

13.9 Molybdenum Precipitation and Recovery

The initial July 2011 testwork evaluated various methods for the recovery and precipitation of molybdenum. It was determined that direct precipitation of molybdenum from an SDU filtrate would be the best method compared to others tested. This method consists of acidifying the SDU filtrate to convert the molybdenum oxide ion to molybdenum sulfate. The subsequent addition of NaHS converts the sulfate to sulfide, resulting in the precipitation of molybdenum tri-sulfide when brought below a pH of approximately 2. This method for the recovery of molybdenum is well known and referred to as the AMAX process. The molybdenum product generated thereby would subsequently be filtered and washed prior to packaging as a final product. Recovery of molybdenum from the leach liquor via IX yielded very poor recoveries of 76 percent in the July 2011 testwork. As such, IX was discarded from consideration as a viable means of recovery.

Precipitation of molybdenum from a liquor composed of the combination of solutions remaining from direct precipitation and IX trials using the AMAX process yielded poor results during the July 2011 testwork with respect to molybdenum product grade 25.7 percent molybdenum, as well as a low recovery of 59 percent. The arsenic levels were relatively high at 1.62 percent, which could prove problematic. Subsequent HRI testwork reported in January 2012 yielded similarly poor results. The grade of the molybdenum product from the SDU filtrate was approximately 10.6 percent. Unlike the previous testwork; however, very high molybdenum recoveries of 99.6 percent were achieved. It is possible that the formation of hydrated salts and other sulfide precipitates could be to blame for the poor molybdenum grade observed in the tests. Additional investigation is necessary to evaluate the optimal recovery method for molybdenum from the leach solution. The suitability of alternate recovery methods such as solvent-extraction (SX) for recovery of molybdenum should also be examined in subsequent test programs.

14.0 RESOURCE ESTIMATE

The mineral resources stated in this section for the Project have been classified according to the CIM Standards on Mineral Resources and Reserves: Definitions and Guidelines (CIM, 2005). Accordingly, the resources have been classified as indicated or inferred. Currently there are no measured mineral resources or mineral reserves defined for the Project.

EUU has conducted exploratory drilling at the Kuriskova Uranium Project since 2005. The 2010-2011 Drill Hole Listing (Table B.6) provides a drilling summary including holes drilled in 2010 to 2011, indicating drilling updated in this report.

This June 2011 resource estimate incorporates the results of 18 diamond drill holes totaling 4,548 m that were drilled between September 2010 and March 2011 subsequent to the last resource estimate of March 24, 2010. The updated estimate also reflects an enhanced understanding of Kuriskova geology, which has allowed the modeling of structures controlling uranium mineralization. The new resource has been updated for the following zones: Main Zone North, Main Zone South, Zone 45. Zone 2 North and Zone 3 North in the hanging wall north of fault J8. The remaining zones remain the same as there is no addition of drill data included in this update. This resource update only applies to zones where new drill holes have been added or where more detailed structural modeling has been incorporated.

The Main Zone South resource update is based on three infill holes drilled between January 2011 and March 2011. Zone 45 resource update is based on 14 holes drilled between August 2010 and December 2010. One hole drilled in Main Zone South of historic hole 1226 was not used for extending resource as it has intersected Main Zone 150 m from existing resource and is considered to have undue influence on resource tonnes. This block will be followed up in the future. The resource update of Main Zone North is based on incorporation of structural modeling. With enhanced understanding of geology and structures, hanging wall north resource has been reclassified; the grade estimate for hanging wall has not been changed.

The database and geological/domain modeling described in this section is for the updated Main Zone South, as well as all other zones of Kuriskova.

14.1 Database

The database described in Section 7.0 through Section 11.0 above was compiled by Mr. Cisovsky. Database management and data collection were carried out under the supervision and review of Mr. Sharma, EUU's resource manager. The database was compiled in a spreadsheet and maintained in Microsoft (MS) Access format. Detailed database verification and QA/QC were conducted as described in Section 12.0 Data Verification. The database comprises of collar, downhole survey, geology, assay, and density data for 151 surface drill holes. Geological records and assay data are handled through the spreadsheet and a MS Access data entry system. Validation queries were created in MS Access and MS Excel to perform data validation before the data were input to Datamine Studio3, a mine modeling software. Datamine built-in validation rules also checked for errors while importing. The final, verified, and validated database is password protected, demonstrating the rigor of EUU's security protocols.

The drill hole information imported in Datamine Studio3, consisted of 151 drill holes, including 18 holes drilled between summer 2010 and March 2011. As described in Section 12.0, this is a "mixed" database; gamma eU% values are used only for 27 historical drill holes. While the mixing of data types is undesirable, it is necessary as the 27 historic drill holes have only eU%

values available. The justification of using eU% for these 27 holes is based on detail data verification by EUU (Tournigan, June 2008) and closed can analysis review report by SRK in 2009, for comparing radiometric and assay data and to arrive at the conclusion that using radiometric data in absence of chemical assay is acceptable. Comparison of chemical assay and radiometric data by closed can method are shown in Figure 12.1 and Figure 12.2 and described in Section 12.1. These studies indicate a relative state of equilibrium exists with a 6 percent bias, where radiometric analyses are 6 percent lower than when compared to chemical analyses. Table 14.1 and Figure 12.4 illustrate the comparison between chemical and radiometric data with review and conclusions by SRK, 2009. As described in detail in Section 12, radiometric and chemical assay data are separately representing the same volume of rock; thus, radiometric assays were not adjusted for disequilibrium. Where it is not explicitly distinguished, any tabulation reporting or listing of percent uranium, in the following discussion relating to the resource estimation, is that of the mixed percent uranium and eU% database field. As described below, the resource estimation was constrained to the wireframed domains of mineralized structures interpreted and constructed by EUU. The characteristics of this database are summarized in Section 14.2 below.

As described in Section 11.0, there are several methods that have been used for uranium analysis of core samples, including ICP, bf-XRF, and pp-XRF. Each was used at various times and for various reasons. EUU has evaluated the applicability of each method and has developed rules to select the value to be used in the resource database. In the following sections, these values are referred to as the “assay” value of a sample interval. While it is preferable to be consistent throughout a database with one analytical method, SRK’s 2008 review of the data evaluations by EUU concurs that the best analytical method, not necessarily the best assay, was used to determine the value of percent uranium used in the database. The rules established by EUU are:

- If bf-XRF is greater than or equal to 1.0 percent, then bf-XRF will be the valued used;
- If bf-XRF is less than 1.0 percent and pp-XRF greater than 0, then pp-XRF will be used; and
- If there is no analysis value by bf-XRF or pp-XRF, then the ICP (also noted as ICM) value will be used.

14.2 Exploration Data Analysis and Model Zone Redefinition

In 2010 to 2011, EUU carried out detailed exploratory data analysis. EUU drilled 122 holes between 2005 to 2011, which enhanced the understanding of geology and structures controlling uranium mineralization. Although the exploratory data analyses were part of the entire previous resource estimate, in the absence of sufficient data, a high grade mineralization trend in previous studies was not readily apparent. Variograms were erratic indicating highly mixed grade population. Very high grade samples were treated with top cut to restrict their undue influence on lower grade population and vice versa. Due to less data and, thus, lack of clear understanding it was not possible to delineate high grade from low grade. Exploration Data Analysis (EDA) exercises carried out by EUU in 2010 to 2011, improved understanding of structures controlling mineralization. An apparent grade boundary that approximates the statistical grade break between low grade, medium grade, and high grade mineralization for Main Zone 1 North and a grade boundary between high grade and low grade mineralization for Zone 45 can be apparent. The upward continuation of a plane between the high grade and low grade in Main Zone North through the 614 Fault intersects the hanging wall at approximately the areas of high-grade mineralization in the Hanging Wall Zone. This does not appear to be a coincidence with the highest grade concentration in the Hanging Wall Zone around drill hole

KG-J-21A, which is possibly due to intersection of fault structures. The interpreted grade boundaries were examined with respect to geology to determine their significance. These data were checked against geological sections in conjunction with drill core to see if these grade boundaries are due to any structural features and to investigate the possibility of modeling these as structures limiting grade. It is suggested that higher grades are likely controlled by these interpreted structural orientation, as the highest grades encountered are close to these interpreted plane. It was concluded that these interpreted planes could be used as a boundary in grade interpolation by partitioning data. Basic statistics after data partitioning using an interpreted plane to define zones of low grade, medium grade, and high grade in Zone 1 North and high and low grade in Zone 45 confirmed the findings of EDA exercises.

The following figures are included to illustrate different EDA exercises carried out to understand grade distribution and mineralization control in Main Zone North, Hanging Wall north and Zone 45. Table 14.1 and Table 14.2 show basic statistics on percent uranium and molybdenum after data partitioning using interpreted plane to define zones of low grade, medium grade, and high grade in Zone 1 north. Table 14.3 and Table 14.4 show basic statistics on percent uranium and molybdenum after data partitioning using interpreted plane to define zones of low grade and high grade in Zone 45. In Table 14.1 to Table 14.4, the mean of percent uranium and molybdenum for different grade zones in Zone 1 North and Zone 45 clearly indicates 3 different grade population in Main Zone North and two different population in Zone 45. The location of different grade zones for Main Zone North is apparent in Figure 14.1, map of full length composites within the Main Zone North wireframe. Figure 14.2 is a 3D view of grade distribution also indicating three discrete grade mineralization in Main Zone North. Figure 14.3, Figure 14.4, and Figure 14.5 are log histograms of three distinct grade zones in Main Zone North. Figure 14.6 is a log probability plot indicating three distinct grade zones in Main Zone North. Figure 14.7 is a cumulative frequency plot for the three grade zones composites of Main Zone North. These figures clearly indicate three discrete zones of mineralization having distinct boundary separating zones of high grade, medium grade, and low grade mineralization. Figure 14.8 shows high grade and low grade zones separated by Fault 45 in Zone 45 and clearly indicates Fault 45 as boundary demarcation between high grade on east and low grade on the west. Figure 14.9 shows fault structures limiting grade zones in Main Zone North and Zone 45.

Table 14.1. Statistics by Sub Domain (Grade Domain) on Percent Uranium (Combined Database Percent Uranium and eU% values)

Description	Domain (Zcode)	Sub Domain (Gcode)	Number of Values	Min	Max	Mean	Variance	Standard Deviation	Coefficient of Variance
High grade zone of Main Zone North	1	10.1	274	0.00	14.50	0.985	2.871	1.695	1.720
Medium grade zone of Main Zone North	1	10.2	169	0.00	4.65	0.369	0.428	0.654	1.774
Low grade zone of Main Zone North	1	10.3	276	0.00	4.85	0.188	0.166	0.408	2.164
All grade zones combined of Main Zone North	1	(10.1+10.2+10.3)	719	0.00	14.50	0.502	1.200	1.095	2.183

Table 14.2. Statistics by Sub Domain (Grade Domain) on Percent Molybdenum for Main Zone North

Description	Domain (Zcode)	Sub Domain (Gcode)	Number of Values	Min	Max	Mean	Variance	Standard Deviation	Coefficient of Variance
High grade zone of Main Zone North	1	10.1	118	0.00	2.62	0.110	0.073	0.270	2.451
Medium grade zone of Main Zone North	1	10.2	159	0.00	3.76	0.069	0.070	0.265	3.830
Low grade zone of Main Zone North	1	10.3	110	0.00	1.01	0.025	0.009	0.093	3.655
All grade zones combined of Main Zone North	1	(10.1+10.2+10.3)	387	0.00	3.76	0.069	0.055	0.235	3.424

Table 14.3. Statistics by Sub Domain (Grade Domain) on Percent Uranium for Zone 45

Description	Domain (Zcode)	Sub Domain (Gcode)	Number of Values	Min	Max	Mean	Variance	Standard Deviation	Coefficient of Variance
Zone 45 high grade zone (east of Fault 45)	5	50.1	53	0.00	5.03	0.769	1.259	1.122	1.459
Zone 45 low grade zone (west of Fault 45)	5	50.2	15	0.00	0.49	0.145	0.022	0.148	1.024
Zone 45 all (high grade +low grade zones)	5	50.1+50.2	68	0.00	5.03	0.652	1.086	1.042	1.599

Table 14.4. Statistics by Sub Domain (Grade Domain) on Percent Molybdenum for Zone 45

Description	Domain (Zcode)	Sub Domain (Gcode)	Number of Values	Min	Max	Mean	Variance	Standard Deviation	Coefficient of Variance
Zone 45 high grade zone (east of Fault 45)	5	50.1	53	0.00	3.66	0.521	0.614	0.788	1.513
Zone 45 low grade zone (west of Fault 45)	5	50.2	15	0.00	2.31	0.240	0.253	0.503	2.096
Zone 45 all (high grade +low grade zones)	5	50.1+50.2	68	0.00	3.66	0.468	0.558	0.747	1.596

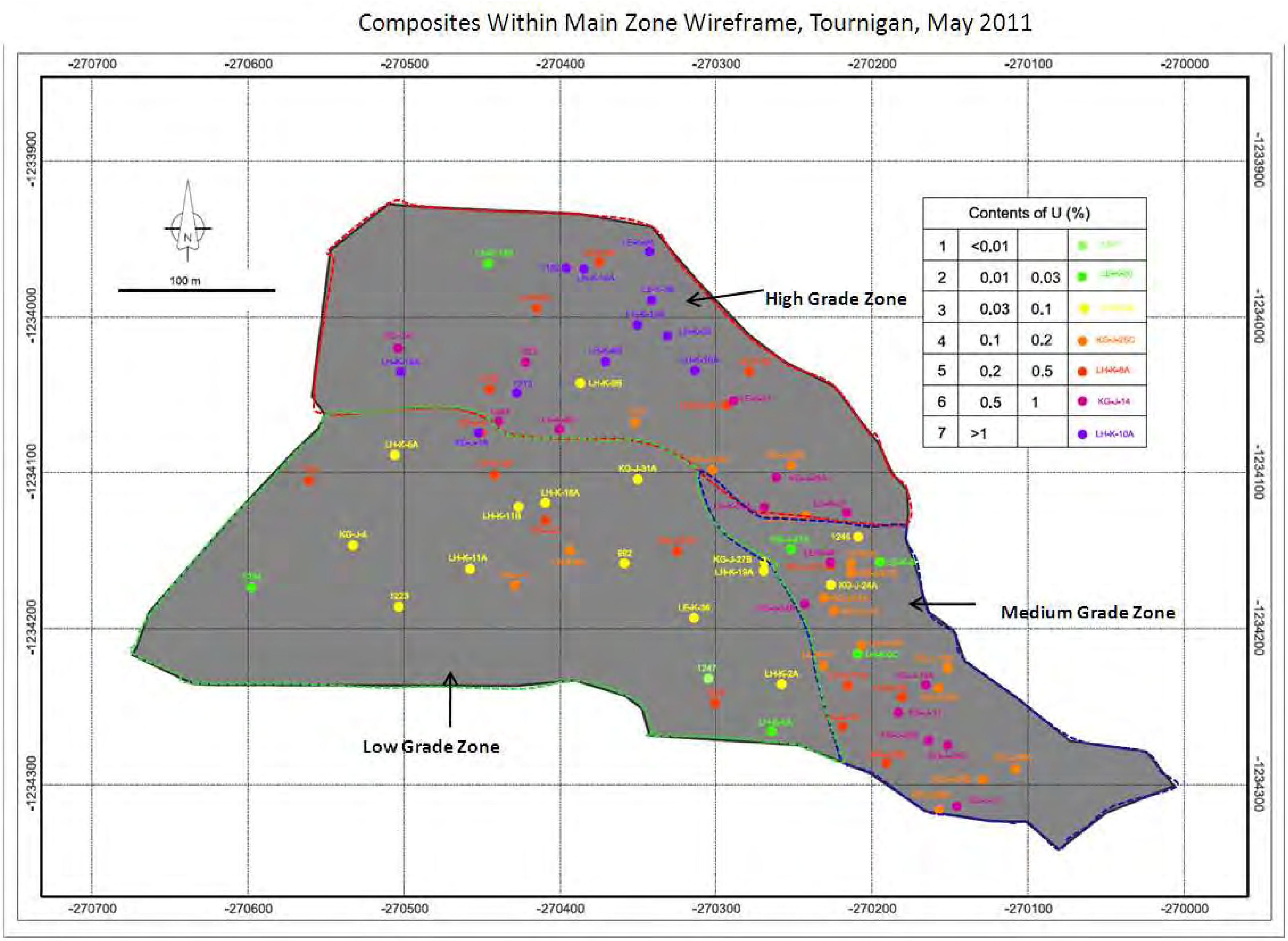


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Figure 14.1
Full Length Composite Map Main Zone North Showing 3 Discrete Zones

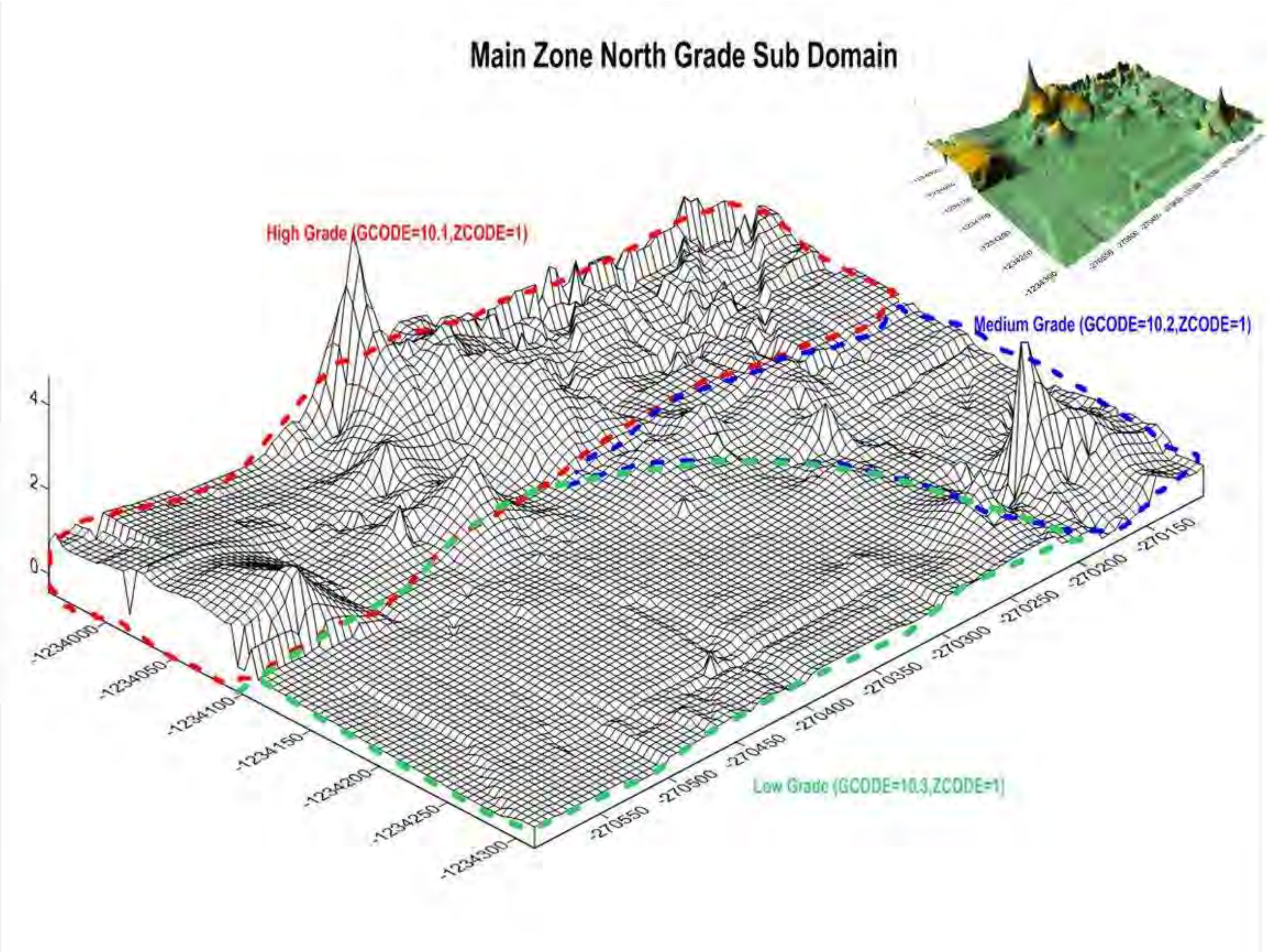

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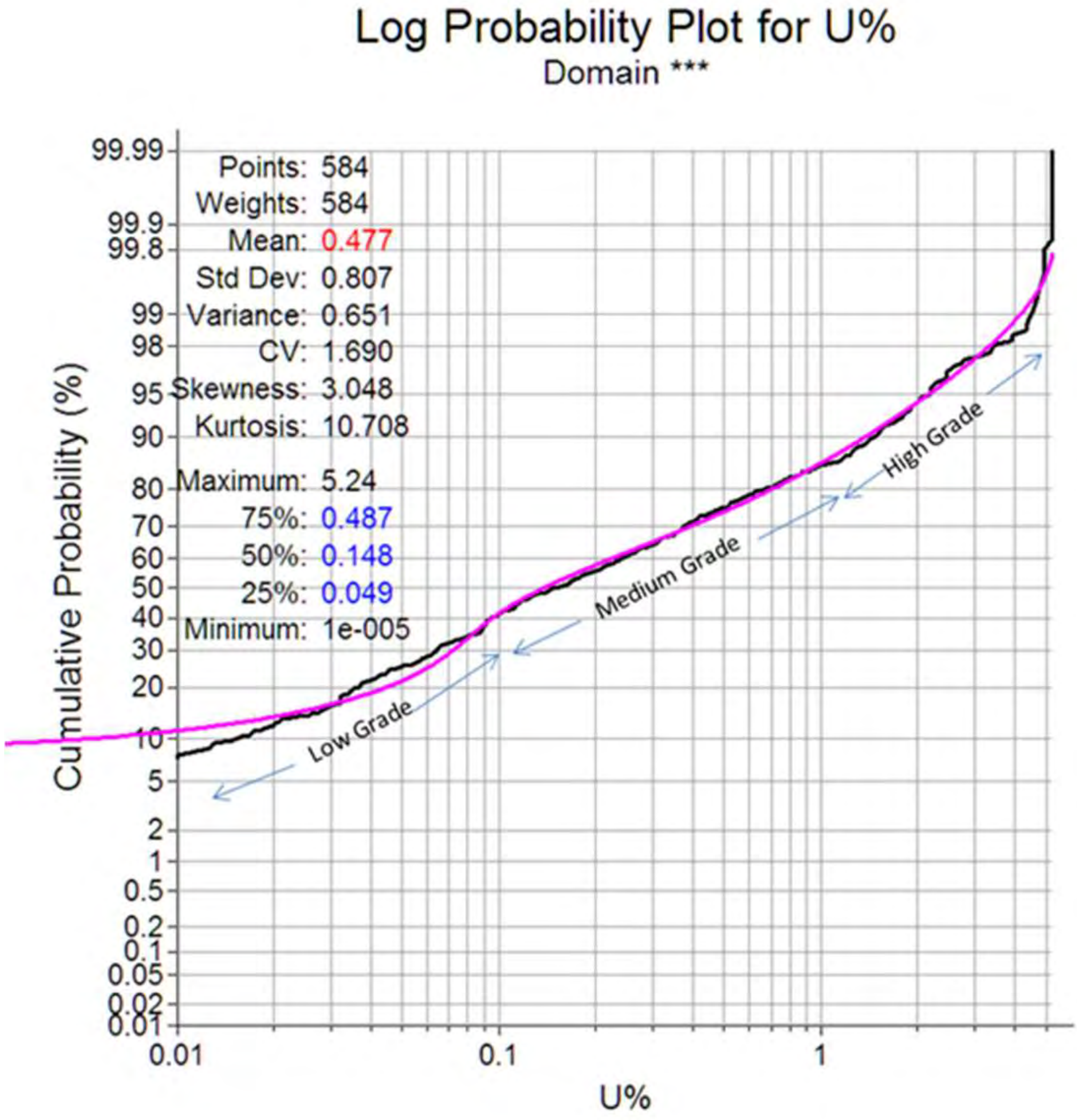
Figure 14.2

3D View of Grade Distribution Main Zone North



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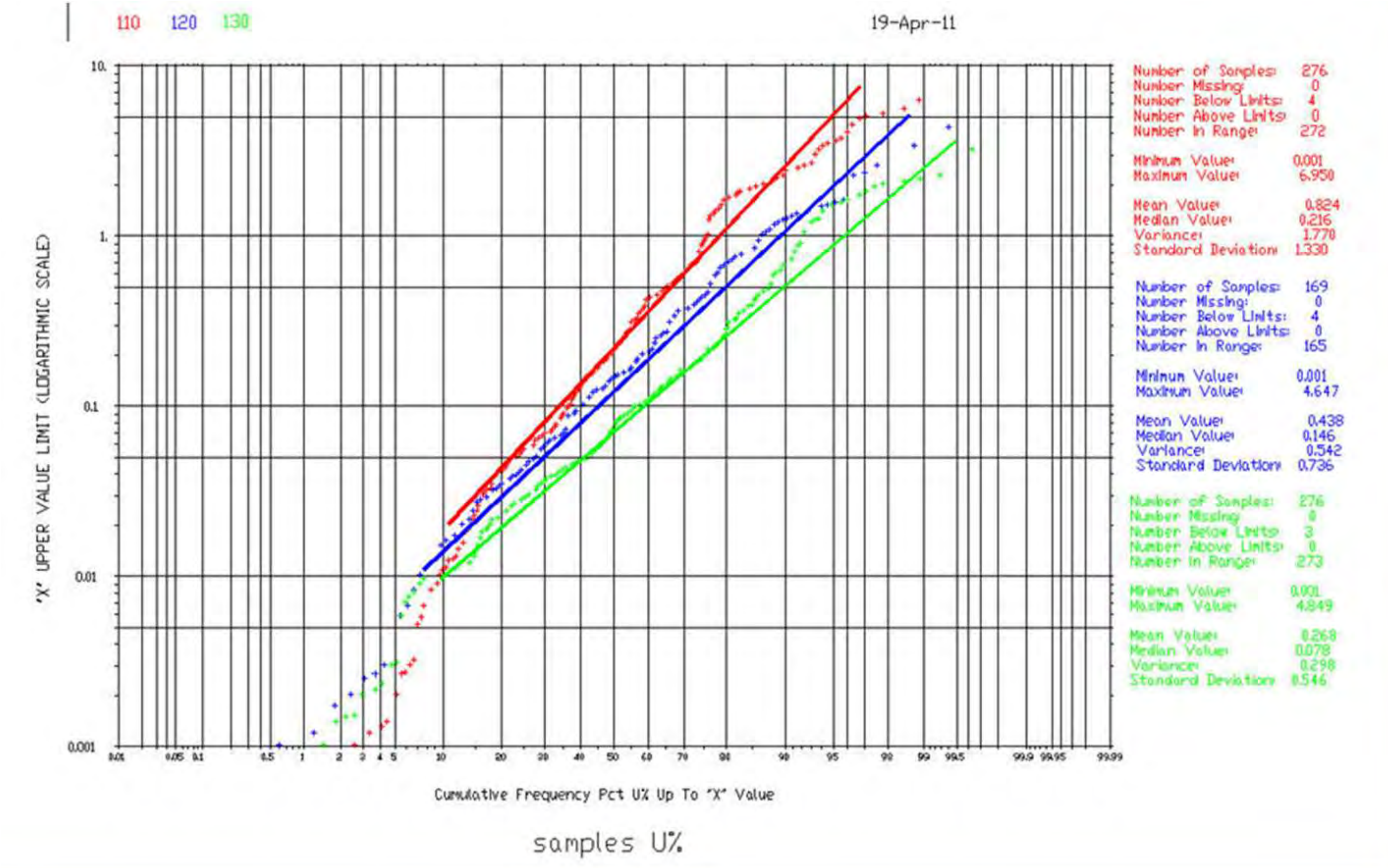
	Project: Kuriskova Uranium Project	Project Number: 114-310990	
	Project Location: Slovak Republic	Date of Issue: June 2011	



Indicating high grade (HG) with mean U% = .824 %, medium grade (MG) with mean U%= 0.438 % and low grade (LG) with mean U%= .268 %.

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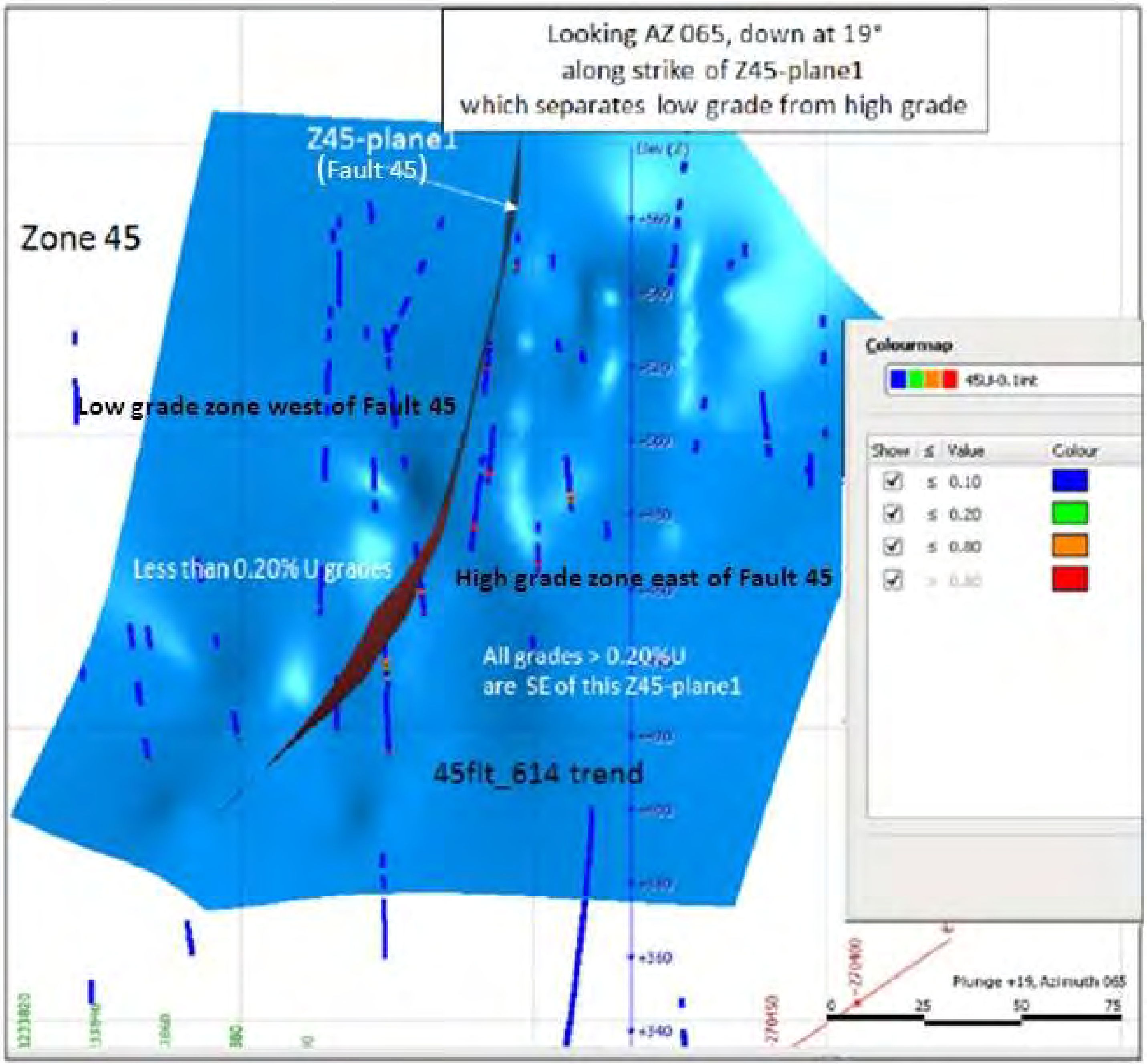
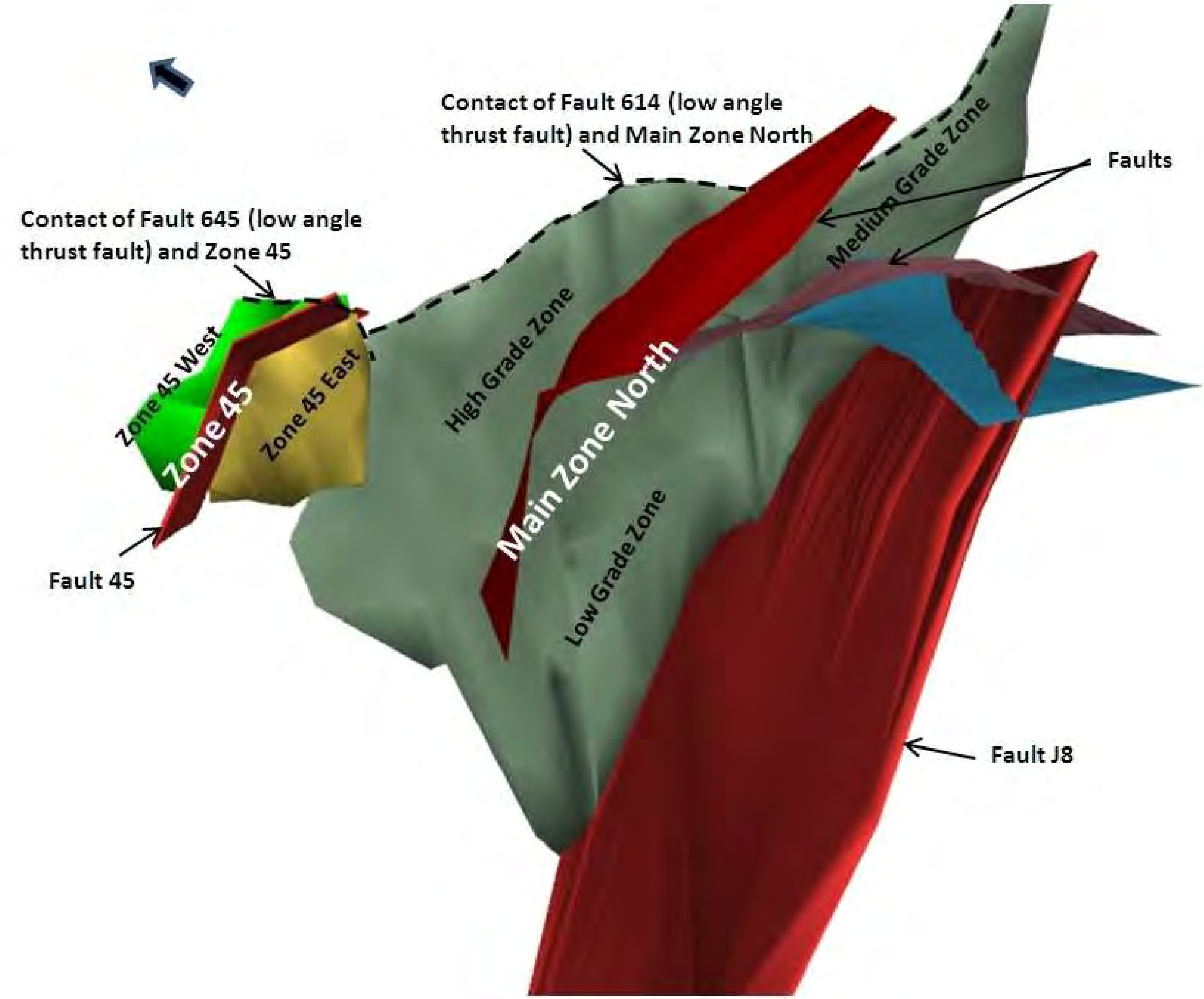
TETRA TECH 350 Indiana Street, Suite 500 Golden, CO 80401 (303) 217-5700 (303) 217-5705 fax	***European Uranium Resources Ltd.***	Fig 14.7.jpeg	**Figure 14.7**
	Project: Kuriskova Uranium Project	Project Number: 114-310990	**Cumulative Frequency Plot for 3 Grade Zones**
	Project Location: Slovak Republic	Date of Issue: June 2011	**Composites of Main Zone North**



Issued by: TETRA TECH 350 Indiana Street, Suite 500 Golden, CO 80401 (303) 217-5700 (303) 217-5705 fax	Prepared for: ***European Uranium Resources Ltd.***	File Name: Fig 14.8 6A.jpeg	**Figure 14.8** **High Grade and Low Grade Zones Separated by Fault 45 in Zone 45**
	Project: Kuriskova Uranium Project	Project Number: 114-310990	
	Project Location: Slovak Republic	Date of Issue: June 2011	



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TETRA TECH 350 Indiana Street, Suite 500 Golden, CO 80401 (303) 217-5700 (303) 217-5705 fax	***European Uranium Resources Ltd.***	Fig 14.9.jpeg	**Figure 14.9** **Fault Structures Limiting Grade Zones in Main Zone North and Zone 45**
	Project: Kuriskova Uranium Project	Project Number: 114-310990	
	Project Location: Slovak Republic	Date of Issue: June 2011	

14.3 Geologic Model/Domain Model

EUU has interpreted three primary geological domains along with a number of sub-domains and few grade domains for the Kuriskova deposit; a Main Zone, a Hanging Wall Zone, and Zone 45.

The Main Zone: which is in general a basal mineralized zone of the Kuriskova uranium deposit and hosts most of the high grade mineralization is divided by the J8 Fault into the sub-domains Main Zone North and the Main Zone South. There is also an Upper Main Zone sub-domain, which is above the 614 Fault. Main Zone North is further subdivided into high, medium, and low grade domains based on positions of structures controlling uranium mineralization as identified by recent exploratory data analysis.

The Hanging Wall Zone: is startigraphically above the Main Zone and is in general lower grade than the Main Zone and includes a stock work mineralization in the andesites and discrete mineralization in tuffegenic sediment above andesites. The Hanging Wall North Zones are north of J8 Fault and stratigraphically above Main Zone North. The Hanging Wall South Zones are zones south of J8 fault and are also stratigraphically above Main Zone South.

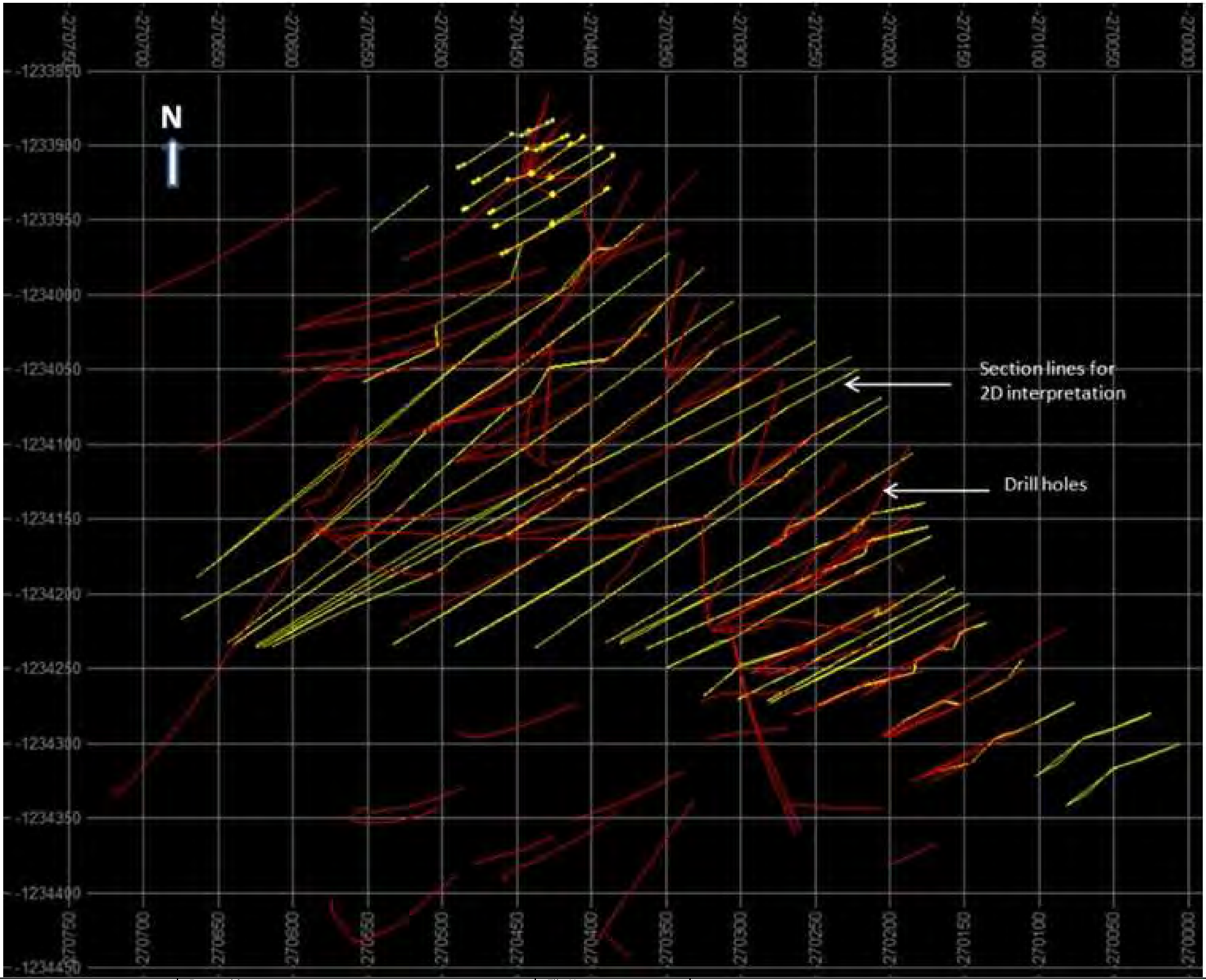
Zone 45: discovered during drilling in 2009 to 2010 is cauterized by high-grade mineralization, similar in grade to the Main Zone. Zone 45 occurs at a shallower depth (100 to 150 m from surface) then the Main Zone and is in a different geological setting, occurring in interformational schist horizons in Hanging Wall, rather than the volcano sedimentary contact which contains the Main Zone. Zone 45 is further subdivided by Zone 45 Fault into Zone 45 East and Zone 45 West sub domains. In general, Zone 45 East is characterized by high grade and Zone 45 West is low grade mineralization.

Two-dimensional structural interpretation and outlining of mineralization was done section-by-section by incorporating geological, structural, and assay information from drill holes for each geological domain. While performing section interpretations, hanging wall and footwall contact points were “snapped” to drill hole locations (points on 3D line segments are created by using the exact assay top or bottom locations) to preserve as accurately as possible representation of volumes for each domain. Since the drilling at Kuriskova is not, in most cases, on regular grid section lines, non-orthogonal or oblique section interpretations were used. In general, holes within 10 m of the center line of the sections were included to interpret section geology, representing a “clipping window” of 10 m in each direction. Figure 14.10 shows a plan view of the general orientation of the northeast-southwest sections for Zone 1 North and Zone 45, created for interpretation.

Figure 14.11 illustrates a typical 3D (north-south) view. The fault structures are the primary controls for modeling domains. For example, the Main Zone North is bound between two faults, horizontal thrust Fault 614 and vertical Fault J8; similarly Zone 45 is bound by horizontal thrust Fault 645. The structures interpreted in exploratory data analysis and subsequently identified in geological cross sections were linked to create wireframe planes. Based on positions of these planes, 3D wireframes for high grade, medium grade, and low grade zones in Main Zone North were created to partition and filter data for estimating these grade zones separately (Figure 14.9).

In all cases, these structural geometrical interpretations were discussed with the EUU senior project geology staff before creating the 3D wireframes, and the resulting shapes were presented to the staff for review. Some members of EUU’s in-country geological staff have considerable experience with the Kuriskova deposit, and their input to the structural modeling was considered essential. The cross-sectional domain outlines were linked by wireframing in

Datamine Studio3, to create 3D mineralized geological domain models. These were verified and validated before creating the 3D block model. Verifications included face and edge overlap checks, surface intersection checks, and visual cross section inspections by slicing to check for any point snapping or digitization error.



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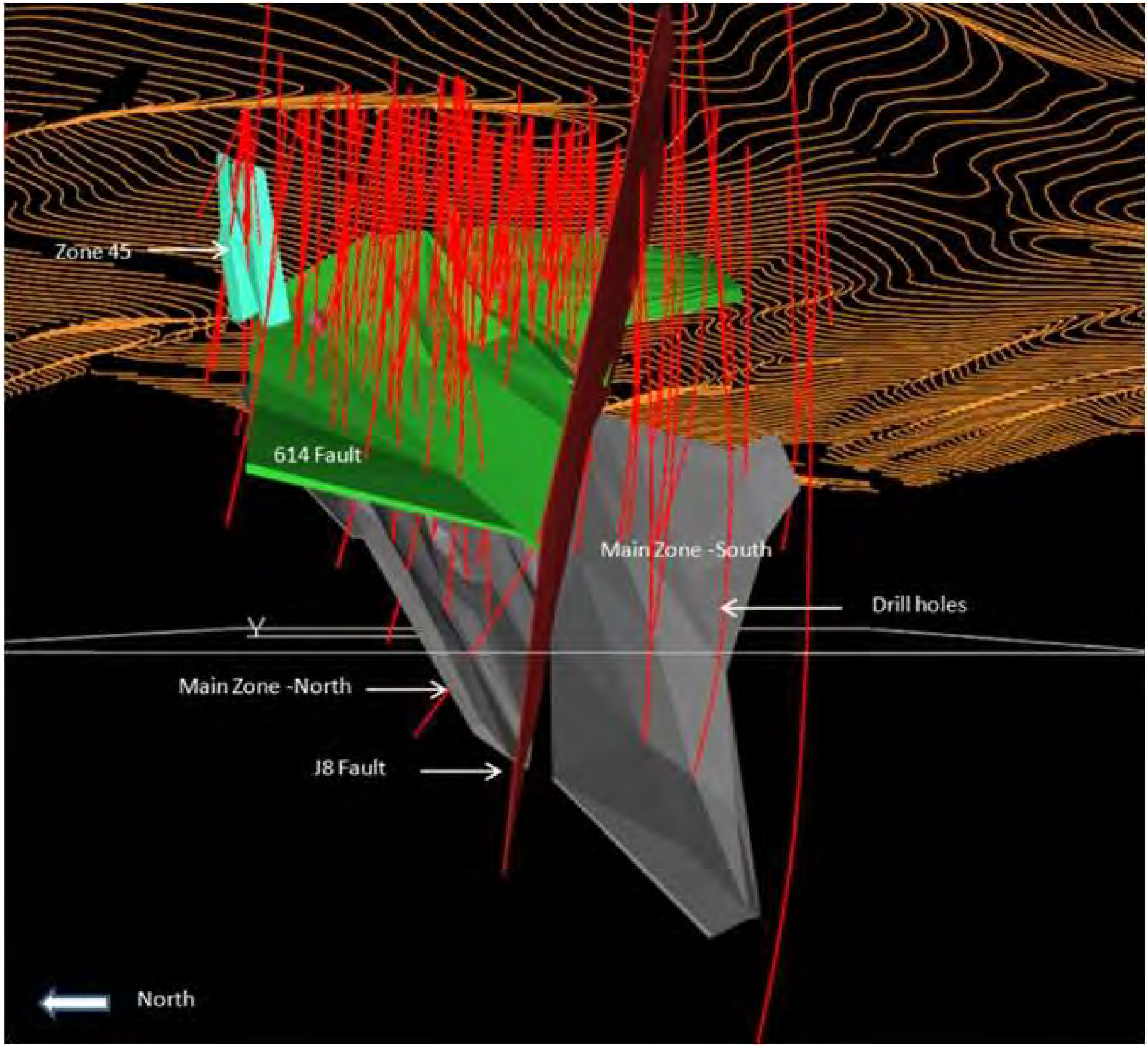
TETRA TECH 350 Indiana Street, Suite 500 Golden, CO 80401 (303) 217-5700 (303) 217-5705 fax	***European Uranium Resources Ltd.***	Fig 14.10.jpeg	**Figure 14.10** **Plan View of Sections for Zone 1 North**
	Project: Kuriskova Uranium Project	Project Number: 114-310990	
	Project Location: Slovak Republic	Date of Issue: May 2010	



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TETRA TECH 350 Indiana Street, Suite 500 Golden, CO 80401 (303) 217-5700 (303) 217-5705 fax	***European Uranium Resources Ltd.***	Fig 14.11.jpeg	**Figure 14.11** **Typical 3D (North South), View Looking East**
	Project: Kuriskova Uranium Project	Project Number: 114-310990	
	Project Location: Slovak Republic	Date of Issue: June 2011	

Where wireframes are not bound by faults, a hard grade shell boundary was applied. From an inspection of the cumulative frequency distribution diagram (see Figure 8.3A and 8.3B) an inflection at 0.03 percent uranium is interpreted as a population break for the mineralized versus non mineralized populations. In general, the sub-domain wireframes, where used as “hard” boundaries; percent uranium values within a particular domain were used only to estimate grade in that domain. While this is appropriate in many cases, such as preventing the extrapolation of higher grades from the Main Zone into proximate hanging-wall andesites, in others cases it may or may not be appropriate. Zone 2 North (Zcode 2) is stock work mineralization and Zone 3 North (Zcode 3) is discrete, discontinuous mineralization above andesite. These two zones were modeled without hard boundary wireframe by creating prototype domain blocks within respective geology domain. The prototype blocks were created around samples with percent uranium value greater than 0.03 percent, which is the same cut off criteria used for the hard boundary wireframe in other zones. These domain blocks were created with search ellipse criteria of X=20 m, Y=15 m, and Z=2 m and were forced to see a minimum of four samples with a maximum of three samples from one drill hole; thereby, forcing two holes for creating a block with a tight search ellipse mentioned above. The wireframes were created for these two zones to filter blocks in the central area where mineralization is better understood and drill intersection density is close to 15 m. These blocks were only included in the resource. The blocks outside of this wireframe were not included in the final resource model and are left as future upside potential. Other than Zone 2 North and Zone 3 North in the Hanging Wall Zone, the wireframes of other domains were used to constrain the grade estimation within the geological domains, and they constitute the primary control for grade estimation and entirely control the domain volumes.

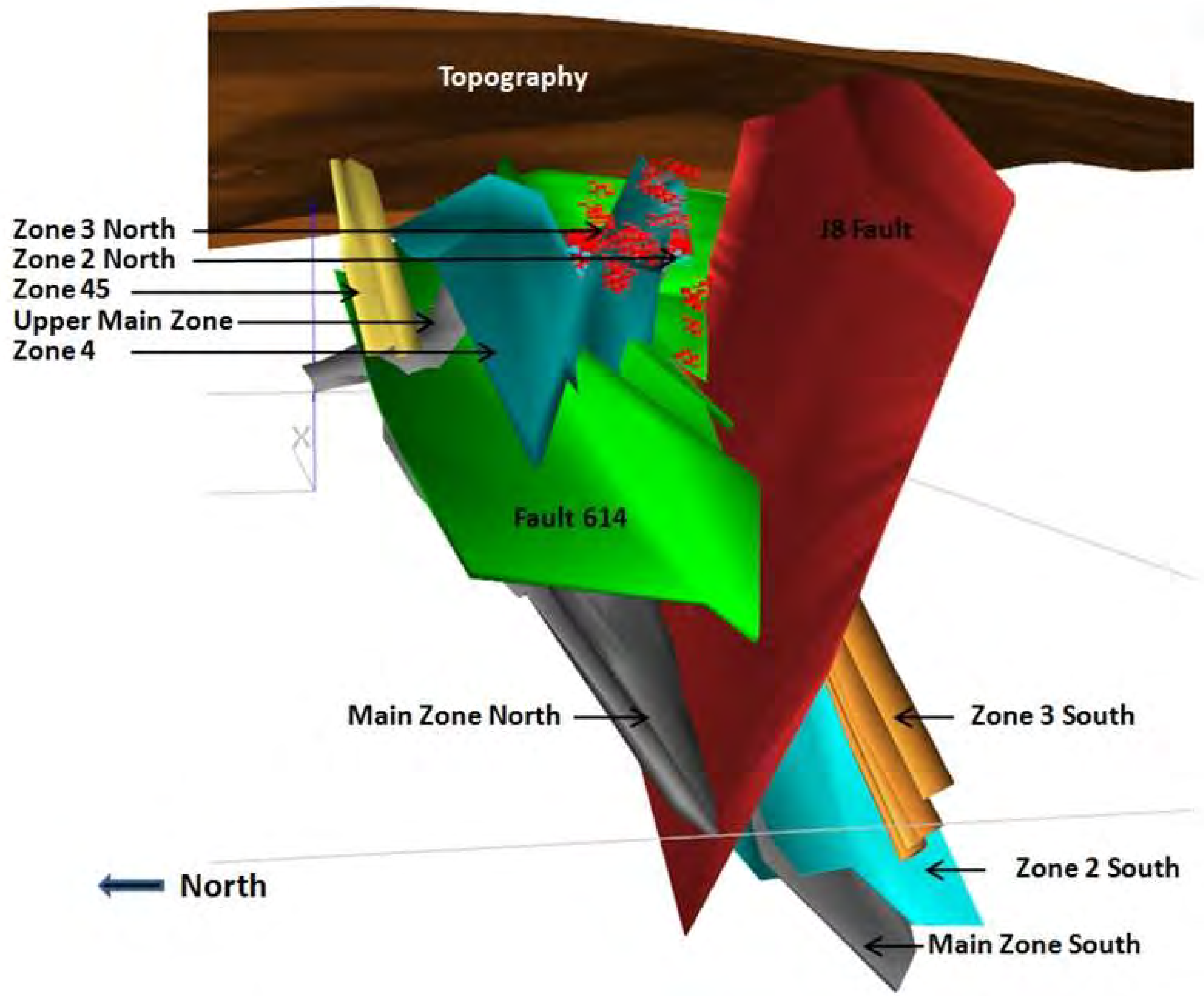
The main structures (Faults J-8 and 614) were modeled first as they significantly influence the position of mineralized zones on the northern side of Fault J8, especially in the Main Zone. Figure 14.12 is a 3D perspective view of the all domain wireframe models and blocks for Zone 2 North and Zone 3 North.

Wireframes and the drill hole samples within sub domain wireframes were coded with numeric ZCODE values to form domain drill hole databases. Similarly, numeric GCODE values were assigned to grade zones of sub domains identified in exploratory data analysis. In the Main Zone North domain, high, medium, and low grade sub domains were assigned GCODE values. Similarly GCODE values were assigned for the high grade sub domain, which is east of Fault 45 in Zone 45 and for the low grade sub domain west of Fault 45 in Zone 45. Except for Main Zone North and Zone 45 where GCODE values have been assigned to further separate grade zones and estimated separately to avoid grade smearing effect. The GCODES used for the remaining zones are not significant for this resource update as no separate grade zones are apparent in these zones; thus, do not require separate estimation within the domains. These are assigned to maintain uniformity and for functioning of estimation macro in Datamine Studio3. Table 14.5 is a summary of sub-domain names, the numeric ZCODE and GCODE values assigned to each.

Waste units internal to the Main Zone North wireframe, with a drill hole intercept thickness greater than 1 m were considered to be separable mineable units of waste and were modeled with internal waste wireframes. Most of the waste thickness is greater than 2 m. These separable internal waste zones were digitized on sections, and strings were projected to approximately 20 m distance on either side of the section to create 3D internal waste wireframes (Figure 14.13). In few cases in the northeast, this projection was 10 m. Since drill hole values designated as separable internal waste are not used for grade estimation, the volumes of the separable waste wireframes need to be representative of expected mining selectivity.

Table 14.5. Summary of Modeling Domains

Domain	Description	Sub-Domains	GCODE	ZCODE
Main Zone	Laterally continuous strata-bound basal mineralized zone, occurring at the main meta-andesite/meta-sediment contact.	High grade: northern part of Main Zone north. Main Zone North (zone1n) is basal mineralized zone, north of Fault J-8	10.1	1
		Medium grade: eastern to central part of Main Zone North. Main Zone North (zone1n) is basal mineralized zone, north of the Fault J-8	10.2	
		Low grade: southern part of Main Zone North. Main Zone North (zone1n) is basal mineralized zone, north of the Fault J-8	10.3	
		Main Zone South (zone1s): basal mineralized zone south of the Fault J-8	11.1	1.1
		Upper Main Zone (upmainzone): Main Zone above Fault 614	12.1	1.2
Hanging Wall Zone	Semi-continuous and discrete mineralized zones hosted within hanging wall meta-andesite.	Zone 2 North (zone2n): mineralized Andesite stratigraphically above the Main Zone, north of J-8 a	20.1	2
		Zone 2 South (zone2s): mineralized Andesite, south of J-8 and above Main Zone South	21.1	2.1
		Zone 3 North (zone3n): discrete mineralized Andesite zone, stratigraphically above Zone 2 and north of J-8	30.1	3
		Zone 3 South (zone3s): discrete zone mineralized andesite zone, stratigraphically above Zone 2 south and south of J-8.	31.1	3.1
		Zone 4: minor, mineralized zones in tuffs above andesite	40.1	4
Zone 45	Laterally continuous high-grade U and Mo mineralization in Upper transitional layer of the Hanging Wall	East of Zone 45 cross fault with high grade mineralization. Zone 45 is north west of Main Zone resource	50.1	5
		West of Zone 45 cross fault, low grade mineralization Zone 45 is north west of Main Zone resource	50.2	



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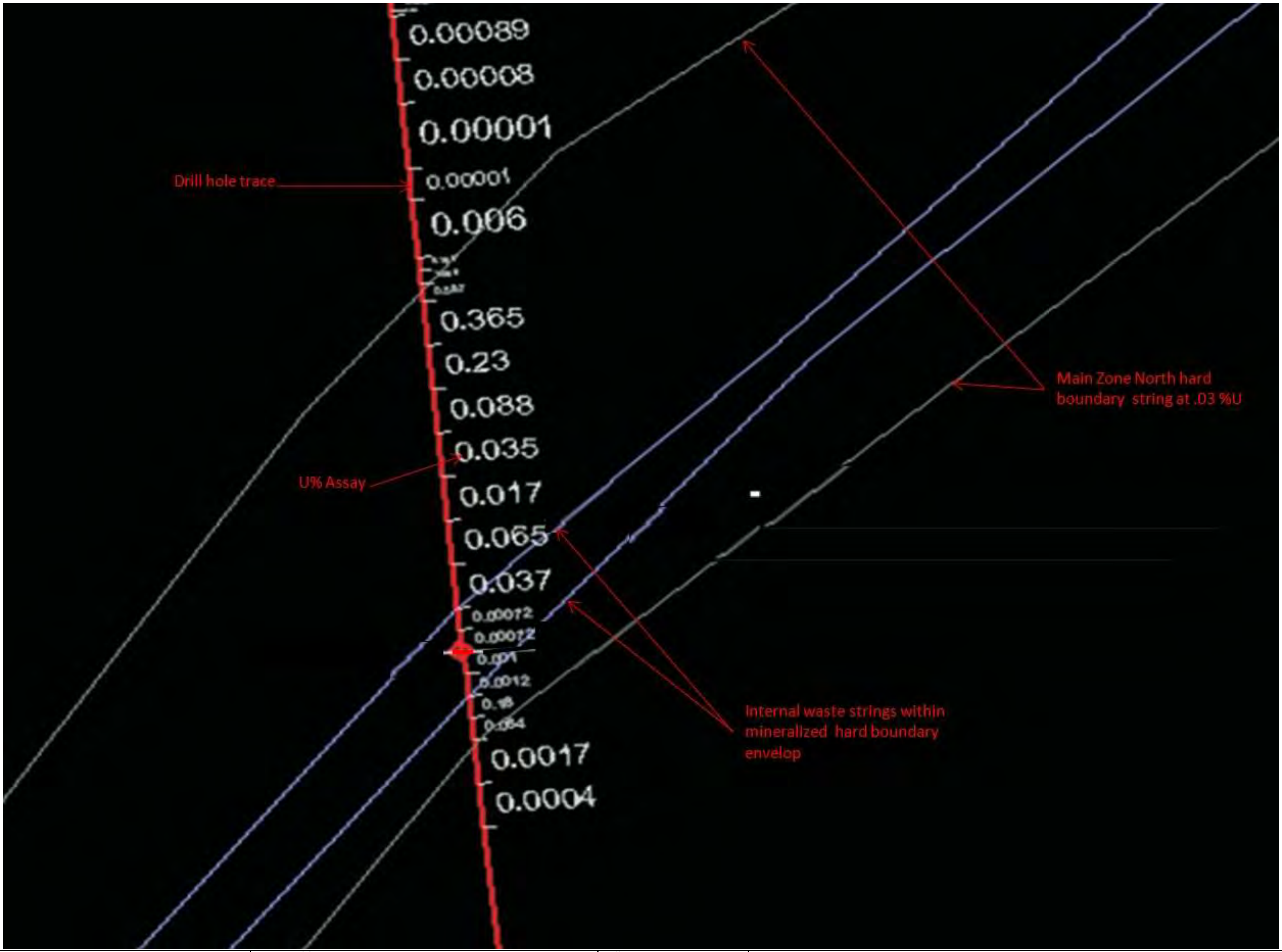
TETRA TECH 350 Indiana Street, Suite 500 Golden, CO 80401 (303) 217-5700 (303) 217-5705 fax	***European Uranium Resources Ltd.***	Fig 14.12.jpeg	**Figure 14.12** **3D Perspective View of all Domain Wireframes**
	Project: Kuriskova Uranium Project	Project Number: 114-310990	
	Project Location: Slovak Republic	Date of Issue: May 2010	



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TETRA TECH 350 Indiana Street, Suite 500 Golden, CO 80401 (303) 217-5700 (303) 217-5705 fax	***European Uranium Resources Ltd.***	Fig 14-13.jpeg	**Figure 14.13** **Main Zone North Internal Waste Boundary**
	Project: Kuriskova Uranium Project	Project Number: 114-310990	
	Project Location: Slovak Republic	Date of Issue: May 2010	

Table 14.6 details the statistical analysis of the drill hole assay values by domain for uranium with no top cutoff grade and only above a lower cutoff grade of 0.05 percent uranium. In addition, there is no distinction made between percent uranium and eU% grades. (Note: 1 – the Main Zone North assay values are excluding internal waste. 2 – Zone 2 and Zone 3 are without hard boundary wireframes as described in Section 10.2).

Table 14.6. Statistics by Domain on Percent Uranium (Combined Database Percent Uranium and eU% values)

Domain Statistics On % U (No Cutoff)							
Domain (Zcode)	**Num of Values**	**Minimum**	**Maximum**	**Mean**	**Variance**	**Standard Deviation**	**Coefficient of Variance**
All	2,034	0.000	14.500	0.310	0.576	0.759	2.445
1	719	0.000	14.500	0.502	1.200	1.095	2.183
1.1	74	0.000	1.642	0.266	0.134	0.366	1.377
1.2	119	0.000	2.300	0.200	0.175	0.419	2.095
2	338	0.030	2.398	0.215	0.105	0.324	1.507
2.1	84	0.001	0.349	0.038	0.004	0.063	1.658
3	334	0.030	3.586	0.190	0.126	0.355	1.868
3.1	147	0.000	0.802	0.054	0.011	0.103	1.907
4	151	0.000	1.003	0.058	0.018	0.132	2.276
5	68	0.001	5.030	0.652	1.086	1.042	1.598
Domain Statistics On % U (0.05 % U Cutoff)							
Domain (Zcode)	**Num of Values**	**Minimum**	**Maximum**	**Mean**	**Variance**	**Standard Deviation**	**Coefficient of Variance**
All	1,224	0.05	3.760	0.493	0.854	0.924	1.875
1	493	0.05	3.760	0.751	1.636	1.279	1.702
1.1	55	0.0501	1.296	0.342	0.150	0.387	1.133
1.2	38	0.051	0.610	0.342	0.256	0.506	1.480
2	256	0.05	0.328	0.294	0.131	0.363	1.235
2.1	18	0.051	0.031	0.138	0.007	0.082	0.594
3	225	0.05	0.780	0.268	0.173	0.416	1.552
3.1	51	0.05	0.140	0.136	0.023	0.150	1.103
4	31	0.05	0.500	0.176	0.043	0.207	1.176
5	57	0.0526	3.660	0.768	1.197	1.094	1.425

The 2010 to 2011 Drill Hole Listing (Table B.6) details the statistical analysis of the drill hole assay values for molybdenum by domain at no cutoff grade and above a cutoff grade of 0.05 percent uranium. Because the Project will also produce molybdenum, as well as uranium, the statistics are tabulated above the 0.05 percent uranium cutoff grade as uranium is the primary mineral of importance, and the molybdenum will be produced as a by-product and not as a primary product.

Table 14.7. Statistics by Domain on Percent Molybdenum

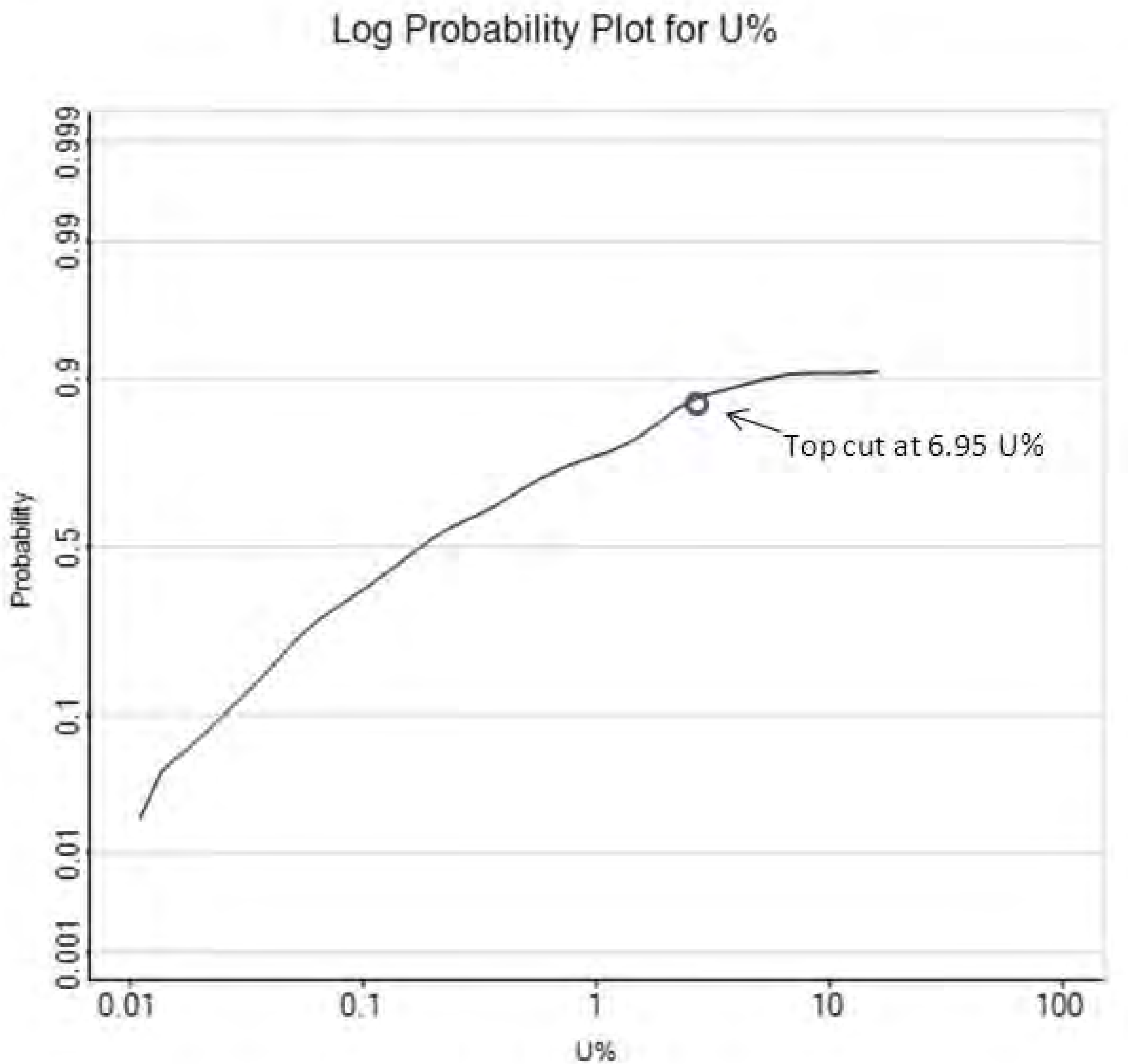
Domain Statistics On Mo (No Cut off)							
Domain (Zcode)	**Num of Values**	**Minimum**	**Maximum**	**Mean**	**Variance**	**Standard Deviation**	**Coefficient of Variance**
All	977	0.000	3.760	0.068	0.066	0.257	3.775
1	387	0.000	3.760	0.069	0.055	0.235	3.424
1.1	59	0.000	1.296	0.054	0.029	0.171	3.181
1.2	54	0.000	0.610	0.042	0.014	0.116	2.762
2	124	0.000	0.328	0.015	0.001	0.035	2.333
2.1	33	0.000	0.031	0.002	0.000	0.004	2.000
3	172	0.000	0.780	0.035	0.007	0.084	2.400
3.1	37	0.000	0.140	0.019	0.001	0.031	1.632
4	43	0.000	0.500	0.029	0.004	0.060	2.069
5	68	0.004	3.660	0.468	0.558	0.747	1.596
Domain Statistics On Mo (0.05% Uranium Cut off)							
Domain (Zcode)	**Num of Values**	**Minimum**	**Maximum**	**Mean**	**Variance**	**Standard Deviation**	**Coefficient of Variance**
All	660	0.000	3.760	0.105	0.101	0.319	3.032
1	268	0.001	3.760	0.103	0.081	0.284	2.759
1.1	48	0.001	1.296	0.068	0.037	0.191	2.812
1.2	38	0.000	0.610	0.059	0.018	0.134	2.271
2	86	0.000	0.328	0.021	0.002	0.041	1.952
2.1	10	0.001	0.031	0.006	0.000	0.006	1.000
3	121	0.001	0.780	0.047	0.010	0.101	2.149
3.1	14	0.001	0.140	0.041	0.002	0.046	1.122
4	18	0.000	0.500	0.060	0.007	0.083	1.383
5	57	0.011	3.660	0.552	0.613	0.783	1.418

Based on an examination of the cumulative frequency distribution diagram of assay values within the Main Zone North wireframe domain (Figure 14.14), a population break is interpreted at approximately 6.95 percent uranium. Grades in excess of this value are considered anomalous, or “outliers” to the distribution and approximately three values in excess of 6.95 percent uranium were “set back” to 6.95 percent uranium. This represents a “cap” or “top cut.” The conservative top cut of 4.2 percent uranium was applied in previous resource estimate of March 2010 to restrict undue influence of high grade and avoid grade smearing. This was changed to 6.95 after three discrete grade zones in Main Zone North were identified and estimated separately representing three separate grade populations as explained in EDA discussion. Table 14.8 is sensitivity illustration of change in composite grade and estimated grade by changing Top cut from 4.2 percent uranium to 6.95 percent uranium.

A top cut of 3.12 percent uranium was applied to the assay values from Zone 45 in the 2010 resource estimate. The discrete high and low grade zones were identified and removed and estimated separately.

Table 14.8. Main Zone North Top Cut Sensitivity

	Mean of % U @ 4.2% U Topcut (March 2010 Resource)	Mean of % U @ 6.95 Topcut (April 2011 Resource)	% Difference
Composite Grade	0.456	0.487	6.80%
Block Model	0.43	0.440	2.29%



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Prepared for: ***European Uranium Resources Ltd.***	File Name: Fig 14.14.jpeg	**Figure 14.14**	
Project: Kuriskova Uranium Project	Project Number: 114-310990	**Log Probability Plot for %U**	
Project Location: Slovak Republic	Date of Issue: June 2011	**Main Zone North**	

14.4 Compositing, Composite Statistics, and Domain Analysis

As discussed in Section 10.1 above, the majority of the drill hole intercept values used for modeling will be assay percent uranium values. Figure 14.15, is a histogram of sample lengths within Main Zone North wireframe and shows a clustering of assay sample lengths at 0.5 m. To preserve the integrity of the primary assay data, a composite length of 0.5 m was selected and a downhole composite database was created. Compositing was controlled by domain ZCODE (each composite has a single ZCODE) with a minimum composite length of 0.1 m.

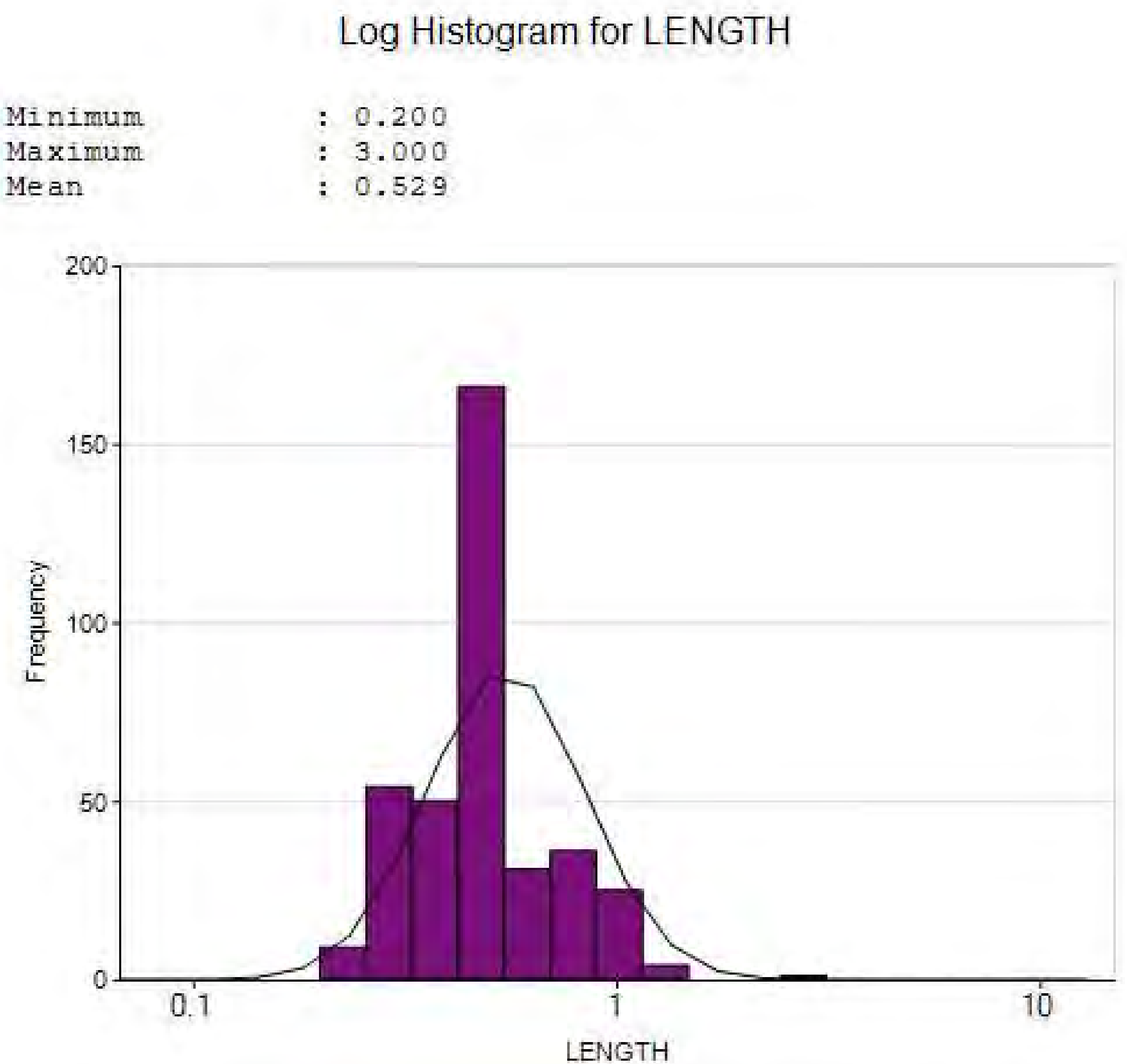
Table 14.9 and Table 14.10 are a summary of the combined percent uranium and eU% composite statistics and percent molybdenum composite statistics by domain. As expected the Main Zone North (ZCODE 1) and Zone 45 (ZCODE 5) have significantly higher grades than the other domains. The coefficient of variation for the separate domains is in general lower than that for all domains, which is an indication that the population segregation by domain is reasonable.

Table 14.9. Composite Statistics by Domain on Percent Uranium

Domain Statistics On % U - Composites (No Cutoff Grade)							
Domain (Zcode)	Number of Values	Minimum	Maximum	Mean	Variance	Standard Deviation	Coefficient of Variance
All	1,383	0.00	6.13	0.276	0.363	0.602	2.179
1	495	0.00	6.13	0.487	0.743	0.862	1.769
1.1	56	0.00	1.21	0.267	0.090	0.301	1.126
1.2	75	0.00	2.30	0.091	0.087	0.295	3.242
2	212	0.03	1.92	0.217	0.086	0.293	1.348
2.1	53	0.00	0.29	0.027	0.002	0.050	1.852
3	266	0.03	3.59	0.190	0.123	0.351	1.846
3.1	74	0.00	0.61	0.054	0.008	0.091	1.685
4	87	0.00	0.48	0.042	0.009	0.094	2.238
5	65	0.00	4.27	0.653	0.850	0.922	1.412
Domain Statistics On % U - Composites (0.05% U Cutoff Grade)							
Domain (Zcode)	Number of Values	Minimum	Maximum	Mean	Variance	Standard Deviation	Coefficient of Variance
All	892	0.05	6.13	0.461	0.539	0.734	1.595
1	353	0.05	6.13	0.674	0.918	0.958	1.423
1.1	46	0.05	1.21	0.321	0.094	0.307	0.956
1.2	44	0.05	2.30	0.345	0.249	0.499	1.446
2	155	0.05	1.94	1.916	0.280	0.101	0.317
2.1	11	0.05	0.29	0.126	0.005	0.068	0.540
3	176	0.05	3.59	0.263	0.166	0.407	1.552
3.1	23	0.05	0.61	0.138	0.017	0.130	0.942
4	27	0.05	0.48	0.169	0.021	0.145	0.858
5	57	0.05	4.27	0.733	0.902	0.949	1.295

Table 14.10. Composite Statistics by Domain on Percent Molybdenum

Domain Statistics On Mo Composites (No Cutoff Grade)							
Domain (Zcode)	Number of Values	Minimum	Maximum	Mean	Variance	Standard Deviation	Coefficient of Variance
All	1144	0.00	3.31	0.068	0.054	0.177	2.592
1	424	0.00	2.38	0.068	0.039	0.199	2.901
1.1	52	0.00	0.91	0.054	0.020	0.141	2.604
1.2	61	0.00	0.61	0.042	0.013	0.115	2.745
2	162	0.00	0.28	0.015	0.001	0.033	2.242
2.1	41	0.00	0.01	0.002	0.000	0.002	1.053
3	227	0.00	0.78	0.035	0.007	0.084	2.422
3.1	49	0.00	0.14	0.019	0.001	0.029	1.551
4	63	0.00	0.50	0.029	0.003	0.055	1.884
5	65	0.01	3.31	0.469	0.453	0.673	1.435
Domain Statistics On Mo Composites (0.05% U Cutoff Grade)							
Domain (Zcode)	**Number of Values**	**Minimum**	**Maximum**	**Mean**	**Variance**	**Standard Deviation**	**Coefficient of Variance**
All	760	0.00	3.31	0.108	0.079	0.281	2.590
1	300	0.00	2.38	0.095	0.053	0.231	2.426
1.1	43	0.00	0.91	0.065	0.027	0.153	2.357
1.2	44	0.00	0.61	0.060	0.018	0.134	2.241
2	113	0.00	0.28	0.020	0.001	0.038	1.931
2.1	10	0.00	0.01	0.005	0.000	0.004	0.833
3	153	0.00	0.78	0.047	0.010	0.100	2.153
3.1	16	0.00	0.14	0.040	0.002	0.042	1.042
4	24	0.00	0.50	0.060	0.005	0.072	1.202
5	57	0.01	3.31	0.527	0.481	0.694	1.316



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Prepared for: ***European Uranium Resources Ltd.***	File Name: Fig 14.15.jpeg	**Figure 14.15**
Project: Kuriskova Uranium Project	Project Number: 114-310990	**Histogram of Sample Lengths**
Project Location: Slovak Republic	Date of Issue: June 2011	**Within Main Zone North Wireframe**

14.5 Bulk Density Measurements (Specific Gravity)

A total of 4,845 samples were analyzed for bulk density (specific gravity) by wet methods. In 2007, EUU conducted bulk density tests on 155 samples by wet method and paraffin wax method. When compared, the two tests showed good correlation. Based on the results of this test, EUU decided to use wet method for all the future samples.

Table 14.11 summarizes the average bulk density by domain (within domain wireframes). While there is some variation, it was not considered significant and an average density of 2.75 tonnes per cubic meter (t/m^3) was used for all domains in the calculation of the geologic resources. Table 14.12 summarizes the average bulk density for all samples analyzed to date.

Table 14.11. Bulk Density (Specific Gravity) by Domain

Domain Statistics On Specific Gravity							
Domain (Zcode)	Number of Values	Minimum (t/m^3)	Maximum (t/m^3)	Mean (t/m^3)	Variance	Standard Deviation	Coefficient of Variance
All	650	2.42	3.18	2.76	0.01	0.07	0.03
1	268	2.48	3.06	2.77	0.01	0.07	0.03
1.1	22	2.66	3.18	2.78	0.01	0.09	0.03
1.2	30	2.53	2.97	2.76	0.01	0.09	0.03
2	114	2.54	2.98	2.74	0.00	0.06	0.02
2.1	29	2.70	2.84	2.76	0.00	0.02	0.01
3	116	2.59	2.98	2.76	0.00	0.06	0.02
3.1	26	2.74	3.07	2.78	0.00	0.06	0.02
4	29	2.61	3.01	2.80	0.01	0.10	0.03
5	16	2.42	3.01	2.77	0.02	0.15	0.06

Table 14.12. Domain Statistics On Bulk Density For All Samples

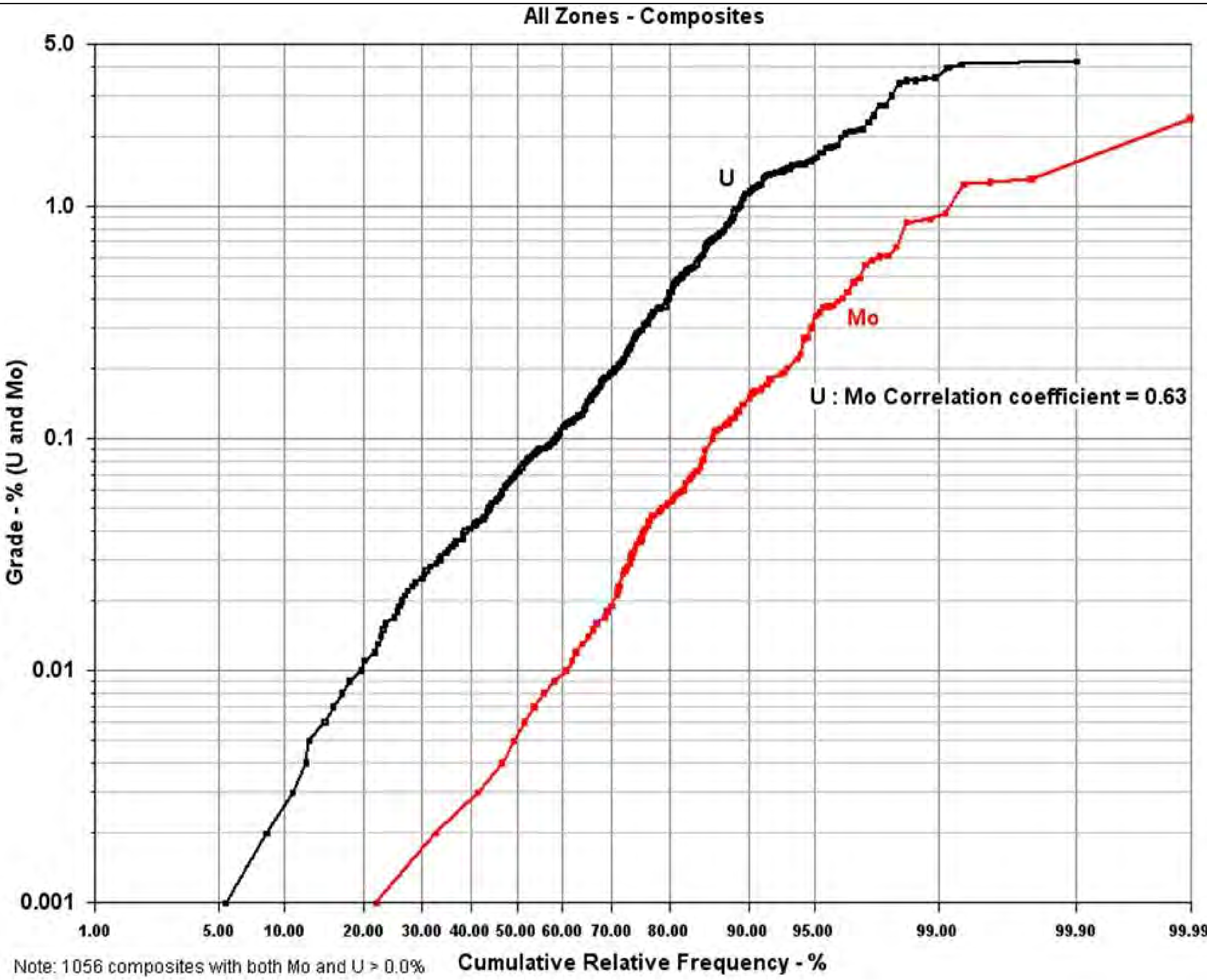
Rock Code	Number of Values	Minimum (t/m^3)	Maximum (t/m^3)	Mean (t/m^3)	Variance	Standard Deviation	Coefficient of Variance
All	5141	1.07	5.26	2.75	0.01	0.11	0.04

14.6 Grade Estimation and Resource Classification

Grades for both uranium and molybdenum were estimated. No attempt was made to develop a separate set of parameters for molybdenum estimation. Molybdenum grades are estimated and coded to the block model as an associated metal with uranium. The resource tabulates molybdenum that is associated with uranium blocks above the uranium 0.05 percent uranium cutoff grade; there is no estimation of molybdenum grades outside the uranium wireframes.

Molybdenum and uranium are associated and are directly proportional, but not on a one-to-one basis (Figure 14.16). Molybdenum values are derived from EUU's 2005 to 2011 drilling. No molybdenum assays are available for historical drilling.

Table 14.13 displays the search parameters and resource confidence classification used for the resource estimation at Kuriskova. The ellipsoidal search volume (SVOL) is initially 50 m, 50 m, and 25 m, reflecting the assumed preferential directions of continuity along strike and down dip, with a two-to-one anisotropy. The first axis with a 50 m search is oriented down dip. The second orthogonal axis, also with a 50 m search, is oriented along strike. For all the zones other than Zones 2 and 3, only model block positions within the wireframed domains were estimated and only the relevant domain composites were used. The wireframe boundaries are exact as drill hole were “snapped” to during their creation and there is no extrapolation beyond these boundaries. Zones 2 and 3 were estimated without hard boundary wireframe using domain blocks created within tight search ellipse as described earlier in Section 14.1. The ellipsoidal SVOL for these two zones is 20 m, 15 m, and 2 m with no second and third search. This approach was taken to be conservative and avoids getting extrapolated blocks in the resource. A variety of grade estimation weighting methodologies were tested including inverse to the distance, with various powers, and kriging were used. Inverse to the distance power of two was used to estimate resource of Main Zone North, Main Zone South, and Zone 45. In the remaining zones, inverse to the power of three was used in previous resource estimates. Since these zones are not updated, it was not felt necessary to change the estimation method for these zones. In future drilling, estimation methods will be reviewed and changed to the best suited method for Kuriskova. Again, to preserve local grade variation, a search neighborhood strategy with three SVOLs of increasing volumes was also used. Only blocks not estimated with the first set of parameters were estimated with the subsequent expanded search. In order to preserve this local variation of grades and have a requirement for grade assignment using data from more than one drill hole, a minimum of four 0.5 m composites were required, with a maximum of three from any given hole, for estimation with the first two SVOLs.



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TETRA TECH 350 Indiana Street, Suite 500 Golden, CO 80401 (303) 217-5700 (303) 217-5705 fax	***European Uranium Resources Ltd.***	Fig 14.16.jpeg	**Figure 14.16** **Uranium and Molybdenum Cumulative Frequency Plot – All Zones**
	Project: Kuriskova Uranium Project	Project Number: 114-310990	
	Project Location: Slovak Republic	Date of Issue: May 2010	

Table 14.13. Search Neighborhood/ Confidence Classification

SVOL	Search Distance (m)				Minimum Number of Composites	Maximum from One Drill Hole	Distance from Nearest Drill Hole
	Class	X	Y	Z			
1	Indicated	50	50	25	4	3	<30
1	Inferred	50	50	25	4	3	>30
2	Indicated	100	100	50	4	3	<15
2	Inferred	100	100	50	4	3	>15
3	Inferred	200	200	100	1	3	

The interpolation methodology and search neighborhood strategy were selected subsequent to experimentation and are intended to preserve the variation of grades observed primarily in the Main Zone. The search ranges were defined based on results of variogram and jackknifing validation of variogram parameters. Figure 14.17 and Figure 14.18 are pair wise relative variograms showing strike direction and down dip direction.

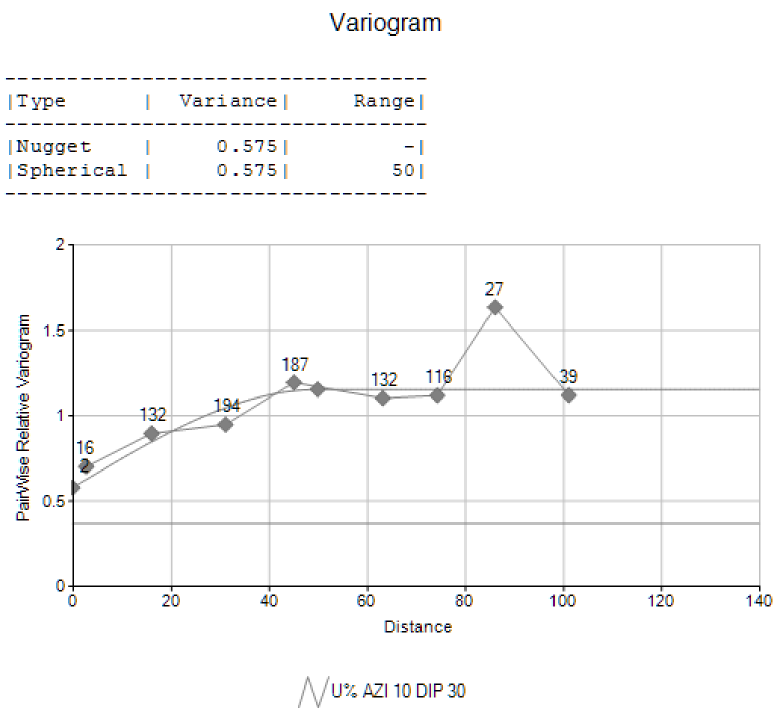


Figure 14.17. Pairwise Relative Variogram in Strike Direction

Variogram

Type	Variance	Range
Nugget	0.157	-
Spherical	0.579	68

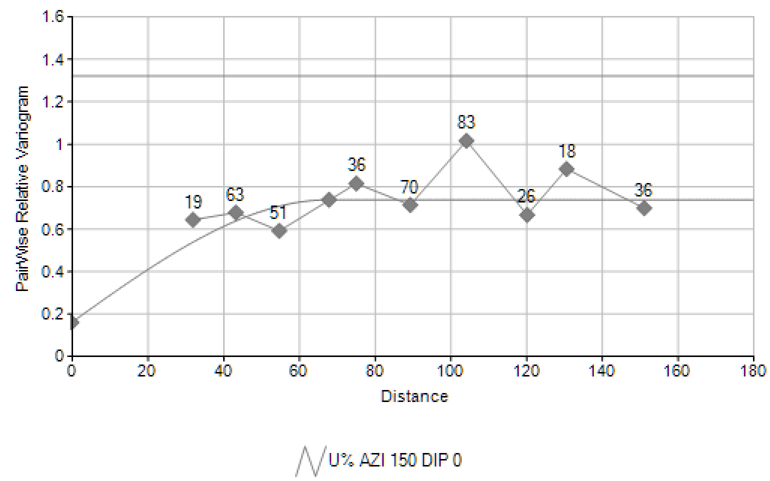
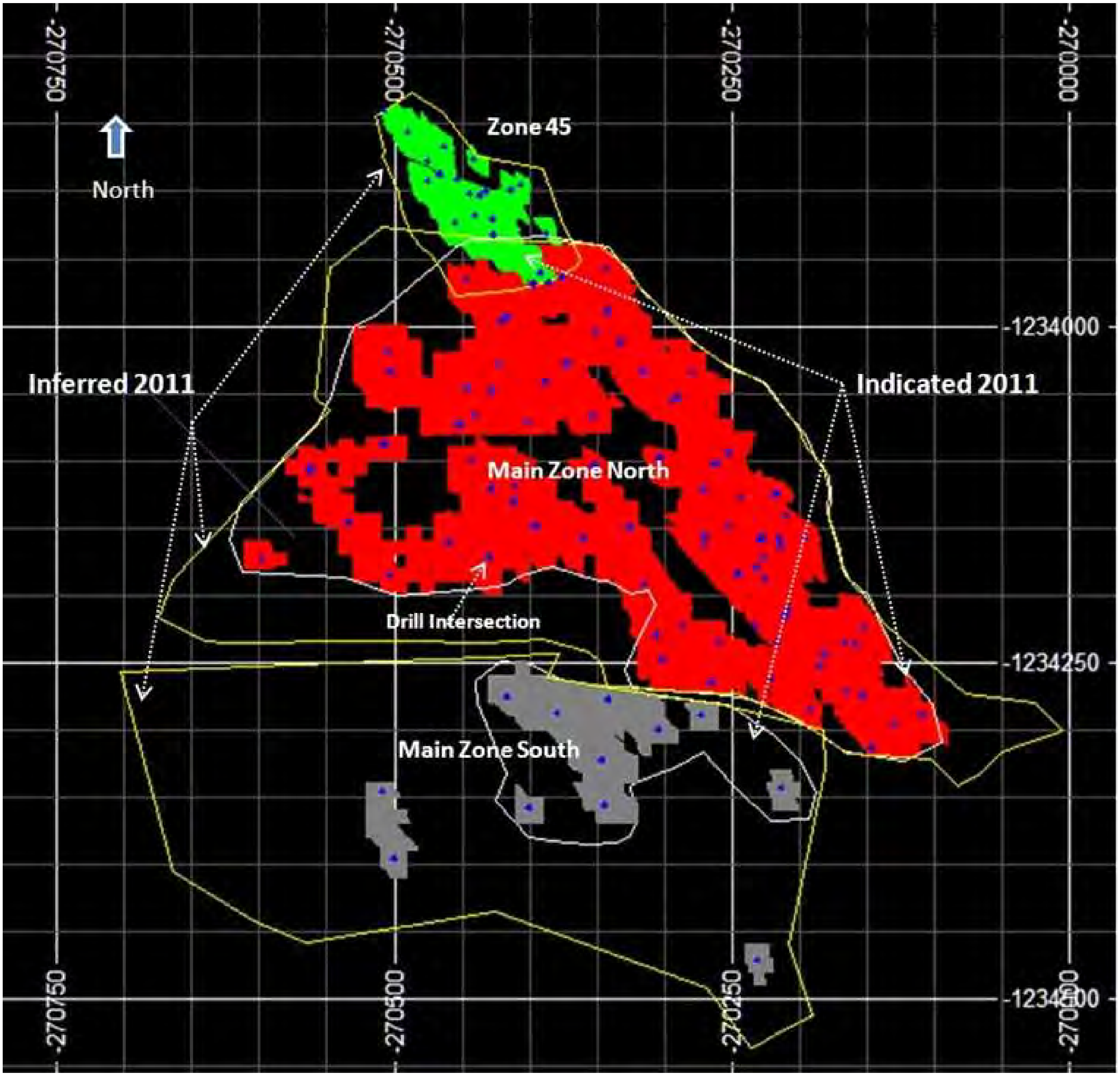


Figure 14.18. Pairwise Relative Variogram In Down Dip Direction

EUU supplemented numerical and statistically derived resource classifications with geological interpretation to avoid a “spotty” representation. For indicated classification using numerical rules, a block grade must be estimated with the rules of the first SVOL, with the additional requirement that at least one drill hole is within 30 m of the block, or estimated with the rules of the second SVOL with one drill hole within 15 m of the block. Parent cells were estimated; that is, sub-cells of the initial 10 m by 10 m block all have the same value. Geological and data considerations were used to adjust (smooth) the numerical and statistical derived classification to avoid a “spotty” representation. Wireframes, based on block estimation attributes and broader geological and data considerations were constructed and used to adjust the classifications. With the numerical classification as a background and with consideration to geologically interpreted mineralization continuity, strings were created restricting the indicated classification of the Main Zone North, Main Zone South, and Zone 45 (Figure 14.19). Using these strings, an indicated classification wireframe was created. Blocks within this wireframe were assigned FCLASS=2 for indicated, and blocks outside this wireframe were assigned FCLASS=3 for inferred. Taking into account the amount, distribution, and quality of data, Tetra Tech is of the opinion that this has produced a result reflecting the level of geological and resource estimation confidence and is commensurate with CIM Standards. All of the estimated indicated resource is restricted to the Main Zone, Zone 45, and Zones 2 and 3 in Hanging Wall North. The percentage of total indicated resource by these zones is given Table 14.14.



Issued by:	Prepared for:	File Name:	Figure 14.19 Representation of the Main Zone and Zone 45 Indicated Block Assignments
TETRA TECH 350 Indiana Street, Suite 500 Golden, CO 80401 (303) 217-5700 (303) 217-5705 fax	***European Uranium Resources Ltd.***	Fig 14.19.jpeg	
	Project: Kuriskova Uranium Project	Project Number: 114-310990	
	Project Location: Slovak Republic	Date of Issue: June 2011	

Table 14.14. Indicated Resources By Zones

Domain	% of Total Indicated Resource
Main Zone	90.29%
Zone 45	3.29%
Hanging Wall	6.42%

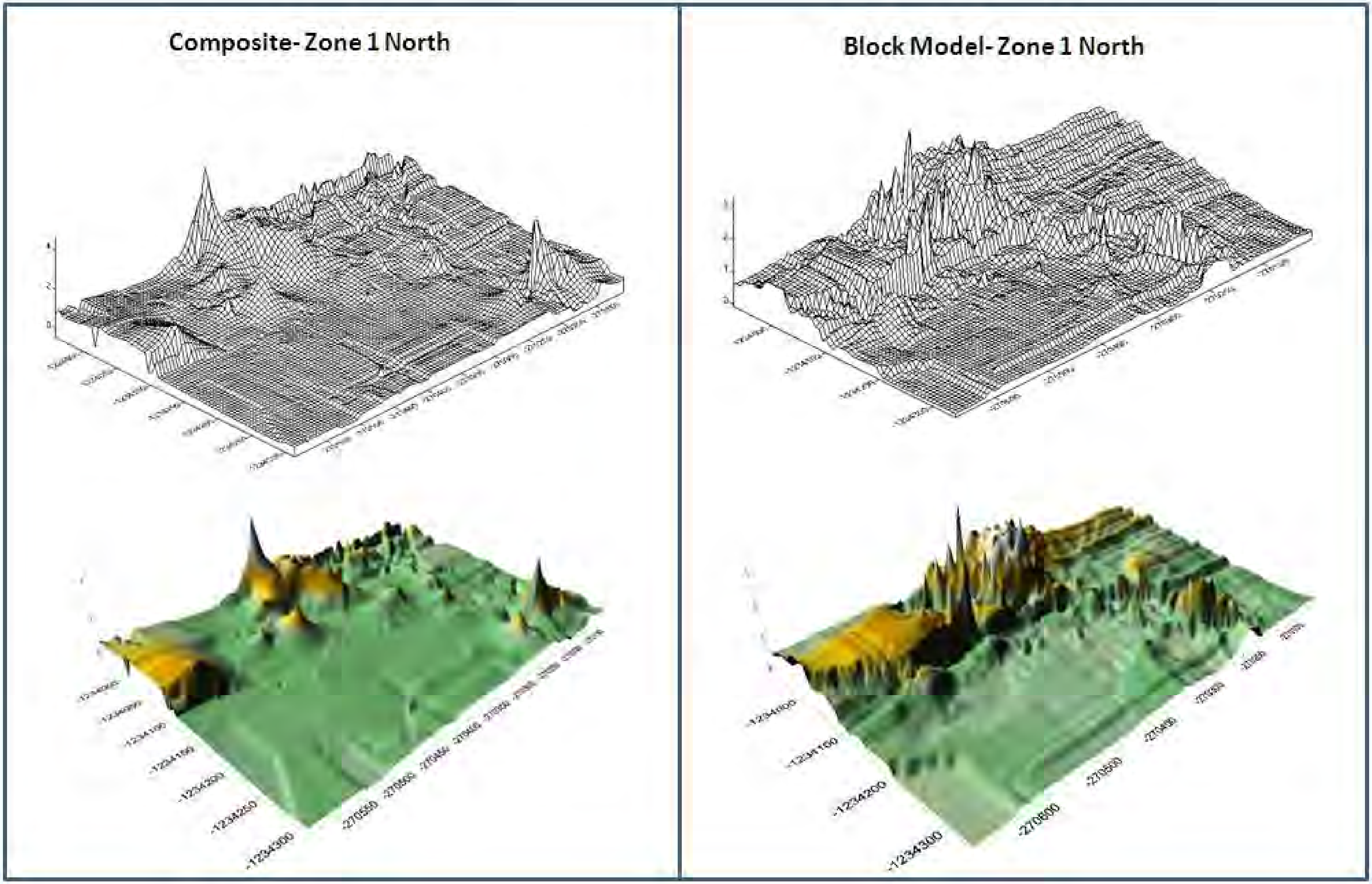
A Datamine Studio3 block model was created with the origins and extents noted in Table 14.15.

Table 14.15. Block Model Parameters

Parameter	Northing	Easting	Elevation
Minimum Coordinates	-1,234,550	-270,800	-280
Maximum Coordinates	-1,231,950	-269,000	690
Block Size	10	10	2

14.7 Resource Model Validation

The Kuriskova block model was validated through a visual comparison between the estimated block grades and the grades of the composites. These were examined in some detail on screen and the distribution of grades in the model appears to honor the distribution of composited values given the controlling anisotropies and wireframed domains derived from geological interpretations. The local variation of grades appears to be relatively well preserved. Figure 14.20 (3D representation of percent uranium distribution) is comparison of percent uranium between raw data and estimated block model grade distribution. It can be seen that drill hole grades are preserved in block estimate with very slight smoothening at places relative to the original assay data. The comparison of domain composite and model block average on Table 14.16 is reasonable.



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(303) 217-5700 (303) 217-5705 fax

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Project Location: Slovak Republic	Date of Issue: June 2011

Figure 14.20
Visual Comparison %U Between Composites and Block Model

Table 14.16. Comparison of Composite Average Grades with Block Model Grades by Domain

Domain	Composite (% U)			Model (% U)		
	No Cut off	% U > 0.03	% U > 0.05	No Cut off	% U >0.03	% U > 0.05
1	0.487	0.593	0.751	0.411	0.421	0..441
1.1	0.267	0.294	0.342	0.1775	0.178	0.179
1.2	0.091	0.319	0.342	0.107	0.169	0.21
2	0.217	0.217	0.294	0.166	0.166	0.189
3	0.190	0.190	0.268	0.167	0.167	0.193
4	0.042	0.126	0.176	0.042	0.074	0.093
5	0.653	0.697	0.768	0.445	0.447	0.451
ALL	0.332	0.373	0.598	0.294	0.309	0.323

14.8 Resource Statement

Table 14.17 and Table 14.18 detail the classified resources at the Project. Resources are stated at a 0.05 percent uranium cutoff grade, which is approximately 0.06 percent U_3O_8. The 0.05 percent uranium cutoff equates to approximately 1.18 lbs of U_3O_8 per tonne of in situ-mineralized material. At a uranium price of US$60/lb U_3O_8, the cutoff grade equals an in situ value of approximately US$70/tonne; which is deemed by Tetra Tech to be sufficient to define a "reasonable potential for economic extraction;" a necessary condition for resource statement. Tetra Tech cautions that it may be appropriate to use either a higher or lower cutoff grade to state resources, and that will only be determined from the mining scoping studies. Tetra Tech believes that this uranium resource update for the Project is NI 43-101 compliant and meets CIM standards and definitions for calculating mineral resources.

Table 14.17. Summary of Indicated Classified Resources at 0.05 Cut Off Percent Uranium

Geological Domain	Sub-Domain	Model Zone (ZCODE)	% U	Tonnes ('000)	% U_3O_8	U_3O_8 ('000 lbs)	% Mo	Tonnes ('000)	Mo ('000 lbs)	Current Resource Update (Year)	Previous Resource Update (Year)
Main Zone	ZONE1N (Main Zone North)	1	0.507	1790	0.598	23,601	0.056	1,790	2,210	2011	2010
	UP MAIN ZONE	1.2	0.211	54	0.248	296	0.033	54	39	2010	2008
	ZONE1S (Main Zone South)	1.1	0.339	207	0.400	1,824	0.073	207	333	2011	2009
Hanging Wall (North)	ZONE2N(43) (HW North)	2	0.279	109	0.329	791	0.016	82	29	2011	2010
	ZONE3N(44) (HW North)	3	0.403	99	0.475	1,037	0.025	99	55	2011	2010
Zone 45	ZONE45 (New Zone)	5	0.523	69	0.617	938	0.425	69	647	2011	2010
Total Main Zone		1+1.1+1.2	0.482	2,051	0.569	25,721	0.057	2,051	2,582		
Total Hanging Wall (North)		2+3	0.338	208	0.399	1,828	0.021	181	83		
Total Zone 45		5	0.523	69	0.617	938	0.425	69	647		
Total Indicated		All	0.471	2,328	0.555	28,487	0.065	2,301	3,312		

Table 14.18. Summary of Inferred Classified Resources at 0.05 Cut Off Percent Uranium

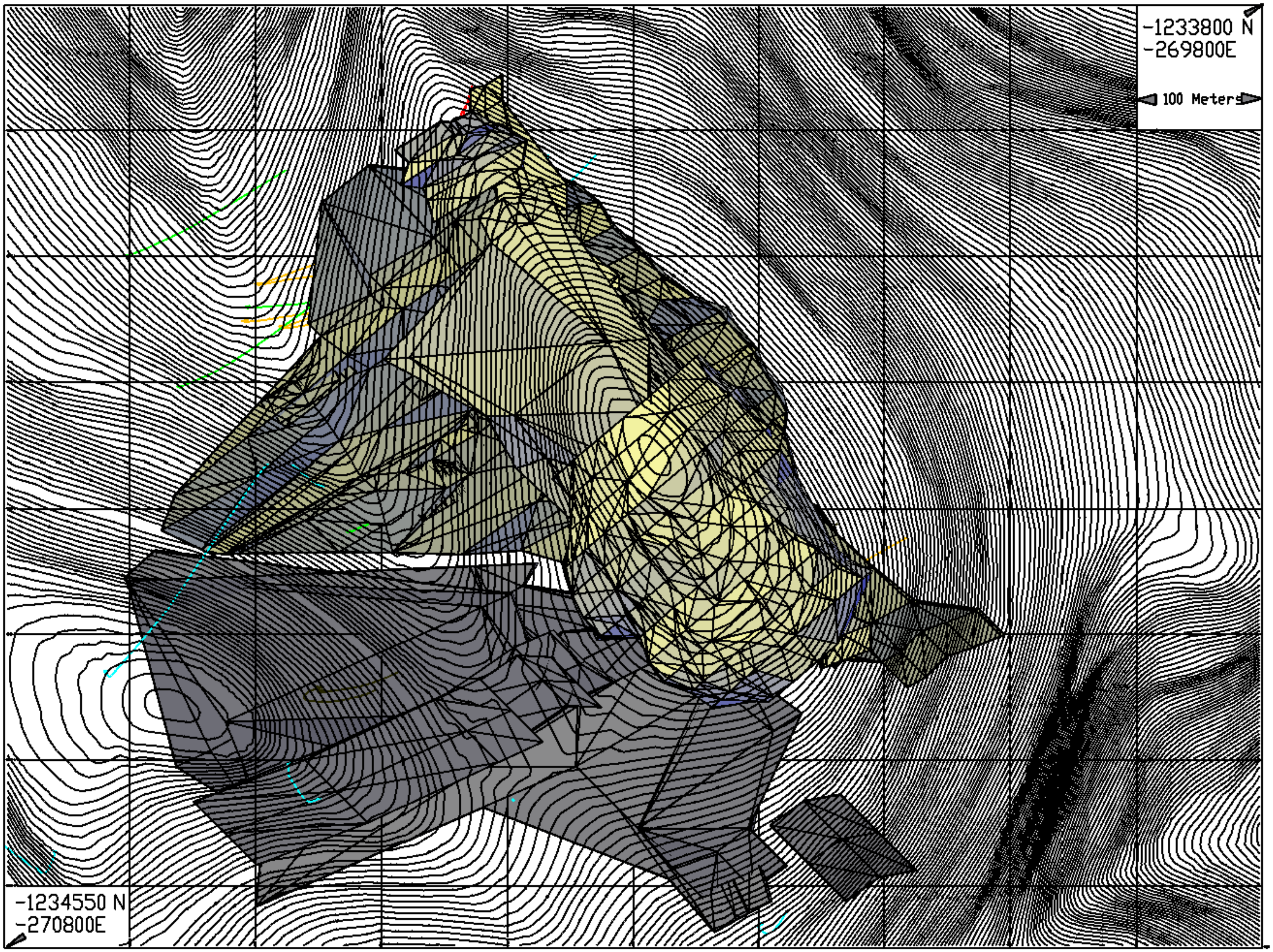
Geological Domain	Sub-Domain	Model Zone (ZCODE)	% U	Tonnes ('000)	% U_3O_8	U_3O_8 ('000 lbs)	% Mo	Tonnes ('000)	Mo ('000 lbs)	Current Resource Update (Year)	Previous Resource Update (Year)
Main	ZONE1N (Main Zone North)	1	0.194	490	0.229	2,471	0.017	490	184	2011	2010
	UP MAIN ZONE	1.2	0	0						2010	2008
	ZONE1S (Main Zone South)	1.1	0.156	1,641	0.184	6,655	0.024	1,612	853	2011	2009
Hanging Wall	ZONE2N(43) (HW North)	2	0.215	130	0.254	727	0.024	110	58	2011	2010
	ZONE3N(44) (HW North)	3	0.153	230	0.180	915	0.047	185	192	2011	2010
	ZONE 4 (HW North)	4	0.095	52	0.112	128	0.071	52	81	2010	2008
	ZONE2S (HW South)	2.1	0.087	181	0.103	409	0.003	181	12	2008	2008
	ZONE3S (HW South)	3.1	0.106	336	0.125	926	0.024	288	155	2008	2008
Zone 45	ZONE45 (New Zone)	5	0.426	39	0.502	432	0.378	39	325	2011	2010
Total Main Zone		1+1.1+1.2	0.165	2,131	0.194	9,127	0.022	2,102	1,037		
Total Hanging Wall		2+3+4+2.1+3.1	0.129	929	0.152	3,105	0.044	855	823		
Total Zone 45		5	0.426	39	0.502	432	0.378	39	325		
Total Inferred		All	0.157	3,099	0.185	12,664	0.033	2,996	2,185		

Note: In situ uranium resources refers to global in-place resources to which a mine design has not yet been applied; although the above stated resources meet the definition of having the "potential for economic extraction" at the cutoff provided.

14.9 Tetra Tech Review of Resource Estimates for 2010 and 2011

Tetra Tech has reviewed the Kuriskova 2011 resource estimate generated by EUU. In doing so, Tetra Tech is relying on its previous 2010 mineral resource estimate of the Kuriskova uranium deposit using independent software and methodologies. Tetra Tech feels that incremental refinement of the 2010 model along with reinterpreted wireframes and new drilling data do not alter the previous conclusion that EUU's estimate has been professionally done to accepted standard practices. The results are prudent and reasonable and are in compliance with 43-101 standards. Sections 14.9 through 14.16 describe the independent work performed by Tetra Tech in 2010. It has been abstracted from Tetra Tech's 2010 report with the exception of the resource table comparing Tetra Tech's with EUU's 2010 resource estimates. This omission was done on purpose to help minimize the reader's confusion with multiple resource tables. Section 14.17 contains a figure that compares EUU's 2010 and 2011 resource estimates. The Tetra Tech 2010 model used geologic wireframe and drill hole interval coding and assays produced by EUU. The Tetra Tech resource model was created in commercial mining software, MicroModel. Also used was GemCom for 3D visualization and the calculation of the percentage of blocks within wireframes. The resource model extents are shown in Figure 14.21.

Tetra Tech Model Extents



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(303) 217-5700 (303) 217-5705 fax

Prepared for: *European Uranium Resources Ltd.*	File Name: Figure 14.21.dwg
Project: Kuriskova Uranium Project	Project Number: 114-310990
Project Location: Slovak Republic, Europe	Date of Issue: 05/09/2010

Figure 14.21
Tt Model Extents Showing Topography, Wireframes, and Drillhole Traces

Interpolation characteristics used in the model have been defined based on the geology, drill hole spacing, and geostatistical analysis of the data. The mineral resources have been classified by a combination of their proximity to the sample locations and kriging error and are reported, as required by NI 43-101, according to CIM standards on Mineral Resources and Reserves. This model review section presents:

- A resource model was set up in MicroModel, based on 5x5x1-m blocks.
- Tetra Tech coded drill hole assays and onehalf m composites inside of 3D wireframes received from the EUU geologists
- One-half meter “rock zone” composites were calculated to be within drill hole interval zone designations
- Zone designations were recoded as integer numbers for use in MicroModel.
- Statistics for drill hole assay and composite data were generated and analyzed.
- A separate analysis was done to confirm a high cut cap of 4.2 percent uranium on composites employed by EUU. This cap was applied to 12 composite values.
- Correlation between uranium and molybdenum grades and their similar distribution shapes was analyzed. The future use of regression of missing molybdenum values is suggested.
- Indicator variograms based on a median cut were chosen to best show the spatial structure of uranium.
- Model validation (jackknifing) was used to help determine estimation parameters, such as the anisotropy ranges along with additional search parameters to be used in estimation.
- Ordinary kriging was used to estimate uranium and molybdenum. Molybdenum estimation was based on uranium kriging parameters.
- The statistical relationship between assays, composites, and kriged estimates was compared. It was determined that there was no apparent anomaly in the sequence of going from assay to composite to blocks.
- A resource classification of indicated and inferred was developed based on the combination:
- Selecting a series of increasing search ranges via jackknifing.
- Adjustment of assigned resource classes using kriging errors.
- Validation of the kriged model was performed using statistics and visual inspection of blocks to composite values in section and plan.
- Bulk density of 2.75 t/m^3 was applied to all zones.
- Grade-tonnage tables and graphs were developed from the block model at various cutoff grades and resource classification codes.
- A comparison of the 2010 and 2011 resources is shown in table form.

14.10 Tetra Tech Block Model

Table 14.19 shows the Tetra Tech block model parameters. The model block size was chosen to respect the complex shapes of the wireframes. The modeling was done primarily with

MicroModel. In addition, due to thin zones, Tetra Tech utilized GemCom to determine the proportion of the block falling within the wireframes.

Table 14.19. Tetra Tech Block Model Parameters

Parameters	Northing	Easting	Elevation
Minimum Coordinates	-1,234,550	-270,800	-280
Maximum Coordinates	-1,233,800	-269,800	660
Block Size	5	5	1
Number of Blocks	150	200	940

14.11 Drill Hole Assay and Composite Data

The drill hole sample lengths varied, but were nominally were 0.1-m to 0.3-m long with the mode being 0.3 m. Two metals, uranium measured in percent (% U), and molybdenum also measured in percent (% Mo) were analyzed. Table 14.20 shows the drill hole statistics for depth and orientation for the 133 drill holes.

Table 14.20. Drill Hole Statistics

	NORTHING	EASTING	ELEVATION	AZIMUTH	DIP	DEPTH
MINIMUM	-1234579.2	-271030.1	504.3	0.0	56.0	32.2
MAXIMUM	-1233449.5	-270132.1	619.0	353.9	90.0	956.0
AVERAGE	-1234170.0	-270393.9	588.2	100.9	83.1	369.1
RANGE	1129.7	898.0	114.7	353.9	34.0	923.8
TOTAL COUNT	133					
TOTAL LENGTH	49088.5					

Table 14.21 shows the raw assay sample statistics for the drill holes. Note that molybdenum assays are missing in a great percentage of the intervals. These missing molybdenum assays have an impact on the estimated average grades in the Tetra Tech model.

Table 14.21. Drill Hole and Assay Sample Statistics

TOTAL DRILLHOLES = 133

AVERAGE VALUES OF SELECTED DATA

LABEL	NUMBER	AVERAGE	STD DEVIATION	MIN. VALUE	MAX. VALUE	# MISS.
U%	146398	0.00631	0.09931	0.00000	14.50000	0
Mo%	6831	0.01545	0.11227	0.00002	3.76000	139567

Table 14.22 shows the recoding of the mineral zones as required by MicroModel. The Tetra Tech zone codes are essentially the original codes multiplied by 10. Two particular cases should be noted. The original zones 20 and 30 have been combined to a Tetra Tech code of 23. Zone 50, has been given a Tetra Tech code of 45

Table 14.22. Tetra Tech Zone Recoding

Zones Re-Coded			
TT Composite Codes	Original Zone Codes	TT Block Codes	Zone Name
10	1	10	Main Zone North (1N)
11	1.1	11	Main Zone South (1S)
12	1.2	12	Main Zone Up
20,30	2,3	23	HW North (2N,3N)
21	2.1	21	HW South (2S)
31	3.1	31	HW South (3S)
40	4	40	HW North
50	5	45	New Zone

Table 14.23 shows the assay statistics for uranium broken out by Tetra Tech zone codes. Zone 10 (Main Zone) appears to be bimodal lognormal (peaks at 0.001 and 0.08 percent uranium). The maximum uranium assay is 14.5 percent Uranium. The drill hole dataset has zone codes were assigned by EUU. These zones were recorded with the Tetra Tech zone codes. The codes were then used with MicroModel's Rock Unit compositing method. This method prioritizes each composite such that their lengths are optimized to fall within each rock unit with a target length of 0.5 m. The method set a minimum acceptable composite length of 0.1 m. The maximum acceptable composite length was set at 0.75 m. Table 14.24 shows the 0.5 meter composite statistics for uranium by zone. Compositing has averaged the two modes into a single one with its peak at approximately 0.06 percent Uranium.

Table 14.23. Sample Assay Statistics for Percent Uranium (By Zone)

DATA TYPE IS SAMPLE
CURRENT LABEL : U%

	SAMPLE COUNT				UNTRANSFORMED STATISTICS						LOG-TRANSFORMED STATS			LOG-DERIVED	
ROCK TYPE	MISSING	BELOW LIMITS	ABOVE LIMITS	INSIDE LIMITS	MINIMUM	MAXIMUM	MEAN	VARIANCE	STD. DEV.	COEF. OF VAR	LOG MEAN	LOG VAR.	LOG STD.DEV	MEAN	COEF. OF VAR.
10	0	8	0	825	0.000010	14.500	0.50690	1.2265	1.1075	2.1848	-2.3356	4.8273	2.1971	1.0812	11.1297
11	0	0	0	64	0.000440	1.6420	0.22277	0.09407	0.30671	1.3768	-2.3397	2.3182	1.5226	0.3071	3.0262
12	0	7	0	119	0.000400	2.3000	0.12492	0.12094	0.34776	2.7839	-4.8445	5.9769	2.4448	0.1563	19.8299
20	0	0	0	338	0.03000	2.3979	0.22774	0.11596	0.34054	1.4953	-2.0902	1.0619	1.0305	0.2103	1.3754
21	0	1	0	84	0.000620	0.34900	0.03866	0.00396	0.06292	1.6274	-4.3654	2.5305	1.5907	0.0450	3.3999
30	0	0	0	334	0.03000	3.5860	0.16086	0.08845	0.29740	1.8488	-2.4107	0.8406	0.9168	0.1366	1.1479
31	0	0	0	147	0.000360	0.80200	0.07057	0.01882	0.13718	1.9439	-3.9931	3.4053	1.8453	0.1012	5.3965
40	0	19	0	151	0.000500	1.0030	0.04916	0.01617	0.12718	2.5869	-4.7268	3.0135	1.7359	0.0400	4.3998
50	0	0	0	61	0.00100	3.1200	0.51184	0.78886	0.88818	1.7353	-3.0938	7.2830	2.6987	1.7293	38.1362
ALL	0	35	0	2123	0.000010	14.500	0.29688	0.57752	0.75995	2.5598	-2.8360	4.2309	2.0569	0.4865	8.2330

```
LOWER BOUND   UPPER BOUND        40        80       120       160       200       240       280       320       360       400
    >=             <      +---------+---------+---------+---------+---------+---------+---------+---------+---------+---------+
  0.0000          0.0000|
  0.0000          0.0000|
  0.0000          0.0000|
  0.0000          0.0001|
  0.0001          0.0001|
  0.0001          0.0002|
  0.0002          0.0003|
  0.0003          0.0004|*
  0.0004          0.0007|****
  0.0007          0.0011|*************************
  0.0011          0.0018|*******************
  0.0018          0.0029|*****************
  0.0029          0.0047|***************
  0.0047          0.0075|************
  0.0075          0.0120|***************
  0.0120          0.0193|************
  0.0193          0.0310|*****************************
  0.0310          0.0498|***********************************************************************
  0.0498          0.0798|****************************************************************
  0.0798          0.1281|***********************************************************
  0.1281          0.2056|**************************************************
  0.2056          0.3299|*************************************
  0.3299          0.5294|************************************
  0.5294          0.8494|**********************
  0.8494          1.3630|*************
  1.3630          2.1872|******************
  2.1872          3.5097|********
  3.5097          5.6318|****
  5.6318          9.0371|*
  9.0371         14.5015|*
                          +---------+---------+---------+---------+---------+---------+---------+---------+---------+---------+
                          0        40        80       120       160       200       240       280       320       360       400
```

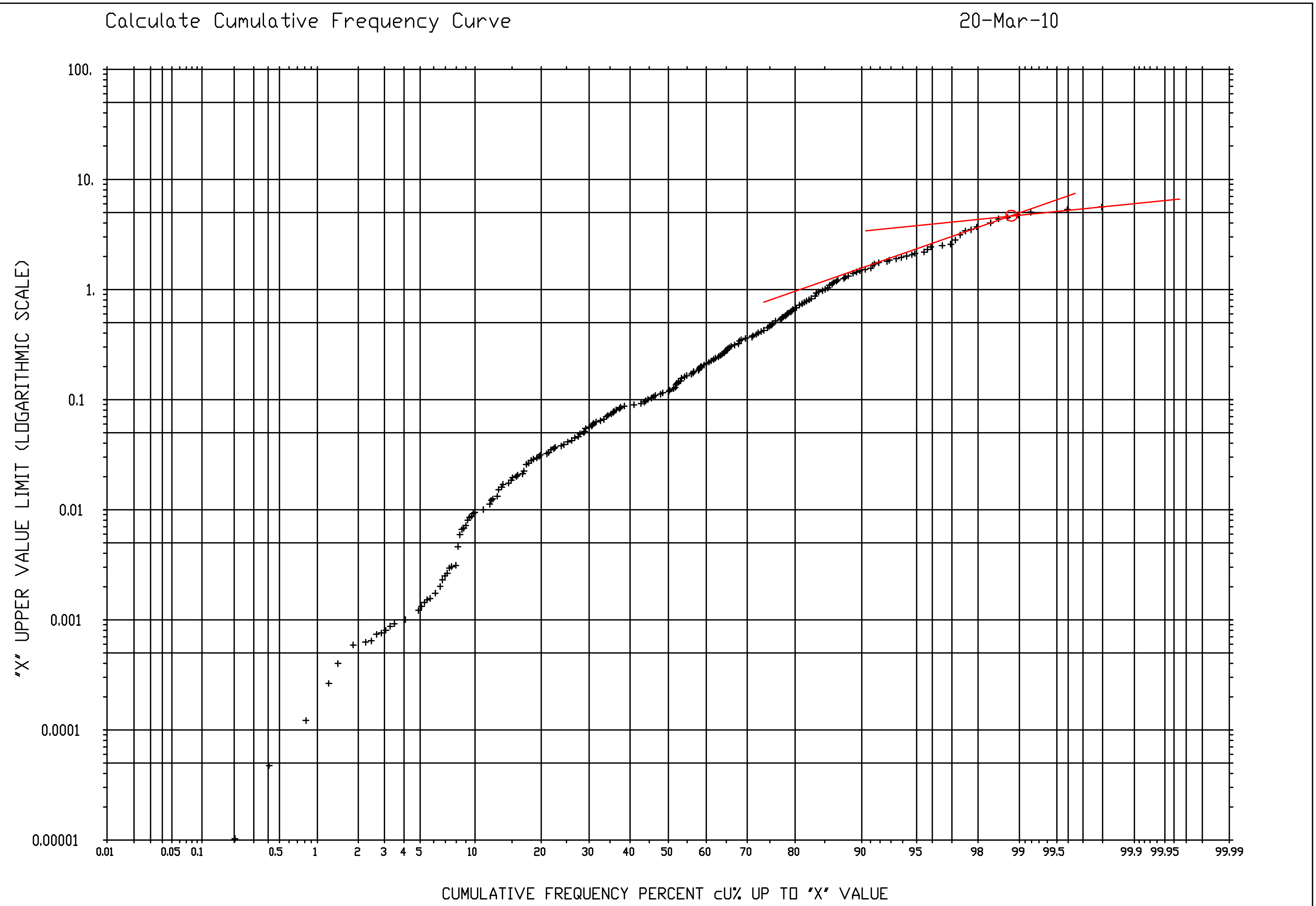
Table 14.24. 0.5 Meter Composite Statistics for Percent Uranium (By Zone)

DATA TYPE IS COMPOSITE
CURRENT LABEL : cU%

ROCK TYPE	COMPOSITE COUNT MISSING	BELOW LIMITS	ABOVE LIMITS	INSIDE LIMITS	UNTRANSFORMED STATISTICS MINIMUM	MAXIMUM	MEAN	VARIANCE	STD. DEV.	COEF. OF VAR	LOG-TRANSFORMED STATS LOG MEAN	LOG VAR.	LOG STD.DEV	LOG-DERIVED MEAN	COEF. OF VAR.
10	0	0	0	490	0.000010	10.615	0.49819	0.92852	0.96360	1.9342	-2.1679	4.4317	2.1052	1.0491	9.1147
11	0	0	0	49	0.00375	1.0938	0.19139	0.05060	0.22494	1.1753	-2.2736	1.4441	1.2017	0.2119	1.7995
12	0	83	0	73	0.000450	2.1072	0.20041	0.15365	0.39198	1.9559	-3.3775	5.5284	2.3512	0.5415	15.8345
20	2	0	0	211	0.03000	1.9160	0.20993	0.08750	0.29581	1.4091	-2.1761	1.0735	1.0361	0.1941	1.3877
21	0	20	0	52	0.000725	0.28346	0.04192	0.00386	0.06216	1.4828	-4.0493	2.0081	1.4171	0.0476	2.5395
30	1	0	0	261	0.03000	3.5860	0.18561	0.11674	0.34167	1.8408	-2.3408	0.9695	0.9847	0.1563	1.2793
31	0	0	0	70	0.000407	0.58209	0.05296	0.00805	0.08970	1.6938	-3.9045	2.3787	1.5423	0.0662	3.1290
40	1	39	0	85	0.000750	0.60562	0.06233	0.01294	0.11375	1.8248	-4.0870	2.9567	1.7195	0.0736	4.2701
50	0	0	0	43	0.00190	3.1200	0.62676	0.67788	0.82334	1.3136	-2.0684	5.5519	2.3562	2.0289	16.0223
ALL	4	142	0	1334	0.000010	10.615	0.29910	0.44214	0.66494	2.2232	-2.5567	3.3798	1.8384	0.4203	5.3259

```
LOWER BOUND   UPPER BOUND         20        40        60        80       100       120       140       160       180       200
    >=             <       +---------+---------+---------+---------+---------+---------+---------+---------+---------+---------+
   0.0000         0.0000|*
   0.0000         0.0000|
   0.0000         0.0000|
   0.0000         0.0001|*
   0.0001         0.0001|
   0.0001         0.0002|*
   0.0002         0.0003|
   0.0003         0.0004|**
   0.0004         0.0006|*****
   0.0006         0.0010|*********
   0.0010         0.0016|**************
   0.0016         0.0026|**********
   0.0026         0.0041|*************
   0.0041         0.0065|***************
   0.0065         0.0103|******************
   0.0103         0.0164|*************
   0.0164         0.0260|**********************
   0.0260         0.0413|*************************************************************************************
   0.0413         0.0655|**********************************************************************************************
   0.0655         0.1041|***************************************************************************************
   0.1041         0.1653|*********************************************************************
   0.1653         0.2625|***************************************************
   0.2625         0.4168|****************************************************
   0.4168         0.6619|*********************************
   0.6619         1.0511|****************************
   1.0511         1.6692|************************
   1.6692         2.6508|******************
   2.6508         4.2096|*****
   4.2096         6.6851|****
   6.6851        10.6164|*
                           +---------+---------+---------+---------+---------+---------+---------+---------+---------+---------+
                           0        20        40        60        80       100       120       140       160       180       200
```

The maximum composited uranium grade in Zone 10 is 10.6 percent uranium. Figure 14.22 shows the cumulative frequency curve for the uranium composite data. This type of graph has been designed to display a lognormal distribution as a straight line. Breaks in the slope of the curve, such as shown by the red construction lines on the graph represent a significant deviation from a simple lognormal model. The modeled break point is at 4.2 percent uranium has been used as the selected “top-cut” grade for uranium in order to reduce a possible high grade bias over portions of the deposit. All composite grades above this value are assigned the top cut value (i.e., capped). There were 12 values out of 1,334 composites that were capped. This is the same value that was employed by the EUU resource estimate.



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350 Indiana Street, Suite 500
Golden, Colorado 80401
(303) 217-5700 (303) 217-5705 fax

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Project Location: Slovak Republic, Europe

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Figure 14.22

Cumulative Frequency Curve for %U Composites Showing 4.2 %U Break

Table 14.25 shows the statistics uranium composites with the 4.2 percent uranium cut. This is the maximum value.

Table 14.25. Composite Statistics with High Cut for Percent Uranium (By Zone)

DATA TYPE IS COMPOSITE
CURRENT LABEL : cU%cut

ROCK TYPE	COMPOSITE COUNT MISSING	BELOW LIMITS	ABOVE LIMITS	INSIDE LIMITS	UNTRANSFORMED STATISTICS MINIMUM	MAXIMUM	MEAN	VARIANCE	STD. DEV.	COEF. OF VAR	LOG-TRANSFORMED STATS LOG MEAN	LOG VAR.	LOG STD.DEV	LOG-DERIVED MEAN	COEF. OF VAR.
10	0	0	0	490	0.000010	4.2000	0.45797	0.57139	0.75590	1.6506	-2.1780	4.3624	2.0886	1.0032	8.8001
11	0	0	0	49	0.00375	1.0938	0.19139	0.05060	0.22494	1.1753	-2.2736	1.4441	1.2017	0.2119	1.7995
12	0	83	0	73	0.000450	2.1072	0.20041	0.15365	0.39198	1.9559	-3.3775	5.5284	2.3512	0.5415	15.8345
20	2	0	0	211	0.03000	1.9160	0.20993	0.08750	0.29581	1.4091	-2.1761	1.0735	1.0361	0.1941	1.3877
21	0	20	0	52	0.000725	0.28346	0.04192	0.00386	0.06216	1.4828	-4.0493	2.0081	1.4171	0.0476	2.5395
30	1	0	0	261	0.03000	3.5860	0.18561	0.11674	0.34167	1.8408	-2.3408	0.9695	0.9847	0.1563	1.2793
31	0	0	0	70	0.000407	0.58209	0.05296	0.00805	0.08970	1.6938	-3.9045	2.3787	1.5423	0.0662	3.1290
40	1	39	0	85	0.000750	0.60562	0.06233	0.01294	0.11375	1.8248	-4.0870	2.9567	1.7195	0.0736	4.2701
50	0	0	0	43	0.00190	3.1200	0.62676	0.67788	0.82334	1.3136	-2.0684	5.5519	2.3562	2.0289	16.0223
ALL	4	142	0	1334	0.000010	4.2000	0.28432	0.30562	0.55283	1.9444	-2.5603	3.3515	1.8307	0.4129	5.2483

```
   LOWER BOUND   UPPER BOUND         20        40        60        80       100       120       140       160       180       200
       >=            <      +---------+---------+---------+---------+---------+---------+---------+---------+---------+---------+
      0.0000        0.0000|*
      0.0000        0.0000|
      0.0000        0.0000|
      0.0000        0.0001|*
      0.0001        0.0001|
      0.0001        0.0001|*
      0.0001        0.0002|
      0.0002        0.0003|*
      0.0003        0.0005|**
      0.0005        0.0007|*****
      0.0007        0.0012|**********
      0.0012        0.0018|**************
      0.0018        0.0027|*********
      0.0027        0.0042|************
      0.0042        0.0065|***************
      0.0065        0.0100|***************
      0.0100        0.0154|*************
      0.0154        0.0237|***********************
      0.0237        0.0364|********************************************************
      0.0364        0.0561|**********************************************************************************************
      0.0561        0.0864|*******************************************************************************
      0.0864        0.1330|****************************************************************************
      0.1330        0.2047|********************************************************
      0.2047        0.3152|*********************************************
      0.3152        0.4854|*********************************************
      0.4854        0.7473|******************************
      0.7473        1.1507|**************************
      1.1507        1.7718|********************
      1.7718        2.7280|****************
      2.7280        4.2004|********
                            +---------+---------+---------+---------+---------+---------+---------+---------+---------+---------+
                            0        20        40        60        80       100       120       140       160       180       200
```

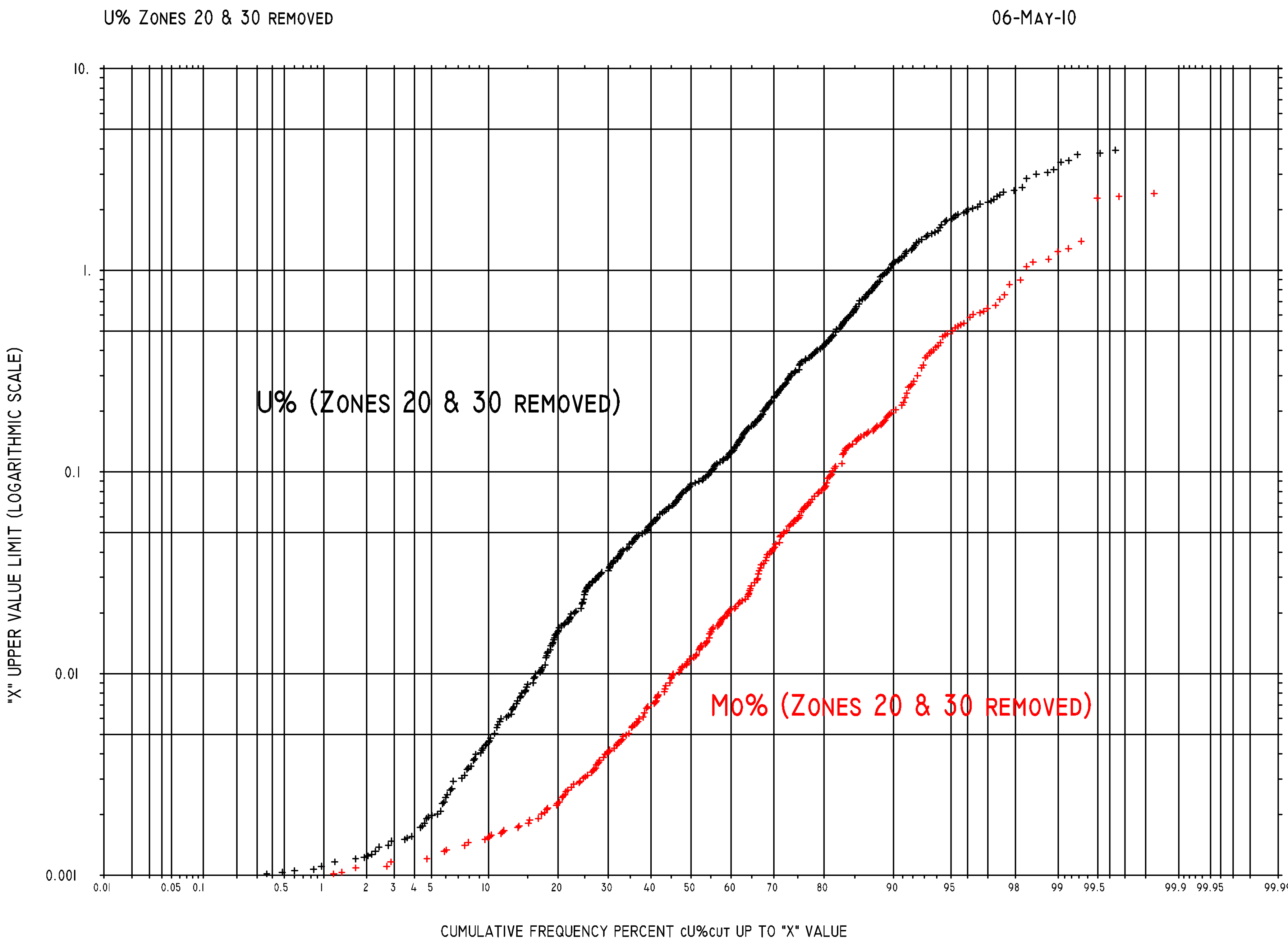
Table 14.26 shows the statistics for molybdenum. There has been no top cutting for molybdenum. Figure 14.23 shows the cumulative frequency plots of uranium and molybdenum side-by-side. The zones 20 and 30 have been removed from this plot as their distributions appear potentially anomalous. This issue of zones 20 and 30 is discussed in more detail in Section 8.0, Section 3.0. Figure 14.23 shows that two metals appear to parallel each other. Molybdenum has grade values at approximately one-tenth that of uranium.

Table 14.26. 0.5 Meter Composite Statistics for Percent Molybdenum (By Zone)

DATA TYPE IS COMPOSITE
CURRENT LABEL : cMo%

ROCK TYPE	COMPOSITE COUNT MISSING	BELOW LIMITS	ABOVE LIMITS	INSIDE LIMITS	UNTRANSFORMED STATISTICS MINIMUM	MAXIMUM	MEAN	VARIANCE	STD. DEV.	COEF. OF VAR	LOG-TRANSFORMED STATS LOG MEAN	LOG VAR.	LOG STD.DEV	LOG-DERIVED MEAN	COEF. OF VAR.
10	92	0	0	398	0.000100	2.3801	0.07361	0.04241	0.20595	2.7977	-4.5034	3.9646	1.9911	0.0804	7.1903
11	3	0	0	46	0.00100	0.11000	0.02007	0.000879	0.02964	1.4774	-4.9489	2.0708	1.4390	0.0200	2.6328
12	97	0	0	59	0.000100	0.60447	0.04095	0.01258	0.11217	2.7392	-5.4326	4.7580	2.1813	0.0472	10.7474
20	54	0	0	159	0.000100	0.32800	0.01522	0.00128	0.03573	2.3469	-5.5886	2.8734	1.6951	0.0157	4.0861
21	31	0	0	41	0.000300	0.03090	0.00274	0.000029	0.00539	1.9706	-6.7170	1.2061	1.0982	0.0022	1.5299
30	38	0	0	224	0.000300	0.78000	0.03444	0.00681	0.08254	2.3969	-4.6359	2.3615	1.5367	0.0316	3.0994
31	22	0	0	48	0.000200	0.12625	0.01866	0.000829	0.02879	1.5433	-5.0301	2.6161	1.6174	0.0242	3.5612
40	63	0	0	62	0.000400	0.50000	0.03678	0.00609	0.07804	2.1215	-5.1014	3.9484	1.9871	0.0438	7.1313
50	6	0	0	37	0.00700	3.1688	0.54465	0.48891	0.69922	1.2838	-1.3581	1.9035	1.3797	0.6661	2.3894
ALL	406	0	0	1074	0.000100	3.1688	0.06165	0.04372	0.20910	3.3917	-4.7960	3.9028	1.9756	0.0582	6.9673

```
LOWER BOUND   UPPER BOUND        10        20        30        40        50        60        70        80        90       100
    >=             <      +---------+---------+---------+---------+---------+---------+---------+---------+---------+---------+
  0.0001         0.0001|**************
  0.0001         0.0002|
  0.0002         0.0003|******
  0.0003         0.0004|************
  0.0004         0.0006|************************************
  0.0006         0.0008|*******************************
  0.0008         0.0011|**********************************************************
  0.0011         0.0016|**********************************************************************************
  0.0016         0.0022|*****************************************************************************************
  0.0022         0.0032|**********************************************************************
  0.0032         0.0045|***********************************************************
  0.0045         0.0063|********************************************************************
  0.0063         0.0089|**************************************************************
  0.0089         0.0126|******************************************************************
  0.0126         0.0178|******************************************************
  0.0178         0.0251|*******************************************************************
  0.0251         0.0355|*****************************************
  0.0355         0.0502|************************************************
  0.0502         0.0709|******************************************
  0.0709         0.1001|***********************************
  0.1001         0.1415|**********************************
  0.1415         0.1998|******************************
  0.1998         0.2823|*****************
  0.2823         0.3988|*************
  0.3988         0.5634|**************
  0.5634         0.7958|************
  0.7958         1.1242|*****
  1.1242         1.5881|*****
  1.5881         2.2434|
  2.2434         3.1691|****
                          +---------+---------+---------+---------+---------+---------+---------+---------+---------+---------+
                          0        10        20        30        40        50        60        70        80        90       100
```



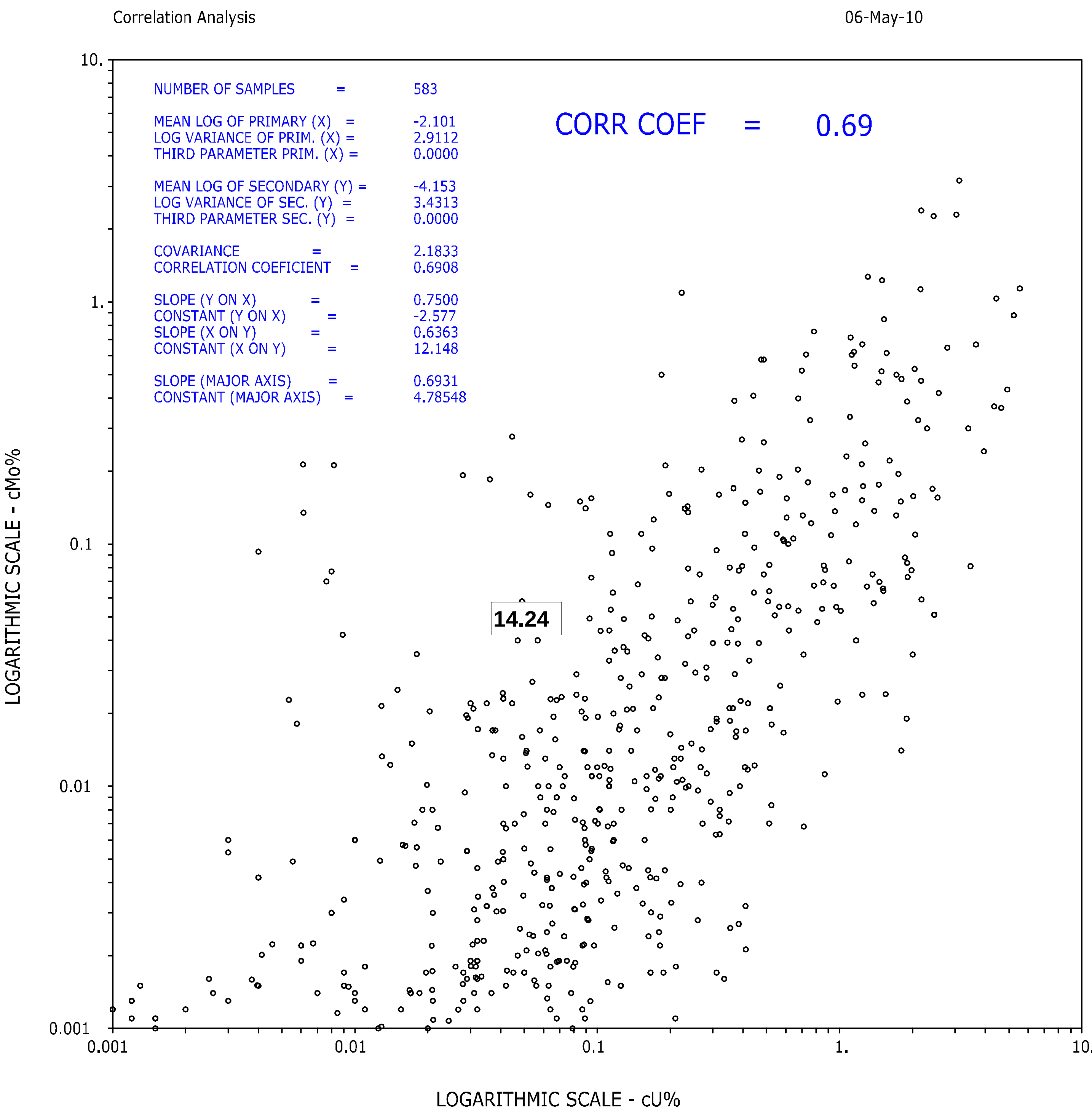
Issued by:	Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	*European Uranium Resources Ltd.*	Figure 14.23.dwg	**Figure 14.23** **Cumulative Frequency Curve for %U & %Mo Composites Side-by-Side**
	Project: Kuriskova Uranium Project	Project Number: 114-310990	
	Project Location: Slovak Republic, Europe	Date of Issue: 05/09/2010	

While their distributions appear to have similar shapes, the two metals have a moderate correlation of 0.69 percent within a range of 0.001 to 10 percent. This is shown in the correlation plot in Figure 14.24. In a correlation plot, the uranium and molybdenum composite values are plotted as a point. In such a graph, as the correlation coefficient approaches unity, the plotted scatter of points becomes tighter and more linear.

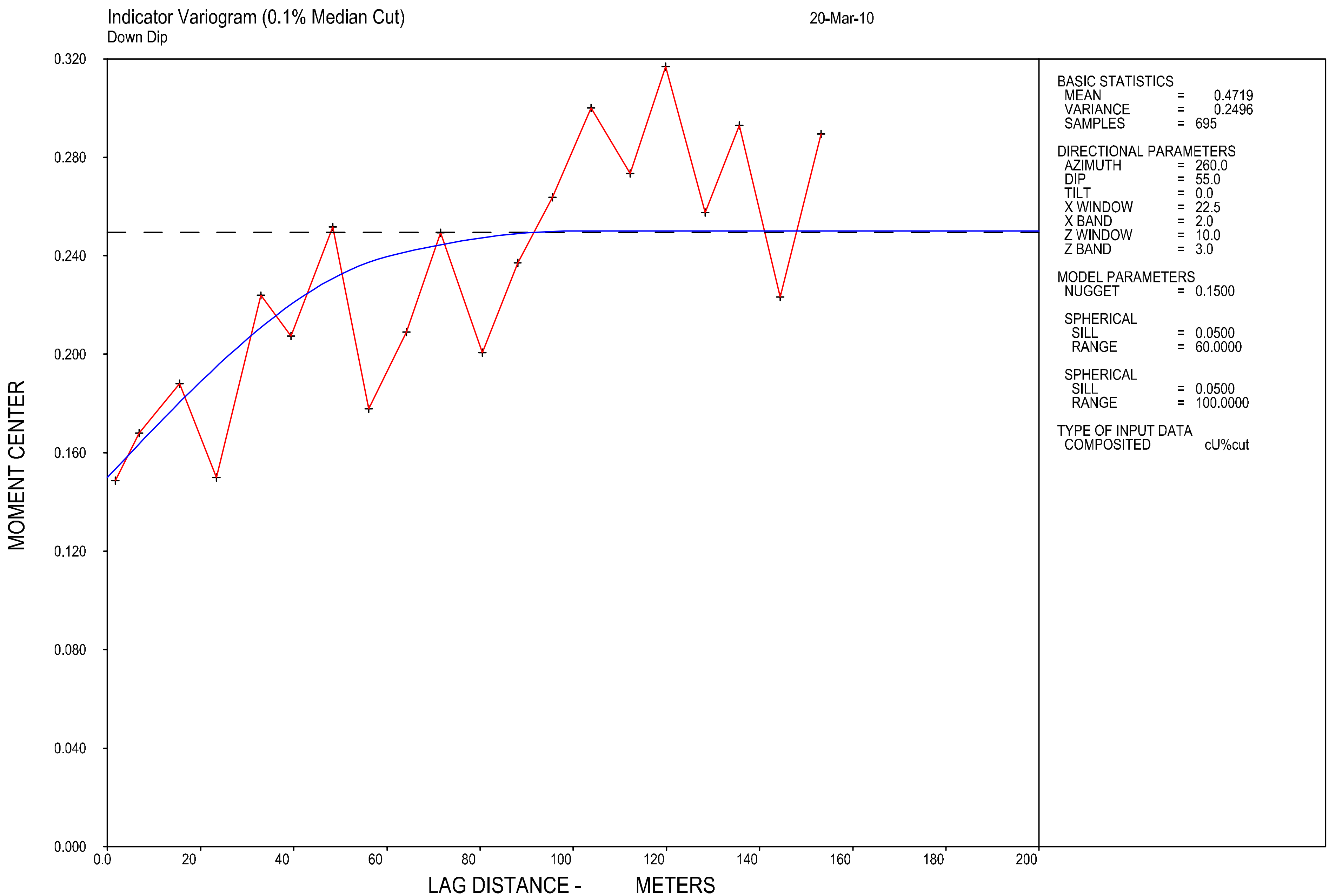
The correlation may be strong enough to consider employing a regression equation between uranium and molybdenum. This would be a good way of missing molybdenum values. In this current study, no regression was performed.

14.12 Variography and Kriging Parameters

Numerous log-variograms, relative and indicator variograms were generated and interpreted. These variograms were calculated in 14 directions such that all directions in three-dimensional space were explored. It was discovered that the variograms echoed the interpreted direction of the deposits modeled by wireframes. Figure 14.25 shows a directional indicator variogram that is down dip from the Main Zone's deposit structure. The experimental variogram was modeled with two spherical structures and a nugget. These ranges and variances are used as kriging parameters listed in Table 14.27. Ordinary kriging was used to estimate blocks 5x5x1 m in size. The kriging was constrained estimate blocks within the wireframes, with composites also within the wireframes. The result of this estimation is not classifiable by 43-101 standards.



Issued by: TETRA TECH, 350 Indiana Street, Suite 500, Golden, Colorado 80401, (303) 217-5700 (303) 217-5705 fax	**Prepared for:** *European Uranium Resources Ltd.*	**File Name:** Figure 14.24.dwg	
	Project: Kuriskova Uranium Project	**Project Number:** 114-310990	**Figure 14.24 Correlation Plot (Scatter) Between %U & %Mo**
	Project Location: Slovak Republic, Europe	**Date of Issue:** 05/09/2010	



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Golden, Colorado 80401
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Prepared for: *European Uranium Resources Ltd.*	File Name: Figure 14.25.dwg
Project: Kuriskova Uranium Project	Project Number: 114-310990
Project Location: Slovak Republic, Europe	Date of Issue: 05/09/2010

Figure 14.25
Down Dip %U Indicator Variogram
Main Zone 10, 11 and 12 (cut 0.1% %U)

Table 14.27. Kriging Parameters Table

Matching Codes[6]				Anisotropy				MIF Search Ranges					Variogram Parameters							
U% (Total uranium value composited to 0.5m , 4.2% Top Cut)																				
TT Composite Codes	Original Zone Codes	TT Block Codes	Zone Name	Axis	Anisotropy Axis Length (m)	Anisotropy Rotation	Type[3]	Resource Class[4]	Resource Code[2]	Maximum Search Range	Number Closest Pts /Max Pts Single Drillhole	Min Pts Required to Estimate	Rotation	Length	Nugget[1]	Nested	Model Type[4]	Sill[1]	Range (m)
				Primary	50	260	Az	M	NA	NA	NA	NA	260	100		1	Sph	0.05	60
10	1	10	Main Zone North (1N)	Second	50	55	Dip	I	2	65	4/3	4	55	100	0.15	2	Sph	0.05	100
				Tertiary	25	0	Tilt	F	3	300	4/3	1	0	50		3			
				Primary	50	260	Az	M	NA	NA	NA	NA	260	100		1	Sph	0.05	60
11	1.1	11	Main Zone South (1S)	Second	50	55	Dip	I	2	65	4/3	4	55	100	0.15	2	Sph	0.05	100
				Tertiary	25	0	Tilt	F	3	300	4/3	1	0	50		3			
				Primary	50	260	Az	M	NA	NA	NA	NA	NA	100		1	Sph	0.05	60
12	1.2	12	Main Zone Up	Second	50	55	Dip	I	2	65	4/3	4	30	100	0.15	2	Sph	0.05	100
				Tertiary	25	0	Tilt	F	3	300	4/3	1	0	50		3			
				Primary	50	260	Az	M	NA	NA	NA	NA	NA	100		1	Sph	0.05	60
20,30	2,3	23	HW North (2N,3N)	Second	50	55	Dip	I	NA	NA	NA	NA	NA	100	0.15	2	Sph	0.05	100
				Tertiary	25	0	Tilt	F	3	25	4/3	1	0	50		3			
				Primary	50	260	Az	M	NA	NA	NA	NA	NA	100		1	Sph	0.05	60
21	2.1	21	HW South (2S)	Second	50	55	Dip	I	NA	NA	NA	NA	NA	100	0.15	2	Sph	0.05	100
				Tertiary	25	0	Tilt	F	3	125	4/3	1	0	50		3			
				Primary	50	260	Az	M	NA	NA	NA	NA	NA	100		1	Sph	0.05	60
31	3.1	31	HW South (3S)	Second	50	55	Dip	I	NA	NA	NA	NA	NA	100	0.15	2	Sph	0.05	100
				Tertiary	25	0	Tilt	F	3	200	4/3	1	0	50		3			
				Primary	50	260	Az	M	NA	NA	NA	NA	NA	100		1	Sph	0.05	60
40	4	40	HW North	Second	50	55	Dip	I	NA	NA	NA	NA	NA	100	0.15	2	Sph	0.05	100
				Tertiary	25	0	Tilt	F	3	300	4/3	1	0	50		3			
				Primary	50	260	Az	M	NA	NA	NA	NA	NA	100		1	Sph	0.05	60
50	5	45	New Zone	Second	50	55	Dip	I	2	65	4/3	4	30	100	0.15	2	Sph	0.05	100
				Tertiary	25	0	Tilt	F	3	300	4/3	1	0	50		3			
Second Pass Krige Error[2]				0.17															
Mo% uses U% parameters for estimation																			

Notes

1 Indicator variogram nugget and sill values used from Main Zone (0.1 Cut) for all zones

2 Kriging Error is used to adjust preliminary class 2,3 to 2,3 & 4 by post-kriging filter (Class 4 = Inferred by Geologic Interpretatoin)

3 Az=Azimuth is clockwise (CW) from North, Dip is positive when downward, Tilt rotates CW around primary axis.

4 Sph=Spherical, Lin=Linear, Exp=Exponential, Gau=Gaussian

5 M=Measured (Class 1), I=Indicated (Class 2), F=Inferred by Variogram (Class 3), D=Inferred by Geologic Interpretaion (Class 4)

Table 14.28 shows the count of blocks that fall with 1 percent within each wireframe. GemCom was used to assign the zone codes and calculate the proportion of the block that is within the wireframe. These two data files are used by MicroModel with its proportional block method of estimation and tabulation of grades and tonnages.

Table 14.28. Block Count by Zone

```
ROCK COUNT FOR BLOCK MODEL (R200)
PROPORTIONAL BLOCK MODEL USED: G504
    MINIMUM COLUMN:    1
    MAXIMUM COLUMN:  200
       MINIMUM ROW:    1
       MAXIMUM ROW:  150
     MINIMUM LEVEL:    1
     MAXIMUM LEVEL:  940

     NUMBER OF ROCK TYPES FOUND =   8
```

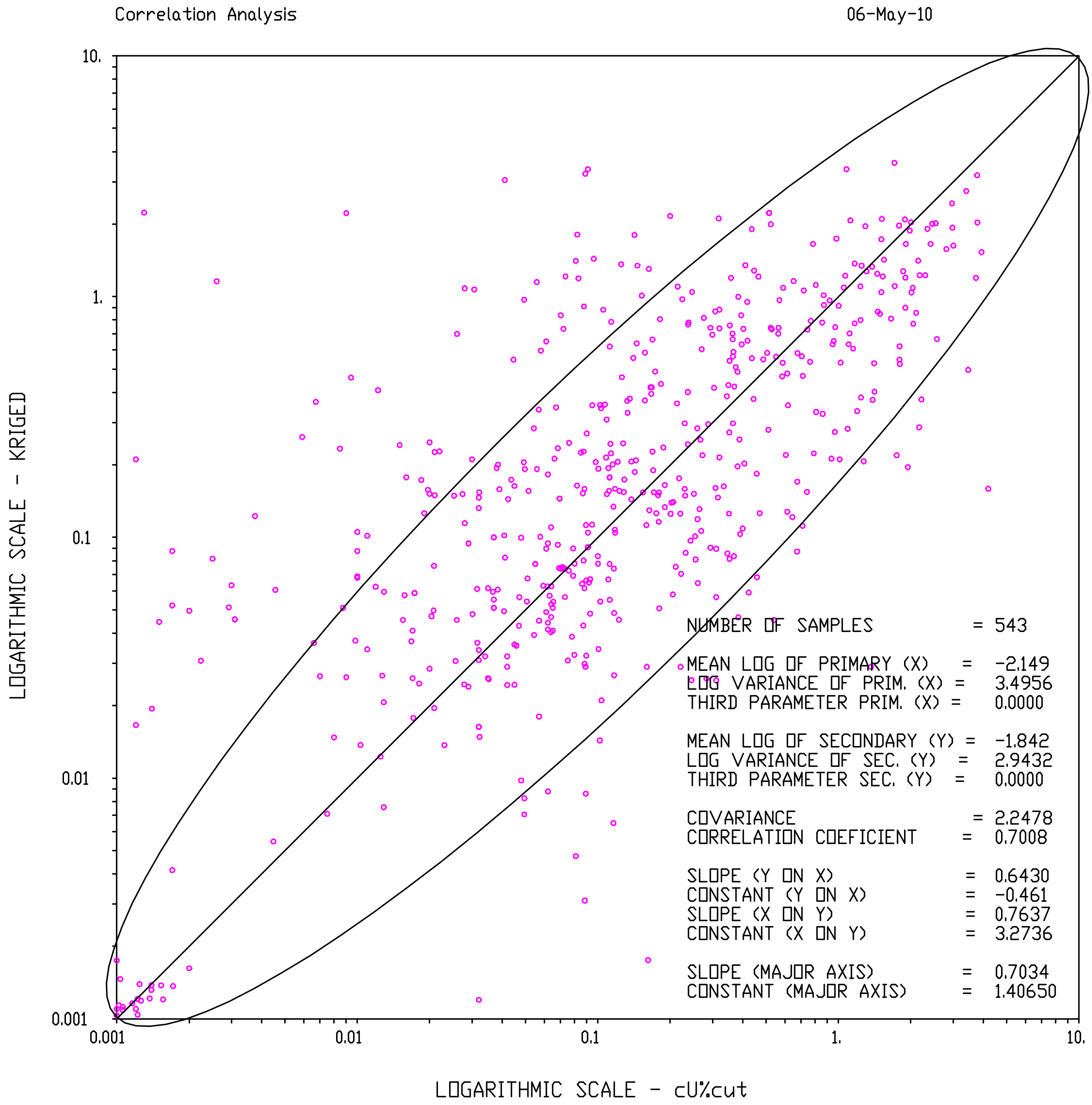
CODE	COUNT	MINCOL	MAXCOL	MINROW	MAXROW	MINLEV	MAXLEV
10	63717	26	159	42	125	165	812
11	47390	20	124	4	61	11	552
12	2601	75	138	45	139	578	808
21	18149	21	126	8	63	48	599
23	290577	88	138	39	95	516	841
31	11801	32	145	9	52	127	730
40	10753	61	125	49	123	500	824
45	2752	63	83	116	133	614	813

14.13 Classification of Blocks into Indicated and Inferred

The estimation utilized a two-pass protocol to assign indicated and inferred classes. In this study, it will be shown that no measured resources exist.

14.13.1 Pass 1

The first pass utilized jackknifing of composite values. Jackknifing or model validation is a computer technique that removes samples one at a time and then predicts what its value is using samples that utilize the search and variogram parameters being investigated. The estimate is then compared to the real value. Figure 14.26 shows a plotted original composite percent uranium values versus the estimated value based on estimates using a 10 meter search radius. Note that if the estimate were perfect, then points would fall on the 45-degree line. This jackknife study produced a correlation between the target and the estimated uranium values of 0.70. The figure has a reference ellipse plotted which is wide enough to contain 80 percent of the points falling adjacent to the 45-degree line. Note that the longer range of 65 m produces a correlation of .66. This correlation is still considered in the realm of indicated. Drill hole spacing is not close enough to produce a correlation that could be classified as measured. Table 14.29 lists the results of the three studies. Zones 20 and 30, re-coded for kriging as Zone 23 presents a particular case (discussed in Chapter 14, Section 9). Here a relatively small number of composites have been selected to be above 0.03 percent uranium, in the largest wireframe containing 290,577 blocks shown in Table 14.29. Within this zone, the search range for an inferred estimate has been limited to a maximum of 25 m.



Issued by:	Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	*European Uranium Resources Ltd.*	Figure 14.26.dwg	**Figure 14.26** **Plot of Jackknife Results at a 60-Meter Search Range**
	Project: Kuriskova Uranium Project	Project Number: 114-310990	
	Project Location: Slovak Republic, Europe	Date of Issue: 05/09/2010	

Table 14.29. Resource Classification – First Pass

Search Range*	Search Criteria Max. Composites per DH / Min Required	Correlation0	Initial Class Index	Initial Class Designation
0-10 m	4/3	0.70	2	Indicated
0-65 m	4/3	0.66	2	Indicated
65-300 m	4/3	0.2	3	Inferred

* Zone 23 (20+30) have a search radius with a maximum of 25 m.

14.13.2 Pass 2

In addition, kriging generates an estimation error (kriging error), which contains a measure of reliability. Figure 14.27 shows a cumulative frequency plot of the kriging error. At an error of 0.17, there is a dramatic break in the curve. This is also shown in Table 14.30, with the kriging errors above 0.17 highlighted in yellow. Note this population of errors deviates from the approximate normal distribution. The second pass uses this information of kriging error to adjust the initial class based strictly of search distance and number of samples used. Any estimate that has a kriging error above 0.17 will be demoted in class.

Figure 14.28 shows graphically how the two-pass method employs not only the search distance/jackknife study but also the kriging error. The top portion of the figure shows the variogram being used to establish the first pass search ranges. The middle portion shows the results of three jackknife studies at the increasing ranges. And finally, the bottom part shows how kriging error break-point is used. This bottom panel (B) of Figure 14.28 is the second pass. It shows kriging error plotted as a log-probability graph. Note that this particular graph shows that at a kriging error of 0.17 there is a break in the plot. This break point is where kriging error shifts from a lower population (better estimates) to a higher one (worse estimates). This break suggests that kriging produces a sub-population of estimations that are particularly poor in quality. To acknowledge these poor estimations, the second pass simply shifts those blocks by adding one to the class code. For example, a block that is classified initially as indicated with a 2 would now be classified as inferred with a class of 3. In the same manner, an initially classified inferred block would be shifted into a class of 4. This 4 class has been given the designation of Inferred-Geology. Blocks of this class are still within the interpreted wireframes, hence there is still a basis classifying them as inferred for tabulation purposes.

Table 14.30 shows the statistics for the block class assignment. The upper portion shows the count for the first pass. Indicated class of 2 has 38.2 percent of the blocks. After the second pass, the indicated blocks proportion falls to 34.8 percent. The initial inferred class has 61.8 percent of the blocks. After the second pass, nearly half of those are shifted into the inferred-geologic class 4 (Table 14.31).

Calculate Cumulative Frequency Curve

06-May-10

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TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	***European Uranium Resources Ltd.***	Figure 14.27.dwg	**Figure 14.27**
	Project: Kuriskova Uranium Project	Project Number: 114-310990	**Cumulative Frequency Plot of**
	Project Location: Slovak Republic, Europe	Date of Issue: 05/09/2010	**Kriging Error Showing Break in Curve at 0.17 %U**

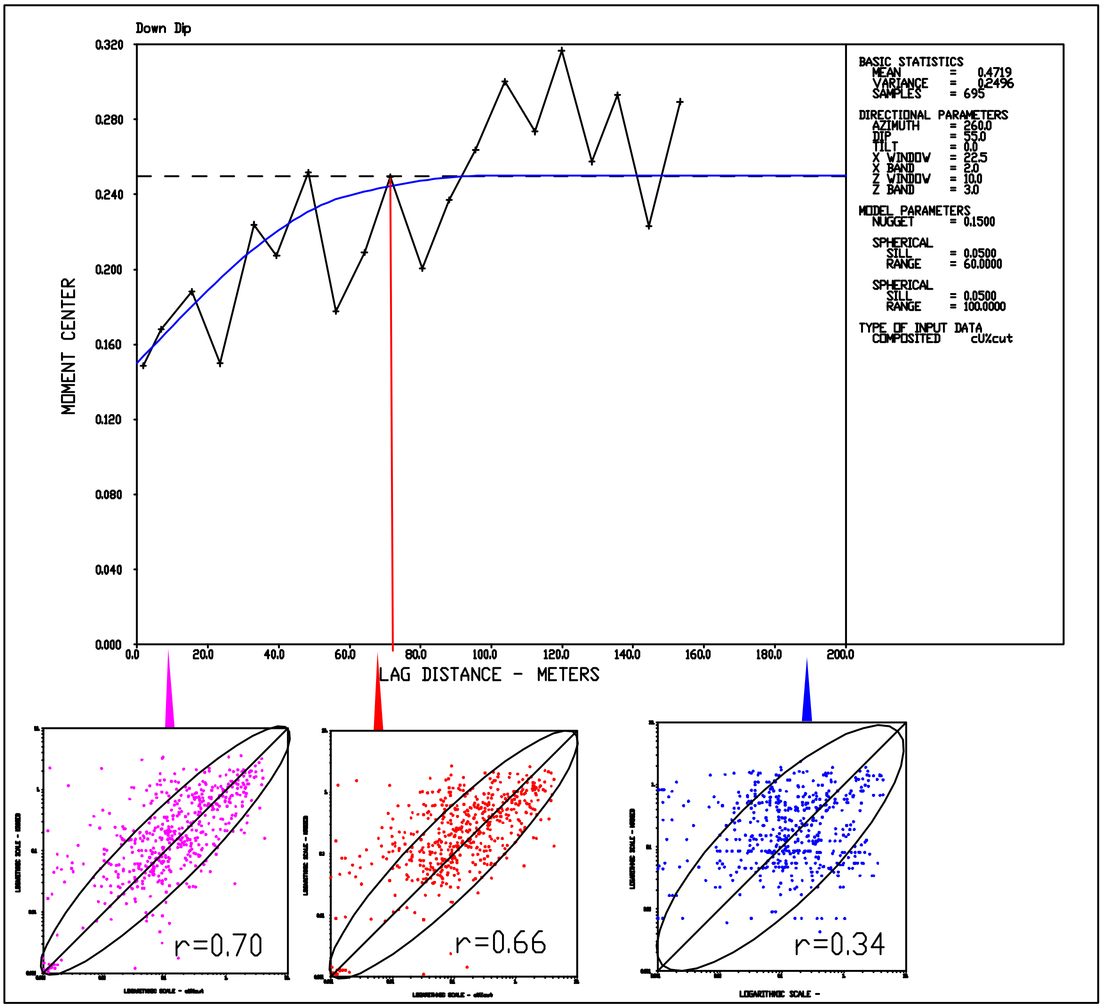
Table 14.30. Statistics of Kriging Error Showing Break at 0.17 Percent Uranium

CURRENT LABEL : (G203) Kriged Error kU%cut

	BLOCK COUNT				UNTRANSFORMED STATISTICS						LOG-TRANSFORMED STATS			LOG-DERIVED	
ROCK TYPE	MISSING	BELOW LIMITS	ABOVE LIMITS	INSIDE LIMITS	MINIMUM	MAXIMUM	MEAN	VARIANCE	STD. DEV.	COEF. OF VAR	LOG MEAN	LOG VAR.	LOG STD.DEV	MEAN	COEF. OF VAR.
10	0	0	0	63717	0.05233	0.24498	0.13304	0.00177	0.04211	0.3165	-2.0681	0.1045	0.3233	0.1332	0.3319
11	9	0	0	47381	0.05776	0.39563	0.17977	0.000992	0.03150	0.1752	-1.7350	0.0422	0.2053	0.1802	0.2075
12	0	0	0	2601	0.05648	0.30854	0.11078	0.00126	0.03545	0.3200	-2.2395	0.0709	0.2662	0.1104	0.2710
21	8811	0	0	9338	0.05805	0.20427	0.16079	0.00107	0.03270	0.2033	-1.8525	0.0552	0.2350	0.1612	0.2382
23	278745	0	0	11832	0.05099	0.15570	0.09869	0.000552	0.02349	0.2381	-2.3451	0.0599	0.2448	0.0988	0.2486
31	0	0	0	11801	0.05844	0.26166	0.17618	0.00273	0.05225	0.2966	-1.7838	0.1001	0.3165	0.1766	0.3246
40	0	0	0	10753	0.05318	0.20328	0.12104	0.000993	0.03151	0.2603	-2.1478	0.0753	0.2744	0.1212	0.2797
45	0	0	0	2752	0.05320	0.21211	0.12733	0.00209	0.04571	0.3590	-2.1298	0.1433	0.3785	0.1277	0.3925
ALL	287565	0	0	160175	0.05099	0.39563	0.14786	0.00214	0.04626	0.3129	-1.9657	0.1156	0.3400	0.1484	0.3501

```
 LOWER BOUND   UPPER BOUND      4000      8000     12000     16000     20000     24000     28000     32000     36000     40000
     >=            <       +---------+---------+---------+---------+---------+---------+---------+---------+---------+---------+
    0.0510        0.0549|
    0.0549        0.0590|*
    0.0590        0.0635|***
    0.0635        0.0683|*****
    0.0683        0.0735|*******
    0.0735        0.0791|*********
    0.0791        0.0851|************
    0.0851        0.0916|**************
    0.0916        0.0985|******************
    0.0985        0.1060|********************
    0.1060        0.1140|***********************
    0.1140        0.1227|*************************
    0.1227        0.1320|**************************
    0.1320        0.1420|*************************
    0.1420        0.1528|************************
    0.1528        0.1644|************************
    0.1644        0.1769|*****************************
    0.1769        0.1903|****************************************
    0.1903        0.2048|***************************************************************
    0.2048        0.2203|************************
    0.2203        0.2371|**
    0.2371        0.2551|**
    0.2551        0.2744|******
    0.2744        0.2953|
    0.2953        0.3177|
    0.3177        0.3418|
    0.3418        0.3677|
    0.3677        0.3957|
                           +---------+---------+---------+---------+---------+---------+---------+---------+---------+---------+
                           0      4000      8000     12000     16000     20000     24000     28000     32000     36000     40000
```

A. Pass 1



B. Pass 2

Issued by:	Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	*European Uranium Resources Ltd.*	Figure 14.28.dwg	**Figure 14.28** **Two Pass Classification Method**
	Project: Kuriskova Uranium Project	Project Number: 114-310990	
	Project Location: Slovak Republic, Europe	Date of Issue: 05/09/2010	

Table 14.31. Block Count by Class Before and After Second Pass

Resource Class Block Count Before Second Pass

LOWER BOUND >=	UPPER BOUND <	FREQ	PERCENT	MEAN	CUM FREQ	PERCENT	CUM MEAN	CUM FREQ	PERCENT	CUM MEAN
					(ALL VALUES < UPPER BOUND)			(ALL VALUES >= LOWER BOUND)		
1.1000	2.1000	61184	38.20	2.0000	61184	38.20	2.0000	160175	100.00	2.6180
2.1000	3.1000	98991	61.80	3.0000	160175	100.00	2.6180	98991	61.80	3.0000
3.1000	4.1000	0	0.00	0.0000	160175	100.00	2.6180	0	0.00	0.0000

```
  LOWER BOUND   UPPER BOUND      10000     20000     30000     40000     50000     60000     70000     80000     90000    100000
      >=            <       +---------+---------+---------+---------+---------+---------+---------+---------+---------+---------+
     1.1000          2.1000|*************************************************************
     2.1000          3.1000|***************************************************************************************************
     3.1000          4.1000|
                            +---------+---------+---------+---------+---------+---------+---------+---------+---------+---------+
                            0      10000     20000     30000     40000     50000     60000     70000     80000     90000    100000
```

Resource Class Block Count After Second Pass (Kriging Error of 0.17)

LOWER BOUND >=	UPPER BOUND <	FREQ	PERCENT	MEAN	CUM FREQ	PERCENT	CUM MEAN	CUM FREQ	PERCENT	CUM MEAN
					(ALL VALUES < UPPER BOUND)			(ALL VALUES >= LOWER BOUND)		
1.1000	2.1000	55738	34.80	2.0000	55738	34.80	2.0000	160175	100.00	2.9962
2.1000	3.1000	49307	30.78	3.0000	105045	65.58	2.4694	104437	65.20	3.5279
3.1000	4.1000	55130	34.42	4.0000	160175	100.00	2.9962	55130	34.42	4.0000

```
  LOWER BOUND   UPPER BOUND       6000     12000     18000     24000     30000     36000     42000     48000     54000     60000
      >=            <       +---------+---------+---------+---------+---------+---------+---------+---------+---------+---------+
     1.1000          2.1000|*********************************************************************************************
     2.1000          3.1000|**********************************************************************************
     3.1000          4.1000|********************************************************************************************
                            +---------+---------+---------+---------+---------+---------+---------+---------+---------+---------+
                            0       6000     12000     18000     24000     30000     36000     42000     48000     54000     60000
```

14.14 Kriged Block Statistics

Table 14.32 and Table 14.33 show the block statistics for uranium and molybdenum.

Table 14.32. Statistics of Percent Uranium Kriged Blocks

CURRENT LABEL : (G103) Kriged Grade kU%cut

MINIMUM CUT-OFF ENTERED = 0.001000
MAXIMUM CUT-OFF ENTERED = 5.000000

ROCK TYPE	BLOCK COUNT MISSING	BELOW LIMITS	ABOVE LIMITS	INSIDE LIMITS	UNTRANSFORMED STATISTICS MINIMUM	MAXIMUM	MEAN	VARIANCE	STD. DEV.	COEF. OF VAR	LOG-TRANSFORMED STATS LOG MEAN	LOG VAR.	LOG STD.DEV	LOG-DERIVED MEAN	COEF. OF VAR.
10	0	23	0	63694	0.00101	3.6320	0.37177	0.20856	0.45668	1.2284	-1.6770	1.5013	1.2253	0.3960	1.8675
11	0	0	0	47390	0.02002	0.69813	0.14056	0.01060	0.10294	0.7324	-2.1957	0.4733	0.6880	0.1410	0.7780
12	0	309	0	2086	0.00100	0.46212	0.13983	0.01547	0.12436	0.8894	-2.4952	1.5090	1.2284	0.1754	1.8768
21	8811	12	0	9326	0.00107	0.16764	0.04551	0.00122	0.03498	0.7685	-3.3816	0.6190	0.7868	0.0463	0.9258
23	278745	0	0	11832	0.03099	1.9482	0.20911	0.05314	0.23053	1.1024	-1.9771	0.7536	0.8681	0.2018	1.0605
31	0	0	0	11801	0.00351	0.32825	0.07486	0.00414	0.06434	0.8594	-2.8456	0.4684	0.6844	0.0734	0.7729
40	0	363	0	10390	0.00101	0.27832	0.04082	0.00196	0.04422	1.0832	-3.7827	1.4861	1.2191	0.0479	1.8493
45	0	0	0	2270	0.00688	2.0083	0.40995	0.10023	0.31659	0.7723	-1.1170	0.4902	0.7002	0.4181	0.7954
ALL	287556	707	0	158789	0.00100	3.6320	0.22526	0.11008	0.33178	1.4729	-2.1817	1.4026	1.1843	0.2276	1.7509

```
LOWER BOUND   UPPER BOUND      4000      8000     12000     16000     20000     24000     28000     32000     36000     40000
    >=             <      +---------+---------+---------+---------+---------+---------+---------+---------+---------+---------+
  0.0010         0.0013|*
  0.0013         0.0018|*
  0.0018         0.0023|*
  0.0023         0.0031|*
  0.0031         0.0041|
  0.0041         0.0055|*
  0.0055         0.0073|*
  0.0073         0.0097|*
  0.0097         0.0129|***
  0.0129         0.0171|******
  0.0171         0.0227|************
  0.0227         0.0302|************
  0.0302         0.0401|*********************************
  0.0401         0.0532|********************************
  0.0532         0.0707|*****************************
  0.0707         0.0939|**********************************
  0.0939         0.1248|**********************************************
  0.1248         0.1657|**********************************************************
  0.1657         0.2201|*************************
  0.2201         0.2924|******************
  0.2924         0.3884|***************
  0.3884         0.5159|*************************
  0.5159         0.6853|***********
  0.6853         0.9103|*********
  0.9103         1.2091|*********
  1.2091         1.6061|******
  1.6061         2.1334|*****
  2.1334         2.8338|*
  2.8338         3.7642|
  3.7642         5.0000|
                          +---------+---------+---------+---------+---------+---------+---------+---------+---------+---------+
                          0      4000      8000     12000     16000     20000     24000     28000     32000     36000     40000
```

Table 14.33. Statistics of Percent Molybdenum Kriged Blocks

CURRENT LABEL : (G102) Kriged Grade kMo%

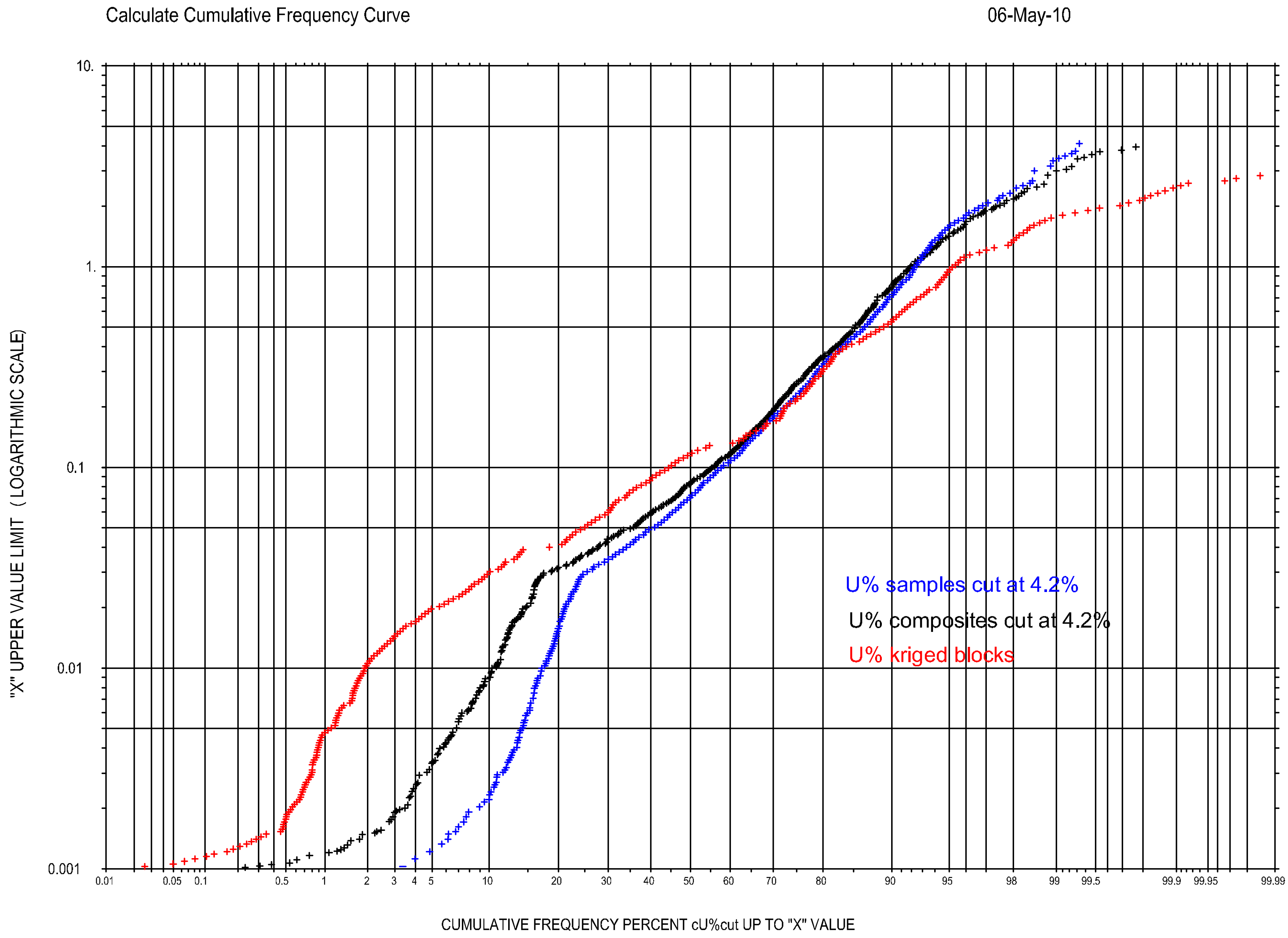
ROCK TYPE	BLOCK COUNT MISSING	BELOW LIMITS	ABOVE LIMITS	INSIDE LIMITS	UNTRANSFORMED STATISTICS MINIMUM	MAXIMUM	MEAN	VARIANCE	STD. DEV.	COEF. OF VAR	LOG-TRANSFORMED STATS LOG MEAN	LOG VAR.	LOG STD.DEV	LOG-DERIVED MEAN	COEF. OF VAR.
10	0	0	0	63717	0.000375	1.2226	0.04611	0.01203	0.10966	2.3782	-4.3597	2.3012	1.5170	0.0404	2.9977
11	0	0	0	47390	0.00135	0.09534	0.01825	0.000446	0.02112	1.1574	-4.6857	1.4124	1.1884	0.0187	1.7623
12	0	0	0	2395	0.00344	0.29382	0.01394	0.000191	0.01383	0.9925	-4.5029	0.4337	0.6586	0.0138	0.7369
21	13533	0	0	4616	0.000858	0.01710	0.00626	0.000020	0.00446	0.7125	-5.4592	0.9351	0.9670	0.0068	1.2439
23	280237	0	0	10340	0.000100	0.46127	0.02267	0.00179	0.04228	1.8649	-4.6805	1.7628	1.3277	0.0224	2.1974
31	3262	0	0	8539	0.000667	0.07009	0.02526	0.000180	0.01342	0.5311	-3.8620	0.4483	0.6696	0.0263	0.7521
40	0	0	0	10753	0.000655	0.33374	0.05638	0.00569	0.07542	1.3376	-4.0846	3.2458	1.8016	0.0853	4.9680
45	0	0	0	2270	0.06631	1.7034	0.42067	0.04776	0.21853	0.5195	-0.9452	0.1336	0.3655	0.41543	0.3780
ALL	297032	0	0	150020	0.000100	1.7034	0.03917	0.00897	0.09472	2.4179	-4.4211	2.1127	1.4535	0.0346	2.6964

```
 LOWER BOUND  UPPER BOUND       4000      8000      12000     16000     20000     24000     28000     32000     36000     40000
     >=            <      +---------+---------+---------+---------+---------+---------+---------+---------+---------+---------+
   0.0001       0.0001|
   0.0001       0.0002|
   0.0002       0.0003|
   0.0003       0.0004|
   0.0004       0.0005|*
   0.0005       0.0007|
   0.0007       0.0010|**
   0.0010       0.0013|********
   0.0013       0.0019|******************
   0.0019       0.0026|*****************
   0.0026       0.0036|*******************************************************
   0.0036       0.0049|***************************
   0.0049       0.0068|***********************
   0.0068       0.0094|*****************************
   0.0094       0.0131|*************************
   0.0131       0.0181|***************************
   0.0181       0.0250|***************************
   0.0250       0.0346|******************************
   0.0346       0.0478|*****************
   0.0478       0.0662|*******************
   0.0662       0.0916|****************
   0.0916       0.1268|************
   0.1268       0.1754|******
   0.1754       0.2427|****
   0.2427       0.3358|*****
   0.3358       0.4647|******
   0.4647       0.6430|**
   0.6430       0.8898|*
   0.8898       1.2312|*
   1.2312       1.7036|
                          +---------+---------+---------+---------+---------+---------+---------+---------+---------+---------+
                          0       4000      8000      12000     16000     20000     24000     28000     32000     36000     40000
```

14.15 Block Model Validation

14.15.1 Validation Test 1

A validation test was done in determining if the estimated results appear correct statistically through the sequence of assays, composites and kriged blocks. The statistical distribution of composites should be statistically similar to assay values. Also blocks should follow a similar distribution as composites. Figure 14.29 shows the log-probability plots for samples, composites and blocks. The graph show the three log-probability plots overlaid. The differing slopes of the probability plots indicates there is the expected successive lowering of variance of the distributions as one proceeds from samples to blocks. This successful overlaying the plotted distributions indicate that the sequence of samples to composites to blocks appears statistically valid.



Issued by:	Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, Colorado 80401 (303) 217-5700 (303) 217-5705 fax	***European Uranium Resources Ltd.***	Figure 14.29.dwg	**Figure 14.29** **Cumulative Frequency Plot of %U Samples, Composites & Blocks Side-by-Side**
	Project: Kuriskova Uranium Project	Project Number: 114-310990	
	Project Location: Slovak Republic, Europe	Date of Issue: 05/09/2010	

14.15.2 Validation Test 2

Sections with blocks, composites and drill hole data were created and visually inspected.

The model passed this qualitative test successfully.

14.16 Bulk Density

A bulk density of 2.75 was used for the all zones.

14.17 Resource Table Comparing 2010 and 2011 and Grade-Tonnage Plots for the 2011 Resource

The following resource table presents the results of the resource estimation study by deposit, by cutoff grade and by resource class. Table 14.34 shows the EUU 2011 and 2010 estimations for uranium and molybdenum at a 0.05 percent uranium cutoff. EUU's 2011 estimation is on the left side of the table highlighted in pink. The 2010 estimations are on the right side of the table highlighted in blue. The yellow highlighted indicates net difference on 2 percent in total inventory (28 percent gain in indicated, and 26 percent reduction in inferred). The conversion of inferred to indicated has been accomplished by more dense drilling in Zone 45 and Main Zone South. Better geologic interpretation has provided data partitioning and led to improved local estimation. Improved geological interpretation of Main Zone North has allowed for defined zones of high, medium and low grade. It has also reduced smearing across the previously undivided zones. This more refined model more accurately portrays they distribution of high medium and low grade areas. The change of top cut from 4.2 percent uranium in 2010 to 6.95 percent uranium in this resource update has had a negligible influence on the mean percent uranium of the block model. The 2011 block model mean is 2.3 percent higher than in 2010.

Figure 14.30 shows the grade-tonnage curve for the Main Zone North.

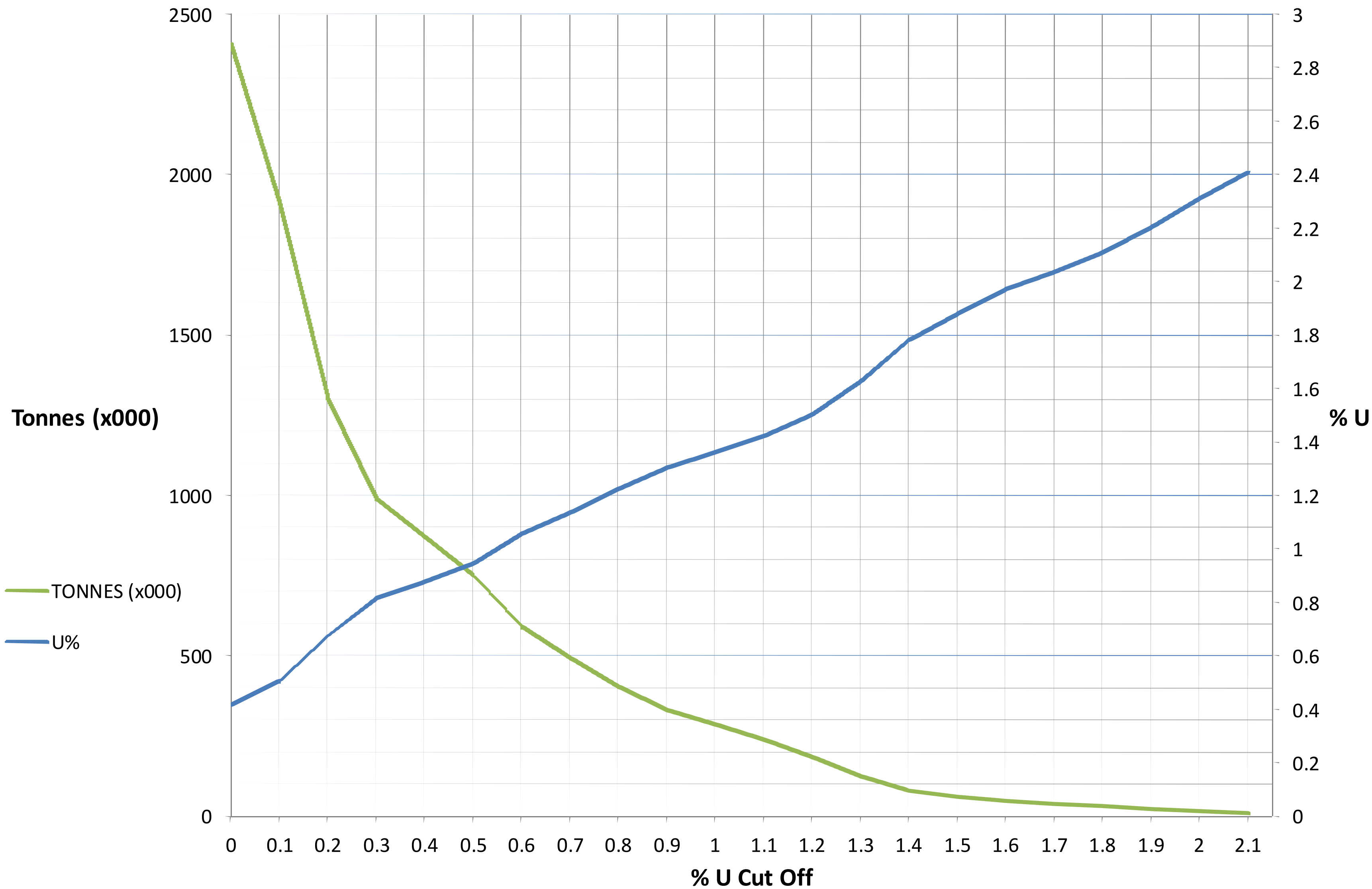
Figure 14.31 shows the grade-tonnage curves for indicated and inferred uranium resources for the all zones.

TABLE 14.34: COMPARISON OF URANIUM AND MOLY RESOURCE (2010 AND 2011) STATED AT 0.05%U CUT OFF

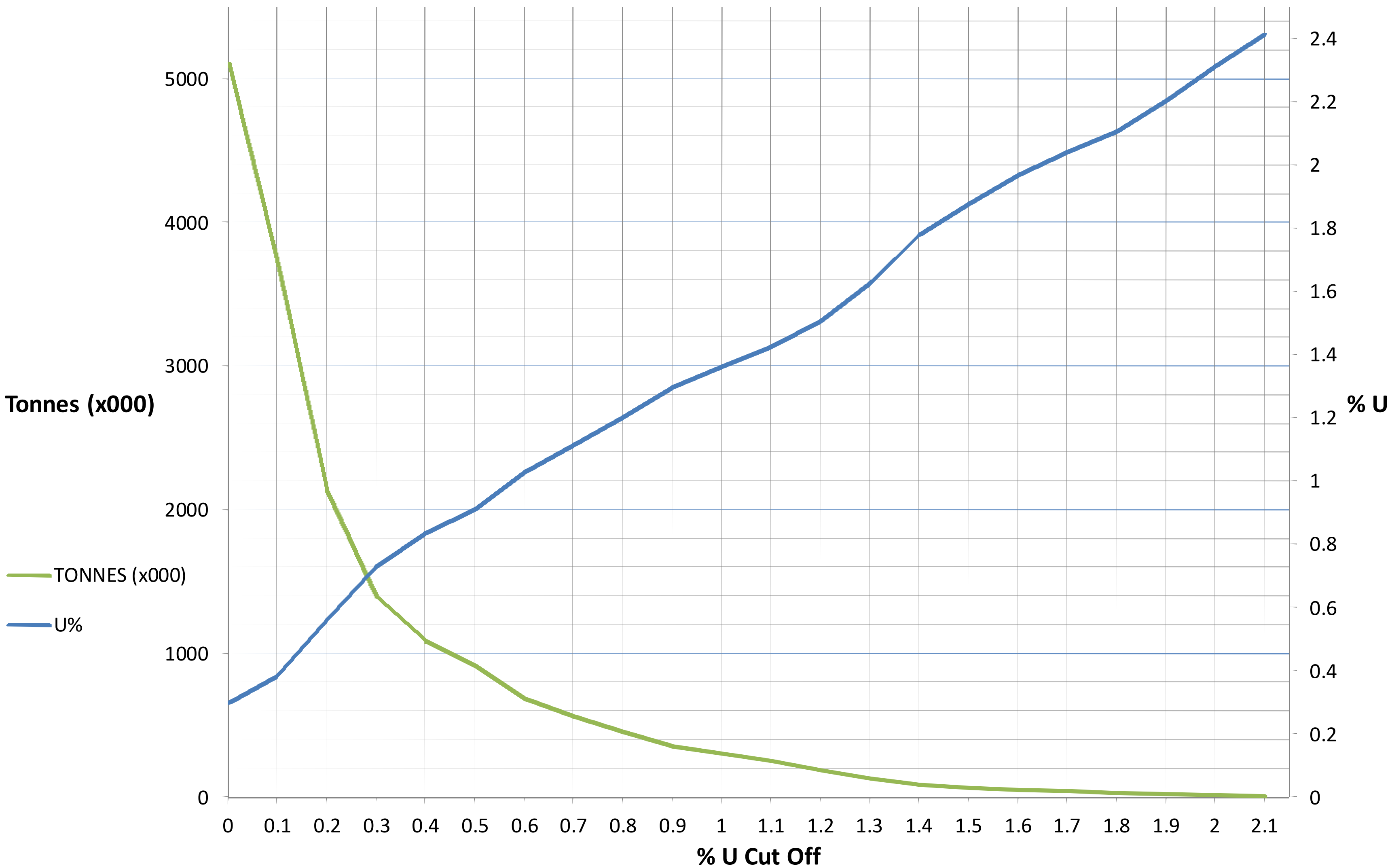
EUROPEAN URANIUM RESOURCES LTD. – KURISKOVA URANIUM PROJECT

June 2011

Geology Domain	Sub-Domain	Model Zone	U%	Tonnes ('000)	$\%U_3O_8$	U_3O_8 ('000 Pounds)	Mo %	Tonnes ('000)	Mo ('000 Pounds)	U%	Tonnes ('000)	$\%U_3O_8$	U_3O_8 ('000 Pounds)	U3O8 lbs % Diff	Mo %	Tonnes ('000)	Mo ('000 Pounds)	Mo lbs % Diff
Indicated Resources, April 2011										**Indicated Resource, March 2010**								
	ZONE1N (Main Zone North)	1	0.507	1790	0.598	23,601	0.056	1,790	2,210	0.502	1,477	0.592	19,276	22%	0.070	1,477	2,279	-3%
	UP MAIN ZONE	1.2	0.211	54	0.248	296	0.033	54	39	0.211	54	0.248	296	0%	0.033	54	39	0%
Main Zone	ZONE1S (Main Zone South)	1.1	0.339	207	0.400	1,824	0.073	207	333	0.269	67	0.317	469	289%	-	-	-	100%
	ZONE2N(43) (HW North)	2	0.279	109	0.329	791	0.016	82	29	-	-	-	-	100%	-	-	-	100%
Hanging wall north	ZONE3N(44) (HW North)	3	0.403	99	0.475	1,037	0.025	99	55	-	-	-	-	100%	-	-	-	100%
	ZONE 4 (HW North)	4	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	ZONE2S (HW South)	2.1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	ZONE3S (HW South)	3.1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Zone 45	ZONE45 (NEW ZONE)	5	0.523	69	0.617	938	0.425	69	647	0.574	33	0.677	498	89%	0.607	33	442	46%
Main Zone total indicated		1+1.1+1.2	0.482	2,051	0.569	25,721	0.057	2,051	2,582	0.482	1,598	0.569	20,041	28%	0.069	1,531	2,318	11%
Zone 45 total indicated		5	0.523	69	0.617	938	0.425	69	647	0.574	33	0.677	498	89%	0.607	33	442	46%
HW north total indicated		2+3	0.338	208	0.399	1,828	0.021	181	83	-	-	-	-	100%	-	-	-	100%
Total Indicated (All Domains)			**0.471**	**2,328**	**0.555**	**28,487**	**0.065**	**2,301**	**3,312**	**0.484**	**1,631**	**0.571**	**20,539**	39%	**0.080**	**1,564**	**2,760**	20%
Inferred Resources, April 2011										**Inferred Resource, March 2010**								
	ZONE1N (Main Zone North)	1	0.194	490	0.229	2,471	0.017	490	184	0.291	770	0.343	5,825	-58%	0.017	770	297	-38%
	UP MAIN ZONE	1.2	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Main Zone	ZONE1S (Main Zone South)	1.1	0.156	1,641	0.184	6,655	0.024	1,612	853	0.162	1,543	0.191	6,499	2%	0.014	1,586	496	72%
	ZONE2N(43) (HW North)	2	0.215	130	0.254	727	0.024	110	58	0.244	239	0.288	1,516	-52%	0.020	191	86	-33%
Hanging wall north	ZONE3N(44) (HW North)	3	0.153	230	0.180	915	0.047	185	192	0.229	329	0.270	1,957	-53%	0.039	285	248	-23%
	ZONE 4 (HW North)	4	0.095	52	0.112	128	0.071	52	81	0.095	52	0.112	128	0%	0.071	52	81	0%
	ZONE2S (HW South)	2.1	0.087	181	0.103	409	0.003	181	12	0.087	181	0.103	410	0%	0.003	181	12	0%
	ZONE3S (HW South)	3.1	0.106	336	0.125	926	0.024	288	155	0.106	336	0.125	924	0%	0.024	288	155	0%
Zone 45	ZONE 45	5	0.426	39	0.502	432	0.378	39	325	0.332	31	0.392	268	61%	0.756	32	533	-39%
Main Zone total inferred		1+1.1+1.2	0.165	2,131	0.194	9,127	0.022	2,102	1,037	0.205	2,313	0.242	12,324	-26%	0.015	2,356	793	31%
Zone 45 total inferred		2+3+4+2.1+3	0.129	929	0.152	3,105	0.044	855	823	0.167	1,137	0.197	4,936	-37%	0.049	1,029	1,115	-26%
HW north total inferred		5	0.426	39	0.502	432	0.378	39	325	0.332	31	0.392	268	61%	0.756	32	533	-39%
Total Inferred (All Domains)			**0.157**	**3,099**	**0.185**	**12,664**	**0.033**	**2,996**	**2,185**	**0.194**	**3,481**	**0.228**	**17,528**	-28	**0.032**	**3,417**	**2,442**	-11%



Issued by:	Prepared for:	File Name:	
TETRA TECH 350 Indiana Street, Suite 500 Golden, CO 80401 (303) 217-5700 (303) 217-5705 fax	***European Uranium Resources Ltd.***	Fig 14.30.jpeg	**Figure 14.30**
	Project: Kuriskova Uranium Project	Project Number: 114-310990	**%U Grade Tonnage Curve**
	Project Location: Slovak Republic	Date of Issue: June 2011	**Main Zone North (ZCODE=1)**



Issued by: **TETRA TECH** 350 Indiana Street, Suite 500 Golden, CO 80401 (303) 217-5700 (303) 217-5705 fax	Prepared for: ***European Uranium Resources Ltd.***	File Name: Fig 14.31.jpeg	**Figure 14.31** **%U Grade Tonnage Curve** **All Zones**
	Project: Kuriskova Uranium Project	Project Number: 114-310990	
	Project Location: Slovak Republic	Date of Issue: June 2011	

15.0 RESERVE ESTIMATE

The cutoff grade is defined as the grade at which mining one tonne of material would monetarily break even with the costs of production, transportation, smelting, refining, environmental, general and administrative costs, and associated production taxes and royalties. Therefore, one would expect mining material above the cutoff grade to generate a profit, and mining material below the cutoff grade to generate a net loss.

15.1 Underground Cutoff Method

It was determined that the cutoff grade for the project was 0.15 percent U_3O_8 with a value of US$68/lb of U_3O_8 which converts to a raw ore grade of 0.13 percent uranium. Table 15.1 and Table 15.2 display the parameters used to calculate the cutoff grades and break even values for the deposit.

Table 15.1. Kuriskova Cutoff Grade Costs

Category	Value	Unit
Labor	33.00	US$/tonne mined
UG Consumables	29.77	US$/tonne mined
Paste Backfill	13.48	US$/tonne mined
Process plant Consumables	48.65	US$/tonne mined
G&A and Royalties	62.00	US$/tonne mined
Total Cost	189.84	US$/tonne mined

Table 15.2. Kuriskova Cutoff Grade Calculation

Category	Value	Unit
Cost	189.84	US$/tonne mined
Process Recovery	92	% Recovered
U_3O_8 Price	68	US$/lb
Cutoff Grade U_3O_8	0.15	% U_3O_8/tonne mined
Conversion from % U_3O_8 to % U	0.848	
Cutoff Grade % U	0.13	% U/tonne mined

15.2 Mineral Reserve

The mineral reserves for the project were developed by applying the relevant economic and design criteria to the resource model in order to define the economically extractable portions of the resource. The reserves were developed in accordance with CIM Best Practice Guidelines for Estimation of Mineral Resources and Mineral Reserves Reserve, and CIM Definition Standards for Mineral Resources and Mineral Reserves. They are disclosed in this report in accordance with NI 43-101.

Mineral reserves are subdivided in order of increasing confidence into probable mineral reserves and proven mineral reserves. A probable mineral reserve has a lower level of confidence than a proven mineral reserve.

15.2.1 Probable Mineral Reserve

A probable mineral reserve is the economically mineable part of an indicated, and in some circumstances, measured mineral resource demonstrated by at least a preliminary feasibility study. This study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified.

15.2.2 Proven Mineral Reserve

A proven mineral reserve is the economically mineable part of a measured mineral resource demonstrated by at least a preliminary feasibility study. This study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction is justified.

Application of the proven mineral reserve category implies that the qualified person has the highest degree of confidence in the estimate with the consequent expectation in the minds of the readers of the report. The term should be restricted to that part of the deposit where production planning is taking place and for which any variation in the estimate would not significantly affect potential economic viability. There are no proven reserves at Kuriskova.

15.2.3 Underground Mineral Reserve Calculations

The estimation of the mineable reserves involved the application of several parameters against the indicated mineral resource values. The parameters included cutoff grade determination, stope design, external dilution, and mining recovery. Each parameter is explained in more detail in the following sections. Tables were included where applicable to demonstrate the various parameters effects on the values.

The first step was to determine a cutoff grade. The cutoff grade as defined in Section 15.1 was set at 0.13 percent uranium or a finished product grade of 0.15 percent U_3O_8. Once the cutoff grade was determined, stope design was undertaken.

A block model and digital geologic wireframes were supplied for the project. The block model was supplied in a sub-blocked format of various block sizes which allowed the model to better follow the geologic wireframes. The largest block was 10 m, 10 m, 2 m (x, y, z) while the smallest was 1.25 m, 1.25 m, 0.002 m (x, y, z). The blocks were found to be spatially contained within the geologic wireframes. Block model attributes included; uranium percent, molybdenum percent, sulfur percent and resource class. Both the block model and geologic wireframes were loaded into Maptek's Vulcan Mine Planning Software (Vulcan). Once loaded into Vulcan the project resource was computed and validated against the resource statement.

Stope design was accomplished by using Maptek's Stope Optimizer within Vulcan. The Stope Optimizer is a tool for stope design that is comparable to Learch-Grossman open pit shell optimization. Economic parameters, stope geometry, geological and geotechnical constraints are input into the program and mineable 3-D stope triangulations are created. The stope shapes are then queried against the block model and the block model attributes contained within the shapes are reported. The Stope Optimizer allows for automation of stope shape creation in a repeatable format. In an effort to reduce the incorporation of dilution into the stope shapes the geometry of the stopes was aligned with the dip and strike of the foot and hanging walls supplied by the geologic wireframes.

Table 15.3 lists the parameters used to generate the Kuriskova stope shapes included in the following:

Table 15.3. Kuriskova Stope Parameters

Category	Value	Unit
Cutoff grade	0.13	% U
Stope orientation (stope wall strike and dip)	Zone 1N	Main zone north wireframe
Stope size in the X direction	36	Meters, with 4 sub units
Stope height	5	meters
Stope width	3 to 5	meters
Minimum dip angle	45	degrees
Maximum dip angle	135	degrees

Due to the narrow vein nature of the deposit the stopes were orientated transversely. Stope strike length was limited to 36 m long with a minimum size of 9 m. These dimensions were based on the available geotechnical data. The optimum stope run yielded a total of 2,030 stope shapes. Weight averaged stope uranium grades ranged from 0.13 percent to 1.39 percent uranium. Figure 15.1 displays a histogram of the stope shapes with respect to the uranium grade. Stope tonnage included 5 percent external dilution. Total diluted stope tonnage delivered to the underground process plant is estimated to be 2.5 million tonnes with an average grade of 0.36 percent uranium and 0.046 percent molybdenum.

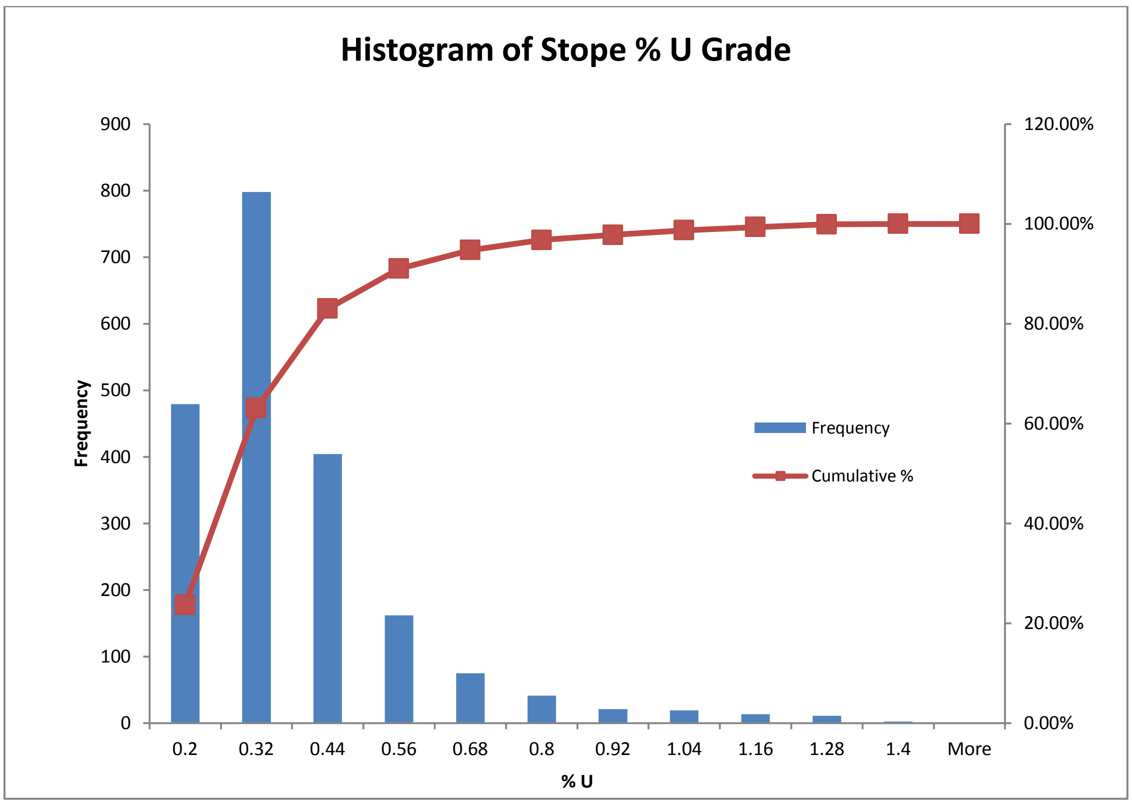


Figure 15.1. Histogram of Stope Uranium Percent Grade

Figure 15.2 and Figure 15.3 display the stope shapes as generated by the stope optimizer:

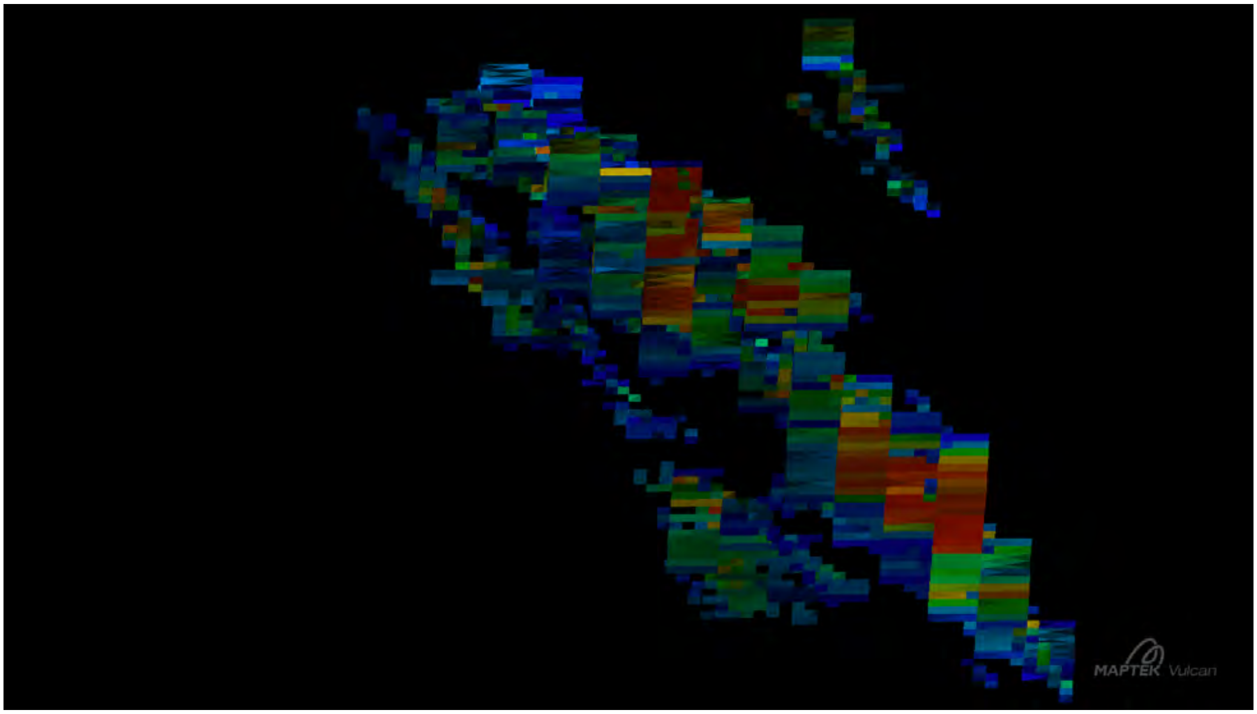


Figure 15.2. Stope Shapes as Viewed from the Footwall Looking Southwest

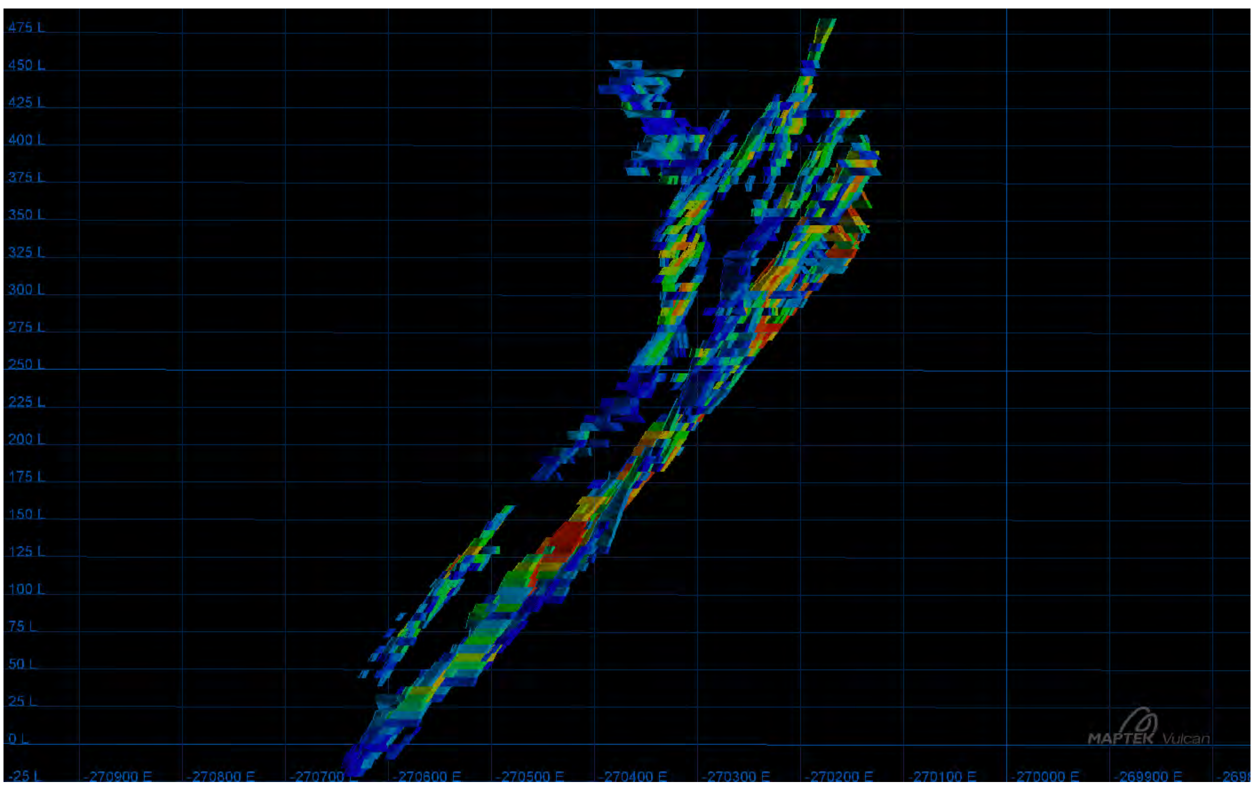


Figure 15.3. Stope Shapes as Viewed on Section Looking Northwest

Mining recovery is defined as the percentage of the planned diluted mined material which was able to be delivered to the process plant. Due to the rectangular nature of the stopes bottom some material was expected to remain during the loading cycle. Small losses were also expected during hauling and crushing. The overall estimated recovery of the planned mined material for the project was 96 percent.

15.2.4 Underground Mineral Reserve Statement

The mineral reserve listed in Table 15.4 was generated from the indicated mineral resource after the application of the cutoff grade of 0.13 percent uranium, stope design, external dilution, and recovery parameters. The reserves have been shown to be economic and Tetra Tech believes that they are reasonable for the statement of probable reserves.

Table 15.4. Kuriskova Mineral Reserves

Classification	Tonnes	Grade % U	Grade % Mo
Proven	0	N/A	N/A
Probable	2,528,000	0.346	0.046
Total	2,258,000	0.346	0.046

16.0 MINING METHODS

The deposit is planned to be extracted by an underground mine. The underground mine plan was designed around the steeply dipping mineralized zone, with an average thickness of 2.5 m and an approximate strike length of 650 m. Underhand cut and fill was chosen as the mining method after consideration was given to the geometry and grade of the deposit, rock mass strength and the tonnage requirements. The mine will be accessed by a 2.6 km decline, which will intersect a spiral ramp in the footwall of the deposit. Access drifts will be driven from the spiral ramp into the mineralized zone for production mining. Due to the low rock mass strength and rock quality designation (RQD) of the mineralized zone drill and blast within the ore body may be difficult to achieve, so a road headed was chosen as the primary production method. Once mined, rock will be transported to the process plant by 30 tonne underground haul trucks.

The process plant along with reagent storage, electrical rooms, control rooms, and the paste plant will all be located underground. This underground infrastructure was located approximately 250 m to the northeast of the deposit and will be accessed from the surface by the main decline.

16.1 Underground Mine Design

The underground mine design was completed using Maptek's Vulcan software and focused on safe and efficient extraction of the deposit. Once the stopes were created 3D mine development was laid out using centerlines. Underhand cut and fill was chosen as the production method. The decision to use underhand cut and fill involved considering the project's rock mass strengths, deposit geometries, grade distribution and project tonnage requirements. An advantage to a cut and fill mine plan at this stage of the project life is the de-risking of uncertainty around the ore body boundaries. Cut and fill mining allows stope shapes to be adjusted to follow irregularities of the ore body, and the ability to avoid low grades areas.

The deposit is steeply dipping and has a strike length of around 800 m. This shape provides for a classic mine design of a spiral ramp located in the footwall, with access drifts for stope entry. Access drifts were designed in a fan (benched) pattern to limit the potential for a high back while performing the next underhand cut access. Access drifts into the stopes were placed every 120 m in an effort to decrease haul distances and minimize open void spans. Stopes were aligned parallel to the strike of the deposit and are to be mined using a road header. The stope shapes are large enough to allow truck loading by a stacking conveyor off of the back of the road header. A small LHD machine will be required for cleanup and assisting the road header.

Once a stope has been mined, roof bolts will be wired standing on the floor in an upright position. A bulkhead will be constructed at the stope access and the stope will be backfilled with paste delivered by pressurized piping. Paste will contain a mixture of tailings, cement, and water. Figure 16.1, Figure 16.2, and Figure 16.3 display a general arrangement of the mine layout.

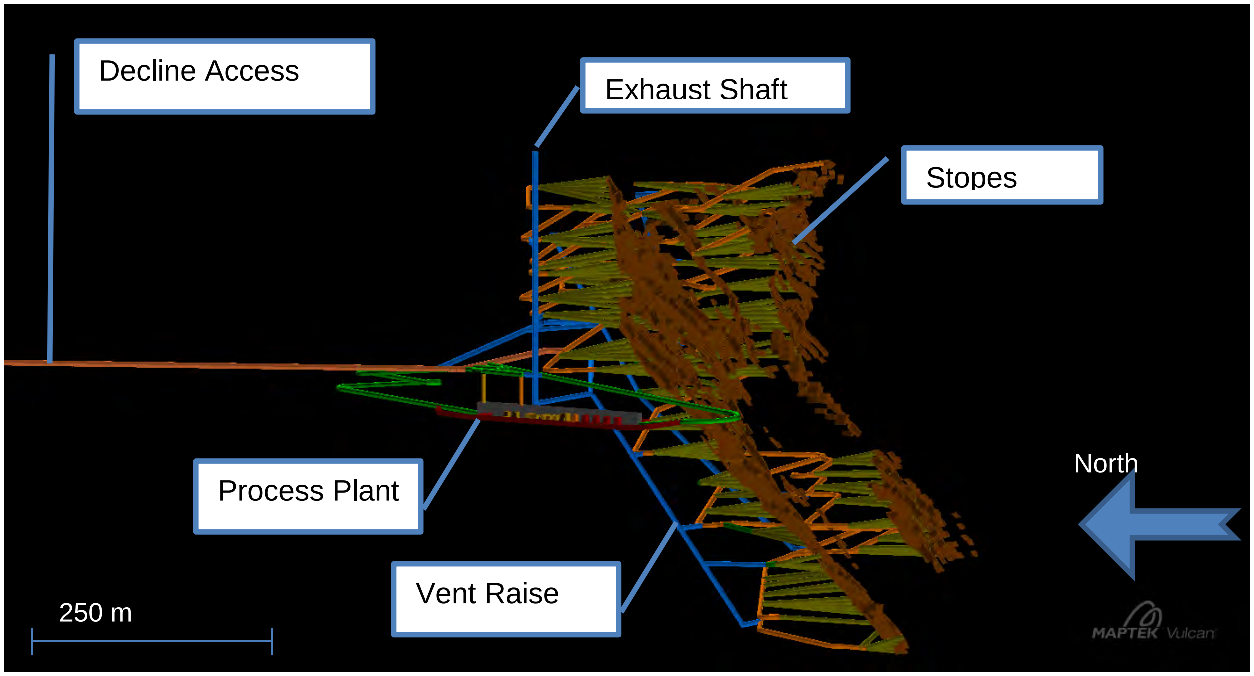


Figure 16.1. General Arrangement Figure Looking East

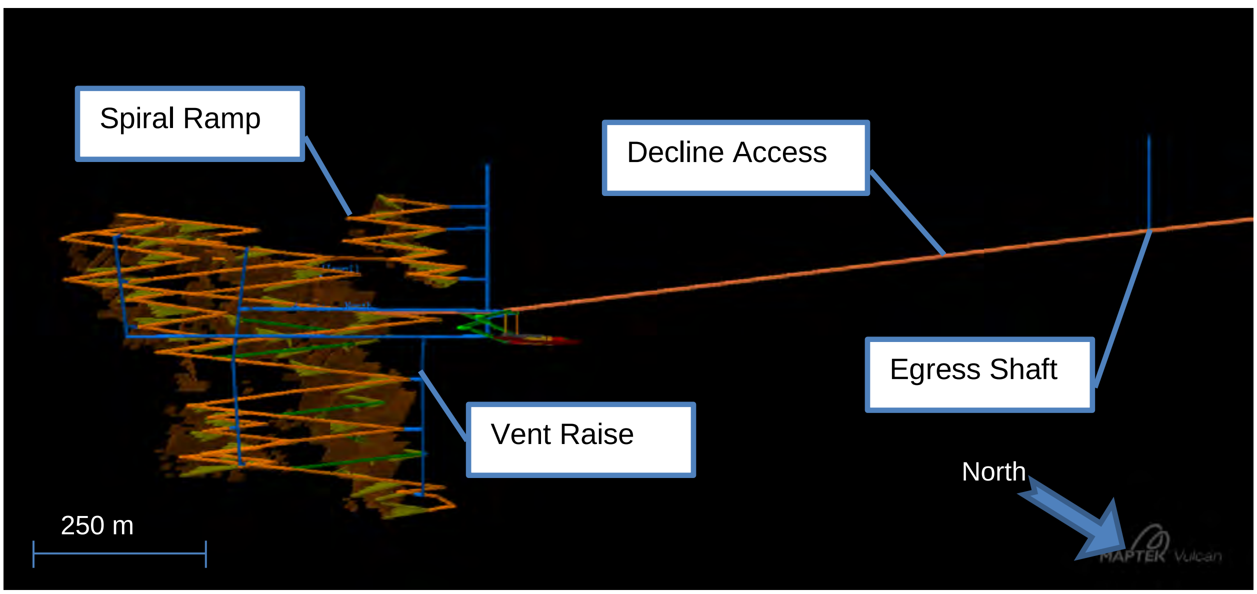


Figure 16.2. General Arrangement Figure Looking South West

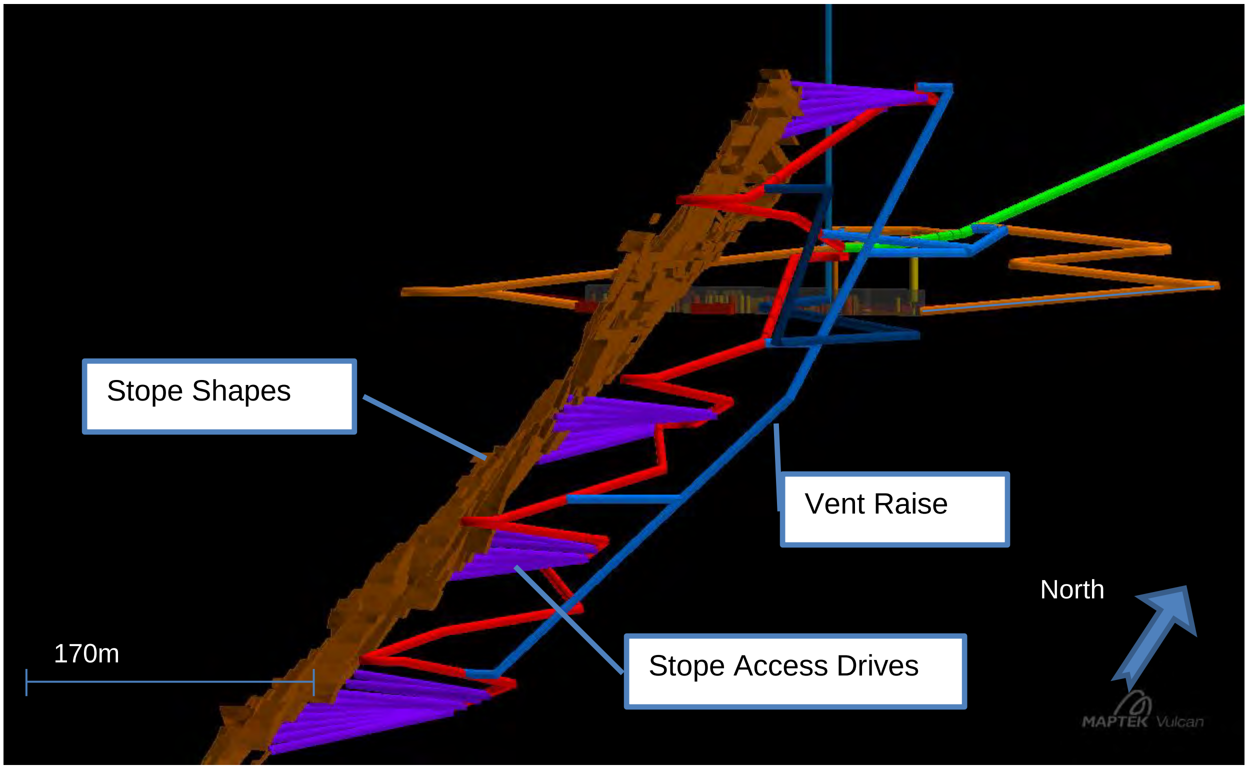


Figure 16.3. General Arrangement Figure Looking North West

16.1.1 Selected Mining Method

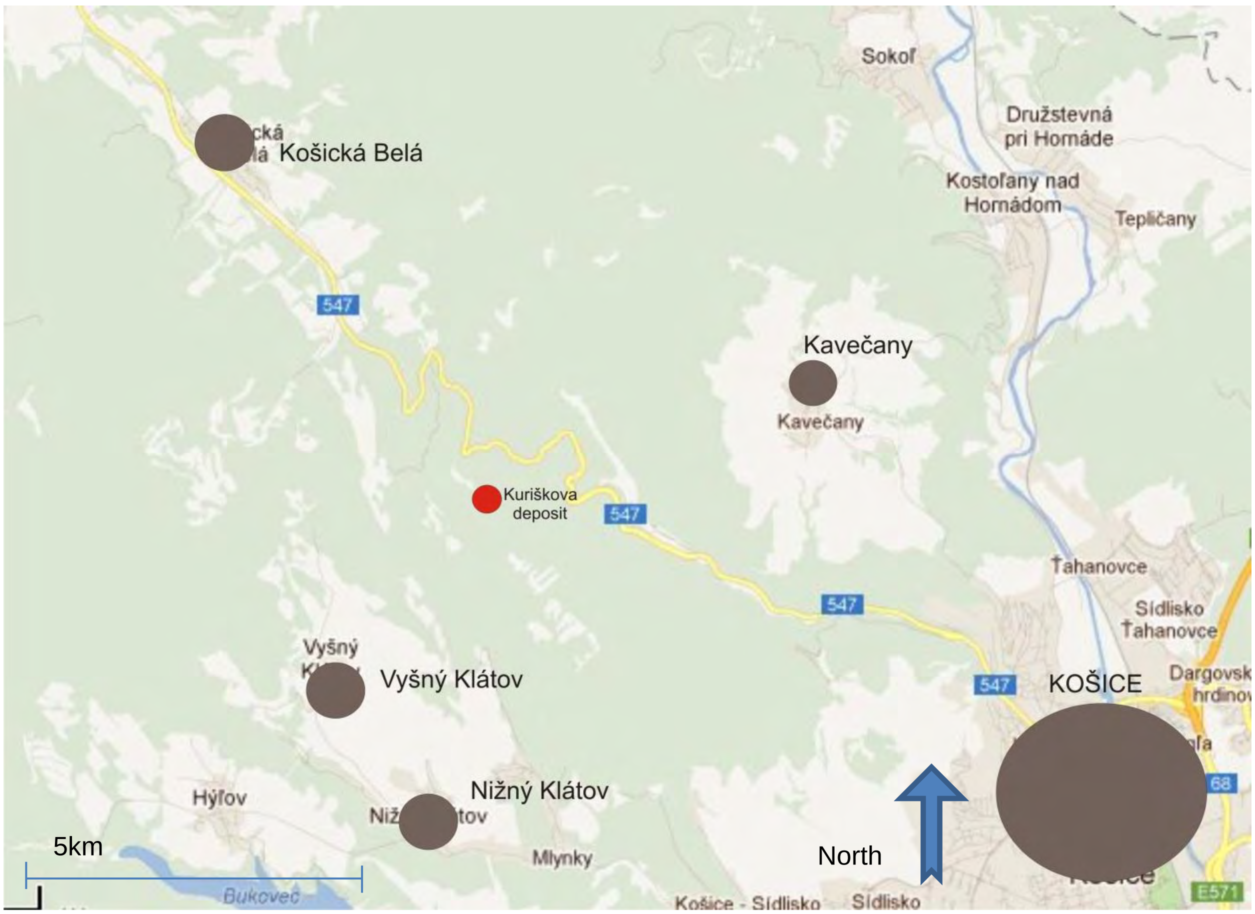
Underhand cut and fill mining with paste backfill was chosen to be the mining method for the deposit. The tonnage requirement of 600 tpd allowed for a lower production mining method. The primary driver was that rock quality was not high enough to support large open stopes, or to allow working under unsupported ground within the ore body. Once mined, a cut will be filled with paste backfill and allowed to cure. The next cut will be performed beneath a roof of cured paste backfill. Mined material is planned to be hauled from the stope and dumped into the run of mine surge bin which feed into crushers. The following criteria were considered during the mine design phase of the project:

- Geotechnical rock mass information
- Overall recovery of the deposit
- Deposit geometry
- Equipment capabilities
- Development capital expenses
- Development drifting
- Mine access

16.1.2 Mine Access and Development

The mine will be accessed by a 6x6 m decline that is approximately 2.6 km in length with an average grade of 9.5 percent. The decline extends from the surface to a depth of 292 m sea level elevation where it intersects the deposits footwall. Two ventilation shafts will also access the underground workings, a 4 m diameter egress shaft located mid-point along the decline and a 4 m exhaust ventilation shaft located adjacent to the underground process plant. The ventilation shafts will be fitted with emergency escape hoists, and are not planned to be used as man trips or for material hoisting purposes. The preferred mine access decline, as determined in the mine trade-off study, is in a northerly direction from the Kuriskova deposit. Additional costing and geotechnical studies to be performed in the feasibility study will determine its exact route and location. Figure 16.4 shows a general diagram of the Kuriskova deposit and facilities location.

Figure 16.4. General Diagram of the Kuriskova Deposit and Population Centers



Mine development for footwall drifts, access drifts, and spiral ramps are planned to be 5x5 m and horseshoe shaped. All drifting was expected to be done conventionally using drill jumbos in conjunction with 30 tonne underground haul trucks and 6 m^3 long haul dumps (LHDs). Utilities to be included with development included; water, electricity, ventilation ducts, communications and paste backfill pipes. Development grades were typically set at 10 to 12 percent and did not exceed 15 percent. Figure 16.5 displays a typical cross section for a development drift.

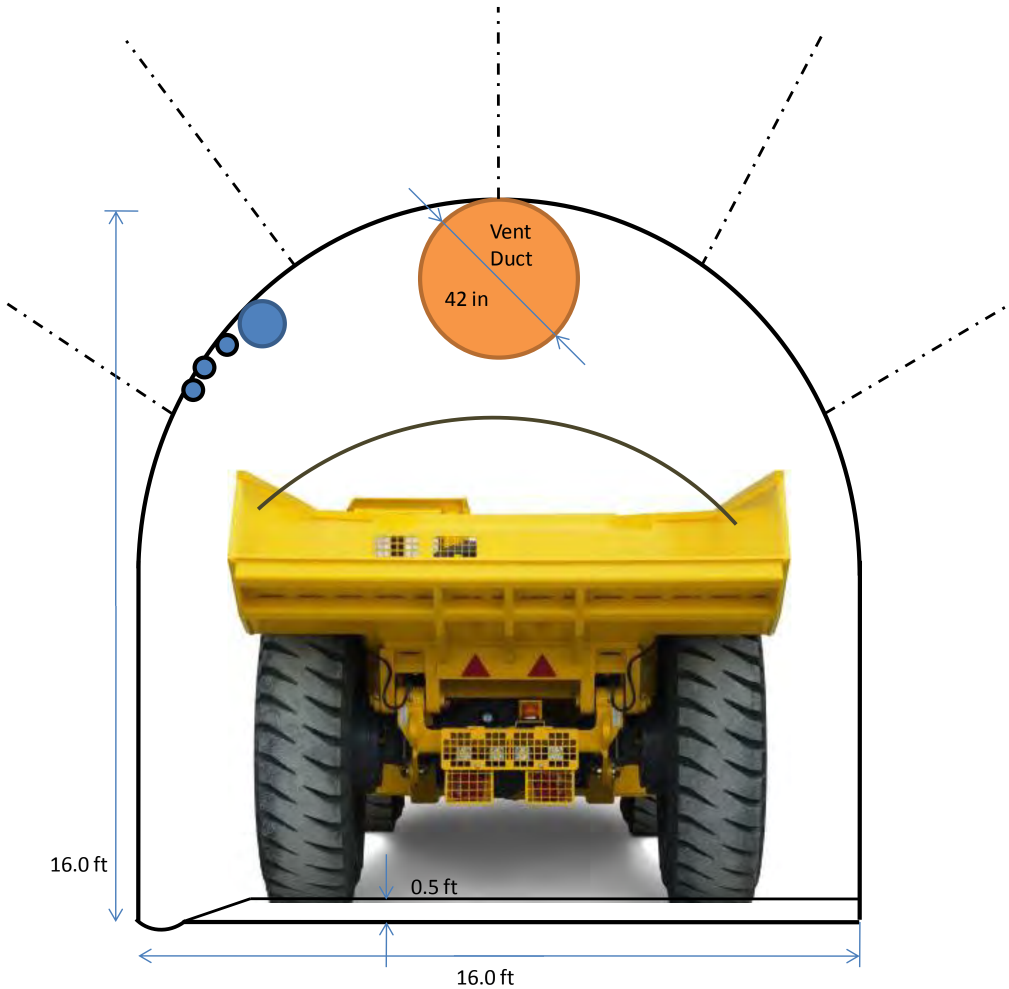


Figure 16.5. Cross Section of a 5x5 Drift

16.1.3 Shafts and Raises

Two shafts are required for the mine plan to meet the needs of secondary escape and ventilation. The planned diameter of the shafts is 4 m and construction is planned to be completed by a raise bore contractor. Each shaft will be equipped with an emergency escape hoist. One shaft known as the Egress shaft will be located midway along the decline and has total length of 153m. This shafts primary role will be to provide additional fresh air intake for the ventilation plan. The second shaft is an exhaust shaft and is located adjacent to the process plant and has a total length of 274 m. The ventilation plan will also require raises to distribute fresh air through the mine workings. The estimated length of ventilation raises for the mine plan was 1,280 m of 3 m diameter raises.

16.1.4 Ventilation Design

The ventilation system for the Kuriskova project has a total air flow of approximately 500,000 cu ft per minute during full production. One main exhaust fan is located in the exhaust shaft near the surface. The fresh air will intake through the decline and egress shaft. A system of ventilation raises is located behind the main spiral ramps in the deposits footwall. There are multiple connections from the ventilation raises to the spiral ramps. These connections will provided enough air flow through the mine without excessive ventilation restrictions. Air doors and auxiliary fans will be required throughout the mine.

The design basis for the ventilation system at Kuriskova was the volume of air requirement to dilute and remove radon gas along with exhaust gases produced by underground diesel equipment. Equipment utilization factors were used to represent the amount of diesel equipment in use at any time. Special ventilation consideration was given to the underground process plant. Fresh air will constantly be flowing through the process plant and immediately out the exhaust shaft.

The ventilation system design was modeled using Ventsim Mine Ventilation Simulation Software (Ventsim). This software allows input parameters including resistance, k-factor (friction factor), length, area, perimeter, and fixed quantities (volume) of air. Underground ventilation control requires several sets of ventilation control doors, regulators, and auxiliary fans to direct air quantities to the workings. Figure 16.6 and Figure 16.7 show the layout of the ventilation network when the mine is in full production.

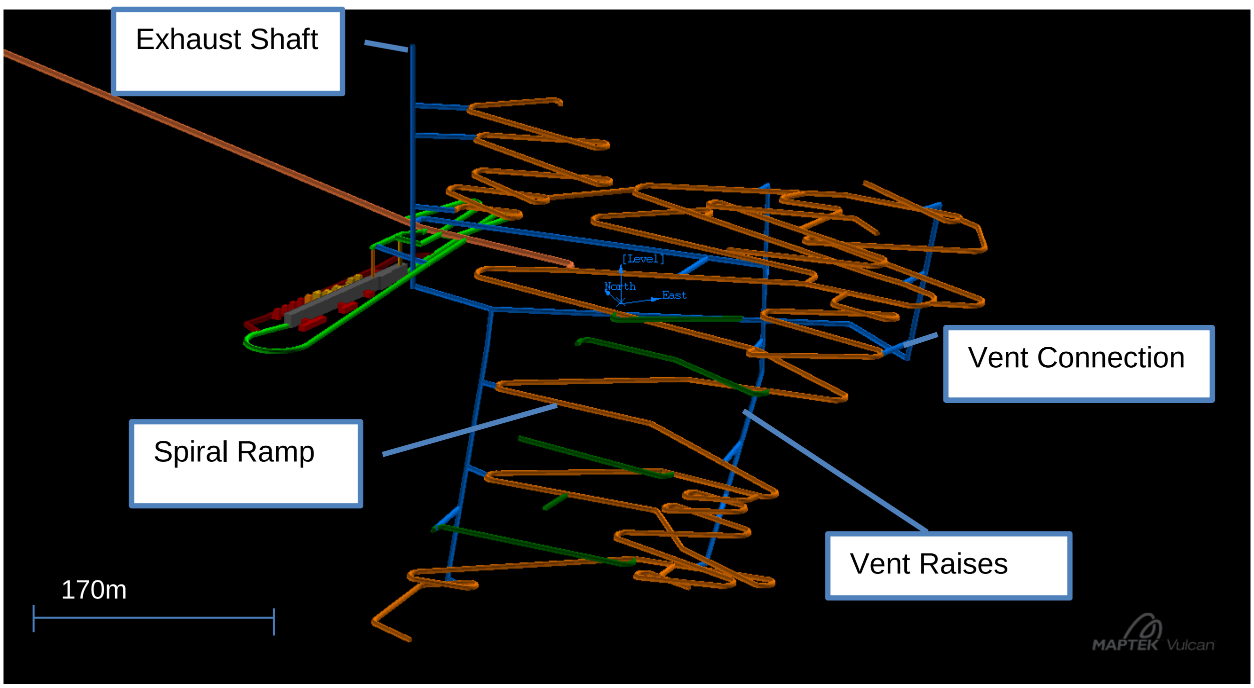


Figure 16.6. General Ventilation Arrangement Figure Looking North East

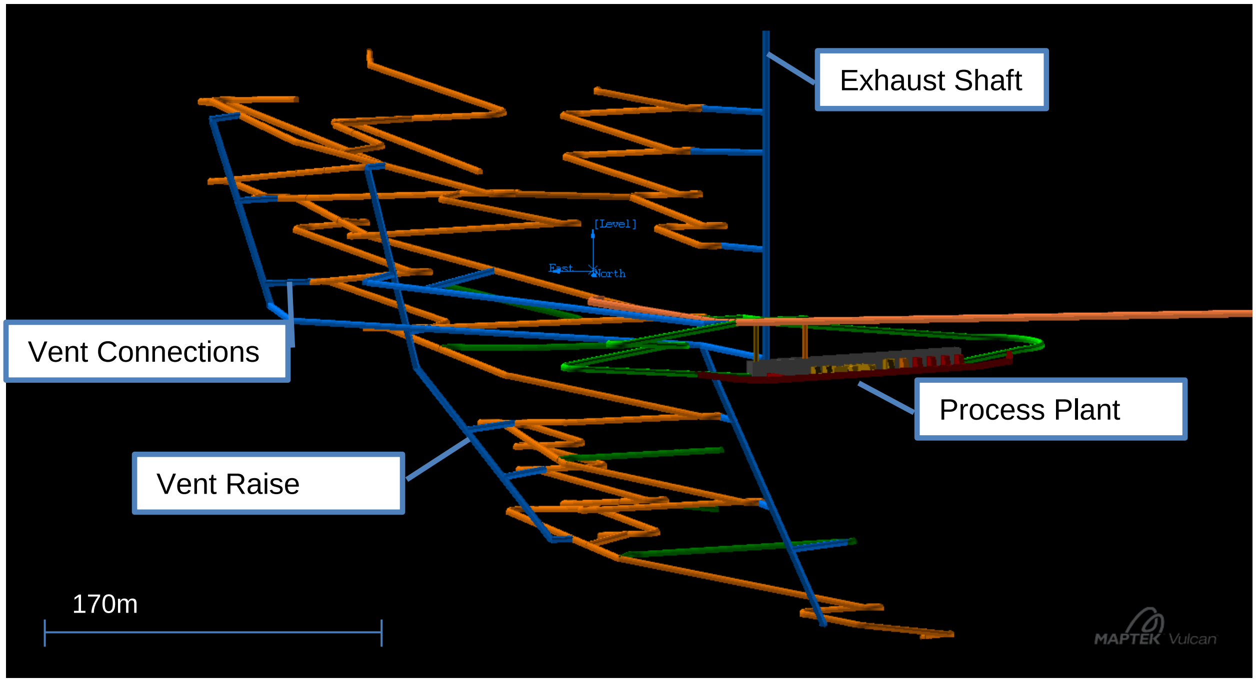


Figure 16.7. General Ventilation Arrangement Figure Looking South

16.1.5 Mine Services

Design consideration was given to mine services and included:

- Underground explosive storage
- Fuel storage and distribution
- Compressed air
- Water supply
- Mine Dewatering;
- Transportation of Personnel and Materials Underground;
- Under Maintenance and Wash Bays; and
- Mine Safety including:
- Fire Suppression
- Mine Rescue
- Refuge Stations
- Emergency Egress

16.2 Mine Equipment

The Kuriskova underground mine is planned to be highly mechanized and completely trackless. All mobile equipment is planned to be purchased new, and replaced at manufactured specified expected life (in terms of hours operated). Once the equipment has reached its first life expectancy it is planned to be rebuilt at 60 percent of the cost to purchase new. The second time the equipment has reached its life expectancy the machine will be replaced.

Equipment hours were generated from first principles cost model; hours were calculated for all equipment on the project specified time line. The overall mobile equipment list is shown in Table 16.1.

Table 16.1. Underground Mine Major Equipment

Equipment	Qty
Road Header	2
Scissor Truck	2
ANFO Loader	2
ANFO Truck	1
U/G Personnel Carrier	3
Service Vehicle	1
Drill Jumbo – 3 boom	1
Rock Bolt Jumbo	2
Drill Jumbo – 2 boom	2
LDH Units – 6 m^3	2
Mine Trucks – 30 Tonne	6
LHD Units – 3 m^3	2
Motor Grader	1
Mechanics Truck with Jib	1
Fuel and Lube Truck	1
Boss Buggies	5
Shotcrete Unit	1
Skid Steer Loader	1
Telehandler Forklift	1

16.2.1 Development Equipment

Drift development fleet was established by matching the client advised advance rate of 6 m per day per heading with the mine plan required drifting lengths and headings on an annual basis. To achieve an advance rate of 6 m per day during decline construction, and taking into consideration the labor limitation of 8 hour shifts, a three boom drill jumbo was required. After the decline is complete and multiple development headings are available two development equipment spreads will be required. Each equipment spread will include a 2 boom drill jumbo, 6 m^3 load haul dump, and rock bolter. Two anfo loaders were included, and one anfo transport. Thirty tonne haul trucks will be assigned to development crews as needed per calculated haul distances. After production Year 1 the development requirements are such that only one development equipment spread will be required.

16.2.2 Production Equipment

The ore body rock lends itself to the use of a roadheader as the main production mining machine. Based on the current understanding of the ore body an Alpine WS200CS was chosen.

The roadheader will be equipped with an axial cutting head, gathering apron and a conveyor off of the back to load haul trucks. One Apline WS200CS is capable of achieving a production rate of 600 tonnes per day, however do to stope sequencing and tramming requirements two machines were included in the mine plan. From an operational stand point a second production machine provides flexibility if one of the machines must be down for planned maintenance.

Support equipment for the production fleet will include a small LHD to help prep and clean up stopes for the roadheader. Haul trucks will be assigned to a production crew as determined by the haul distance to the dump point. Blasting is not anticipated to be required for production minin; however, if the need arises, a development crew will be able to assist.

16.2.3 Support Equipment

Support equipment includes major equipment that is required to install mine services, transport personnel, pump water or compress air, and provide temporary ventilation. Mechanic trucks, fuel and lube trucks along with skid steers, and forklifts are included as service equipment.

16.3 Ground Support/ Rock Mechanics

16.3.1 Introduction

The purpose of these sections is to report on the 2011 rock mechanics program and its use in mine planning for the Kuriskova project and includes the rock mechanics data collection, the application to mine planning, the paste backfill testing program and its application to paste backfill design. The analysis includes five hole geotechnical drilling program with laboratory testing for paste backfill design.

16.3.2 Geology

The mineralization at Kuriskova is a re-deposition of uranium and base metals in fractures controlled by folding and thrusting. Regional mountains were built from the tectonic action. The Main Zone uranium mineralization is a stratabound zone of mineralization following the once-horizontal contact between lower sandstones and shales and overlying andesites and volcaniclastics. Mineralization occurs in the fractured andesite tuffs immediately above the contact, and extends into the hanging wall andesites for variable distances. Mineralization is fairly continuous, high grade, and varies in thickness from 2 to 8 m. The zone has been explored to date over 650 m of strike length and to 550 m depth. Both transverse and thrust faults have segmented the body into blocks, with displacements of up to tens of meters. Mineralization along zones cut by thrust faults are enriched by later remobilization. In the hanging wall andesites, the uranium mineralization occurs in the form of stockwork veins and thin stringers that form irregular clusters. Stringers range from several millimeters to 10 to 15 centimeters in width. Uranium grade tends to increase with increasing proximity to major faults and fracture zones.

The second mineralized zone is stockwork uranium mineralization that occurs in the approximate centre of the hanging wall andesite unit, approximately 10 to 50 m stratigraphically above the tabular Main Zone. The thickness of the zone is variable from 1 to 10 m (maximum of 20 m) that is roughly concordant with lithologic layering. The zone appears to occur in the rheological transition from competent andesite over schistose tuffaceous volcaniclastics and sediments. Faults segment the stratabound zone into blocks. The mineralization is lensoidal with thicknesses to 4.5 m, and generally hosts lower grade mineralization in contrast to Main Zone mineralization. The uranium mineralization occurs in irregular quartz-carbonate stringers with apertures of 1 to 5 mm (to 5 cm maximum). From a regional exploration perspective, the stockwork mineralization offers the potential of significant tonnage expansion, albeit of lower grade mineralization.

The third recognizable zone of uranium-molybdenum mineralization occurs within the tuffs and tuffaceous rocks overlying the andesite and volcaniclastic units. Mineralization is disseminated, very low grade and discontinuous, occurring 20 to 40 m above the andesite–tuffaceous contact. Figure 16.8 offers a generalized cross section of the geology.

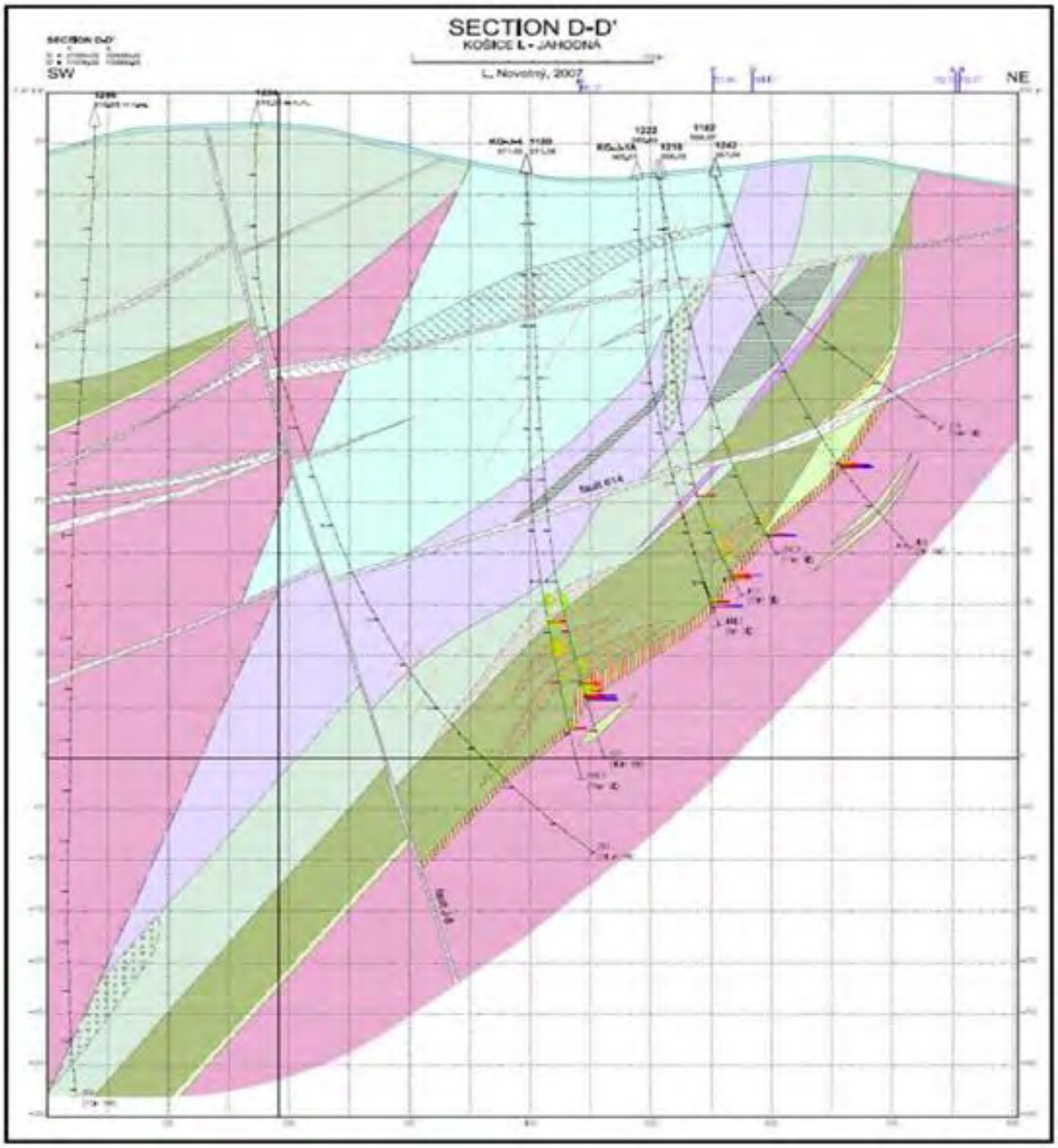


Figure 16.8. General Lithologic Cross Section

16.3.3 Rock Stress Conditions

Rock stress in mining environments is derived from horizontal and vertical stresses. Vertical stresses are typically the result of gravity load or the weight of the rock above the area of interest. In situ vertical loading for Kuriskova operations is projected to achieve a maximum of 18.7 MPa for the 750 m of deposit depth found in hole LE-K-71. In checking the World Stress map (Heidbach 2009), the setting of Kuriskova finds no extraordinary evidence of excess horizontal stress. The southern portion of Poland bounding Slovakia has some stress evidence, but sufficient distance exists between Kuriskova and the Polish border. The Kuriskova deposit is depositional controlled followed by structural alteration. The alteration came from the regional mountain building lateral stresses. Excess horizontal stress has dissipated as evidenced by the shear zones found in the Kuriskova drill core. Based on these criteria, it is assumed horizontal stress is not excessive and is equal to the vertical in situ loading. Further to support this characteristic is that any excess stress would be carried in the highest modulus rock, which in the case of Kuriskova is the violet schist which is above the deposit.

16.3.4 Rock Mass Classification

The rock mass rating (RMR) utilized in the analysis was based on five drill holes and testing data from the 2011 program. Laboratory analysis including physical strength testing on rock specimens was performed at the Ingeo-Envilab at Zilina, Slovakia (Janis, 2011) and Advanced Terra Testing at Lakewood, Colorado, USA (ATT, 2011). RMR is a standardized method of accessing rock characteristics use din mine design. Defining the RMR is done by logging the drill core measuring strength, joint frequency and condition, water, and strike and dip of structure. Figure 16.9, Figure 16.10, Figure 16.11, Figure 16.12, and Figure 16.13 show the results of these measurements.

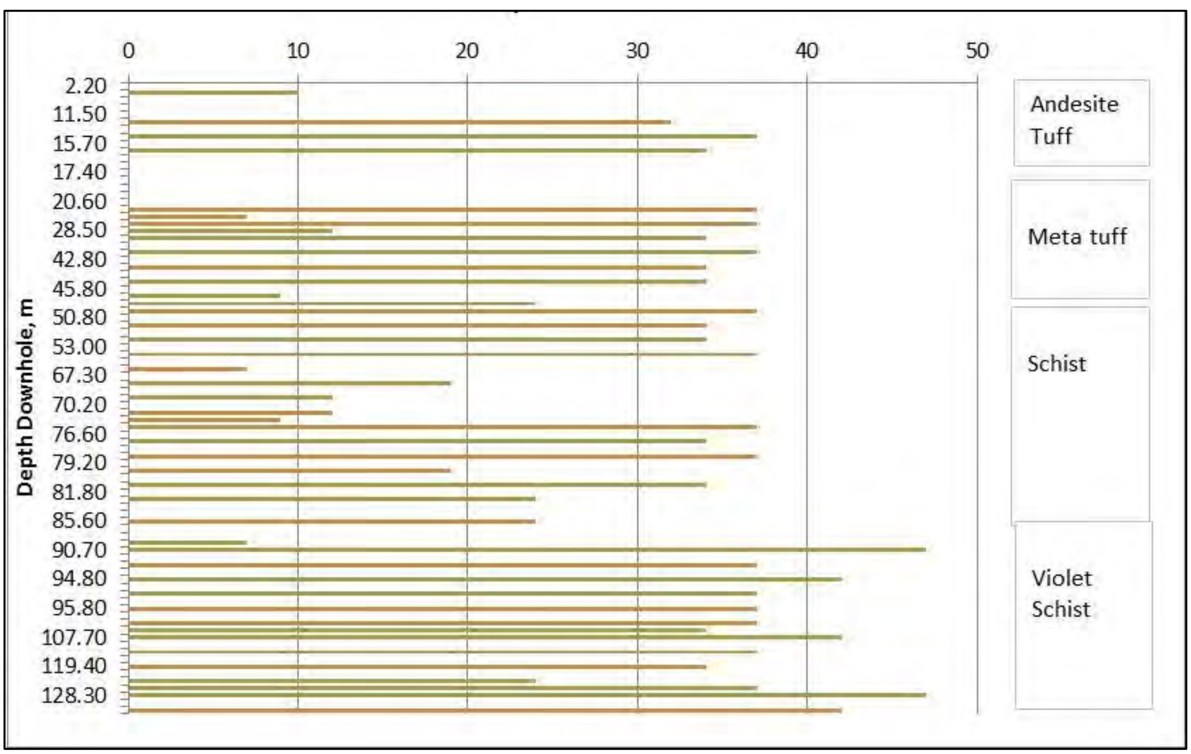


Figure 16.9. 2011 Geotechnical Holes Mine Access (KB-2-G)

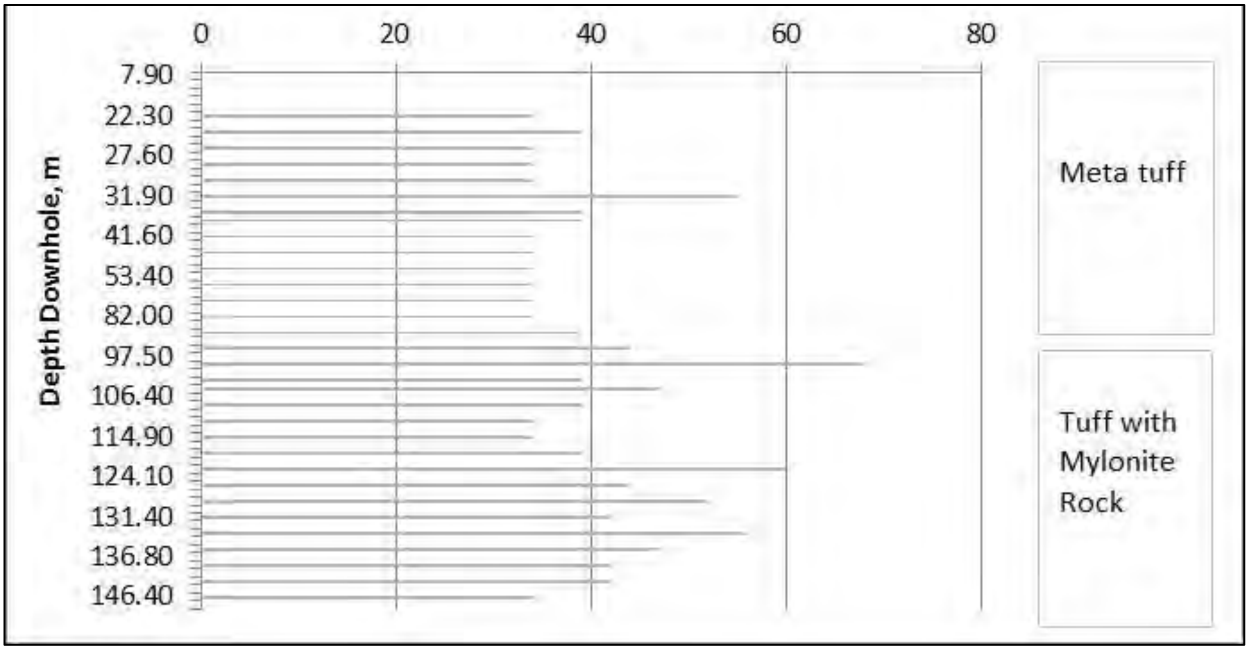


Figure 16.10. 2011 Geotechnical Holes, Mine Access (KB-3-G)

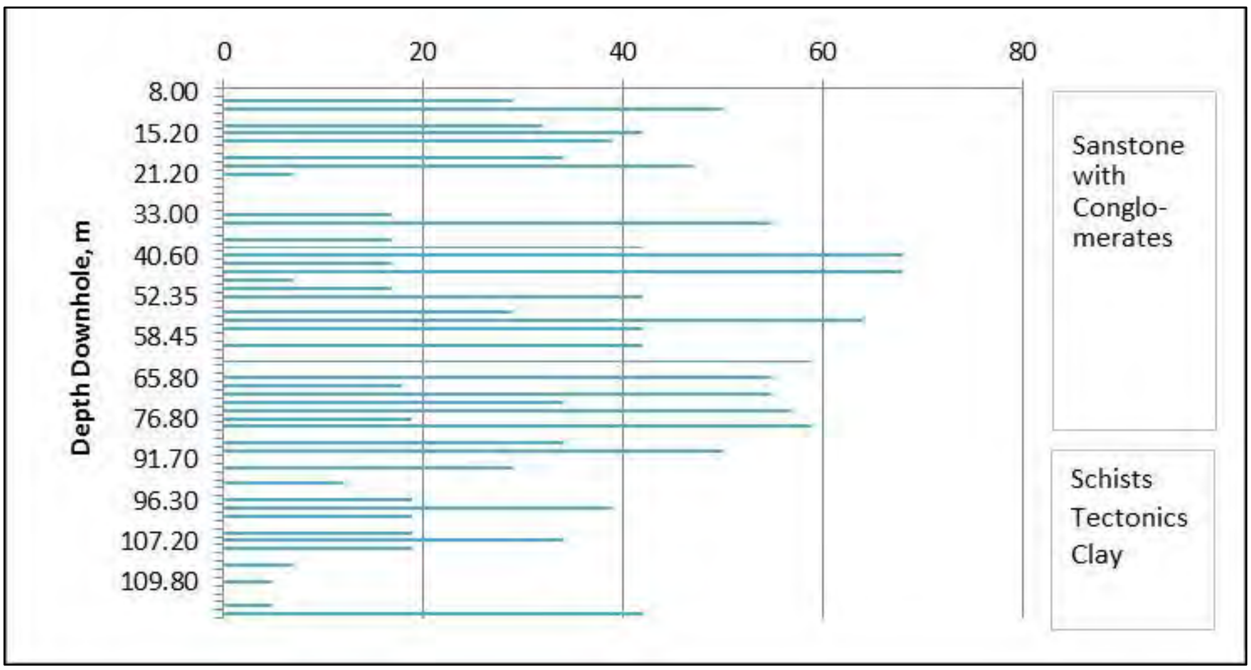


Figure 16.11. 2011 Geotechnical Holes, Kuriskova Deposit (KB-4-G)

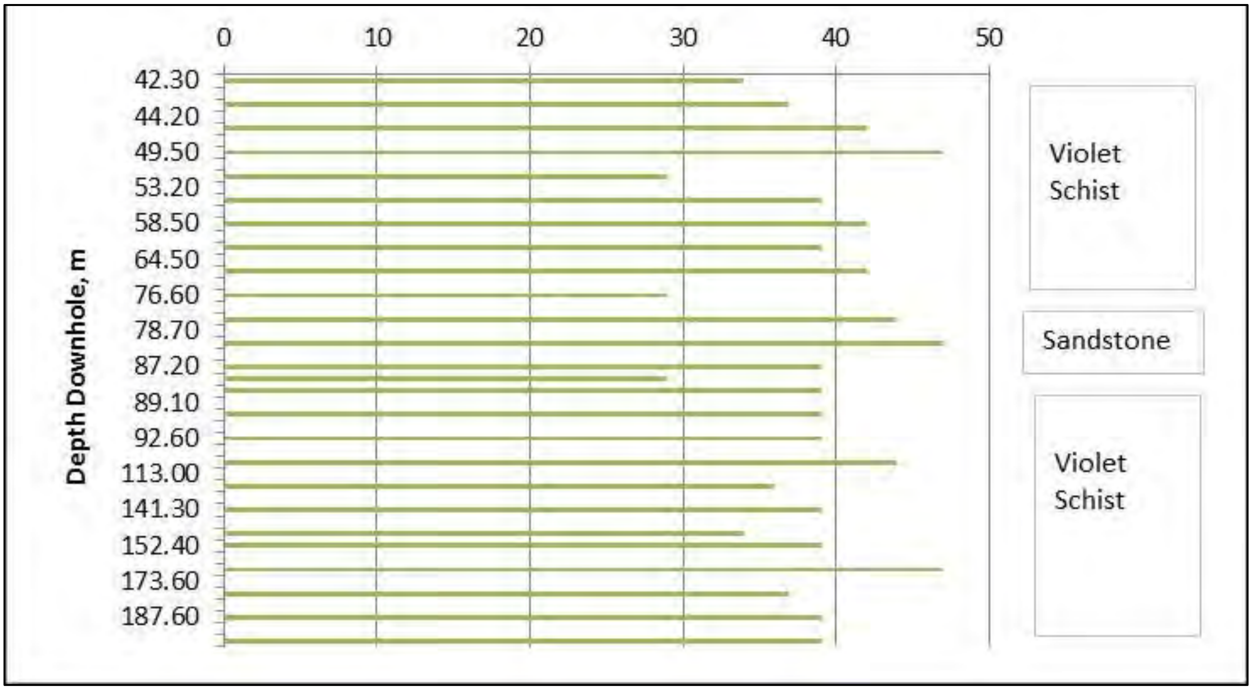


Figure 16.12. 2011 Geotechnical Holes, Mine Access (KB-6-G)

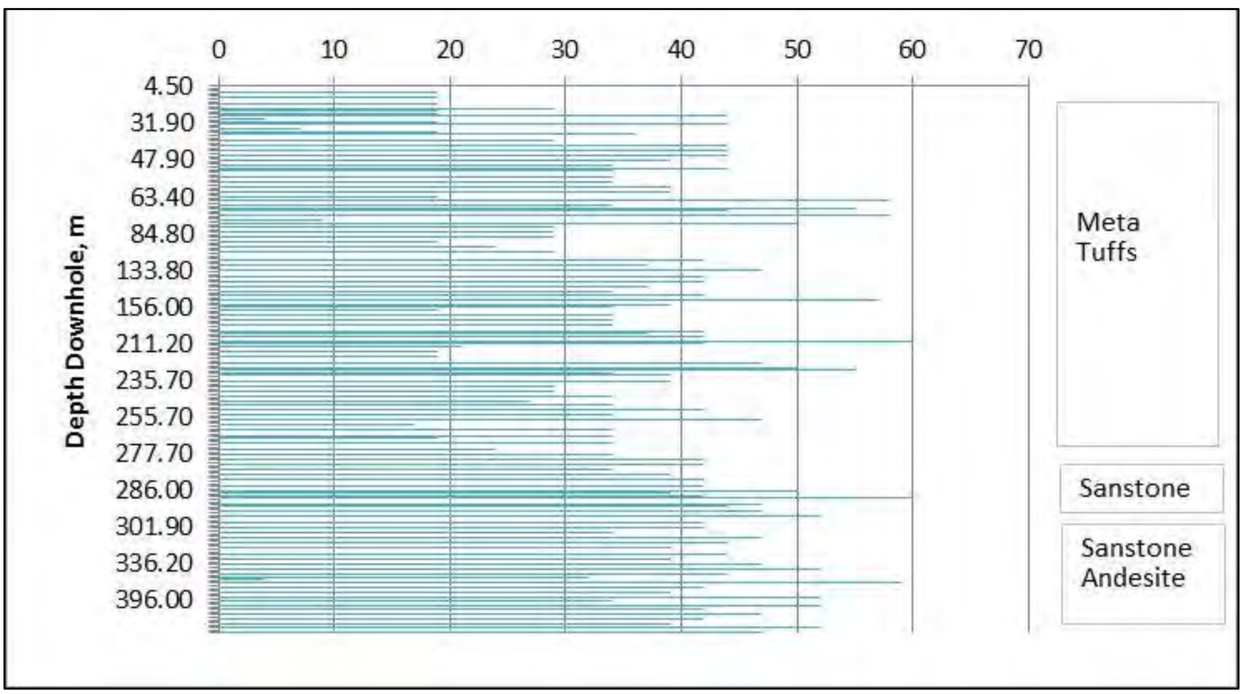


Figure 16.13. 2011 Geotechnical Holes, Kuriskova Deposit (LE-K-70-G)

The immediate highwall and footwall of Kuriskova underground is a sandstone/andesite best shown in hole LE-K-70-G. This unit has a range in RMR of 35 to 45, and is thusly classified as weak rock. The overburden rock that the decline will drive through is shown in KB-2-G and KB-6-G at 0 to 150 m depth. The rock types encountered are schists, violet schists, and sandstones having a range in RMR of 30 to 45 with bands of broken rock having as low as 0 RMR. The broken rock is shear zones created from tectonic force. Figure 16.14 contains an example picture of the ore host.

Figure 16.14. Ore Host in LH-NH-4 541 to 556 m Downhole

Using 2011 drilling results, Table 16.2 summarizes the RMR system for the rock types using Bienawski (1989).

Table 16.2. RMR Classification

Rock Unit (Top Down)	RMR Range	RMR Median	RMR/ Type
Alluvium	Soil	Soil	Soil
Andesite Tuff	30/40	35	Upper IV, poor rock
Meta Tuff	30/45	38-40	Upper IV, poor rock
Schist	30/40	35	Upper IV, poor rock
Violet Schist	35/45	36-39	Upper IV, poor rock
Sandstone	35/50	42-44	Lower III, fair rock
Sandstone/Andesite	35/50	42-44	Lower III, fair rock

16.3.5 Major Structure

It is anticipated that structural geology will affect underground operations because of the numerous and continuous cores of broken rock. The structures traverse through the mining areas and are a direct factor in applying an underhand cut and fill mining method in that structure has fractured the rock. The mine plan does not attempt to specifically delineate structure, but rather accounts for it by applying conservative ground control systems, the mining method of underhand cut and fill, and paste backfill.

16.3.6 Hydrogeology as Applied to Ground Control

Hydrogeology was considered from a rock mechanics perspective. Kuriskova sub-surface waters are contained in shear and structure systems at shallower depths (less that 500 m). Pump tests performed on various rock intervals revealed the rock although broken is not necessarily permeable. This is probably due to the relative tightness of the joints and fractures. Considering the pump tests and fitting them into a localized portion of the mine layout yielded a water make prediction of 9.5 m^3/sec. As the mine expands and achieves maturity in Years 8 to 12, this water make will probably increase due to the wetted perimeter expanding. Because the pump test yielded a low result, water in and of itself is not expected to impact mining productivity.

16.3.7 Physical Core Testing

A physical core testing program was performed on specimens from the major rock units encountered at Kuriskova. Most of the laboratory testing was undertaken at a certified Slovakia laboratory operated by the Ingeo Construction Company located at Zilina for UCS and tensile strength, and direct shear strength. Samples were also tested for triaxial compressive strength at Advanced Terra Testing at Lakewood Colorado, also a certified laboratory. A balanced program of 72 total tests was implemented to extract physical characteristics for design. Table 16.3 shows the average results for the testing. Individual rock types are not differentiated due to low sample count.

Table 16.3. Rock Physical Testing

Test	Samples	Range (MPa)	Mean (MPa)
UCS	12	5.5-56.1	28.7
Brazilian Tensile Strength	10	1.2-8.4	3.3
Direct Shear	1	9.7	N/A
Elastic Modulus	2	11,511; 13,223	12,367

Two triaxial compressive tests done on meta tuff and meta andesite consisting of three confining pressures. The average of the triaxial tests resulted in the following confining pressures; 4.1, 8.3, and 12.4 MPa. The meta tuff averaged 61.5 MPa compressive strength through the 3 confining pressures and the meta andesite averaged 97.4 MPa. The results support that mine pillars having confinement of the outer layers will be stable.

Measured strength is higher from individual specimens than the rock mass due to jointing and fractures. The schists tested had UCS strengths from 38 to 56. The other rock types had lower UCS strengths in the 5.5 to 20 range. The strongest rock tested was a violet schist having a 56.1 MPa (8,134 psi) UCS.

16.3.8 Paste Backfill Test Results and Design

For efficient environmental performance, the need of structural fill for mine ground control, and to increase recovery, the use of paste backfill was considered in detail. Tetra Tech tested various paste backfill designs using Kuriskova process tailings from metallurgical bench tests. A design target of 3.5 MPa was established by benchmarking successful cut and structural fill underhand stoping worldwide. Strength tests were completed at Agapito and Advanced Terra Testing certified laboratories in Grand Junction and Lakewood Colorado, respectively (Agapito 2011, ATT 2011). Table 16.4 below lists the test results for varying mixtures.

Table 16.4. Paste Backfill Strength Test Results

Test	Cement (g)	Tap Water (g)	Sand (g)	Process Tailings (g)	Type C Fly Ash (g)	Quarry Rock (g)	Total (g)	Mini-Slump (mm)	28-day Strength UCS (Mpa)
Test No 1	202	1,453	0	2,400	0	0	4,055	35	0.8
Test No 2	180	854	0	1,440	0	1,008	3,482	32	1.1
Test No 3	185	732	0	1,036	0	1,480	3,433	44	1.3
Test No 4	304	1,202	0	2,546	0	0	4,052	6	1.8
Test No 5	185	1,204	0	2,035	556	0	3,980	35	2.4
Test No 1F	285	855	0	1,282	143	285	2,850	95	3.0
Test No 2F	360	1,011	0	1,202	185	750	3,509	83	3.4
Reconciliation % by wt									
Test No 1	5.0%	35.8%	0.0%	59.2%	0.0%	0.0%	100.0%	35	0.8
Test No 2	5.2%	24.5%	0.0%	41.4%	0.0%	28.9%	100.0%	32	1.1
Test No 3	5.4%	21.3%	0.0%	30.2%	0.0%	43.1%	100.0%	44	1.3
Test No 4	7.5%	29.7%	0.0%	62.8%	0.0%	0.0%	100.0%	6	1.8
Test No 5	4.7%	30.2%	0.0%	51.1%	14.0%	0.0%	100.0%	35	2.4
Test No 1F	10.0%	30.0%	0.0%	45.0%	5.0%	10.0%	100.0%	95	3.0
Test No 2F	10.3%	28.8%	0.0%	34.3%	5.3%	21.4%	100.0%	83	3.4

The best mix to achieve the objective 3.4 MPa is Test No 2F. This mixture utilizes quarry rock which adds strength, fly ash which gives beneficial use of coal-fired waste stream, and tailings. Geo-chemistry testing subsequent to the strength testing showed the fly ash to produce excessive alkalinity. As a result, final mix used in mine cost calculations was 60 percent tailings, 11 percent cement, and 29 percent water. The quarry rock was dropped from consideration because the mine development rock will produce sufficient material for community and mine beneficial use and the rock component for strength is not necessary. The increased water to the 29 percent level will allow for lower cost paste backfill pumping.

Paste backfill pump sizing was completed by Putzmeister, a world leader in paste pumping. Based on 29 percent water content, an 85 mm mini-slump for rheology, and the mine layout, a pump was designed and costed having a capability to 20 m^3/hr at 100 bar. Operating pressures of the paste backfill are calculated to be in the 30 to 60 bar range.

A survey was done for 33 Canadian mines using mine paste backfill of various types and various applications (Souza). This survey showed that 50 percent of the application was for ground control. The other 50 percent was for a combination of reducing mining costs, environmental protection, fire control, and ventilation. The types of mining methods where paste backfill was applied were 33 percent for various types of cut and fill and 67 percent for non-cut and fill applications.

16.3.9 Geotechnical Feasibility of Underground Mining

The geotechnical feasibility of underground mining is derived from the ground control support required to have stability of the rock mass. Primary determinants of mining method are rock mass strength and ore body shape. De Souza offers a method to identify mining method (De Souza, 1987) shown in Figure 16.15.

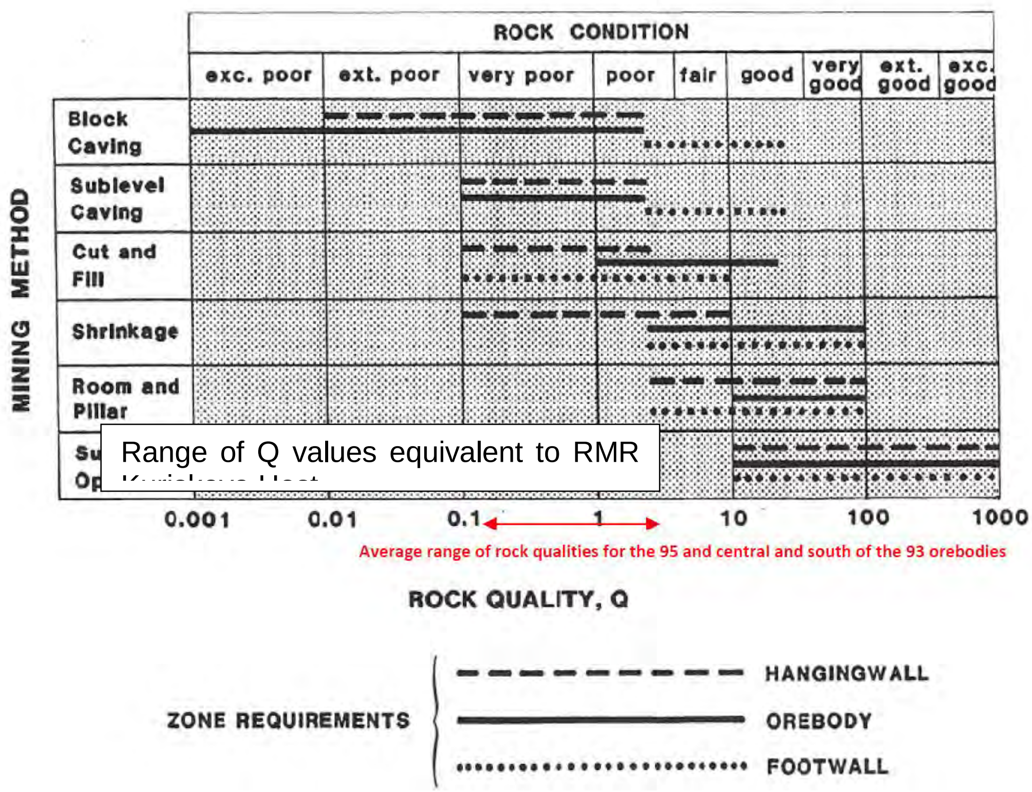


Figure 16.15. Mining Method Selection Based on Rock Quality

The measured RMR of Kuriskova falls in the range of cut and fill mining method. The caving methods are not applicable as control of the ground would be compromised and excessive dilution would result. Another alternative is longhole stoping with immediate paste backfill. This method is not applicable to Kuriskova because of the thin (2.5 m to 8 m) and steeply dipping mineralization.

16.3.10 Primary Roof Support

Primary support in the poor rock of the Kuriskova formation can be estimated using the well proven index of RMR (Table 16.5).

Table 16.5. Primary Roof Support for Rock Mass Ratings.

Rock Mass Class	Excavation	Rock Bolts (20 mm Diameter, Fully Grouted)	Shotcrete	Steel Sets
I - Very good rock RMR: 81-100	Full face, 3 m advance.	Generally no support required except spot bolting.		
II - Good rock RMR: 61-80	Full face, 1-1.5 m advance. Complete support 20 m from face.	Locally, bolts in crown 3 m long, spaced 2.5 m with occasional wire mesh.	50 mm in crown where required.	None.
III - Fair rock RMR: 41-60	Top heading and bench 1.5-3 m advance in top heading. Commence support after each blast. Complete support 10 m from face.	Systematic bolts 4 m long, spaced 1.5 - 2 m in crown and walls with wire mesh in crown.	50-100 mm in crown and 30 mm in sides.	None.
IV - Poor rock RMR: 21-40	Top heading and bench 1.0-1.5 m advance in top heading. Install support concurrently with excavation, 10 m from face.	Systematic bolts 4-5 m long, spaced 1-1.5 m in crown and walls with wire mesh.	100-150 mm in crown and 100 mm in sides.	Light to medium ribs spaced 1.5 m where required.
V – Very poor rock RMR: < 20	Multiple drifts 0.5-1.5 m advance in top heading. Install support concurrently with excavation. Shotcrete as soon as possible after blasting.	Systematic bolts 5-6 m long, spaced 1-1.5 m in crown and walls with wire mesh. Bolt invert.	150-200 mm in crown, 150 mm in sides, and 50 mm on face.	Medium to heavy ribs spaced 0.75 m with steel lagging and forepoling if required. Close invert.

Given that most rock types are in the poor category to very low fair category, systematic bolting, screening, and shotcrete will be required. Although this is a high cost ground control pattern, the roof can be supported.

For all development drifts, the ground control system consisted of 2.5 m long tensionable resin bolts on a 1.5 m square pattern placed on cycle, wire mesh on 505 of all drifts, and 0.1 m shotcrete on 25 percent of all drifts. The tensionable criterion is to avoid the keystone failure mode. Keying considers the force necessary to hold in place loose stones. The keying method utilizes the tension along the bolt axis to add sufficient force such that the resultant force along the plane of weakness is greater than the force pulling the block out of the roof.

16.3.11 Ground Support Feasibility for Underground Access

The underground mine will have an egress shaft, a main ventilation shaft, and decline access to the underground process plant and ore zones. Shafts are all 4 m ID lined with concrete with sufficient thickness based on the Lame formula. The decline because of its long life and critical function is planned to have 100 percent bolts, screened, and shotcreted. It is unlikely that roof support could be reduced to a spot basis anywhere in Kuriskova.

16.3.12 Underground Process Plant and Infrastructure

A variety of infrastructure will be excavated underground including the underground process plant, refuge chambers, and other bays. To avoid over-width excavations in the poor rock, any span greater than 8 m would be reduced by intermediate standing support. Due to the size of the underground process plant, the ground control plan includes standing support decreasing the span, bolted, screened, and 0.2 m of shotcrete.

16.3.13 Pillar Size Feasibility and Span Feasibility

Pillar feasibility was designed using empirical methods. The most fundamental design factors in empirical pillar formulas are the degree of reduction taken for in situ compressive strength versus laboratory strength, and a strength modification for height to width ratio to account for slenderness. Reductions are necessary due to the fact rock is fractured and the laboratory strength of a specimen represents the best case of a homogenous sample. A strength reduction is necessary for slenderness as inherently, taller structures are weaker than squatter structures.

Empirical formulas are validated by comparing formula-predicted pillar performance against mine case histories. Figure 16.16 is a composite offered by Esterhuizen (2006).

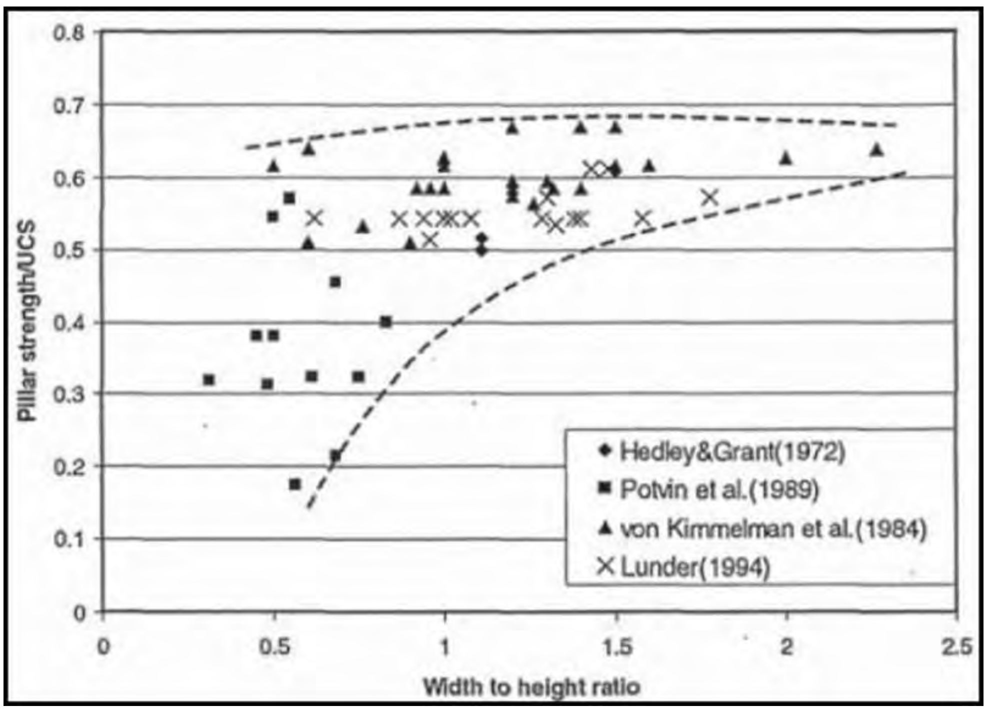


Figure 16.16. Graph Relating Research Projects into Pillar Stability

Figure 16.17, Esterhuizen (2006) compiled three methods of hard rock pillar design. The results allow a method to extract a strength reduction factor from multiple researchers and formula.

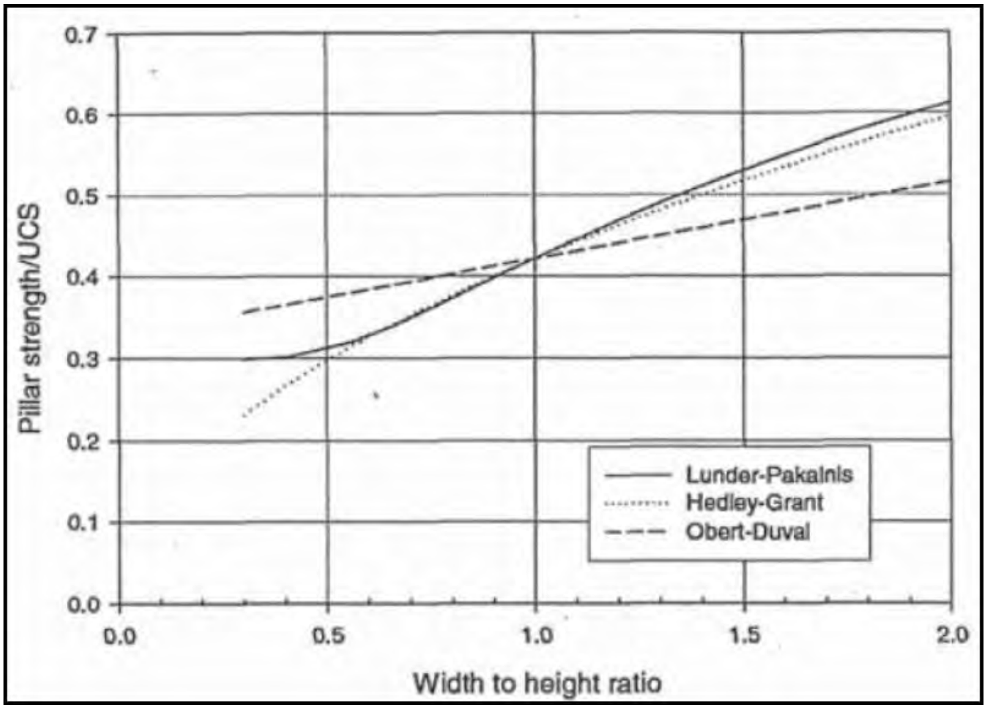


Figure 16.17. Width to Height Ratio to Pillar Strength for Three Empirical Methods

The average of the UCS testing performed in 2011 on Kuriskova / Picnic Tree sample host ore zone is 558 psi.

The determination of the load acting pillars is most commonly done using the concepts of extraction ratio and tributary area load. The tributary area load method assumes that each pillar supports the column of rock over the cross-sectional area of the pillar plus a portion of the room equally shared by the neighboring pillars. Tributary (average) pillar stress is defined as the tributary area load acting on the pillar's area. Pillar analysis for Kuriskova is listed in Table 16.6.

Table 16.6. Pillar Size Feasibility

Area	Process Plant	Mine Pillar min	Mine Pillar med	Mine Pillar max
(Tonnes)	n/a	1,856	1,856	1,856
($US/Tonne)	40.85	40.85	40.85	40.85
(% Fe)	9.74%	9.74%	9.74%	9.74%
(% Pb)	0.28%	0.28%	0.28%	0.28%
(% Zn)	6.24%	6.24%	6.24%	6.24%
Rock SG	2.75	2.75	2.75	2.75
Width (m)	5.00	5.00	5.00	5.00
Average Mine Ht (m)	5.0	5.0	5.0	5.0
Pillar Height (m)	5.0	5.0	5.0	5.0
Cover to Roof (m)	280	86	381	602
Pillar Width (m)	10.0	5.0	8.0	12.0
Max W:H Ratio	2.0	1.0	1.6	2.4

Area	Process Plant	Mine Pillar min	Mine Pillar med	Mine Pillar max
Pillar/UCS Strength	0.42	0.42	0.55	0.73
UCS Strength (Mpa)	50	50	50	50
Span (m)	5	5	5	5
Pillar Strength (Mpa)	21	21	28	36
Overlying Rock S.G.	2.84	2.84	2.84	2.84
Tributary Stress Applied (Mpa)	12	5	18	24
Calculated Safety Factor	1.7	3.9	1.5	1.5

The method used in calculating the pillar size is to first size the pillar for a minimum width such that the width to height ratio is 0.8, and then to calculate a rock strength that replicates confinement. The final step is to calculate the stability factor (stress capacity divided by stress applied). The analysis reveals the pillars will be stable using the dimensions in the work sheet. The weight of the overburden in the calculations was set to correspond to at 0.025 MPa per meter of depth.

Many researchers have suggested reduction factors for pillar strength; of note were Salmon and Munro. Their work in the 1960s set the stage for modern relationships for relating rock mass, laboratory strength, and in-mine pillar performance. Their work related that using the tributary method of pillar loading is satisfactory as long as the recognition is made that, if one pillar fails, the adjacent pillars immediately accept the additional tributary load. In the case of tributary area failure, more pillars fail in a zipper effect. This condition requires the cuts be placed sufficiently apart for each other. This will be the case for mining at Kuriskova with the cut and fill method.

16.3.13.1 Span Feasibility

The primary analysis utilized for unsupported span feasibility is from Bieniawski (1989). The method utilizes RMR to determine the span (Figure 16.18).

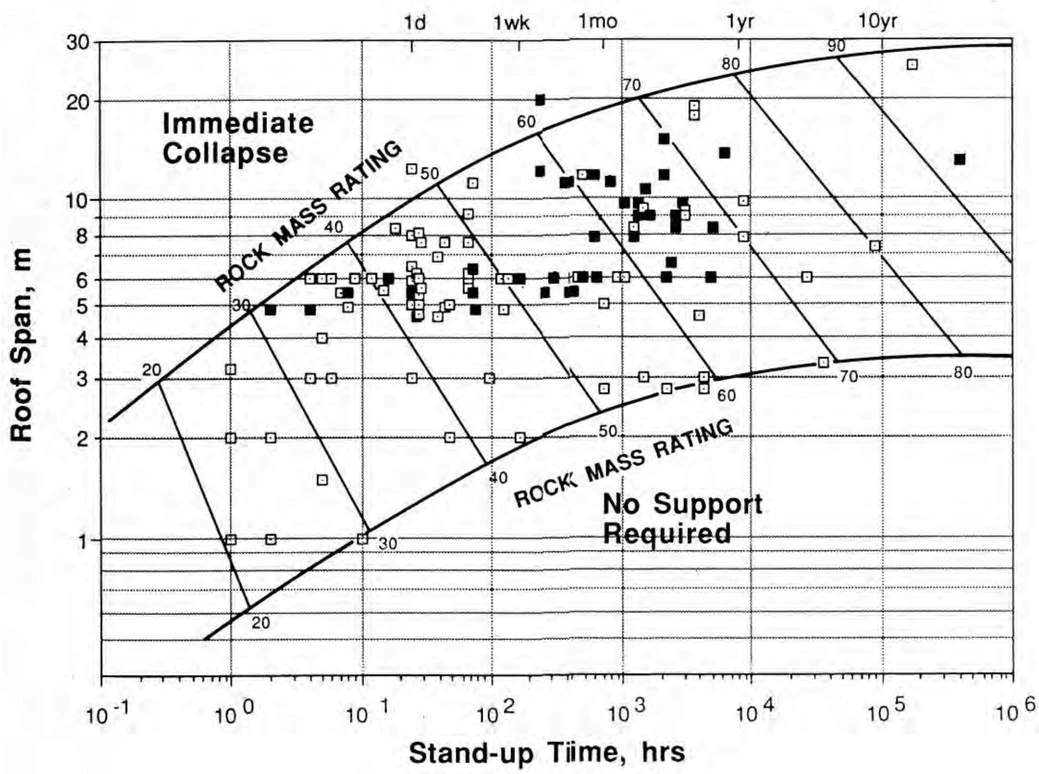


Figure 16.18. RMR Approach to Span

Using a RMR rating of 25 to 40, the unsupported span is in the 2 m range for stability. Because this span is narrower than the intended drift and stope cut width, some dribbling will occur.

Darhnke (2000) offers an approach (Figure 16.19) to check span based on joints per meter. The Kuriskova host was logged for joint spacing as part of the RMR process. Hole LE-L-70 in the center of the mineralization had joint spacing 50 mm or less. Applying this to the Darhnke chart yields a span of 2 m for a beam less than 2 m thick. This supports the conclusion above.

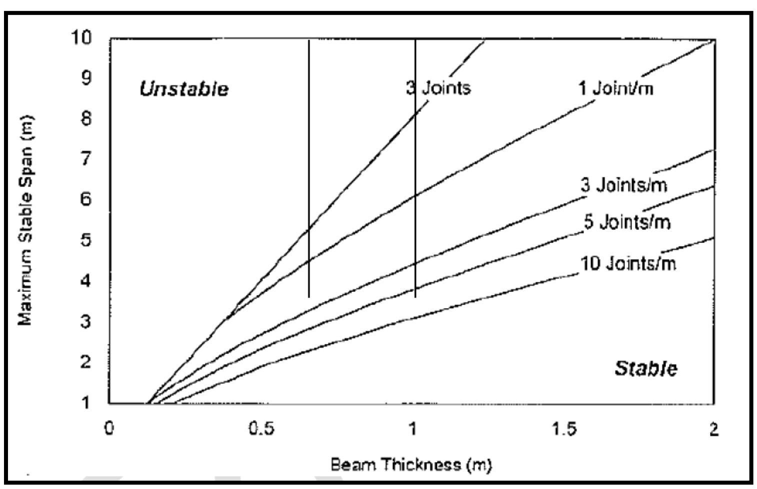


Figure 16.19. Span Approach Using Joints per Meter

Using this input, the stable span for 1.0 m beam is in the approximately 2 to 3 m range.

16.3.14 Comparison of Underground Analysis to Previous Studies

Two scoping studies were performed by Pincock Allen and Holt, and AMEC on the Kuriskova project. In both cases due to the lack of hard geotechnical data, but upon visual inspection of the core, it was noted the fractured ground of the mineralization. Consequently both studies postulated underhand cut and fill would be the correct method.

16.3.15 Paste Backfill Plant

The paste backfill plant will be placed underground adjacent to the processing plant in a 5 m x 25 m x 8 m high room. This placement will minimize the material handling cost as the main feed to the plant is the tailings output from the processing plant. The heart of the plant will be a Putzmeister BSM 1002 (or equivalent) piston pump that will dispatch 50 m^3 per hour of paste though a 20 cm ID pipe to mine stopes. The pump will require 75 kilowatts (kW) of electrical service.

The main system components providing paste backfill to the pump are:

- 9-tonne feeder hopper that feeds the horizontal mixer;
- horizontal mixer to combine the tailings, water, and cement;
- 34,500 L water tank to stage the water before the mixer; and
- 3 m x 10 m x 4 m bin to stage the tailings before the mixer.

The cement tanks to stage the cement before the mixer are discussed below.

The cement requirement for the paste backfill equals 27.2 m^3 per day. The system to mobilize this cement to the underground plant will consist of 26 tonne over-the-road tanks built to be shuttled underground and placed near the premix hopper. Approximately 1.8 tanks of cement will be used per day. The tanks, which are delivered to the surface site by over-the-road semi-truck tractors will be transported underground by a diesel prime mover via the decline to the paste backfill plant.

The capital and operating costs of the plant include the purchase and construction of its components, miners to man the plant, and the material cost and are included in Section 17.0.

16.4 Conclusions and Recommendations

16.4.1 Underground Mining

Tetra Tech has prepared an underground mine plan for the Kuriskova uranium project which included; a mine layout, mine schedule along with the associated operating and capital cost estimates. The project was designed to achieve a production rate of 600 tpd and sustain that rate for a mine life of 12.5 years based on the probable mineable reserves.

From a mine planning perspective it is recommended to examine the factors which contribute to the cost or mine head grade. The use of roadheader mining machine was proposed for this project. Further test work will be needed to identify the specific requirements for the roadheader, including bit spacing, bit wear and bit size, and motor power.

The inclusion of an underground process plant in the mine plan will require more comprehensive geotechnical analysis of the opening to ensure stability.

Special consideration to miner safety must be considered when mining high grade ore. It is advised that a study be conducted to correlate ore grade percent with worker radiation exposure.

16.4.2 Geotechnical

Underground mining at Kuriskova deposit will require pre-operational control due to the weak rock. The rock is weak both in terms of highly fractured and when it tact exhibits poor cementation as evidenced by UCS testing results. The mining method of underhand cut and fill allows for cut dimensions and pre-operational support to be applied as conditions change during the course of mining operations.

It is recommended to perform additional geotechnical drilling and sampling to capture host rock at the shallow, mid, and deep level of production. Also the additional drilling would be done to capture site specific information for shafts and underground process plant design.

Table 16.7 below shows the depth and purpose of the additional recommended drilling and the proposed locations.

Table 16.7. Meters of Geotechnical Drilling: Purpose to Gather Rock Mechanics Data for Mine Design

Hole	Purpose	Orientation, Degrees	TD Along Axis, m
1	Upper ore zone and development	50	410
2	Mid ore zone and development	50	810
3	Lower ore zone and development	50	1,100
4	Main shaft and decline	0	275
5	Underground process plant	0	280
6	Escape shaft and decline	0	160
Total			3,035

16.5 Personnel

Pre-production development for the Kuriskova mine will be carried out by the mine owner with consultant supplied construction management team. All stope production at the Kuriskova underground mine will be owner operated. Engineering and technical support for the project will be completed by mine owner personnel.

16.5.1 Hourly Personnel

The mine will operate twenty four hours a day seven days per week. Three 8 hour shifts per day was used as the staffing basis. The basis for the manpower estimate was done from the amount of manpower required to operate the mine for a shift. This included all mobile equipment, utility work, and maintenance. Table 16.8 displays the hourly manpower required per shift at each mine.

Table 16.8. Underground Mine Hourly Labor Schedule

Position	Yr -3	Yr -2	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13
Mine Superintendent	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine Maintenance Superintendent	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Construction Foreman	3	3	3	0	0	0	0	0	0	0	0	0	0	0	0	0
Mine Maintenance Foreman	1	3	3	4	4	4	4	4	4	4	4	4	4	4	4	4
Mine Shift Foremen	2	3	3	4	4	4	4	4	4	4	4	4	4	4	4	4
Drill and Blast Foreman	1	1	3	4	4	4	4	4	4	4	4	4	4	4	4	4
Surveyor	2	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Surveyor Helper	2	3	3	6	6	6	6	6	6	6	6	6	6	6	6	6
Jumbo Operator	1	2	2	2	1	1	1	1	1	1	1	1	1	1	1	1
Jumbo Offsider	1	2	2	2	1	1	1	1	1	1	1	1	1	1	1	1
Roadheader Operator	0	0	0	4	4	4	4	4	4	4	4	4	4	4	4	1
LHD Operator	1	1	2	4	4	4	4	4	4	4	4	4	4	4	4	1
General Laborer	6	9	11	17	14	14	14	15	14	14	14	14	14	13	14	7
Backfill Crew	0	0	0	3	3	3	3	3	3	3	3	3	3	3	3	3
Truck Driver	2	6	9	14	11	11	11	10	10	10	11	11	11	9	11	4
Blasting Crew	1	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1
Bolter	1	1	2	3	3	3	3	3	3	3	3	3	3	3	3	1
Grader	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Diamond Driller	2	2	2	3	3	3	3	3	3	3	3	3	3	3	3	2
Utility Vehicle Operator/Nipper	1	2	2	3	3	3	3	3	3	3	3	3	3	3	3	2
Mechanic	1	1	2	4	3	3	3	3	3	3	3	3	3	3	3	1
Electrician	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	3
Welder	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Tireman	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Surface Haul Truck Driver	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Surface Haul Loader	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Maintenance Worker	1	1	1	5	5	5	5	5	5	5	5	5	5	5	5	1
Hourly Sub-Total	40	56	66	98	89	89	89	89	88	88	89	89	89	85	85	57

16.5.2 Salary Personnel

With the exception of initial construction management the mine owner will be responsible for providing all technical services for the mine. This includes engineering surveyors, and management. The basis of manpower for salaried staff was completed for each project on an annual schedule. Table 16.9 lists the salaried personnel.

Table 16.9. Underground Salary Labor

Position	Yr -3	Yr -2	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13
Mine Geologist	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Long Range Planner	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine Planner	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Drill and Blast Foreman	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Chief Geologist	0	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine Maintenance Foreman	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Shop Foreman (Day Shift Only)	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine Maintenance	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Backfilling	0	0	2	2	2	2	2	2	2	2	2	2	2	2	2	0
Warehouse Security	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Nurse/EMT	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine Trainers	0	1	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Salaried Sub-Total	7	10	19	19	19	19	19	19	19	19	19	19	19	19	19	14

16.6 Pre-Production Development

The majority of Kuriskova underground mine pre-production development is planned to be carried out by the mine owner. Underground mine pre-production work to be completed includes driving the decline from the surface to the deposit, raise boring two shafts, development of the underground chamber that will house the process plant, and underground process plant rooms and associated infrastructure facilities. All development is expected to be done using mechanized mining equipment. Drift driving and chamber development will be done using the drill, blast, load, haul cycle. All raise development will be carried out by contractors.

An advance rate of 6 m per day was chosen for the decline and development drifts. In order to achieve this rate in the decline (single heading) a three boom jumbo drill and a three boom rock bolting machine are required. The three-boomed fleet will continue to operate during process plant chamber construction. Once a second heading is available a second development fleet will be required. The second development fleet will consist of a 2 boom jumbo and a 2 boom bolter. Haul trucks will be assigned as dictated by the mine rock haul distances. All non-mineralized mine rock from pre-production development will be hauled to the surface where it will be crushed and screened.

Process plant chamber construction will be done from two headings that will converge in the middle of the chamber. Aside from the chamber that will house the process plant several rooms for reagent storage, electrical components, a control room, a truck bypass lane, and 45 days of tailings storage will need to be excavated during the pre-production development. Raise boring from underground drifts will be required for an ore storage bin and the vertically orientated pachuca tank. It is anticipated that raise bore openings will need to be supported by a concrete liner.

An egress shaft and exhaust ventilation shaft will be raise bored from the surface. Work on these installments is schedule to begin once the underground working reached their locations. Pre-production development is planned to last for three years. Table 16.10 displays the pre-production development lengths and process plant chamber volumes. Figure 16.20 displays a 3D general arrangement of the development required for process plant pre-production.

Table 16.10. Pre-Production Development

Development	Year -3	Year -2	Year -1
Decline (m)	464	2,058	0
Tailings Storage (m)	0	0	303
Drift Development – Waste (m)	0	1,078	4,631
Total Development (m)	464	3137	4934
Raise Bore	Year -3	Year -2	Year -1
Egress Shaft (m x 4 m dia10)	153	0	0
Exhaust ventilation shaft (m x 4 m dia)	274	0	0
Ore storage/Pachuca Tank (m x 4 m dia)	0	0	80
Process plant ventilation (m x 4 m dia)	0	0	40
Mine ventilation (m x 3 m dia)	0	0	322
Total Raise Bore (m)	427	0	442
Process Plant Area	Year -3	Year -2	Year -1
Process plant Excavation (m^3)	0	38,436	0
Truck Bypass (m^3)	0	10,469	0
Side Excavations (m^3)	0	10,123	0
Electrical Room 1 (m^3)	0	1,950	0
Electrical Room 2 (m^3)	0	1,200	0
Explosive Magazine (m^3)	0	1,200	0
Total Process Plant Area (m^3)	0	63,378	0

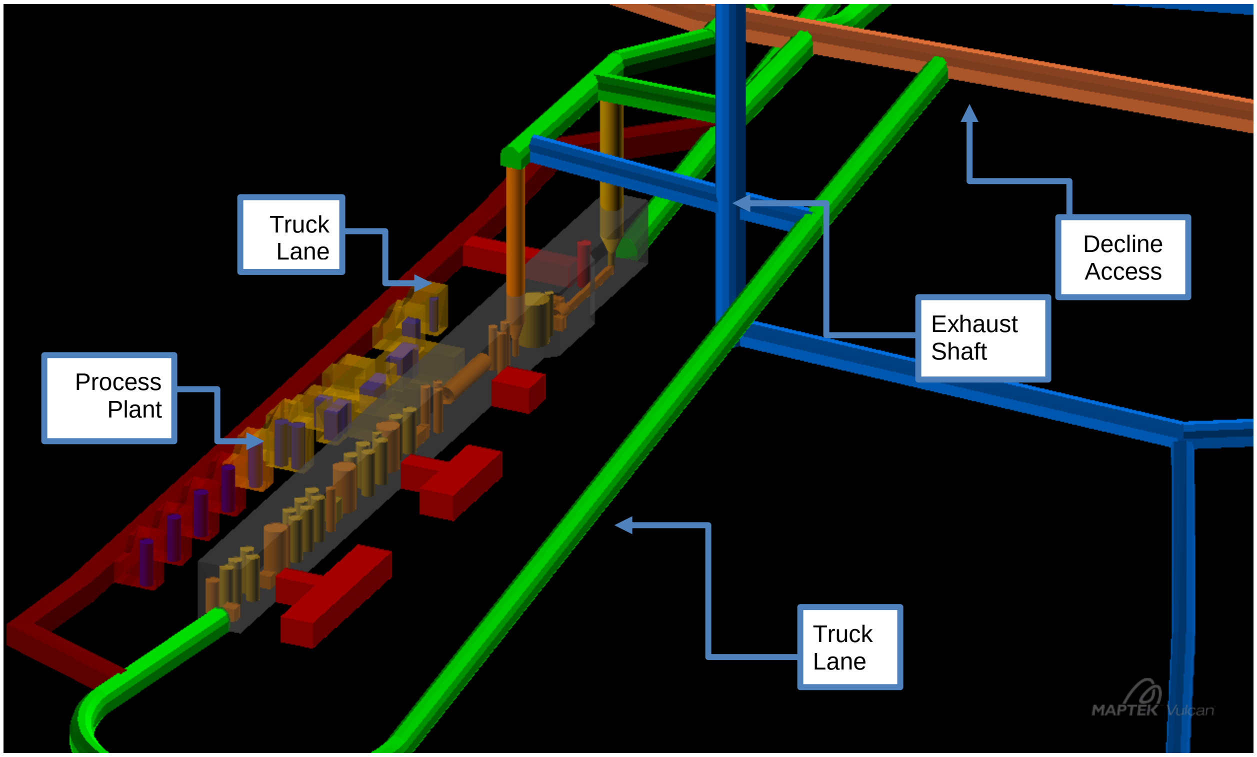


Figure 16.20. Process Plant Pre-Production Development Looking Southeast

16.7 Production Development

After the first year of stope production, drift development will be advanced far enough to allow for the operation of only one development fleet. Raise boring for ventilation purposes will be required throughout Year 3 of the mine life. All waste rock from development was planned to be hauled to the surface where it will be crushed and screened. Table 16.11 lists the development required on an annual basis.

Table 16.11. Production Development

Development	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13
Tailings Storage (m)	688	815	810	816	814	812	808	809	811	807	814	812	0
Drift Development - Waste (m)	4,966	2,261	2,252	2,225	2,362	2,161	2,220	2,280	2,239	2,239	1,284	2,017	1,083
Total Development (m)	5,654	3,075	3,062	3,041	3,176	2,973	3,028	3,088	3,050	3,047	2,098	2,828	1,083
Raise Bore	Yr 1	Yr 2	Yr 3	Yr 4	YR 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13
Ventilation Raises (m x 3m dia)	239	213	79	0	0	0	0	0	0	0	0	0	0

16.8 Mining Trade-off Studies

Three trade-off studies were completed as part of the prefeasibility process. They included:

- The comparison and decision of alternative mine access options;
- Optimize the underground haulage truck size; and
- Compare main mine access using a decline versus a shaft.

The results of the trade-off studies were used in the technical and cost analysis of the PFS.

17.0 RECOVERY METHODS

17.1 Underground Processing Facility

The Kuriskova UPF has been designed to process 600 tpd uranium bearing ores. Results from metallurgical testwork performed on select ore grade samples from the Kuriskova deposit indicate that a carbonate leach using pressure oxidation is the best alternative available for uranium and molybdenum extraction. The overall process plant and units operations therein are designed to produce a uranium peroxide yellowcake as well as a molybdenum sulfide concentrate.

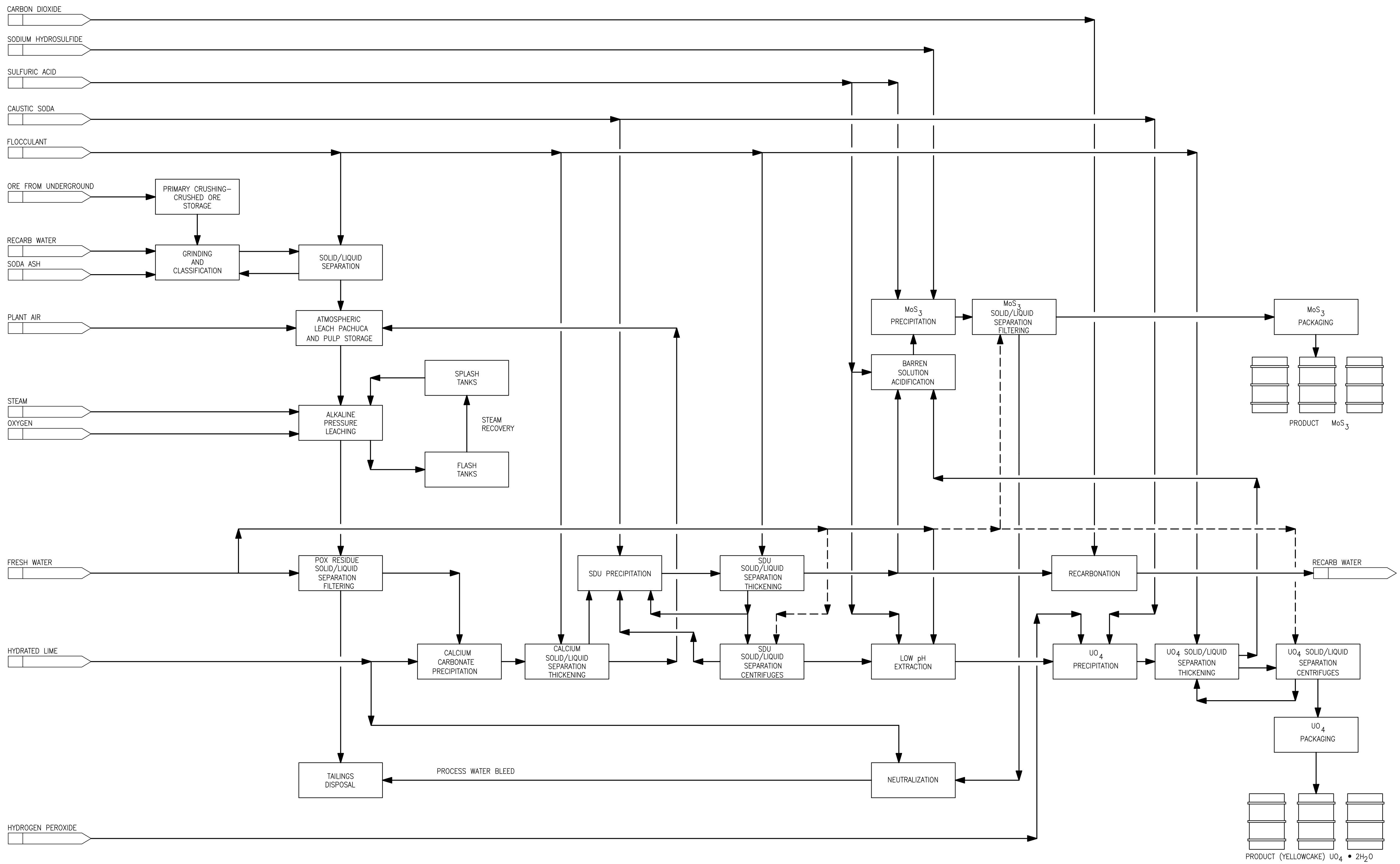
The UPF consists of a single stage crushing, the product from which is directed to a crushed ore storage bin. Ore will be drawn from this storage bin into a single stage ball mill grinding circuit. Ground ore will be directed to a conditioning tank which will provide surge capacity ahead of the leach circuit. Slurry from the conditioning tank will be pumped under pressure to the POX circuit. Uranium and molybdenum constituents are extracted using carbonate leach chemistry in conjunction with POX carbonate leaching to increase leach rate. Pregnant solution is separated from the leach residue and precipitated as SDU and is further purified by re-leaching and precipitation of a uranium peroxide as yellowcake. Molybdenum sulfide is precipitated from a bleed stream of the process solution.

All of the tailings generated during the mine life are disposed of as 100 percent paste backfill to the mine or in underground excavations of inert rock. This is the preferred method of tailings disposal because it minimizes the amount of radio-nuclide bearing material that is transported to the surface.

The process plant will consist of the following unit operations and facilities:

- Primary Crushing and Ore Storage
- Grinding, Classification, and Thickening
- Pressure Oxidation and Leaching
- High pH SDU Precipitation
- SDU Releach and Low pH Uranium Precipitation
- Molybdenum Precipitation
- Paste Backfill
- Reagent Storage and Handling
- Water Supply and Distribution
- Assay and Metallurgical Laboratory

A block flow diagram is presented in Figure 17.1.



NO.	DATE	DESCRIPTION	PREPARED	REVIEW	DESIGN	AUTHORIZE
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CLIENT: EUROPEAN URANIUM RESOURCES LTD.

TETRA TECH

PROJECT NAME: KURISKOVA URANIUM PROJECT

DRAWING DESCRIPTION: FIGURE 17.1 UNDERGROUND PROCESS FACILITY BLOCK FLOW DIAGRAM

DRAWING NO: 114310990-DWG-P0100-C(2) — REV: C

3D MODEL REF No:

17.2 Major Design Criteria

The process design criteria were derived using results from metallurgical testwork in conjunction with expert opinion and additional information regarding the equipment components as provided by the various vendors. The UPF is designed to process 600 tpd, equivalent to 219,000 tonnes per year (tpy). The major design criteria used in the design are outlined in Table 17.1, below.

Table 17.1. Major Design Criteria

Criteria	Unit	
Operating Year	d	336
Plant Availability	%	92
Process Throughput	tpd	600
Process Throughput	tpy	219,000
Bond Ball Mill Work Index	kWh/t	13.2
Autoclave Retention Time	hrs	2
Autoclave Temperature	^{o}C	200
Uranium Extraction	%	94
Molybdenum Extraction	%	87
Sulfur Oxidation	%	100
Overall Uranium Recovery	%	92
Overall Molybdenum Recovery	%	86.8

17.3 Operating Schedule and Availability

The UPF is designed to operate on the basis of three, eight-hour shifts per day, 345 days per year. Process operations crews will work in conjunction with mine operations crews to man-trip to and from the surface facilities so as to efficiently move the UMF labor force in and out of the underground area. Process crews will consistently “hot change” between crews to ensure the process plant has adequate supervision and staffing 24 hours a day.

Overall UPF availability is expected to be 92 percent or 335.8 operating days per year. This will allow sufficient downtime for scheduled and unscheduled maintenance process plant equipment.

Major scheduled maintenance annually accounts for 20 days of downtime for maintenance of the mill and autoclave. The remaining 9.2 operating days per year reflect a combination of minor scheduled maintenance and unscheduled maintenance.

17.4 Process Plant Description

17.4.1 Primary Crushing and Ore Storage

Primary crushing of ore occurs on the crushing level 40 m above the main process plant excavation. A cement-lined raise-bore excavation will be used as a coarse ore bin. Mined ore is fed through a bar grizzly to remove any material greater than 200 mm. Free of boulders, the ore falls onto an apron feeder directly feeding a horizontal shaft impact crusher. Ore draw from the apron feeder into the crusher is regulated in such a way so as to maintain a half meter dead-bed of ore on the pan at all times thereby preventing the ore falling through the bar grizzly from hammering the pans. Ore is removed from the apron feeder pocket and crushed to a nominal P_{80} of 12 mm before being discharged directly into the coarse ore bin.

Major equipment for the Crushing area includes:

- ROM oversize grizzly
- Apron feeder
- Horizontal shaft impact crusher
- Crushed ore bin

17.4.2 Grinding, Classification, and Thickening

The grinding, classification, and thickening circuit consists of a single ball mill in closed circuit with vibrating screens. Ore reclaimed from the coarse ore bin is fed into the ball mill using a single mill feed conveyor. Carbonate in the form of pulverized soda ash is added to the ore on the belt conveyor prior to the ball mill allowing carbonate leaching to commence therein. The ball mill operates with a 300 percent circulating load at 75 percent of critical speed.

The ball mill discharges to the vibrating screen pumpbox from which the slurry is pumped to a bank of screens for classification. Grind size classification size is designed at a P_{80} of 75 µm. Oversize material from the vibrating screens is returned to the ball mill feed for additional grinding.

Undersize material from the vibrating screens discharge into the dewatering cyclone feed pumpbox at a slurry density of approximately 20 percent and is dewatered using hydrocyclones working in conjunction with a high rate thickener. Approximately 90 percent of the water along with approximately one-third of the solids reports to cyclone overflow. This slurry is directed to the cyclone overflow clarifier to remove the rest of the solids; the clarified water being recycled back to the grinding circuit. Solids from the clarifier are combined with cyclone underflow producing slurry with a target density of 40 percent solids to be fed to the leach circuit.

The rational for using cyclones ahead of the grind thickener was to reduce the size of the thickener. The use of dewatering cyclones before the grind thickener reduced the required thickener diameter from 15 m to 8 m which is substantially more acceptable given the underground confines within which the mill is positioned.

Major equipment for the Grinding and Classification area includes:

- Crushed ore apron feeder
- Ball mill feed conveyor
- Belt weightometer
- Ball mill: 3 m diameter x 4 m long, 375 kW
- Vibrating screen pumps and pumpbox
- Vibrating screens
- Dewatering cyclone pumps and pumpbox
- Dewatering cyclones
- Cyclone overflow clarifier

- Cyclone overflow pump
- Cyclone underflow pumps
- Grind thickener

POX conditioning feed pumps and pumpbox

17.4.3 Pressure Oxidation and Leaching

Milled slurry is pumped from the grind thickener into the POX conditioning tank, the feed end to which is located on the crusher level. This tank is a steel lined raise-bore excavation which acts as a storage tank. It provides 16 hours of residence and conditioning time between the comminution and leach circuits.

Slurry is pumped from the conditioning tank through a series of three splash tanks. From the splash tanks, the slurry is directed into a pressure leach autoclave where the ore is oxidized and leached. A leach solution concentration of 69 g/L Na_2CO_3 and 23 g/L $NaHCO_3$ is sufficient to obtain the extractions quoted; however, process optimization testwork may reduce these concentration requirements. The autoclave has a six chamber design and operates at 200^{o}C and 2.8 MPa with an oxygen overpressure of 0.7 MPa. The autoclave discharges to a series of three flash tanks before being pumped to the leached slurry belt filter to separate leach residue from the pregnant solution.

The flash and splash tanks are designed to recover heat from the autoclave discharge and pre-heat the slurry entering the autoclave, respectively. Splash tanks utilize steam from their corresponding flash tanks to heat slurry before the autoclave. Each set of splash and flash tanks operates at different pressures to provide different ranges of steam transfer thereby allowing increased overall heat recovery. The low temperature splash/flash tanks operate in a vacuum, 25 kPa absolute pressure, to provide a steam temperature of 60°C. The intermediate temperature tanks operate under atmospheric pressure, 101 kPa absolute pressure, to provide 100°C steam. The high temperature tanks operate under a pressure of 543 kPa and provide a temperature of 155°C. The final pre-heated temperature of the slurry before the autoclave is estimated at 140°C requiring an additional 60°C of heating in the autoclave proper.

Additional heat energy for the autoclave is supplied by direct steam injection, supplied by electrical steam boilers, to maintain an operating temperature of 200°C. Oxygen, being the main process oxidant, is supplied through a pipe running from the surface facilities along the main access drift. In addition to the oxidation of uranium (IV) to uranium (VI) for leaching, molybdenite (MoS_2) and pyrite (FeS_2) are the primary oxygen and reagent consumers in the process. These exothermic oxidation reactions add heat to the circuit that would need to be provided otherwise by steam.

Leached slurry is pumped from the final flash tank to a horizontal belt filter to separate the leachate solution from the leached tailings. The belt filter is equipped with a counter current wash system to increase recovery of leachate solution. Filtered leachate solution is pumped to the SDU Precipitation Circuit.

Major equipment in the POX and leach circuit area includes:

- POX conditioning tank
- Pressure leach feed pumps
- Splash Tanks

- Pressure Leach Autoclave
- Flash Tanks
- Electrical steam boiler
- Leach slurry feed pumps
- Leach slurry belt filter
- Pregnant solution discharge pumps and pumpbox

17.4.4 High pH SDU Precipitation

Pregnant solution pumped from the leach circuit reports to bicarbonate trim tanks where finely powdered hydrated lime is metered into the stream to react with excess bicarbonate to produce calcium carbonate before SDU precipitation. Trimming the bicarbonate with lime prior to SDU precipitation reduces caustic soda consumption during precipitation. Precipitated calcium carbonate is separated from the leachate solution in the bicarbonate trim thickener. Underflow is directed back to the leach circuit where it is disposed of in tailings with the filtered leach residue.

Overflow from the bicarbonate trim thickener is pumped to a series of SDU precipitation tanks where caustic soda is added to precipitate the SDU. SDU precipitate is separated from the solution in the SDU thickener and further dewatering with a decant centrifuge. A small stream of precipitate is recycled back to the first SDU precipitation tank to provide a nucleation source for the precipitation reaction.

Barren solution from the circuit is sent to the process water system where it is recarbonated with CO_2 to convert carbonate to bicarbonate. A bleed stream from this flow is sent to the Molybdenum Precipitation Circuit to prevent buildup of molybdenum ions in the process solution. SDU precipitate is sent to the Low pH Precipitation circuit.

Major equipment for the SDU precipitation area includes:

- Bicarbonate trim tanks
- Bicarbonate trim circuit pumps
- Bicarbonate trim clarifier
- Bicarbonate trim clarifier underflow pumps
- Bicarbonate trim clarifier overflow pumps
- SDU precipitation tanks
- SDU clarifier
- SDU clarifier underflow pumps
- Barren solution pumps
- SDU residue pumps
- SDU centrifuge

17.4.5 Low pH Uranium Precipitation

SDU precipitate must undergo additional purification to reduce concentrate impurities prior to shipping. This is achieved by re-leaching the precipitate with pH adjustments and precipitating the uranium in a series of tanks with hydrogen peroxide. Precipitated Uranium Peroxide is separated from the solution stream in the low pH thickener and further dewatered using a decant centrifuge.

Final dewatered yellowcake precipitate is packaged in barrels for shipping. Barrels loaded for transport are sent to the surface for short term secure storage before being sent out as a final product.

The barren solution stream from the circuit is sent to the Molybdenum Precipitation Circuit. The stream is combined with the bleed stream from the SDU Precipitation Circuit and is ultimately mixed with tailings for disposal in paste backfill.

Major equipment for the Low pH Precipitation area includes:

- Low pH re-leach and precipitation tanks
- Low pH precipitation pumps
- Low pH precipitation thickener
- Low pH precipitation overflow pumps
- Uranium peroxide pumps
- Uranium peroxide centrifuge

17.4.6 Molybdenum Precipitation

The Molybdenum Precipitation Circuit combines the bleed stream from the SDU Precipitation Circuit and the barren solution from the Low pH Precipitation Circuit. This combined feed solution is adjusted to an approximate pH of 6.0 before adding sodium hydrosulfide. The solution is allowed to equilibrate for a period of one hour before additional pH adjustment is done to rapidly adjust the solution to a pH between 2.0 and 3.0. The rapid pH adjustment of the solution precipitates the molybdenum as a molybdenum sulfide.

The precipitate laden solution is then pumped to a pressure filter for dewatering and production of a final cake that is packaged into drums for shipping. Barrels loaded for transport are sent to the surface for short term storage before being sent out as a final product.

Major equipment for the Molybdenum Precipitation area includes:

- Steady acidification tank
- Molybdenum precipitation tanks
- Molybdenum precipitation pumps
- Molybdenum filter press

17.4.7 Paste Backfill

Leached tailings solution from the horizontal belt filter will report to the paste backfill plant where it will be mixed with cement to produce paste used in backfill of the mine stopes.

17.4.8 Reagent Handling and Storage

Various chemical reagents are added to the process streams to facilitate the extraction and recovery from the uranium and molybdenum minerals from the mined ore. Solid and liquid reagents are stored underground in individual excavations separate from the main plant excavation. These are accessed by a single truck lane running parallel to the main process plant excavation. Reagents are pumped to their respective areas in the UPF by reagent metering pumps to control reagent dosage.

Equipment required for the preparation of the various reagents includes:

- Bulk storage bins
- Reagent mixing systems
- Mix tanks
- Storage tanks
- Reagent metering pumps

The primary reagents and chemicals to be used in the process plant include: soda ash, oxygen gas and carbon dioxide gas for alkaline leaching. Other miscellaneous reagents and chemicals will be used for filtration, uranium, and molybdenum precipitation.

Grinding media is added to the ball mill as required. The estimated consumption rate for grinding media is 1.0 kg/t.

The Reagent Handling and Storage area incorporates a containment design to accommodate 110 percent of the largest tank volume.

17.4.9 Water Supply and Distribution

Water for the process is supplied from underground mine dewatering efforts. Excess mine water not needed for the process is pumped to the surface and processed in a WTP.

Fresh water and process water are stored in separate tanks for use in the UPF. All process water makeup goes through the fresh water system, where the majority is used as wash water for the leach slurry belt filter. Additional fresh water is used as wash water for the uranium and molybdenum final products as well as for reagent preparation.

The process water system supplies recycled water throughout the process as needed, the majority of which is recarbonated with carbon dioxide and fed into the grinding circuit. Process water is recycled to the greatest extent possible so as to reduce reagent consumption. Ionic concentrations are maintained by bleeding portions of the process water to paste backfill through the Molybdenum Precipitation Circuit.

17.4.10 Assay and Metallurgical Laboratory

Several of the operator stations in the UPF are equipped to perform various operational tests and analysis on a real time basis. Marcy buckets, scales and select screens will be available in the UPF to provide operators the ability to determine grind sizes and slurry densities. Titration equipment and pH meters will be available for the operators to perform quick pH and carbonate / bicarbonate analysis.

The formal assay laboratory located with the other surface facilities will be equipped with the necessary analytical equipment and instruments to provide all routine assays for the mine and process with additional preparation capabilities for environmental monitoring.

Major equipment for the laboratory includes:

- Atomic absorption spectrometer
- All lab ware and glassware associated with the tests performed
- Lab crusher and sample bucking equipment

The assay laboratory is located at the surface facilities and is equipped to prepare and process all mine and process metallurgical and assay samples. The lab has capabilities to prepare environmental samples prior to testing at third party laboratories.

17.4.11 Single Process Facility Process Manpower

Process plant salaried personnel estimates were developed to provide adequate supervision and technical support for the daily operation of the process plant. Required salaried personnel for the plant are estimated at 11 persons as detailed in Table 17.2. Salaried personnel will supervise 49 hourly employees as detailed Table 17.3. Process positions, both salaried and hourly, that require 24 hour per day coverage will be staffed by rotating eight-hour shifts.

Table 17.2. Process Plant Salaried Manpower

Description	Manpower
Process Plant Operations Salaried Manpower	
Process Plant Superintendent	1
Senior/Chief Metallurgist	1
Metallurgist	1
Operations General Supervisor	1
Shift Operations Supervisor	4
Process Plant Maintenance Salaried Manpower	
Maintenance General Supervisor	1
Maintenance Supervisor	1
Electrical Supervisor	1
Total Salaried Staff at Process Plant	11

Table 17.3. Process Plant Hourly Manpower

Description	Manpower
Process Plant Operations Hourly Manpower	
Crush/Grind Operator	4
Autoclave/Filtration Operator	4
Precipitation Operator	4
Product Packaging Operator	4
Reagent Operator	4
Operation Helpers	8
Day Laborers	4
Mill Clerk	1
Process Plant Maintenance Hourly Manpower	
Repairman 1st Class	4
Repairman 2nd Class	4
Electrician	5
Instrument Technician	2
Maintenance Planner/Logistics	1
Total Hourly Staff at Process Plant	49

17.4.12 UPF Process Plant Control

17.4.12.1 Plant Control

The type of plant control system will be a PLC based control system that will provide equipment interlocking, process monitoring, control functions, and supervisory control. The control system can will generate production reports and provide real time data and malfunction analysis as well as a log of all process upsets. All process alarms and events will be also logged by the PLC.

Operator interface to the PLC will be via programmable computer (PC) based operator workstations located in the process control room:

The plant control rooms will be staffed by trained personnel 24 hours per day.

Operator workstations will be capable of monitoring the entire plant site process operations, and be capable of viewing alarms and controlling equipment within the plant. An engineering workstation will be provided in the surface facility substation control room.

Where applicable, field instruments will be microprocessor-based "smart" type devices. Instruments will be grouped by process area, and wired to each respective area local field instrument junction boxes. Signal trunk cables will connect the field instrument junction boxes to PLC input/output (I/O) cabinets.

Intelligent-type MCCs will be located in electrical rooms throughout the plant. A serial interface to the PLC will facilitate the MCC's remote operation and monitoring.

17.4.13 Control Philosophy

17.4.13.1 Primary Crushing Control System

The control objective of the primary crushing area will be to provide a crushed product to the coarse ore bin prior to the grinding circuit.

Control and monitoring of the primary crusher will occur at the main process controls room. The control objective of the coarse ore storage bin and reclaim will be to provide a crushed ore delivery buffer and a consistent ball mill feed.

17.4.13.2 Processing Control Systems

All control and monitoring functions for the processes and ancillary operations will be controlled from the PC workstation installed in the main process control room:

The PC workstation will control and monitor the following:

- Ball mill conveyor (zero speed switches, side travel switches, emergency pull cords and plugged chute detection)
- Grinding mill (bearing temperatures, lubrication systems, clutches, motors, and feed rates)
- Particle size monitors (for grinding optimization)
- Pumpboxes, tanks, and bin levels
- Variable speed pumps
- Cyclone feed density controls
- Thickeners (drives, slurry interface levels, underflow density, and flocculant addition)

- Autoclave leaching and steam recovery (temperatures, pressures, oxygen and steam airflow rates, and agitator system)
- Leach slurry belt filters
- Tank levels and agitators
- Centrifuges
- Reagent handling, storage level and distribution systems
- Paste backfill system
- Water storage and distribution, including tank level automatic control
- Vendors' instrumentation packages

An automatic sampling system will collect samples from various product streams for on-line analysis and daily metallurgical balance.

Particle size-based computer control systems will be used to maintain the optimum grind sizes for the primary grinding and concentrate regrinding circuits. The particle-size analyzers described earlier will provide main inputs to the control system.

17.4.13.3 Remote Monitoring

Closed circuit television (CCTV) cameras will be installed at various locations throughout the plant to provide visual surveillance of the operations for both production and safety. The cameras will be monitored from the plant control room as well as the surface facilities.

18.0 PROJECT INFRASTRUCTURE

This section provides an overview of the Project infrastructure for the Project. It covers the surface buildings and facilities, power supply and distribution and WTP. In general, the Project will have an underground mining operation, an underground process plant and process plant and surface facilities to support the operation.

18.1 Surface Facilities

The surface facilities include the administration building, warehouse and all infrastructures required for operations and maintenance. Two additional sites on the surface include the exhaust ventilation shaft to include egress man safety hoist and a second egress shaft and man safety hoist.

At the surface facility entrance a staging area for large trucks is provided for inspection and offloading, as required. All inbound and outbound traffic will register at the security building. The security building also has facilities for site specific safety training, first aid station and the industrial hygiene center. The employee parking lot will be adjacent to the security building. All employees and visitors will also register at this building prior to entering into the site. The facility will have a perimeter security fence with monitoring security cameras.

Four main structures will be located within the fenced area:

- Administrative Building which will also include the mine dry, sample preparation, assay and environmental laboratories.
- Warehouse.
- Truck shop with five truck bays and a 10-ton overhead crane.
- Portal-mine entrance to the decline access.

The surface facility will have a fire protection system to include a yard fire loop and fire hydrants. The security building will be equipped with a remote Fire Alarm Control (FAC) panel interlocked with the local FACs in the other buildings. The Administration building complex and warehouse will be equipped with wet pipe automatic fire protection sprinkler systems.

Other structures to include:

- A covered roof structure to park three man-trips and four boss buggies.
- An undercarriage washer and truck scale.
- Reagent storage.
- Molybdenum and uranium concentrate/products storage.
- Miscellaneous mining supplies storage.
- Potable water and fire water tanks and associated pumping systems.
- A new substation, generator sets for emergency service.
- A WTP.
- A sewage treatment plant.
- A storm water retention pond.

In addition, a 60-m diameter domed enclosure will house a portable rock crusher. All development rock will be brought to the surface, crushed, and used for local area road improvement projects. Figure 18.1 provides details of the surface facility layout.

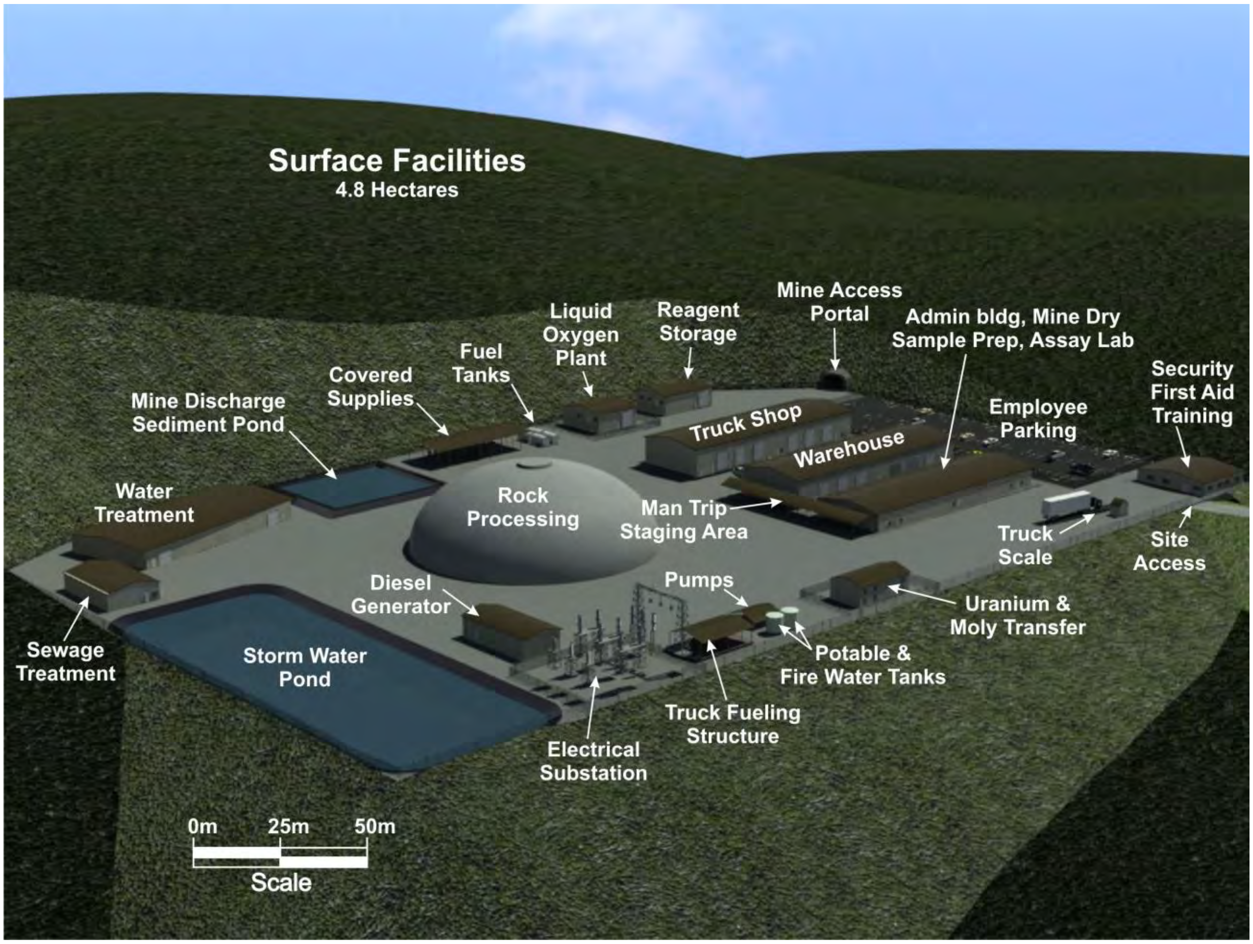

Figure 18.1. Surface Facility Layout

18.1.1 Mine Exhaust Shaft Site

The mine exhaust shaft will be 4 m in diameter and have a perimeter barbed wire fence and gated access road. Facilities include an escape hoist pad with a boom on a swivel and a three man bullet type escape pod. The hoist will be diesel driven. A first aid and firefighting shed and potable water tank will also be located within the fenced area. In addition, an emergency generator will be provided with a bore hole that will provide power to the ventilation fans located underground and other critical process plant equipment

18.1.2 Mine Egress Shaft Site

The egress shaft site will be 4 m in diameter with a perimeter barbed wire fence and a gated access road. Facilities include an escape hoist pad with a boom on a swivel and a three man bullet type escape pod. The hoist will be diesel driven. A first aid and firefighting shed and potable water tank will also be located within the fenced area.

18.2 Power Supply, Distribution, and Control System

The power to the mine site will be provided from a new substation. The power feed will include a 20 MVA, 110-23kV transformer and switchgear for mine and surface area power distribution. Two 23 kV feeders will be routed into the mine via the mine decline access to supply underground substations that provide power to the underground process equipment. In addition, 550 KVA generator sets will be installed for emergency service if electrical power into mine site is disrupted. A new transmission line (approximately 5 km) will be routed to a new mine site substation. This will serve to distribute power to the above ground and below ground facilities.

The control system for the facilities and equipment will comprise of a Programmable Logic Controller (PLC) located in the mine site substation. An Operator Interface Terminal (OIT) locate in the substation is the operator interface to the system. In addition to the OIT, there will be an Engineering workstation/computer used for programming, advanced control, and configuration changes. A second PLC is used for control of the underground facilities. The two PLCs communicate with each other via a fiber optic network for high speed transmission and reliability. The underground facility PLC will also include a similar OIT for operator interface and control. The various analogue and digital inputs/outputs (I/O) for both the surface and underground systems will have remote I/O (RIO) panels located in electrical rooms. An Uninterruptible Power Supply (UPS) will be provided to provide back-up/emergency power supplies to critical systems.

18.3 Mine Water Treatment Plant

The mine water will be treated with standard packaged reverse osmosis equipment and will have a secured area for an oxygen plant that will transport gaseous oxygen (GOX) to the process facilities underground. Mine water will be processed in a WTP. The treated water will be discharged to a sediment discharge pond. The mine discharge sediment pond will have bird netting to protect wildlife.

19.0 MARKET STUDIES AND CONTRACTS

Fifteen countries depend on nuclear power for at least a quarter of their electricity. France is the leader at roughly 75 percent, followed by Slovakia at over 50 percent; Belgium, Ukraine, Hungary, Armenia, Sweden, Switzerland, Slovenia, Czech Republic, Bulgaria, and South Korea derive over one-third of their power requirements from nuclear generation. Japan, Germany, and Finland obtain more than one-quarter of their needs from nuclear, and the United States (with 104 reactors, the most of any country) gets nearly 20 percent of its total through fissionable material.

Presently there are 65 power reactors being constructed in 14 countries, to provide roughly 62 GWe of additional installed capacity. Notably the principal countries are China, South Korea, and Russia. An uprating of existing plants during the past several years has served to improve efficiency or increase output, and upgrades over time have worked to postpone decommissioning.

19.1 Demand and Supply

Uranium production to feed these units has increased substantially in the past decade. Total production throughout the world in 2003 was 35,200 tonnes; by 2010 this figure had risen to 53,700 tonnes, equating to a 4.3 percent per year compounded increase.

As for natural resources to feed these power plants, an estimated five million tonnes of naturally occurring uranium is believed to be recoverable at the present time. Australia leads with more than one million tonnes (+/- 24 percent of the world's known supply), followed by Kazakhstan with 17 percent, Canada at ten percent, and the United States and South Africa at roughly seven percent each. Canada's resources/reserves are the highest grade, and Australia's average is among the lowest. Production forecasts are approximately 63,600 tonnes of uranium which will be required for 2012, an 18 percent increase from the 2010 total presented in 19.1.

The uranium market declined significantly through the 1980s and 1990s because of the end of the Cold War arms race, as well as a cessation in new construction of nuclear facilities. Disarmament of nuclear-weapons stockpiles added surplus highly-enriched uranium (HEU) to the market. A 1993 agreement between Russia and the United States concerned 500 tonnes of Russian HEU that was to be blended down to 15,000 tonnes of reactor-grade fuel for electrical generation. This supplied an estimated 50 percent of United States fuel needs and resulted in an underinvestment in production capacity (enrichment and mining) for nuclear fuel. This agreement expires in 2013 and likely will not be renewed as Russia has signed several contracts directly with United States utilities instead.

Table 19.1. Summary of Production Methods for 2010

Method	% of production
Conventional Underground	28
Conventional Open Pit	25
In situ Leach	41
By Product	5

Mining methods have changed over time. In 1990, approximately 55 percent of world production derived from underground mines; this proportion declined dramatically to 1999 but with new

Canadian high-grade mines and the Olympic Dam operation in Australia conventional underground mining now claims nearly 33 percent of total output. In situ leach mining has steadily increased, mainly because of Kazakhstan production, and currently accounts for 41 percent of uranium production. 19.1 summarizes production methods for 2010.

Four companies accounted for 59 percent of world uranium production in 2010 (Cameco, Areva, KazAtomProm, and Rio Tinto), and the largest ten mines were responsible for 55 percent of the total. Thus there is a notable concentration of supply reposing within a small number of entities.

19.2 Pricing

Most metals are traded on an international exchange, but uranium is distinct in that a formal marketplace does not exist and trading in this commodity is largely conducted through various contract negotiations. Buyers typically elect to purchase some of their requirements from the spot market in an attempt to gain a more favorable price than may exist in their long-term agreements.

The spot market has become more transparent over the past several years and is increasingly acceptable as a proxy for uranium transaction prices. Historical spot prices for U_3O_8 are available from a number of sources and serve to illustrate at least the general behavior of trading levels and volumes over time. Long-term pricing information suffers from the type of contract entered into between seller and buyer, the start date of deliveries, contract term, quantities involved, reliability of supplier, and a number of other factors (such as confidentiality) that make uniform, consistent comparisons difficult at best, if not impossible.

19.2.1 Historical Pricing

By 2000 the uranium mining industry had made few significant uranium discoveries in a decade and only supplied about half of global demand. A series of events including reductions in available weapons-grade uranium, a fire at Australia's Olympic Dam mine, unexpected flooding in Canada's Cigar Lake mine, and the need for fuel at power plants that extended their licenses, all caused substantial increases in uranium prices over the last few years. Figure 19.1 presents annual and averaged spot prices for U_3O_8 from 1988 through 2011. These latter serve to modify extreme swings in annual price levels and may be a more useful tool for forecasting. An average for 2007 reached US$100/lb U_3O_8 (equivalent to roughly US$118/lb uranium), but has since declined to US$56/lb U_3O_8 for 2011. Figure 19.2 shows the monthly variation over the past five years.

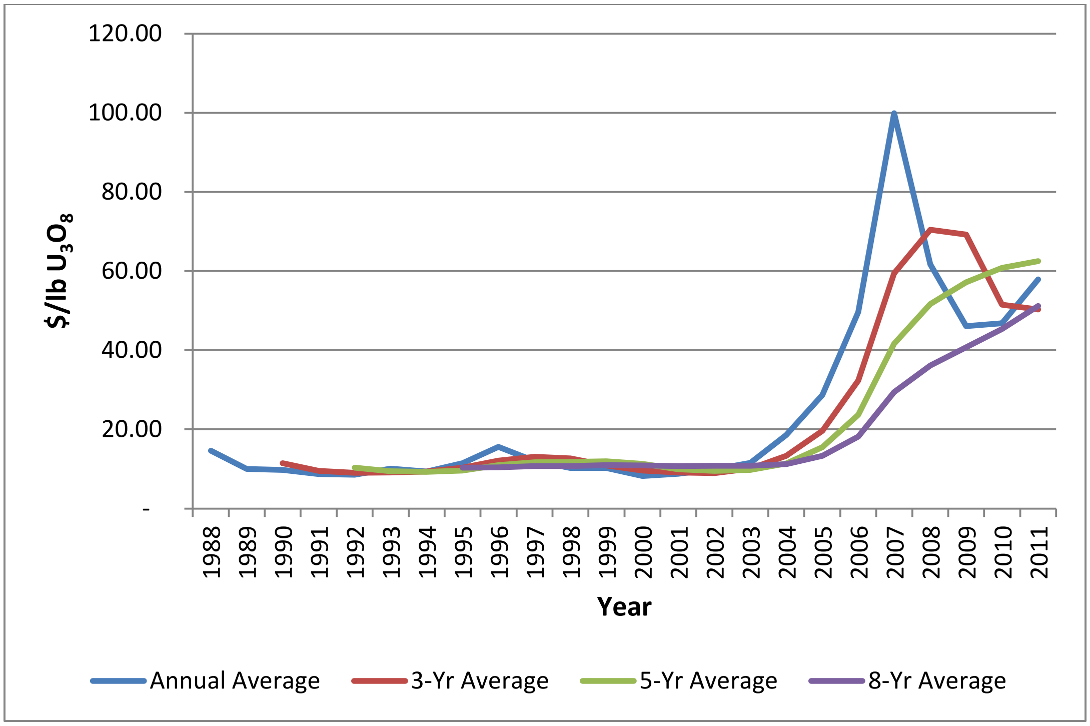

Figure 19.1. Historical U_3O_8 Spot Prices, US$/lb

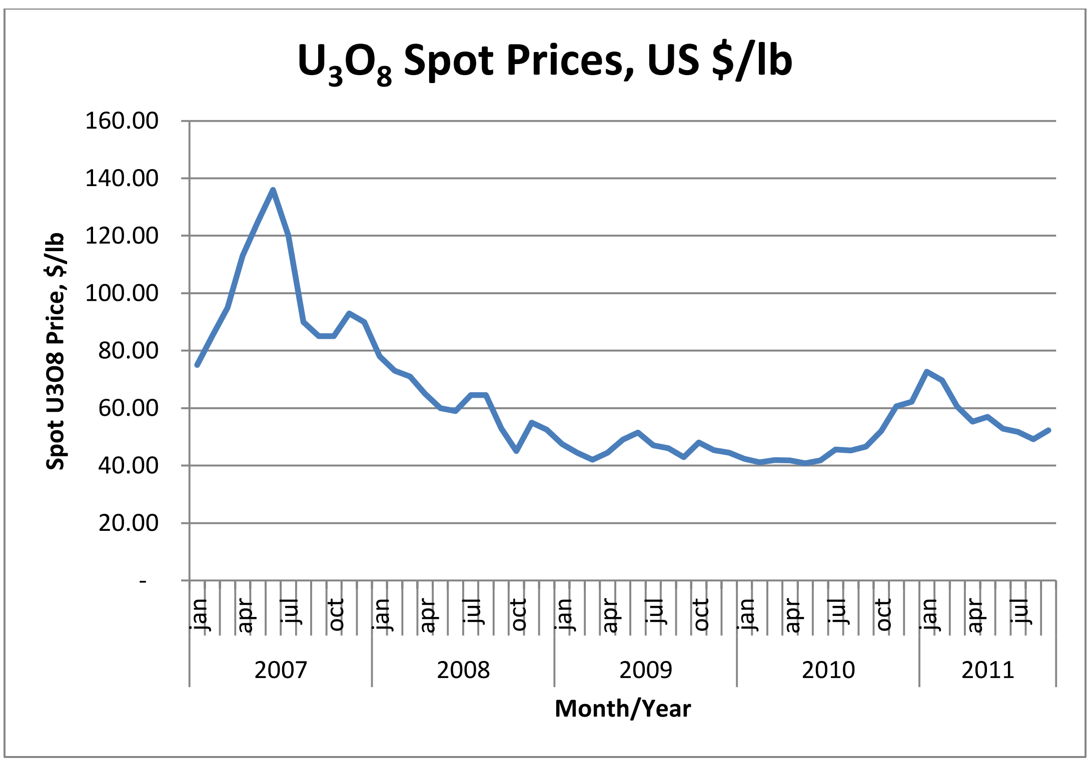

Figure 19.2. Monthly U_3O_8 Spot Prices in last five years, US$/lb

As noted above, a number of concurrent events have contributed to the recent spot price fluctuation including world-wide announcements of significant planned nuclear power generation, and production shortfalls or delays at major uranium mines, both of which caused concern over future supply and drove the spot price upward. Subsequently there has been the disaster at a Japanese generating facility, followed by Germany and some other European countries announcing a reduction in nuclear capacity because of safety concerns. A phasing out of Highly Enriched Uranium as a source of fuel for power generation will require an enhanced focus on exploration in the future; the industry faces infrastructure problems, with enrichment as an example; the success of international efforts to expand use of a nuclear-fuel bank remains uncertain; and environmental concerns will continue to hamper the industry, particularly spent fuel disposal. Difficulties in project financing, the lack of skilled workers, and uncertainty regarding mine expansions in future years all contribute to concerns over supply. The conclusion from this is that the uranium industry from mining through generation is in a state of flux, and therefore price forecasting is best deemed an art at this point, albeit with some foundation for near-term projections.

19.2.2 Forecasted Pricing

Figure 19.3 presents a forecast of spot U_3O_8 prices, again showing annual, three-year, five-year, and eight-year averaging. The next few years reflect the uncertainties noted above, particularly the plans for expanded nuclear power generation throughout the world--in contrast to many countries in central Europe, the Czech Republic is planning on expansion of nuclear power, as is China, Saudi Arabia, and others. The graph exhibits a leveling out of pricing beginning in 2015, with an annual increase in real prices projected thereafter. Such straight-line projections generally stem from uncertainty and impression in the supply/demand realm, but nevertheless serve in this instance as a source for establishing a plausible price base for the Kuriskova project.

Long-term averaging of prices has been used to assess behavior, and in this report the three-year and eight-year average projections are taken as reasonable bounds for future U_3O_8 prices. A single price is preferred, both for establishing a cutoff grade in the deposit and to allow development of a simplified cash flow as part of the Project's economic analysis. In examining the underlying price data, it is determined that the 10-year annual average is US$69.40/lb U_3O_8, whereas the three-year rolling average price from 2015 through 2027 is US$66.90/lb. In this report, a single, constant-dollar future price for U_3O_8 produced on site at Kuriskova is taken at US$68/lb.

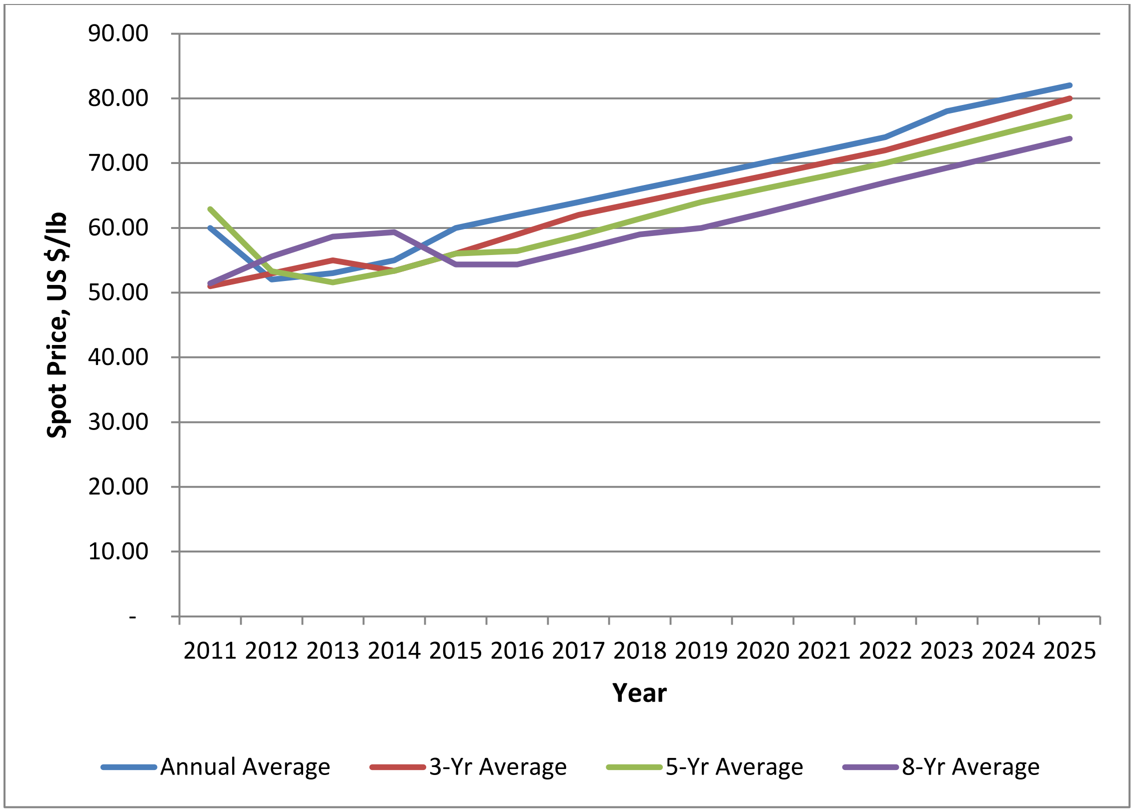

Figure 19.3. Forecast U_3O_8 Spot Prices, US$/lb

20.0 ENVIRONMENTAL

This section provides an overview of environmental and permitting aspects of the Project. The costs associated with the initial permitting, baseline studies and ongoing monitoring and permitting are described in Section 21.0, Capital and Operating Costs.

20.1 Permitting

The permitting strategy is structured to identify and address the various environmental and social requirements and standards applicable to the Project.

The PES will be made public shortly after completion of the feasibility study; thereby, triggering the EIA process under the Slovakian EIA Act (Act No. 127/1994 as amended in Act No. 391/2000). The EIA process will be the primary permitting driver and is anticipated to take 18 to 24 months to complete. A multi-agency regulatory process will be completed to obtain all required permits and approvals necessary to construct, operate and ultimately close the Project. The permitting process in Slovakia is relatively complex and includes participation from the Regional Mining Bureau, Regional Construction Office, the Slovakian environmental agencies, several other government agencies, companies, affected municipalities and the public.

The Project area includes two Natura 2000 ecological protection areas. Natura 2000 is a network of areas designated by EU member countries with the objective of protecting birds and other animal species and their habitat. The specific EU directives include:

- Council Directive 79/409/EEC for the protection of wild birds; and
- Council Directive 92/43/EEC for the protection of habitat.

Volovske Vrchy, established pursuant to the Wild Birds Directive, and Stredne Pohornadie, established pursuant to the Habitats Directive, are designated as Natura 2000 areas, which primarily include the forested areas in and around the Project area and the mountainous regions to the west.

Exploration within the Volovske Vrchy Natura 2000 area requires avoidance of bird fledgling areas during the spring, resulting in suspension of drilling from the beginning of March through June. The presence of Natura 2000 areas does not preclude development activities. For example, active timbering and logging are conducted within the Natura 2000 area by the Kosice Timber Company. Development of the Kuriskova deposit with underground and minimal surface facilities is unlikely to result in impacts that would adversely affect the integrity of the Natura 2000 areas.

Surface disturbances resulting from project development will require meeting of the standards established by Articles 6(3) or 6(4) of the European Union’s Habitats Directive are satisfied. Article 6(3) requires a finding by the State that the Project will not adversely affect the integrity of the overlapping Natura 2000 site; and Article 6(4) will allow a project to proceed despite a conclusion of adverse effects so long as:

- There are no alternative solutions;
- The Project must be carried out for “imperative reasons of overriding public interest,” including those of an economic or social nature; and
- All compensatory measures necessary to ensure the overall coherence of Natura 2000 are taken.

To this end, the Kuriskova deposit will be accessed by means of a decline to an underground mining and process plant with minimal surface disturbances.

20.2 Environmental Liabilities

No environmental liabilities have been identified by Tetra Tech that would materially impede the advancement of the Project to the next engineering study. EUU is responsible for surface disturbances associated with the exploration activities. These activities have been permitted and include financial assurance to cover the costs of reclamation and re-vegetation.

20.3 Baseline Studies

Baseline studies are being conducted with the primary goal of collecting and analyzing technically adequate data that will support the required permit applications and environmental documentation including an EIS. Many of the baseline studies have been initiated as detailed in VODS (2008) and have been advanced since 2009 as the Project moved forward. The primary study areas include:

- Water resources;
- Geochemical characterization;
- Water treatment;
- Ecology (flora and fauna);
- Meteorology, climatology, and air quality;
- Soils; and
- Radiological monitoring.

The baseline study program is summarized in the following sub-sections.

20.3.1 Water Resources

Surface water hydrology, hydrogeology, and the site wide water balance are discussed in additional detail in Section 24.2, Hydrology and Hydrogeology. The Project includes 24 surface water monitoring locations and four groundwater wells. The groundwater monitoring program is being expanded to eight wells as part of the feasibility study. The surface water and groundwater locations are monitored on a quarterly basis for bulk chemistry, anions, metals, and radionuclides including uranium-natural Th-230, Ra-226, Pb-210, Po-210, and other radionuclides.

Background groundwater and surface water quality is generally good with near neutral to alkaline pH and some elevated concentrations of constituents relative to water quality standards/guidelines (European Union, 1998; Miesfera Consult, 2011; WHO, 2006). For example, background radium-226 was at or above the 0.2 Becquerels per liter (Bq/L) guideline value at two of the surface water sampling locations whereas radium-226 concentrations up to 2.6 Bq/L were observed in groundwater samples. In addition, dissolved concentrations of nitrate are generally elevated above the 0.02 mg/L guideline value likely as a result of decaying plant and animal matter.

20.3.2 Geochemical Characterization

Geochemical characterization of mine rock, tailings, and cemented paste backfill was conducted in support of the preliminary feasibility study. The results demonstrate that all samples were not acid generating (non-PAG) and have excess neutralization potential due to the presence of dolomite and/or calcite. When mine rock samples were subjected to water leaching the resulting pH values were alkaline (~pH 8.5 to 9.5) and most constituents were well below the Slovak, EU and WHO regulatory guidelines with some exceptions such as aluminum, iron, and nitrite. Gross alpha and beta content of water extracts from mine rock and tailings samples exceeded the WHO and Slovakian guidelines of 0.50 and 1.0 Bq/L, respectively. The individual radionuclide concentrations should be determined to demonstrate that elevated concentrations are consistent with the naturally elevated background concentrations associated with area soil, rock and water.

The process tailings (conventional and cemented paste backfill) samples were also non-PAG and did not contain sulfidic minerals. The pH from the POX process tailings sample was slightly alkaline (pH=8). The pH values from the cemented paste backfill under atmospheric conditions were above the EU, WHO, and Slovakian guidelines. When a partial pressure of CO_2 (PCO_2) of 2 percent was used, a typical groundwater PCO_2, the pH was well within the acceptable limits.

Placement of tailings as underground paste backfill for geotechnical and materials management purposes is common practice at mine operations throughout the world. The geochemical characterization of cemented paste backfill suggests that underground placement of tailings remains a viable option that should be investigated further as the Project moves through the Preliminary Feasibility stage into feasibility stage.

The characterization program also includes an assessment of the Slovakian Geological Survey (SGUDS) which shows that the quality of the data provided by SGUDS is acceptable and should be used to perform additional/future geochemical testing.

20.3.3 Meteorology, Climatology, and Air Quality

Long-term local data has been recorded from 1952 to 2011 at a meteorological monitoring station located in Kosice. An on-site meteorological tower was established by EUU at the Jahodna, a local ski resort, approximately 1,500 m from the deposit (1.2 km west of the proposed ventilation shaft). The station was established in June 2010 and records wind velocity, wind direction, dew point, rainfall, barometric pressure, air temperature, and humidity. There is a ridge between the location of the ventilation shaft and the meteorological tower. Therefore, a second tower will be installed near the ventilation shaft to collect data on the wind speed, direction, and stability which is more representative of conditions at the ventilation shaft and will be sufficient to conduct atmospheric modeling.

Air quality monitoring was performed at Jahodna during three week-long sampling events and included analysis of sulphur dioxide (SO_2), nitrogen dioxide (NO_2), nitrogen oxides (NOx), carbon monoxide (CO) and suspended particulate matter (PM10), and ozone (O_3).

20.3.4 Water Treatment

Conceptual-level WTP and sediment pond were designed to support the prefeasibility study. The WTP is designed to treat 700 tpd of water from the mine rock leachate and cemented paste backfill decant water. Water is to be pumped from the mine workings, to the surficial sediment pond. A 48-hour retention time will be used for settling, before conveyance to the WTP. At the plant, ceramic microfiltration and reverse osmosis (RO) will treat the water to meet Slovakian surface water discharge limits. With a removal rate of 90-95 percent, the plant permeate (560 to 665 tpd) will be directly discharged to the environment and the concentrated waste (140 to 35 tpd) will be returned to the mine to be incorporated into the paste backfill.

20.3.5 Ecology

Between August 2008 and October 2009, ecological surveying was completed in the Volovske hills (vrchy). The survey area covered roughly 120 km^2 in the Volovske hills for which no known previous published survey is available. The surveying, and subsequent reports, provided a baseline update for the general species diversity and status in the survey area, as well as an assessment of the potential ecological impacts related to the development of the Project. The vegetation and wildlife studies will be continued as the Project advances to the feasibility and permitting stage.

20.3.6 Soils

A soil survey encompassing approximately 182 hectares and included potential mine-related surface disturbance was conducted in 2011. The main goals of the soil survey was to map and classify the soils within the study area, describe the soil profiles, collect soil samples for pedological and geochemical analysis, assess background metals concentrations in soils prior to mining activities, and assess soil salvage depth for reclamation. Tetra Tech conducted oversight on the soils work, including participation in the sampling and surveying, and review and finalization of a soils report.

In general, soils in the survey area should have a sufficient depth and quantity to permit practical salvage, and be of suitable quality and texture for use as primary and secondary plant growth medium. The primary factors limiting soil salvage in the survey area are shallow soils, high coarse rock fragments and steep terrain. The primary factors which may affect the suitability of the soils in the survey area as a plant growth medium for reclamation are soil acidity (low pH) and potentially low nutrient status and high aluminum concentrations. To address these limitations, seeding plants species or planting tree and shrub seedlings that are adapted to low soil pH and nutrient status for interim and final reclamation and/or application of soil amendments (e.g., lime) or fertilizers may be required to establish vegetation.

Some soils sampled had greater metal concentrations than the indices promulgated by Slovakia (Rule Ministry Landscape SR No. 531/1994 – 540). Based on these indices, the trace elements that are elevated in the soils of the survey area include: arsenic, chromium, copper, nickel, lead, vanadium and zinc. Slovakian reference indices were not readily available for the other trace elements analyzed (i.e., uranium, aluminum, iron, manganese, selenium, cadmium, cobalt, and fluorine).

A subset of the surface soil samples collected during the soils survey will define the background concentrations of radionuclides within the Project area. Samples were collected to evaluate the background radionuclide soil concentrations from the surface (0 to 15 cm) and subsurface (15 to 30 cm) from each project-relevant soil map unit identified during the soils survey. Soil and sediment sampling will continue as the Project advances to the FS and EIA stages.

20.3.7 Additional Radiological Monitoring

The baseline radiological monitoring program is designed to provide an assessment of the environmental conditions at the Project site prior to the beginning activities. For planning purposes, the radiological monitoring program is separated into the PFS-level activities and the FS/EIA-level activities. In preparing this proposed baseline monitoring program, guidance was obtained from the U.S. Nuclear Regulatory Commission (NRC) Regulatory Guide 4.14- Radiological Effluent and Environmental Monitoring at Uranium Mills (NRC, 1980) and guidance from the International Atomic Energy Agency (IAEA). Two specific documents included IAEA Safety Report Series No. 27 (IAEA, 2002a) and IAEA Safety Standard Series Safety Guide No.

WS-G-1.2 (IAEA, 2002b). The IAEA guidance is more general in nature; therefore, the more prescriptive NRC Regulatory Guide was largely used as the basis for this program.

Major elements of the baseline radiological monitoring program include:

- Direct gamma radiation measurements, and;
- Determination of radionuclide concentrations in:
- Surface water;
- Groundwater;
- Soil and sediment;
- Radon gas;
- Airborne particulates;
- Meat, milk, vegetation, and fish; and

Seven radionuclide monitoring locations are being considered as part of the feasibility-level monitoring program. In addition to the locations selected for the feasibility-level soils study, soil sampling will also be conducted at the air particulate locations.

20.4 Reclamation and Closure

This section presents the planned Project reclamation and closure activities. Reclamation and closure costs are provided in 21.0, Capital and Operating Cost Estimates.

The primary interim reclamation activities will include stockpiling of the first 300 mm on average of soil during initial site preparation. Approximately 100,000 m^3 of topsoil will be salvaged from the roads and surface facility location. The soil stockpiles will be temporarily revegetated with an approved temporary seed mix until the time of their intended use, after which the soil and its footprint will be fully reclaimed. The soil stockpiles will be located close to the intended reclamation sites to minimize haul distances and associated costs.

Infrastructure and facilities that cannot be converted to a post-mining land use will be decommissioned, demolished and reclaimed. Structural demolition will include disassembling the structural steel and building skeletons, selling steel as scrap and placing construction debris underground. Concrete foundations will be rubblized after the steel infrastructure and other demolition debris have been removed. The reclamation plan currently includes salvage of the majority of the equipment within the Process plant and the structural steel. Items that cannot be salvaged will be cemented in place underground.

The portal will be sealed by pushing fill from the surface facility. Topsoil will be placed over the final slope, to the extent practicable, and the area will be revegetated using an approved seed mixture. The ventilation and egress shafts will be sealed with polyurethane foam (PUF) plugs or similar and steel plates which will be captured in the concrete overslab. Compacted soil will be placed over the concrete overslab followed by topsoil which will be seeded.

Roads that cannot be converted to a post-mining land use will be regraded to reestablish approximate original ground contours, scarified, topsoil will be placed and revegetated using an approved seed mixture.

21.0 CAPITAL AND OPERATING COST ESTIMATES

Capital costs for the mine, process plant, power supply, environmental/reclamation, and surface facilities for the Project have been prepared in accordance with standard industry practices for this level of study and to a level of definition and intended accuracy of ±25 percent. The principal engineer for the Project design, initial and sustaining capital cost estimation was Tetra Tech.

21.1 Initial Capital Costs

The initial capital cost estimate consists of four components: direct costs, indirect costs, contingency and Owner's costs. Owner's costs were estimated with input from EUU.

The initial CAPEX for the Project is approximately US$225 million, subject to qualifications, assumptions, and exclusions.

The initial capital cost summary and distribution are shown in Table 21.1.

Table 21.1. Initial Capital Costs Summary

Item	US$ Millions
Direct Costs	
Mining	$91.56
Processing Plant	$28.37
Environmental/Reclamation	$1.03
Infrastructure	$23.18
Total Direct Costs	$144.14
Project Indirect Costs	$24.12
Other Owners Costs	$25.75
Total Indirect Costs	$49.87
Total Direct and Indirect Costs	$194.01
Contingency	$31.00
Total Initial Capital	$225.01

21.1.1 Direct Initial Capital

The direct initial capital costs include all new equipment, new materials, and installation for all permanent facilities associated with:

- Crushing, material handling, and processing facilities
- Process building and excavation
- Infrastructure roads and site preparation
- Power supply and distribution
- Pre-production development and mining
- Underground tailings storage excavation
- Warehousing
- Administration
- Truck shop

- Yard services and other utilities
- Control and communications systems
- Plant mobile equipment
- Fuel storage
- Explosives storage

21.1.1.1 Pre-production Development and Mining

The initial direct mining CAPEX is estimated at US$91.56 million and is broken down in Table 21.2.

Table 21.2. Mining Initial Capital Expenditures

Items	US$ Millions
UG Mobile Equipment	$16.47
UG Services	$2.27
UG Communication and Electrical	$6.58
Ventilation Equipment	$0.58
UG Capital Development	$27.25
UG Infrastructure	$33.91
Development Rock Surface Crusher	$1.27
UG Contractor Mobilization and Demobilization	$0.08
Mine Salaried Labor	$3.16
Total Initial Direct Mining Capital	$91.56

21.1.1.2 Process Facilities

The initial process facility CAPEX is estimated at US$28.37 million and is shown distributed into the various process areas in Table 21.3.

Table 21.3. Process Facility Initial Capital Expenditures

Area	US$ Millions
Crushing	$0.91
Grinding And Classification	$3.08
Flashing And Cooling	$8.67
Solid Liquid Separation	$3.95
Sodium Diuranate Precipitation	$0.80
Low pH Precipitation	$0.66
Re-Carbonation	$5.36
Molybdenum Precipitation	$2.52
Paste Backfill Plant	$1.56
Reagents	$0.86
Total Initial Process Plant Capital	$28.37

The initial infrastructure CAPEX is estimated at US$23.18 million and is shown distributed into the various process areas in Table 21.4.

Table 21.4. Infrastructure Initial Capital Costs Summary

Item	US$ Millions
Site Preparation	$3.79
Overall Site Electrical	$0.72
Overall Site Controls And Communications	$1.37
Buildings And Structures	$8.52
Site Services And Utilities	$3.03
Plant Mobile Fleet	$1.00
Miscellaneous Concrete	$0.06
Surface Facilities - Offsite	$4.70
Total Initial Infrastructure Capital	$23.18

21.1.1.3 Environmental/Reclamation

Total direct initial capital costs for environment and reclamation is US$1.03 million and includes the water treatment facilities. Other environmental initial capital costs are included in Other Owner's Costs.

21.1.2 Indirect Initial Capital

The total indirect capital costs are US$49.87 million. Indirect costs include the following:

- Temporary construction services including some construction equipment
- Freight
- Vendor representatives
- First fills and capital spares
- Engineering, procurement and construction management (EPCM) services (including travel expenses)
- QA
- Surveying
- Owner's costs
- Start-up and commissioning allowance

Table 21.5 shows the distribution of the indirect capital costs.

Table 21.5. Indirect Initial Capital Expenditures

Area	US$ Millions
Project Indirect Costs	
Construction Indirect Costs	$8.83
Spare Parts	$2.96
Initial Fills	$0.15
Freight and Logistics	$5.30
Commissioning And Start up	$0.43
Engineering and Procurement (EP)	$2.94
Construction and Management (CM)	$2.94
Vendor Assistance	$0.32
Temporary Facilities	$0.25
Project Indirect Costs Total	$24.12
Owner's Costs	$25.75
Total Initial Project Indirect Capital	$49.87

21.1.3 Working Capital

Two months of operating expenses were included as capital costs in the cash flow statement. This capital was recovered at the cessation of operations.

21.1.4 Contingency

The overall contingency for the Project development has been estimated as 16 percent of direct costs and reflects an average of contingencies from each area.

The contingency amount is an allowance that has been added to the capital cost estimate to cover unforeseeable costs within the scope of the estimate.

21.2 Sustaining Capital Costs

Sustaining capital over mine life totals US$70.85 million. Table 21.6 shows a summary of the breakdown of costs.

Table 21.6. LOM Sustaining Capital Expenditures

Area	US$ Millions
Underground Mine	$67.47
Process Plant	$.09
Infrastructure	$1.00
Environmental/Reclamation	$2.29
Total Sustaining Capital	$70.85

21.3 Operating Costs

The OPEX per tonne of ore is US$201. Table 21.7 shows a summary of the breakdown of costs. Table 21.8 shows a breakdown of the operating costs by production year and for the LOM.

Table 21.7. LOM Unit Operating Costs

Operating Costs	US$ / Tonne of Ore
Mining U/G	$86.51
Processing Plant	$92.99
Infrastructure	$2.57
General & Administrative	$18.74
Total LOM Operating Costs	$200.81

Table 21.8. Yearly Operating Costs

Production	US$/Tonne Ore					
Year	Mine	Plant	Infrastructure	G&A	Total	US$/lb U_3O_8 *
1	$131.92	$102.64	$2.86	$20.90	$258.33	$20.68
2	$84.76	$100.03	$2.53	$18.49	$205.80	$14.79
3	$85.46	$96.70	$2.54	$18.57	$203.27	$17.53
4	$84.30	$94.70	$2.53	$18.46	$199.99	$19.81
5	$85.25	$94.09	$2.54	$18.50	$200.38	$22.91
6	$82.59	$92.30	$2.54	$18.54	$195.97	$24.04
7	$84.02	$94.16	$2.55	$18.61	$199.34	$27.16
8	$84.45	$90.55	$2.55	$18.59	$196.14	$30.49
9	$84.46	$88.88	$2.54	$18.55	$194.43	$29.94
10	$84.29	$88.00	$2.55	$18.61	$193.46	$34.70
11	$70.34	$88.84	$2.53	$18.50	$180.21	$35.34
12	$81.45	$86.17	$2.54	$18.54	$188.70	$41.88
13	$86.51	$92.49	$2.78	$20.32	$202.11	$50.48
LOM	$86.51	$92.99	$2.57	$18.74	$200.81	$24.26

*Note: Does not include any molybdenum byproduct of US$1,27 per lb U_3O_8 over LOM.

22.0 ECONOMIC ANALYSIS

22.1 Summary

Economic analysis of the Project was performed to assess the economic viability of constructing and operating the Project as designed. The analysis was based on mine plans and production schedules derived from the most current resource estimates. Yearly LOM metal production averages approximately 786 tonnes of U_3O_8 yellowcake and 84 tonnes of molybdenum as molybdenum sulfide over the 13 years of production. Details of the reserve calculations and production schedules are shown in Section 15.

A proforma cash flow statement projects potential revenues, transport costs and facility operating and capital costs to yield annual net cash flows which are then discounted to determine a project NPV. The cash flow excludes corporate income taxes, but includes the cost of all royalties to the Slovak government and Local Community Support payments. The Base Case NPV, at 8 percent discount rate, and IRR are calculated to be US$276 million and 30.8 percent, respectively. Initial capital costs are US$225 million with a simple payback of 1.9 years. The highest sensitivity for both NPV and IRR is future uranium price. Changes to operating and initial capital costs had less of an effect on project NPV and IRR than uranium price. A detailed analysis of these values and other metrics are contained further in this report.

The economic analysis herein assumes ore production of 600 tpd originating from the Kuriskova underground deposit using underground mechanized cut-and-fill mining methods. The process facility is designed to process the full 600 tpd in a facility located in an underground excavation near the main access to the mine. CAPEX and OPEX were developed by the design team.

22.2 Cash Flow Basis

22.2.1 Mineral Reserves

The Kuriskova mineral deposit will be mined using conventional mechanized cut-and-fill underground mining techniques. Reserves for the deposit are presented in Section 15.

The reserves were developed by applying cutoff grades and underground mine plans using appropriate mining and processing methods and estimated costs. The cutoff grades were 0.13 percent uranium for the underground mine area. Uranium values assessed at a price of 68.00/lb U_3O_8 and an estimated recovery of 92 percent were originally used to estimate cutoff grade. Molybdenum grades were not considered in calculating potential mineable resources.

Total LOM ore mined and processed from the underground mine will be 2.528 million tons at average grades of 0.346 percent uranium and 0.046 percent molybdenum. The total contained recoverable amounts of metals are approximately 20.9 million lbs of U_3O_8 and 2.223 million lbs of molybdenum.

22.2.2 Mine Permitting and Development Schedule

A multi-agency regulatory process will be completed to obtain all required permits and approvals necessary to construct, operate and ultimately close the Project. The permitting process in Slovakia is relatively complex and includes participation from the Regional Mining Bureau, Regional Construction Office, the Slovakian environmental agencies, several other government agencies, companies, affected municipalities, and the public.

The Project baseline studies are well underway and are anticipated to be completed in 2013, prior to issuance of the feasibility study. The PES will follow, which will trigger the EIA process under the Slovakian EIA Act (Act No. 127/1994 as amended in Act No. 391/2000). The EIA process will be the primary permitting driver and is anticipated to take 18 to 24 months to complete.

Construction of the decline from the surface facilities to access the underground deposit is scheduled to begin three years before the start of ore processing. Development of the decline is estimated to take about 17 months. At the completion of the surface decline, development of the underground process excavation will begin in parallel with mine access development. The underground excavations will take about 7 months to complete. Construction of the processing plant will begin at the completion of the excavation and will take 12 months to complete.

22.2.3 Mine Plans and Schedules

The Kuriskova underground deposit area will be accessed prior to the beginning of production with production beginning concurrent with startup of the underground process facility. Production from the deposit will ramp up concurrent with the commissioning of the plant to 90 percent of full production for the first year. Production will continue at full capacity for the remainder of the 13-year underground mine life.

22.2.4 Metals Production

Projected metals productions of uranium into yellowcake and molybdenum as a concentrate are summarized in Table 22.1. Overall plant recoveries are estimated to be 92 percent and 86.8 percent for uranium and molybdenum, respectively, as determined by metallurgical testwork. LOM uranium production as yellowcake is estimated to be 17.75 million lbs (20.93 million lbs U_3O_8).

Table 22.1. Metal Production by Mine Period

Metal	Units	Years 1-5 Annual Average	Years 1-10 Annual Average	LOM Annual Average	LOM Total
Yellowcake Concentrate	tonnes/y	1,121	903	833	10,060
Uranium Production	lbs/yr (000's)	1,977	1,593	1,470	17,746
U_3O_8 Equivalent	lbs/yr (000's)	2,331	1,878	1,733	20,927
Molybdenum Production	lbs/yr (000's)	222	202	185	2,223

The proforma cash flow analysis Base Case uses metal prices of US$68.00/lb U_3O_8 and US$15/lb molybdenum.

22.2.5 Transport and Refining Costs

Transport costs for uranium yellowcake and molybdenum concentrate were estimated by EUU to be US$420.00 and US$280.00 per wet tonne of concentrate, respectively. Transport costs were calculated on the basis of 15 percent moisture for uranium yellowcake and 10 percent for molybdenum concentrate.

Total net mine returns were calculated using a 98.5 percent pay factor for uranium and 80 percent for molybdenum. Additional penalties for impurities in the yellowcake are possible, but are not expected at this time. Additional metallurgical work will be conducted to confirm.

22.2.6 Royalties and Local Community Support Payments

The cash flows calculated in the proforma include the costs of all third-party royalties, as well as a Local Community Support payment, payable on income from the Kuriskova project.

The only royalty is to the Slovak government based on the Mining Act is 10 percent for uranium and 2 percent for molybdenum. Local community support payments are calculated as 1 percent of the payable uranium revenue.

22.2.7 Operating Costs

The LOM operating costs for the Project are estimated at US$507.6 million as summarized below in Table 22.2.

Table 22.2. LOM Operating Cost Summary

Area	US$ Million
Underground Mine	218,693
Process Plant	235,079
Infrastructure	6,490
G&A	47,370
Total OPEX	507,633

22.2.8 Summary of Parameters

Values of key parameters used during preparation of the proforma cash flow statement are presented in Table 22.3. Included are such values as ore tonnage and grade, overall recoveries for the metals, total metal production, pay factors, capital and operating expenses and metal prices (Base Case shown).

This information is also found in Table 22.4; however, the summary in Table 22.3 is provided for ease of use and reference. Sensitivity analyses on metal price, capital costs, and operating costs are presented in Section 22.4. A Monte Carlo Risk Analysis can be found in Section 22.5.

Table 22.3. Base Case Parameters Used for the Cash Flow Analysis

U_3O_8 (tons)	U_3O_8 (%)	U_3O_8 (%)	U_3O_8		U_3O_8
			Uranium \$/lb U_3O_8	\$68.00	
2,527,924	0.346	0.046	Molybdenum \$/lb	\$15.00	
Metal Balance	Contained	Recovery	Recovered	Payable %	Payable
Uranium as U_3O_8 (lbs)	22,747,141	92.0%	20,927,370	98.50%	20,613,460
Molybdenum (lbs)	2,560,628	86.8%	2,222,625	80.00%	1,778,100
Average Annual Production		Avg. Years 1-5	Avg. Years 6-10	Avg. Years 10-13	LOM
Uranium Recovered as U_3O_8 (lbs)		2,331,360	1,424,662	715,755	1,609,798
Molybdenum Recovered (lbs)		222,437	181,366	67,869	170,971
Capital Costs (000's US\$)			\$/t Ore Processed	\$/lb U_3O_8 Equivalent	LOM (US\$ Millions)
Initial	\$225,012	Underground Mining	\$86.51	\$10.45	218.7
Sustaining	\$70,852	Process Plant	\$92.99	\$11.23	235.1
Total	\$295,864	Infrastructure	\$2.57	\$0.31	6.5
		General & Administrative	\$18.74	\$2.26	47.4
		Total	\$200.81	\$24.26	507.6
Pre-Tax Economics					
NPV 0% (US\$ Millions)	**NPV 5% (US\$ Millions)**	**NPV 8% (US\$ Millions)**	**NPV 10% (US\$ Millions)**	**IRR %**	**Payback from Startup**
616.8	373.5	276,4	225.4	30.8%	1.9 years

22.3 Project Cash Flow

Table 22.4 shows the proforma cash flow statement. Project development is shown to commence in three years prior to the start of ore processing operations. Mine and process production ramps up to full production within the first year and continues through into Year 13 of the mine life. Note that there are two years of reclamation work, shown as Sustaining Capital/Closure, required and accounted for beyond Year 13.

The cash flow statement “production summary” summarizes mine production for the deposit. Note in this the annual average metal grades, estimated process plant recoveries, concentrate grades, and amount of concentrate produced.

The “payable metals production” summarizes estimates of scheduled pay factors used to estimate the payable values from each of the metals and the payable values themselves by annual production.

The “cash flow summary” shows the pay factors, transport costs, and royalties deducted resulting in a net mine return after royalties. Capital costs are presented in Section 21.0 of this report.

NPVs were calculated from the resulting cash flow at discount rates of 0 percent, 5 percent, 8 percent, and 10 percent using standard valuations. At 8 percent discount rate, project NPV for the Base Case is US$276.4 million with an IRR of 30.8 percent. A simple payback on the initial capital costs incurred in the first three years occurs in 1.9 years for the Base Case.

Table 22.4

Uranium =	$68.00
Molybdenum =	$15.00

Kuriskova Uranium Project

Underground Mine and Processing F+K92acility

PRODUCTION SUMMARY

			-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	Totals
Mill feed from U/G	tonnes					186,247	210,613	209,683	210,897	210,404	209,976	209,266	209,436	209,843	209,161	210,501	209,969	31,928			**2,527,924**
Mill feed grade: Uranium	%U					0.522%	0.582%	0.485%	0.422%	0.366%	0.341%	0.307%	0.269%	0.271%	0.233%	0.213%	0.188%	0.167%			**0.003**
Mill feed grade: Molybdenum	%Mo					0.067%	0.068%	0.052%	0.046%	0.050%	0.053%	0.049%	0.046%	0.039%	0.040%	0.031%	0.017%	0.012%			**0.000**
Mill feed Total	**tonnes**					**186,247**	**210,613**	**209,683**	**210,897**	**210,404**	**209,976**	**209,266**	**209,436**	**209,843**	**209,161**	**210,501**	**209,969**	**31,928**			**2,527,924**
Mill feed grade: Uranium	%U					0.522%	0.582%	0.485%	0.422%	0.366%	0.341%	0.307%	0.269%	0.271%	0.233%	0.213%	0.188%	0.167%			**0.346051%**
Mill feed grade: Molybdenum	%Mo					0.067%	0.068%	0.052%	0.046%	0.050%	0.053%	0.049%	0.046%	0.039%	0.040%	0.031%	0.017%	0.012%			**0.000**
Mill Recoveries	U %					92%	92%	92%	92%	92%	92%	92%	92%	92%	92%	92%	92%	92%			92%
	Mo %					87%	87%	87%	87%	87%	87%	87%	87%	87%	87%	87%	87%	87%			87%
Concentrate Grade : Uranium	% U					80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%	80%			**80%**
	% Mo					10%	10%	10%	10%	10%	10%	10%	10%	10%	10%	10%	10%	10%			**10%**
CONCENTRATE PRODUCED	dtonne U					1118.3	1408.5	1168.5	1023.5	884.8	822.8	738.4	647.6	655.0	560.5	516.0	454.7	61.5			**10,060**
	dtonne Mo					1086.1	1245.2	954.6	851.0	907.0	957.0	883.7	844.1	709.3	718.5	572.5	317.8	33.1			**10,080**
	wtonne U					1315.7	1657.0	1374.7	1204.2	1040.9	968.0	868.7	761.9	770.6	659.4	607.1	535.0	72.3			
	wtonne Mo					1206.8	1383.5	1060.7	945.5	1007.8	1063.3	981.9	937.9	788.1	798.4	636.2	353.1	36.8			**11,200**

PAYABLE METALS PRODUCTION

			-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	Totals
Contained Recoverable Metals																					
Uranium	lb U					1,972,761	2,484,574	2,061,185	1,805,503	1,560,720	1,451,352	1,302,510	1,142,328	1,155,504	988,736	910,269	802,162	108,408			17,746,012
U_3O_8 Equivalent	lb U_3O_8					2,326,421	2,929,988	2,430,697	2,129,179	1,840,513	1,711,538	1,536,013	1,347,115	1,362,653	1,165,989	1,073,455	945,967	127,842			20,927,370
U_3O_8 Equivalent	tonnes U_3O_8					1,055	1,329	1,102	966	835	776	697	611	618	529	487	429	58			9,491
Molybdenum	lb Mo					239,493	274,564	210,493	187,641	199,995	211,019	194,854	186,117	156,408	158,433	126,247	70,067	7,294			2,222,625
Molybdenum	tonnes Mo					109	125	95	85	91	96	88	84	71	72	57	32	3			1,008
Contained Recoverable Metal Gross Value																					
Uranium	$000s					$158,197	$199,239	$165,287	$144,784	$125,155	$116,385	$104,449	$91,604	$92,660	$79,287	$72,995	$64,326	$8,693			**1,423,061**
Molybdenum	$000s					$3,592	$4,118	$3,157	$2,815	$3,000	$3,165	$2,923	$2,792	$2,346	$2,376	$1,894	$1,051	$109			**33,339**
Total	$000s					$161,789	$203,358	$168,445	$147,599	$128,155	$119,550	$107,372	$94,396	$95,007	$81,664	$74,889	$65,377	$8,803			**1,456,401**
Payable Metal																					
U Pay factor	%	98.5%				98.5%	98.5%	98.5%	98.5%	98.5%	98.5%	98.5%	98.5%	98.5%	98.5%	98.5%	98.5%	98.5%			98.500%
Molybdenum Payfactor	%	80.0%				80.0%	80.0%	80.0%	80.0%	80.0%	80.0%	80.0%	80.0%	80.0%	80.0%	80.0%	80.0%	80.0%			80.00%
Payable Uranium	lb U_3O_8					2,291,524	2,886,038	2,394,237	2,097,241	1,812,905	1,685,865	1,512,973	1,326,909	1,342,213	1,148,499	1,057,353	931,777	125,924			**20,613,460**
Payable Molybdenum	lb Mo					191,594	219,651	168,394	150,113	159,996	168,816	155,883	148,893	125,126	126,746	100,998	56,053	5,835			**1,778,100**
Uranium Prices Used	**$/lb**					**$68.00**	**$68.00**	**$68.00**	**$68.00**	**$68.00**	**$68.00**	**$68.00**	**$68.00**	**$68.00**	**$68.00**	**$68.00**	**$68.00**	**$68.00**	**$68.00**	**$68.00**	
Molybdenum Prices Used	**$/lb**					**$15.00**	**$15.00**	**$15.00**	**$15.00**	**$15.00**	**$15.00**	**$15.00**	**$15.00**	**$15.00**	**$15.00**	**$15.00**	**$15.00**	**$15.00**	**$15.00**	**$15.00**	
Payable Metal Value																					
Uranium	$000s					$155,824	$196,251	$162,808	$142,612	$123,278	$114,639	$102,882	$90,230	$91,270	$78,098	$71,900	$63,361	$8,563			**$1,401,715**
Molybdenum	$000s					$2,874	$3,295	$2,526	$2,252	$2,400	$2,532	$2,338	$2,233	$1,877	$1,901	$1,515	$841	$88			**$26,671**
Total	$000s					$158,698	$199,545	$165,334	$144,864	$125,678	$117,171	$105,220	$92,463	$93,147	$79,999	$73,415	$64,202	$8,650			**$1,428,387**

Table 22.4

Uranium =	$68.00
Molybdenum =	$15.00

Kuriskova Uranium Project

Underground Mine and Processing F+K92acility

CASHFLOW SUMMARY

			-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	Totals
			US$000s	US$000s	US$000s	US$000s	US$000s	US$000s	US$000s	US$000s	US$000s	US$000s	US$000s	US$000s	US$000s	US$000s	US$000s	US$000s	US$000s	US$000s	US$000s
Gross Value of Contained Metals						**$161,789**	**$203,358**	**$168,445**	**$147,599**	**$128,155**	**$119,550**	**$107,372**	**$94,396**	**$95,007**	**$81,664**	**$74,889**	**$65,377**	**$8,803**			**$1,456,401**
Payfactor dedn						$3,091	$3,812	$3,111	$2,735	$2,477	$2,379	$2,151	$1,932	$1,859	$1,665	$1,474	$1,175	$152			28,014
Uranium Transport						$553	$696	$577	$506	$437	$407	$365	$320	$324	$277	$255	$225	$30			4,971
Molybdenum Transport						$338	$387	$297	$265	$282	$298	$275	$263	$221	$224	$178	$99	$10			3,136
Net Mine Return						**$157,807**	**$198,462**	**$164,460**	**$144,094**	**$124,958**	**$116,467**	**$104,581**	**$91,881**	**$92,603**	**$79,499**	**$72,982**	**$63,878**	**$8,610**			**1,420,280**
Federal Uranium Royalty						$7,929	$8,054	$6,821	$5,990	$5,226	$4,814	$4,321	$3,871	$3,951	$3,391	$2,797	$2,725	$365			**60,255**
Federal Molybdenum Royalty						$26	$24	$19	$17	$18	$19	$17	$17	$14	$15	$10	$6	$1			**203**
Local Community Support						$1,555	$1,959	$1,625	$1,423	$1,230	$1,143	$1,026	$900	$910	$779	$717	$633	$86			**13,986**
Net Mine Return after Royalty/Support						**$148,297**	**$188,426**	**$155,995**	**$136,663**	**$118,484**	**$110,490**	**$99,216**	**$87,093**	**$87,728**	**$75,315**	**$69,458**	**$60,514**	**$8,158**			**$1,345,836**
Operating Costs	$/tonne ore																				
Mining U/G	$86.51	$000s				$24,571	$17,851	$17,920	$17,780	$17,936	$17,342	$17,583	$17,688	$17,722	$17,630	$14,807	$17,101	$2,762			**218,693**
Processing	$92.99	$000s				$19,116	$21,067	$20,275	$19,972	$19,797	$19,380	$19,704	$18,963	$18,651	$18,407	$18,701	$18,093	$2,953			**235,079**
Infrastructure	$2.57	$000s				$533	$533	$533	$533	$533	$533	$533	$533	$533	$533	$533	$533	$89			**6,490**
General & Administrative	$18.74	$000s				$3,893	$3,893	$3,893	$3,893	$3,893	$3,893	$3,893	$3,893	$3,893	$3,893	$3,893	$3,893	$649			**47,370**
Total Operating Costs			$0	$0	$0	$48,113	$43,344	$42,622	$42,178	$42,160	$41,150	$41,714	$41,078	$40,800	$40,464	$37,935	$39,621	$6,453			**507,633**
	$/tonne ore					*$258.33*	*$205.80*	*$203.27*	*$199.99*	*$200.38*	*$195.97*	*$199.34*	*$196.14*	*$194.43*	*$193.46*	*$180.21*	*$188.70*	*$202.11*			*$200.81*
	$/lb U_3O_8					*$20.68*	*$14.79*	*$17.53*	*$19.81*	*$22.91*	*$24.04*	*$27.16*	*$30.49*	*$29.94*	*$34.70*	*$35.34*	*$41.88*	*$50.48*			*$24.26*
Operating Cashflow	**$000s**		**$0**	**$0**	**$0**	**$109,694**	**$155,118**	**$121,838**	**$101,915**	**$82,798**	**$75,317**	**$62,866**	**$50,802**	**$51,803**	**$39,035**	**$35,047**	**$24,257**	**$2,157**			**912,647**
Initial Capital Costs			**60,376**	**90,458**	**74,178**	**0**	**0**	**0**	0	0	0	0	0	0	0	0	0	0	0	0	**225,012**
Working Capital	**2.0**	months	0	0	0	8,019	0	0	0	0	0	0	0	0	0	0	0	-8,019	0	0	**0**
Sustaining Capital/closure			0	0	0	17,222	3,069	7,372	4,627	3,690	6,596	4,949	7,196	7,126	4,061	2,660	0	753	1,094	438	**70,852**
Total Capex including WC, Sustaining/Closure			60,376	90,458	74,178	25,241	3,069	7,372	4,627	3,690	6,596	4,949	7,196	7,126	4,061	2,660	0	-7,266	1,094	438	**295,864**
Pre-Tax Net Cashflow			**-60,376**	**-90,458**	**-74,178**	**84,452**	**152,049**	**114,466**	**97,288**	**79,108**	**68,721**	**57,918**	**43,607**	**44,677**	**34,974**	**32,388**	**24,257**	**9,422**	**-1,094**	**-438**	**616,783**
Accum Pre-tax			**-$60,376**	**-$150,834**	**-$225,012**	**-$140,560**	**$11,489**	**$125,955**	**$223,243**	**$302,351**	**$371,073**	**$428,990**	**$472,597**	**$517,274**	**$552,248**	**$584,636**	**$608,893**	**$618,315**	**$617,221**	**$616,783**	

			NPV 000s
		Disc rate	**Pre-tax**
Pre-Tax IRR	**30.8%**	**0%**	**$616,783**
Simple Payback (Yrs)	**1.9**	**5%**	**$373,546**
		8%	**$276,382**
		10%	**$225,373**

The timing and magnitude of cash flows and expenditures are presented in Figure 22.1. Early capital requirements reflect the negative cash flow in the early years; however, these values quickly become positive once operations commence.

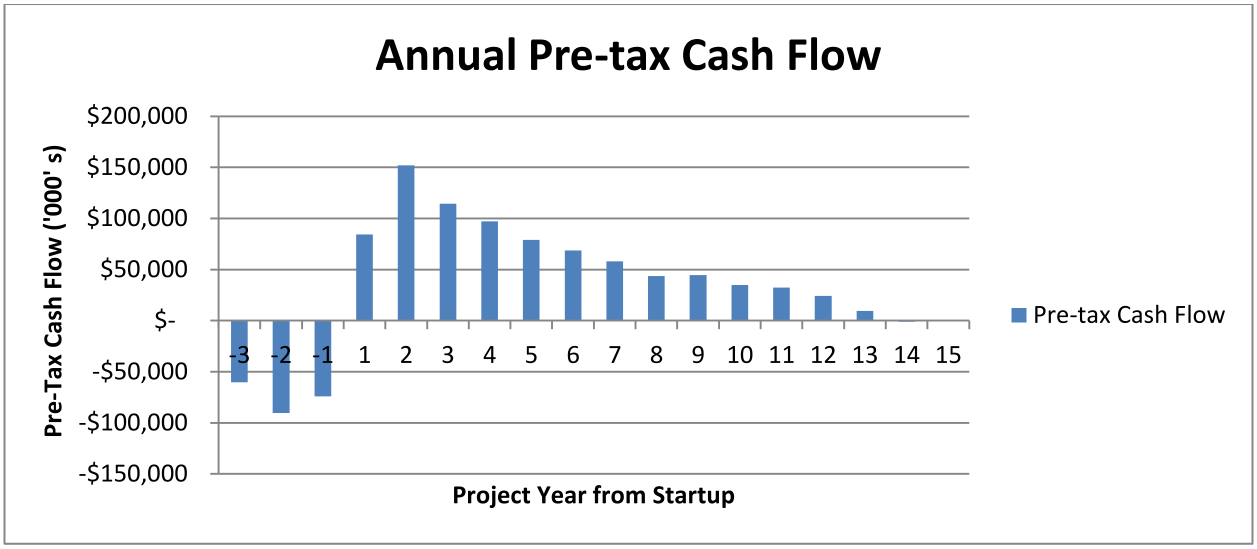

Figure 22.1. Annual Pre-Tax Cash Flow

Presented in Figure 22.2 is the sum of CAPEX costs by project year. Note the large capital expenditure requirements in the early years followed by much smaller sustaining capital requirements in the later years. Negative capital over two years at the cessation of operations is attributable to return of working capital and salvage value recovery from sale of the heavy equipment and other individual heavy equipment items.

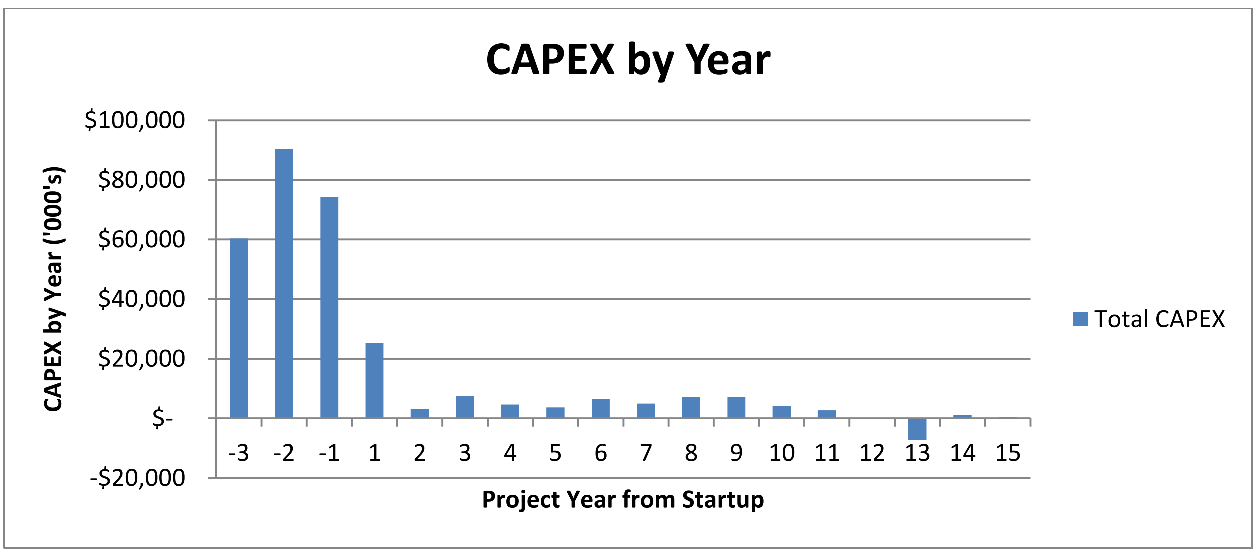

Figure 22.2. Capital Expenditures by Year

Presented in Figure 22.3 is the Operating Cash Flow by Year. As shown, higher cash flows are generated in the earlier years due to the higher grade portions of the deposits being mined as early as possible, resulting in a shorter payback period and larger NPVs.

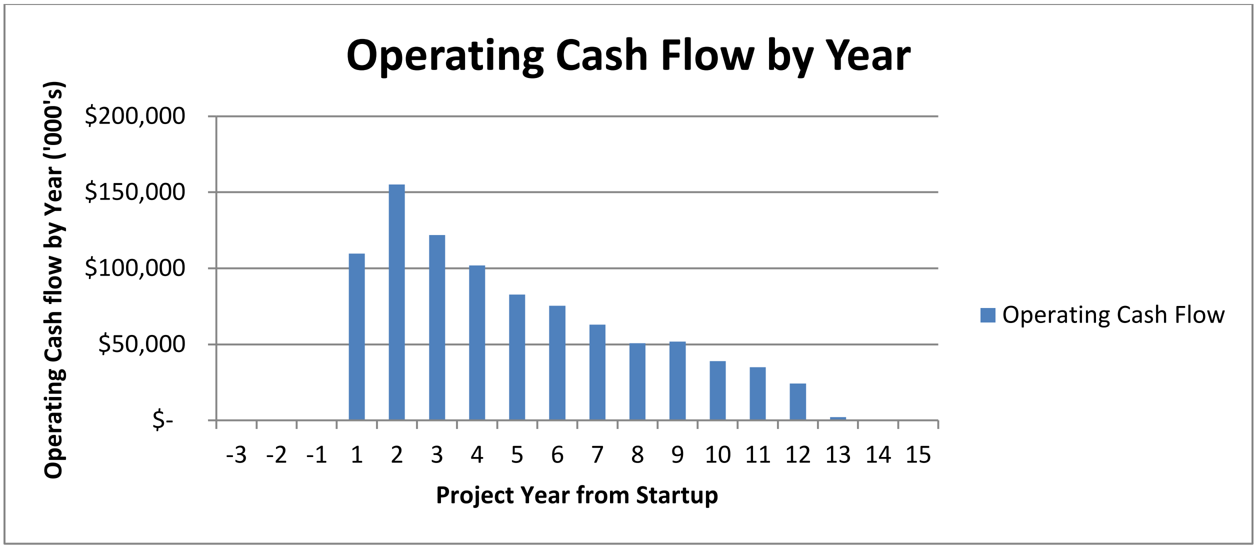

Figure 22.3. Operating Cash Flow by Year

22.4 Sensitivity Analysis

Sensitivity analysis was performed on values for NPV and IRR due to changes in forecasted metals prices, operating costs, and initial capital costs. Values for these three parameters were individually increased and decreased in 10 percent intervals to 30 percent.

Results of these analyses with regard to NPV are presented in Table 22.5. Figure 22.4 depicts this information graphically. These results are also presented for the Project IRR in Table 22.6. Figure 22.5 depicts these results graphically.

Table 22.5. NPV 8 Percent Sensitivity to U_3O_8 Price, OPEX, and Initial CAPEX

U_3O_8 Price	NPV8 (‘000)	OPEX	NPV8 (‘000)	CAPEX	NPV8 (‘000)
	$276,491		$276,491		$276,491
-30%	$49,653	-30%	$365,612	-30%	$334,194
-20%	$125,266	-20%	$335,905	-20%	$314,960
-10%	$200,879	-10%	$306,198	-10%	$295,725
0%	$276,491	0%	$276,491	0%	$276,491
10%	$352,104	10%	$246,784	10%	$257,257
20%	$427,717	20%	$217,078	20%	$238,023
30%	$503,329	30%	$187,371	30%	$218,789

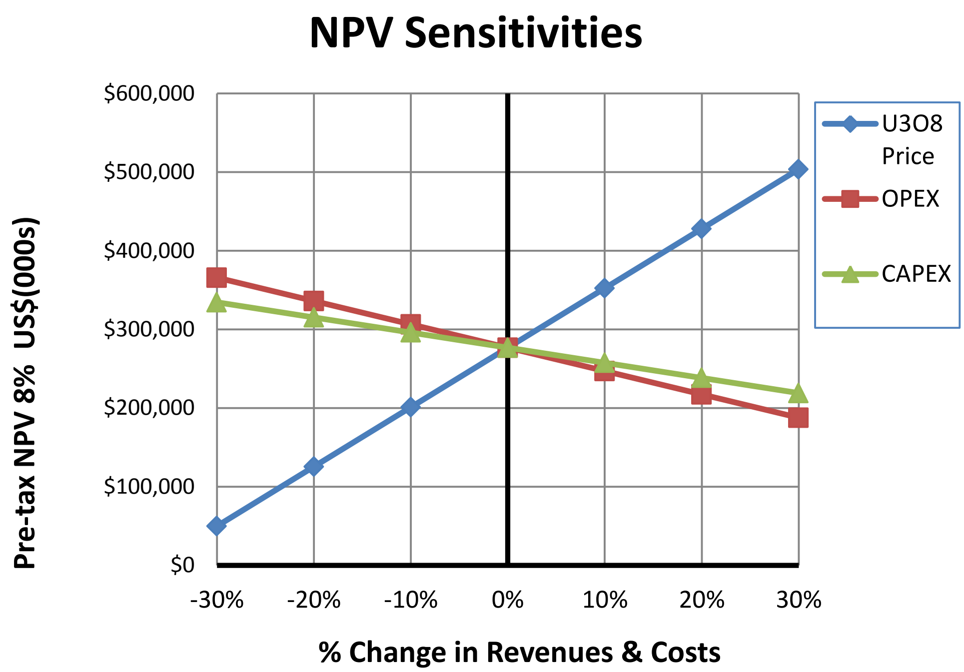

Figure 22.4. NPV Sensitivity to U_3O_8 Price, OPEX, and Initial CAPEX

Table 22.6. IRR Sensitivity to U_3O_8 Price, OPEX, and Initial CAPEX

U_3O_8 Price	NPV8 ('000)	OPEX	NPV8 ('000)	CAPEX	NPV8 ('000)
	30.8%		30.8%		30.8%
-30%	13.3%	-30%	35.8%	-30%	42.9%
-20%	20.0%	-20%	34.2%	-20%	38.1%
-10%	25.7%	-10%	32.5%	-10%	34.1%
0%	30.8%	0%	30.8%	0%	30.8%
10%	35.4%	10%	29.0%	10%	27.8%
20%	39.7%	20%	27.1%	20%	25.3%
30%	43.7%	30%	25.1%	30%	23.0%

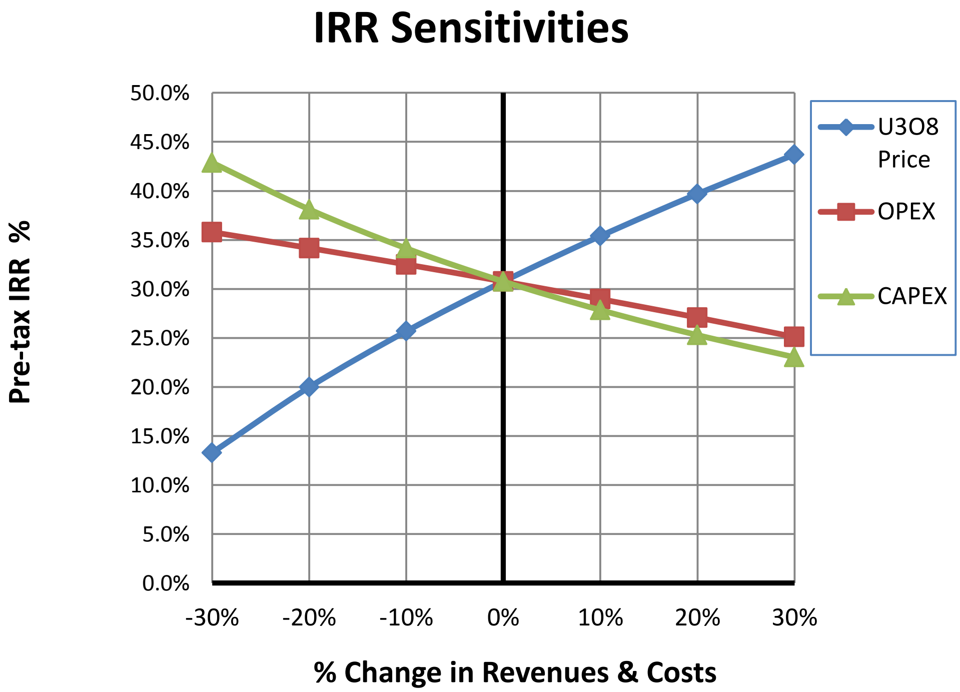

Figure 22.5. IRR Sensitivity to U_3O_8 Price, OPEX, and Initial CAPEX

22.5 Monte Carlo Risk Analysis

Monte Carlo simulation is a system which uses random numbers to measure the effects of uncertainty in a spreadsheet model. This technique performs risk analysis by building models of possible results by substituting a range of values—a probability distribution—for any selected factor that has inherent uncertainty. It then calculates results over and over, each time using a different set of random values from the probability functions. Depending upon the number of uncertainties and the ranges specified for them, a Monte Carlo simulation could involve thousands or tens of thousands of recalculations before it is complete. Monte Carlo simulation produces distributions of possible outcome values.

By using probability distributions, variables can have different probabilities of different outcomes occurring. Probability distributions are a much more realistic way of describing uncertainty in variables of a risk analysis. The basis for this exercise is the Excel file: Kuriskova Proforma CW 120118.xlsx. Probability distributions utilized in the Kuriskova risk analysis are described below and include:

Lognormal – Values are positively skewed, not symmetric like a normal distribution. This distribution is used to represent values that do not go below zero or some pre-selected number, but have unlimited positive potential. The lognormal distribution was applied to uranium price, with the minimum value of US$60/lb, and mean at US$68/lb uranium. This distribution was applied each year of operations.

Uniform – All values have an equal chance of occurring, and the user simply defines the minimum and maximum. Variables which were assigned a uniform distribution were: 1) the uranium grade in the deposit with the variability extending from -10 percent to +10 percent of the mined grade in each year, and 2) the uranium recovery in the process plant which carried a discreet value of 92 percent throughout the mine life and was allowed to vary from 90 percent to 94 percent each year.

Triangular – The user defines the minimum, most likely, and maximum values. Values around the most likely are, of course, more likely to occur, and this point was taken as the single-point figure presented in the Excel spreadsheet for that variable. Those variables that were described by a triangular distribution included: 1) mine production rate, 2) total mine operating costs, 3) unit prices for two major process consumables (power and caustic soda) and, 4) initial capital expenditures during the three pre-production years. The mine production rate was allowed to vary downward by 10 percent from the most likely value expected each year, and upward by 5 percent. Mine operating costs were distributed within a range of -5 percent to +10 percent from the most likely value, and the distribution was applied each year of operation. The two process variable costs also were triangularly distributed from -5 percent to +10 percent of the most likely figure, but these were applied to the base price for the Project; thus, any change in the variables automatically affected the cost throughout the Project life. Initial project capital was also expected to vary from -5 percent to +10 percent in each of the three pre-production years.

Monte Carlo simulation furnishes the decision-maker with a range of possible outcomes and the probabilities they will occur for any choice of action. It shows the extreme possibilities from the worst outcome if all variables in a given trial were the most negative, to the best outcome where just the opposite occurred. A target cell is identified within the Excel spreadsheet which in this case was the NPV of the Project at an 8 percent discount rate. Figure 22.6 presents the results of 2,000 trials as a distribution; also shown on the page are important statistical parameters that suggest a near-normal distribution of the trials. Interesting data suggest a worst-case situation wherein the Project returns an NPV at an 8 percent discount rate of about US$202 million and a best-case scenario with an NPV8 of nearly US$319 million. It is noted that the single-point analysis resulted in an NPV of US$275 million, but under the conditions assumed in this exercise, the median value (50 percent above and 50 percent below) is US$261 million. Table 22.7 illustrates the forecast values by percentile ranges; thus there is a 100 percent chance of achieving an NPV8 of US$202 million, but only a 20 percent probability of attaining or exceeding the base case US$276 million figure presented in the underlying cash flow analysis.

This exercise has assessed the three major factors impacting project economics (i.e., revenues, operating costs, and capital costs). Subjectively, only the more important inputs to the cash flow calculation were chosen for probabilistic distribution, but it is believed that the results would not be significantly different had a more rigorous analysis been performed.

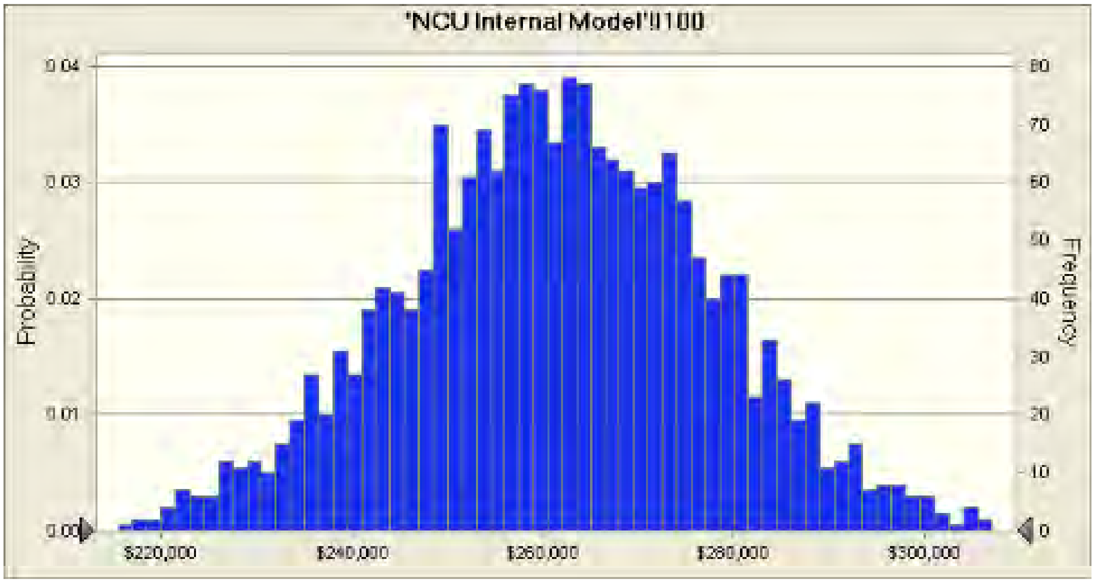

Figure 22.6. NPV Distribution Curve

Table 22.7. Monte Carlo Forecast Analysis

Statistics	Forecast Values
Trials	2,000
Base Case	$276,787
Mean	$261,264
Median	$261,372
Mode	---
Standard Deviation	$16,349
Variance	$267,299,271
Skewness	-0.0534
Kurtosis	2.91
Coefficient of Variability	0.0626
Minimum	$207,390
Maximum	$317,132
Range Width	$109,742
Mean Standard Error	$366
Percentiles	**Forecast Values**
0%	$207,390
10%	$240,259
20%	$247,757
30%	$252,803
40%	$257,227
50%	$261,368
60%	$265,455
70%	$270,224
80%	$274,997
90%	$281,923
100%	$317,132

23.0 ADJACENT PROPERTIES

There are no immediately adjacent properties that have relevance to the Project with respect to geology and/or resources. There are other uranium occurrences in the region, and EUU controls some of them as separate uranium exploration projects. Those exploration properties, and immediately adjacent lands to the Kuriskova exploration license, may have future bearing on the potential development of the Project, from an access/infrastructure or other perspective; however, they have no immediate relevance to the Project.

24.0 OTHER RELEVANT DATA

24.1 Hydrological Studies

24.1.1 Introduction

Water affects the Kuriskova project is two ways. First, it affects the mining operations. The proposed underground mining method will intersect the groundwater and water will report to the underground workings. Specifically, the rate of groundwater inflow anticipated to report into the underground working plays a role in the design of these facilities, the constructions methods, the infrastructure to handle this water, and the associated costs to construct and operate these facilities. Second, water is a natural component of the environment, and as such, how the water interacts with the mine must be considered.

In order to understand these issues, a program was designed to investigate the hydrology and hydrogeology of the site. These studies have been conducted by members of the State Geological Institute of Dionyz Stur (SGUDS), private consultants (such as HES-COMGEO), and staff from Tetra Tech. Analysis of the hydrology and hydrogeology of the site involved collecting and organizing existing data at both the local and regional scale and conducting site-specific investigations. These investigations were designed to evaluate pre-mining baseline conditions and to provide the data and information needed to predict mine-groundwater interactions during and post-mining. This section describes the results of these studies.

24.1.2 Previous Studies

Two key reports contain summaries of the regional and local hydrology and hydrogeology. The first of these is the Environmental Assessment (VODS, 2008). This report contains extensive information on the natural environment (including geology, soils, flora, fauna, surface water, atmosphere and climatic conditions), cultural and natural resources, demographic (including health) data, and a human health impact assessment.

The second report (SGUDS, 2011) was concerned with compiling regional and local hydrologic and hydrogeological data. The study analyzed published information on the geology, hydrology, and hydrogeology of the Project area. This included compiling climate, stream flow, springs, and groundwater well data from the Slovenský hydrometeorologický ústav (Slovak Hydrometeorological Institute, or SHMU). The SGUDS (2011) study was conducted at two scales: regional (based on archival data and less detailed) and local (based on site-specific data and more detailed). Data acquisition also included a field mapping program. The data collected during the first phase of the SGUDS study will be used to further characterize the present conditions and to predict potential future effects and interactions between the mine and the environment This report, to be published in the spring of 2012, will present the results of data evaluation, data interpretation, and numerical modeling.

24.1.3 Climate Data

The best climatological data are from the meteorological station at the Košice airport located south of the town. The Košice meteorological station has over 39 years of high-quality data. A meteorological station was established at the Jahodna Ski area approximately 1,500 meters from the deposit. The station was established on June 18, 2010 and records wind velocity, wind direction, dew point, rainfall, barometric pressure, temperature, and humidity. Another meteorological station is planned to be installed near the location of the proposed ventilation shaft above the deposit.

24.1.4 Surface Water

The only existing surface water monitoring station within the study area is on the Belá River in the village of Košická Belá (SHMU station number 8565) (SGUDS, 2011). Flow rate of the Belá at station Košická Belá in the years 1974 to 2005 ranged from 0.015 to 36.8 m^3/s (VODS, 2008). The seasonal patterns displayed at this station are typical for streams in the area. The source of flow in streams is rain, spring snowmelt and groundwater discharge during the months of February through April. August and September typically shows the lowest flows.

As part of the SGUDS study, surface water flow data were collected from 15 stations within the local study area (Figure 4.6). Monthly flow data are being collected at 10 surface water streams and three springs by hand measurements. Hourly stage data are collected from two monitoring stations equipped with data loggers installed as part of this study: one on the Panská Lúka River, a tributary to the Ĉermeľ Valley east of the deposit and another on the Vrbica River south of the deposit. Data have been collected from these stations between July 2011 and the publication of the SGUDS study in November 2011. It is recommended that flow measurements be continued in all 15 surface water stations through the feasibility study.

The climate between June 2011 and November 2011 was dryer than normal. Additional analyses correlating the stream flow data to the climate record will be completed in the second phase of the SGUDS study due in the spring of 2012. Analysis of the seasonal variation showed that the pattern exhibited in the two instrumented stations mimicked the seasonal pattern exhibited in the Belá River as discussed above (SGUDS, 2011).

24.1.5 Springs

Over 100 springs have been identified near the deposit. As part of the SGUDS study (2011), a detailed map and database of the springs in the local study area was produced. The median discharge of all springs is 0.025 L/s and the average discharge is 0.041 L/s. Spring flow magnitude does not seem to be correlated to rock type or elevation. Springs do seem to be spatially associated with major faults and fractures.

24.1.6 Groundwater

The VODS (2008) study contains a general description of the hydrogeology of the region. They describe the hydrogeology relative to broad terrain divisions. A recurring theme expressed in the study is that the movement and occurrence of groundwater are controlled by the tectonic juxtaposition of rocks of contrasting physical and hydraulic properties and geologic ages. They broadly classify the occurrence of groundwater into five categories. The site itself is hosted in rocks of Paleozoic age. Lithologically, these units consist of metamorphosed shales, wackes, arkoses, conglomerates, meta-basalt tuffs and tuffites, and graphitic and sericitic-chloritic phyllites. Due to the metamorphism, these rocks have lost their original intergranular porosity, so they tend to be aquitards except where fractured and/or faulted.

24.1.7 Existing Borehole Data

The study by SGUDS (2011) concluded that there are no pre-existing boreholes in the local study area. Of all the wells identified by the SGUDS study, the three nearest wells are between 2.2 km and 3.7 km from the deposit. All three of these wells reached their total depth in alluvial or shallow, weathered bedrock. The vast majority of the wells identified in the SGUDS study are located in and around the town of Košice. The SGUDS study (2011) concluded: “Present knowledge about permeability in Paleozoic rock massif in our studied area is generally weak.”

In addition to the wells identified in the SGUDS study, three wells were drilled at the Jahodna ski area in an attempt to develop a reliable supply of water for snowmaking activities (Anonymous, 1991).

24.1.8 2011 Well Installation

In 2011, EUU installed three hydrogeological monitoring wells (LE-K-67, LE-K-68, and LE-K-69, Figures 4.11 to 4.15) to supplement and existing well installed in 2008 (LH-K-16A). The wells installed in 2011 were designed to monitor water levels and to provide the opportunity to sample groundwater for chemical quality. The intention is to install permanent wells for monitoring the system.

24.1.9 Aquifer Characteristics

The SGUDS study briefly discussed the regional hydrogeology, focusing on a regional-scale map of hydrogeological units (Méryová et al., 2005). This map shows that the bulk of the materials around the deposit consist of Permian-aged rhyolite, metatuffs, metatuffites, and metarhyolites with low transmissivity (approximately 1e-6 to 1e-5 m^2/s).

In order to obtain site-specific measurements of aquifer hydraulic properties, the wells drilled in 2011 were tested by HES-COMGEO (2011) using packer tests, short-term pumping tests (slug tests), and long-term pumping test methodologies. Overall, the hydraulic conductivity measured in the well are relatively low with a global median hydraulic conductivity of 3E-8 m/s +/- 1.5 orders of magnitude. There appears to be a crude relationship between hydraulic conductivity and depth, particularly in the upper 200 m of the section. This pattern of vertical decrease in hydraulic conductivity has been noted elsewhere in the Spissko-Gemerske Rudohorie Mountains (Bajtoš, 2007; SGUDS, 2011).

SGUDS (2011) analyzed drilling records from the exploration boreholes for observations of flows. Groundwater inflows were observed in two boreholes and drilling fluids were reported lost in 26 exploration boreholes. In all but six cases, the observed water flows occurred at depths of less than 25 m consistent with the observed relationships of hydraulic conductivity with depth.

Overall, the observed range of hydraulic conductivity falls on the low range of values typically seen in fractured igneous and metamorphic rocks. Based on the observed hydraulic conductivities, the rocks are considered to be aquitards and will not yield water readily.

24.1.10 Water Levels

Equivalent water level pressures have been measured in four transducers in LH-K-16A since October 2008.(Howell and Mayer, 2009). Transducers were deployed in LE-K-68 and LE-K-69 approximately one week prior to the short-term aquifer tests. A data logger was deployed in LE-K-67 to measure the artesian pressure at the surface. Water levels will be collected in these and additional wells to be drilled at the site throughout the life of the mine project.

24.1.11 Groundwater Flow

Tetra Tech prepared a preliminary conceptual map of the groundwater table in the local study area (Figure 24.1). Near the deposit, the water table elevation is based on water levels measured in the three 2011 hydro holes at the time of drilling and the water level in the shallowest transducer in LH-K-16A.

The configuration of contours on the rest of the map is based on the ground topography and a conceptual model of groundwater flow from recharge areas on ridgelines to discharge areas in river valleys.

An estimate of the groundwater velocity and the travel time was done using a Darcys Law approach. Many assumptions were made in order to calculate these values. The results of a 1000-realization Monte Carlo simulation shows that the mean velocities are slow and the mean travel time is approximately 440 years and the median travel time is 185 years. Furthermore, the results show that travel times between the deposit have a 5 percent chance of being less than 22.4 years and a 5 percent chance of being greater than 1,650 years. Given the range of uncertainty expressed in these results, additional data and analysis are warranted prior to issuing the Feasibility Study. Six additional hydrogeological wells are planned to be installed near the deposit. These wells will provide additional information on the nature of the aquifer which will reduce the uncertainty associated with the current estimate. Furthermore, the second part of the SGUDS study, due in the spring of 2012, will present the results of data evaluation, data interpretation, and numerical modeling. A reassessment of groundwater velocity will be made at that time and a decision will be made if additional data and analyses are needed.

Estimates of the rate of inflow into the mine workings were made using an analytical approach. Fourteen analytical models were used in this effort. These solutions each assume slightly different model geometries and boundary conditions, but are all applicable to estimating groundwater inflow into underground workings.

The analytical models predict that on average, approximately 600 L/m may be expected to flow into the working drifts. Dr. Bajtos of SGUDS (personal communication, October 2011) remarked that in his experience, large mines in the region hosted in similar rocks typically produce 10 to 30 liters per second with higher rates of this range associated with shallower mines. This rate represents a relatively small volume of water. Thus, the mine design is assumed to not require a separate, active dewatering system. Instead, the mine design assumes that underground seepage will be collected in underground sumps and mostly used in paste backfill production.

24.1.12 Site-Wide Water Balance

Tetra Tech constructed a site-wide water balance model (SWWB). The primary objectives of the Kuriskova SWWB model include predicting the volume of water sent to the Mine WTP for treatment and predicting the volume of water discharged to the local system.

The sources of inputs to the SWWB model include the mining group, the processing group, the hydrogeologic group, and the WTP planning group. Climate statistics are from the Košice meteorological station historic record.

The model considered four scenarios: low (80 percent) and high (95 percent) efficiency rates of the Mine WTP and low (10 percent) and high (15 percent) tailings moisture content.

The model predicts that the WTP will have to treat approximately 590 to 700 tpd of water depending on the exact value of WTP efficiency and tailing moisture content.

24.1.13 Surface Runoff

Tetra Tech performed preliminary hydrologic and hydraulic analysis of the Project site for the management of surface water run-on to the Project site from the ridge to the east and for on-site storm water management. The primary objectives of the Kuriskova surface water hydrologic modeling include calculating the volume of non-contact-water storm water runoff to be routed around the mine’s surface facility and calculating the volume of contact-water storm water runoff that will report to a storm water pond.

This modeling indicates that the ditches intercepting surface water run-on to the Project site from the ridge to the east would need to have to the capacity to transport between 0.1 and 0.2 m^3/sec, in order to prevent the 100-year storm overland flow from running onto the surface facility site. The stormwater pond would need to have the capacity to contain 14,100 m^3 of water to retain the runoff from the surface facility caused by a 100-year storm.

24.1.14 Conclusions and Recommendations

Rocks in and near the Kuriskova deposit are fractured metavolcanics, metasediments, and sediments. These rocks possess low hydraulic conductivity, even when fractured. Because of this, the mine is not expected to require an active dewatering system. Instead, seepage into underground workings will be handled by sumps within the mine.

The low hydraulic conductivity of the rocks will also impede migration of potential constituents of concern that may be mobilized by underground mining activities.

During the preparation of this PFS, a number of additional data collection efforts with respect to the hydrology and hydrogeology of the site were identified. These are discussed below.

Currently, water levels are being collected by automated transducers from four wells at four-hour intervals. Furthermore, these wells are being sampled quarterly for groundwater quality in conjunction with the surface water sampling program. This program should be continued in order to provide data to characterize the groundwater system prior to, during, and after mining.

An additional six wells near the deposit are recommended to further characterize the hydrogeology of the system and to provide opportunities for ongoing baseline sampling for water quality and water levels prior to, during, and after mining. In addition to these wells, two wells are planned to be installed near the proposed surface facilities in order to provide opportunities for water quality sampling prior to, during, and after mining. These wells should be equipped with automated transducers set to record water levels every four hours. Groundwater quality samples should be collected from these wells quarterly, on the same schedule as the surface water program.

The SGUDS began a detailed study of the hydrology and hydrogeology of the site in late 2011. Phase 1 of this study was designed to collect and organize available data in and near the site. It also included a field program designed to document hydrologic features within a local study area centered on the deposit. As part of this program, surface water flow data are automatically collected hourly from two surface water streams. Monthly surface flows are also manually measured at ten other surface water locations and three springs. It is recommended that monitoring be continued at least through the start of mining at all of these features in order to further characterize the hydrologic system.

A groundwater model will be constructed during the interpretation phase of the SGUDS study. It will cover a relatively small model domain centered over the deposit. Boundaries will consist of no-flow boundaries associated with basin divides or type-3, head dependent flux boundaries along the Ĉermeľ River. The model will have at least two hydrostratigraphic rock types: the surficial saprolite/colluvium and the underlying bedrock consisting of meta-volcaniclastic and metasedimentary rocks. Hydraulic conductivity will be reduced with depth. Major faults and fracture zones will likely be represented as a third rock type based on the equivalent porous media approach. Whether or not smaller-scale, discreet fractures need to be represented remains to be determined.

Spring data were available from three different sources. Duplicates and discrepancies were noted between the three sources. Compiling all of the spring data into a common database is recommended for the feasibility study. Similarly, all borings and wells should be compiled into a common database. The data in these databases should be checked for errors and omissions and should be maintained to be current.

Analysis of the long-term pump test data conducted in LE-K-68 in October 2011 has not been completed at the time of writing this report. Additional single-well and multi-well testing is planned after the new hydrogeological monitoring wells are installed.

A water budget on the regional scale has not been done. Data being collected by SGUDS should provide the information needed to construct a water balance of the hydrologic system. This should be done in time for the feasibility study.

Collection of climate information should continue at the Jahodna weather station. A new station should be established at the deposit and possibly another one at the surface facilities. These two new stations will help refine our understanding of the hydrology of the site.

25.0 INTERPRETATIONS AND CONCLUSIONS

The PFS of the Project indicates that the Project is both technically and economically viable. It appears that the Project could be developed using conventional mining and processing methods. The development of the Kuriskova uranium project could provide Slovakia with a secure source of uranium for approximately 30 years at its current consumption rate. This potential energy source is in line with the Slovakian Strategy of Energy Security. EUU is also encouraged by the extent of uranium exploration and development activities in other European countries including Sweden, Spain and Finland.

25.1 Summary of Results

Tetra Tech has reviewed the Kuriskova resource estimate and believes that it was prepared in accordance with accepted industry standards, sufficient for purposes of the PFS. The current mineral resources at Kuriskova are estimated at 28.5 million lbs of indicated U_3O_8 in 2.3 million tonnes grading 0.555 percent U_3O_8 and 12.7 million lbs of inferred U_3O_8 in 3.1 million tonnes grading 0.185 percent U_3O_8.

Based on estimated indicated mineral resources, mineral reserves were estimated at 2.5 million tonnes and an average grade of 0.346 percent Uranium which was determined to provide an underground mining rate of about 210,000 ore tonnes per year at an economic cutoff of 0.13 percent U_3O_8 for approximately 13 years. No inferred resources were used in reserve calculation or mine plan. The mine plan is based on an underhand drift and fill mining method which utilizes a roadheader as the primary production method and assumed an external dilution (over break) of 5 percent at a grade of 0.03 percent uranium.

Metallurgical test results completed at HRI in Golden, Colorado indicate that uranium and molybdenum recoveries of 92.0 percent and 86.8 percent, respectively, can be achieved using conventional alkaline leaching and precipitation circuits producing separate uranium (yellowcake) and molybdenum concentrates.

The average annual production of uranium as a U_3O_8 concentrate would be approximately 786 tonnes and 84 tonnes of molybdenum in molybdenite with a life-of-mine U_3O_8 production of 20.9 million lbs (9,500 tonnes). Project economics in the base case analysis are based on these figures.

The base case IRR is estimated at 30.8 percent on a pre-tax basis with a 1.9-year payback after the start of production on an estimated initial capital cost of US$225 million including owner's costs and a contingency of US$31 million. At an 8 percent discount rate, the pre-tax NPV is estimated at US$276.4 million.

Total operating costs are estimated to be US$22.98 per lb of U_3O_8 over the mine life and during the first four years of production US$16.68 per lb of U_3O_8. These costs include a byproduct credit for molybdenum of about US$1.27 per lb of U_3O_8. In addition to adding value to the Kuriskova project, molybdenum has been defined as a Class 2 strategic metal by the European Union. During the life of mine there will be sustaining capital requirements of about US$71 million. The operating costs above are exclusive of royalties, which are estimated at US$2.89 per lb of U_3O_8.

Long-term uranium and molybdenum prices of US$68/lb U_3O_8 and US$15/lb molybdenum, respectively, were used in the calculation of the Project economics.

25.2 Potential Opportunities

There are opportunities which may provide improvements and cost savings for the Kuriskova project including the following:

- EUU is planning a surface infill drilling program with the objective of upgrading more of the inferred resource to the indicated category, at a higher grade than the current inferred resource. The 2008 drilling program more than doubled the indicated resource while significantly increasing the grade;
- EUU intends to conduct further step-out exploration drilling where the high-grade mineralization is open along strike and at depth;
- Additional geotechnical and hydrological studies are required to evaluate alternative mine designs and accesses which may improve costs and schedules for construction and mine production.

25.3 Project Improvements since PEA

Project improvements since the publication of the PEA in July 2009 include:

- Shortening of the preproduction construction period by one and one-half years to three years in the PFS from four and one-half years in the PEA;
- Increase in the indicated resources by 39 percent to 28.5 million lbs U_3O_8;
- Increase by 62 percent in the average uranium grade to the process plant from 0.252 percent U_3O_8 to 0.408 percent U_3O_8;
- Increase in the uranium recovery by 2 percent to 92 percent in the PFS from 90 percent in the PEA; and
- Lower LOM operating cost by 26 percent to US$22.98/lb U_3O_8.

25.4 Financial Analysis

The PFS economic evaluation of the Kuriskova project was based on a pre-tax financial model. The following pre-tax financial parameters were calculated based on long-term uranium and molybdenum prices:

- 30.8 percent IRR;
- 1.9-year payback on US$225 million preproduction capital cost including contingency; and
- US$276.4 million NPV at an 8 percent discount rate.

The Project is financially sensitive to the U_3O_8 price and amount of uranium produced per year. This is directly related to the tonnage and grade mined and processed annually as well as to process plant recovery.

26.0 RECOMMENDATIONS

26.1 Mineral Processing and Metallurgical Testing

Tetra Tech recommends the following additional test work to optimize the mineral processing operations:

- Further testwork to optimize reagent concentrations at selected operating conditions.
- Additional testwork evaluating thickening requirements associated with the circuit.
- Testwork to evaluate filtration performance and characteristics of the leach residue.
- Testwork to evaluate the re-precipitation of uranium from re-leached SDU cake so as to provide a higher purity final product.
- Additional testwork evaluating molybdenum recovery methods. This includes generation of a larger mass of molybdenum precipitate to confirm previous direct precipitation results as well as examination of methods not tested such as solvent-extraction.

26.2 Recovery Methods

Additional testwork performed for the feasibility study, as referenced in the metallurgical testing Section 4.1, above, will yield results giving direction to potential process improvements. Overall, the process as defined is robust and is less than likely to require significant upgrades or improvements. The recovery of molybdenum from leach slurry would be the main area in which modifications may be made; again, these being made in response to results from metallurgical testwork.

Tetra Tech recommends the following be considered during the feasibility study phase of work:

- Tests to demonstrate the performance of screens in the grind circuit with resulting reductions to mill size and potential advantages during the leach cycle should be conducted.
- Evaluation of leach slurry filtration techniques should be performed to determine the best filtration equipment and arrangements.
- Evaluation of alternative molybdenum recovery techniques should be performed.
- The use of plate and frame filters for SDU filtration should be considered in comparison to the use of a centrifuge.

26.3 Underground Mining

Tetra Tech has prepared an underground mine plan for the Kuriskova uranium project which included; a mine layout, mine schedule along with the associated operating and capital cost estimates. The Project was designed to achieve a production rate of 600 tpd and sustain that rate for a mine life of 12.5 years based on the probable mineable reserves.

From a mine planning perspective it is recommended to examine the factors which contribute to the cost or mine head grade. The use of roadheader mining machine was proposed for this project. Further test work will be needed to identify the specific requirements for the roadheader, including bit spacing and motor power.

The inclusion of an underground process plant in the mine plan will require more comprehensive geotechnical analysis of the opening to ensure stability.

Special consideration to miner safety must be considered when mining high grade ore. It is advised that a study be conducted to correlate ore grade percent with worker radiation exposure.

26.4 Surface Infrastructure

The recommendations related to the surface infrastructure are provided in the following sub-sections. They cover site location and layout, design and construction approach, power supply, and concentrate shipping and handling.

26.4.1 Site Location

The foundation recommendations for the surface facilities proposed for this study were based on limited subsurface information from five boreholes located across the Project site. Only one of the reference boreholes was located in the immediate vicinity of proposed surface facilities. It is recommended that a more detailed subsurface drilling and sampling program be undertaken for future phases of the Project. A minimum of 14 geotechnical boreholes must be drilled at locations specific to individual surface facilities. Each borehole shall be drilled to a minimum depth of 15 m below ground surface. Additionally, a minimum of four test pits, each 6 m deep (minimum) must be excavated in the vicinity of the proposed surface facilities to characterize surficial soils as borrow materials. The geotechnical boreholes must be advanced in accordance with Standard Penetration Test (SPT) procedures as specified in American Society for Testing and Materials (ASTM) D 1586. A laboratory testing program must be implemented to characterize the geotechnical properties of the samples retrieved from the borehole and test pits.

26.4.2 Site Layout

After further geotechnical investigation, not only the site location, but also the site layout should be better defined. This applies to the access roads, including road base materials, culverts, guard rails, lighting, and turnouts. In addition, the ventilation and egress shaft fuel tank location and its distance from the shaft should be re-evaluated. The topography also needs to be reviewed to address sizing, minimum cover, road crossings, easements, R-O-W issues. In addition, the design of the water supply to the treatment plant at the mine water pond and discharge should be a structure at or in the reservoir.

26.4.3 Design and Construction Approach

For the FS, local contractors should be used to derive the capital and EPCM costs for the Project. The site conditions specification should be updated to including European code references and applications, electrical area classification above and below ground, QA/QC program outline, foundations design basis, ambient conditions, etc. All pre-engineered building sizing should be developed, including interior requirements, HVAC, fire protection, and mine heating design requirements. Knowledge needs to be developed to understand how permits for occupancy are obtained and a better understanding of the liquid oxygen (LOX) supply and requirements needs to be developed.

26.4.4 Power Supply

Electrical designs need to be father developed starting with detailed discussions with the power company about possible substation interfaces and the t-line to the surface facility. In addition, the electrical distribution above and below ground, including grounding, raceway, indoor and outdoor lighting, power and control cabling, fiberoptic, and communications, need to be designed.

26.4.5 Product Transportation

Discussions on product transportation and deliveries will need to commence. Delivery quantities, destinations, security, and pricing need to be identified.

26.5 Environmental and Permitting

The recommendations related to the environmental aspects of the Project are summarized in the following subsection. The client will determine the appropriate time to facilitate permitting discussions with regulatory agencies.

26.5.1 Baseline Studies

The recommendations associated with the baseline are described below.

26.5.1.1 Geochemical Characterization

The following recommendations should be considered as the Project advances to the feasibility and permitting stage:

- Additional mine rock and decline rock samples should be subjected to static testing to substantiate the findings of this study which suggest high neutralization capacity with no potential to generate acid.
- Process tailings and water samples from the optimized process plant flow sheet should be characterized.
- Cemented paste backfill should undergo characterization using passive diffusion testing following ASTM C-1308 (ASTM, 2008). Passive diffusion testing utilizes cemented paste backfill columns immersed in a series of groundwater solutions over time to determine constituent release rates resulting from intact cemented paste backfill.
- Future geochemical characterization should include analysis of natural uranium, Pb-210, Po-210, Ra-226, Th-228, Th-230, and Th-232. This suite may be reduced if testing demonstrates that some radionuclides are not present.
- Once sufficient groundwater quality data is obtained, the leachate quality should be compared to groundwater quality to provide a preliminary assessment of whether mine materials (rock, process tailings, cemented paste backfill) will impact water quality.
- The characterization program focuses on static testing including leachate analysis using SPLP testing. However, it may prove useful to include kinetic testing of mine rock in the program to assist with water quality predictions.

26.5.1.2 Underground Process Tailings Placement

The Project geochemical and hydrogeologic/hydraulic studies should continue with the objective of clearly demonstrating that water quality will not be negatively impacted by underground tailings placement. An assessment of the geochemical characteristics of water that contacts the paste backfill is being conducted to work towards meeting this objective. If the testing shows there is potential for these contact solutions to be elevated in regulated constituents relative to background groundwater quality and/or water quality standards/guidelines than modeling should be conducted to determine if nearby groundwater or surface water quality will be negatively impacted.

26.5.1.3 Water Treatment

Additional data for more detailed evaluation is recommended as development of the WTP proceeds toward design. Of particular importance is silica which can impact the recovery of the RO system. High silica concentrations can polymerize calcium and magnesium in the water and bind off the membranes, rendering them ineffective. Bench scale testing is recommended to prove the water treatment concept. Following proof of concept, pilot testing to prove scalability of the process will be needed before proceeding with treatment.

26.5.1.4 Water Resources

Currently, water levels are being collected by automated transducers from four wells at four-hour intervals. Furthermore, these wells are being sampled quarterly for groundwater quality in conjunction with the surface water sampling program. This program should be continued in order to provide data to characterize the groundwater system prior to, during, and after mining.

An additional six wells near the deposit are recommended to further characterize the hydrogeology of the system and to provide opportunities for ongoing baseline sampling for water quality and water levels prior to, during, and after mining. In addition to these wells, two wells are planned to be installed near the proposed surface facilities in order to provide opportunities for water quality sampling prior to, during, and after mining. These wells should be equipped with automated transducers set to record water levels every four hours. Groundwater quality samples should be collected from these wells quarterly, on the same schedule as the surface water program.

The SGUDS began a detailed study of the hydrology and hydrogeology of the site in late 2011. Phase 1 of this study was designed to collect and organize available data in and near the site. It also included a field program designed to document hydrologic features within a local study area centered on the deposit. As part of this program, surface water flow data are automatically collected hourly from two surface water streams. Monthly surface flows are also manually measured at ten other surface water locations and three springs. It is recommended that monitoring be continued at least through the start of mining at all of these features in order to further characterize the hydrologic system.

A groundwater model will be constructed during the interpretation phase of the SGUDS study. It will cover a relatively small model domain centered over the deposit. Boundaries will consist of no-flow boundaries associated with basin divides or type-3, head dependent flux boundaries along the Ĉermeľ River. The model will have at least two hydrostratigraphic rock types: the surficial saprolite/colluvium and the underlying bedrock consisting of meta-volcaniclastic and metasedimentary rocks. Hydraulic conductivity will be reduced with depth. Major faults and fracture zones will likely be represented as a third rock type based on the equivalent porous media approach. Whether or not smaller-scale, discreet fractures need to be represented remains to be determined.

Spring data were available from three different sources. Duplicates and discrepancies were noted between the three sources. Compiling all of the spring data into a common database is recommended for the feasibility study. Similarly, all borings and wells should be compiled into a common database. The data in these databases should be checked for errors and omissions and should be maintained to be current.

Analysis of the long-term pump test data conducted in LE-K-68 in October 2011 has not been completed at the time of writing this report. Additional single-well and multi-well testing is planned after the new hydrogeological monitoring wells are installed.

A water budget on the regional scale has not been done. Data being collected by SGUDS should provide the information needed to construct a water balance of the hydrologic system. This should be done in time for the feasibility study.

Collection of climate information should continue at the Jahodna weather station. A new station should be established at the deposit and possibly another one at the surface facilities. These two new stations will help refine our understanding of the hydrology of the site.

26.5.1.5 Radiological Monitoring Studies

Groundwater from representative domestic or agricultural use wells within 2 km of the Project proposed operations should be sampled quarterly for the same minimum suite of dissolved radionuclides currently being analyzed. These samples will serve to document the initial groundwater condition in nearby wells prior to initiation of activity with the potential to impact the groundwater. During operations these wells will be monitored to insure that there are no changes attributable to the mining activities.

26.5.1.6 Radon Studies

Preliminary estimates of the radiological dose to the general public from the proposed operation were conducted using MILDOS (ANL, 1989). The model should be updated as part of the feasibility study once sufficient data is obtained from the meteorological tower to be installed near the ventilation shaft.

26.5.1.7 Soils

The following recommendations will help advance the current understanding of the soils in the Project area to support permitting efforts and closure planning:

- Collect and analyze additional surface soil samples to represent the different soil types based on their areal extent and their physiographic position relative to potential project facilities.
- Complete a site-wide soils, closure cover, construction and reclamation material inventory and characterization program to identify material sources, properties and mass balance.
- Complete an erosion and sediment control study.

26.5.2 Reclamation

A more detailed reclamation plan including assessment of salvage values should be developed in support of the feasibility and permitting stage.

26.6 Planned Work Program Costs

The total estimated expenditures over the next two years are estimated by EUU at approximately US$9.6 million. A summary of the estimated costs for the planned work programs is summarized in Table 26.1.

Table 26.1. Planned Work Program Costs Summary

DESCRIPTION	US$ Cost
Preparation of Feasiblity Study	4,250,000
Metallurgical/Environmental Test Work	1,000,000
Drilling - Geotechnical, Metallurgical & Hydrology	3,545,600
Kosice Bela Office Expenses	40,000
Mobile Equipment Purchases	57,000
Hydrology Program	60,000
Geotechnical Test Programs	70,800
Meteorological Stations/Data	203,000
Water Sampling Programs	182,700
Soils Baseline Program	15,000
Fauna/Flora Survey Studies	31,500
EAS Preparation	65,000
Outside Consultants/Translations	75,000
Total Estimated Planned Expenditures	**$9,595,600**

27.0 REFERENCES

Advanced Terra Testing (ATT). "Unconfined Compressive Strength, Pastefill Samples". Advanced Terra Testing, Lakewood Colorado USA, internal report, February 25, 2011.

Advanced Terra Testing. "Tournigan Energy Tri-axial Compressive Strength Test Results". Advanced Terra Testing, Lakewood Colorado USA, internal report, October 10, 2011.

Agapito Associates. "Rock Mechanics Core Testing Results, Pastefill Samples." Agapito Associates, Grand Junction Colorado USA, internal report, January 2011.

Afrouz, Andy A. (1992) Rock Mass Classification Systems and Modes of Ground Failure. CRC Press, Boca Raton Florida.

AMEC, 2008. Kuriskova Targeting Study, May 21, 2008.

Anonymous, 1991. Hydrogeological Exploration Jahodnà 1991. Prepared for Mr. Pampúrik. Report is undated.

ANL, 1998. MILDOS-AREA User's Guide (Draft). Environmental Assessment Division, September.

Argonne National Laboratory (ANL), 1989. MILDOS-AREA: An Enhanced Version of MILDOS for Large-Area Sources, June. ANL/ES-161.

ASTM, 2010. American Society for Testing and Materials Method C143/C143M-10a Standard Test Method for Slump of Hydraulic-Cement Concrete. ASTM, Conshohocken, PA, 4p.

ASTM, 2008. Standard Test Method for Accelerated Leach Test Diffusive Releases from Solidified Mine and a Computer Program to Model Diffusive, Fractional Leaching from Cylindrical Mine Forms. Standard C1308-08, ASTM, Conshohocken, PA, 14p.

Bajtoš, P. 2007. Genesis of Mine Waters in the Spissko-Gemerske Rudohorie Mountains and Evaluation of their Exploitability. PhD Dissertation thesis, Comenius University, Bratislava.

Barfield, B.J, Warner, R.C, and Haan, C.T. 1981. Applied Hydrology and Sedimentology of Disturbed Lands. Oklahoma Technical Press, Stillwater, OK.

Barrick, 2010. New Test Methodology for the Paste Diffusion Testing. Prepared for Nevada Division of Environmental Protection by Barrick Goldstrike Mine, Inc. March 9, 2010.

Beckstead, et all. (1985) Precipitation of Molybdenum Sulfide from Aqueous Solution. Paper presented at the 114th AIME Annual Meeting, New York, NY, February 27.

Beinawski, ZT. Engineering in Rock Mass Classifications. John Wiley & Sons.

Brady, T., Martin, L., and Pakalnis R. Empirical Approaches for Weak Rock Masses. NIOSH, Spokane, Washington, paper on file.

Carney, S., "Czechs Plan Nuclear Push with a $25 Billion Project", Wall Street Journal, October 26, 2011.

Chow, V.T., Maidment, D.R., and Mays, L.W. 1988. Applied Hydrology. McGraw-Hill Companies, Inc. New York, NY.

Commodas Ultrasort (2011) Preliminary Ore Sorting Investigation Benchtop Amenability Test: Radiometric Study for the Kuriskova Uranium Deposit. March 11. Prepared for Tournigan Energy Ltd.

Čurlík, J., Šurina, B. The Guide of Soil Survey and Soil Mapping, SSCRI. Bratislava, 1998.

Daehnke, A., Salamon, M.D,G., and Roberts M.K.C. (2000) Quantifying Stable Hangingwall Spans between Support Units. Journal of the South African Institute of Mining and Metallurgy. October 2000, pp. 14.

DCM Science Laboratory, Inc. Scanning Electron Miscropy/Petrographic Analysis, May 27, 2009, 23 p.

De Souza, E., DeGame D., and Archibald J.F. Minefill Applications, Practices and Trends in Canadian Mines. Queens University, Department of Mining Engineering.

Dominy, S.C., Annels, A. E., Camm, S. G., Wheeler, Barr, S. P., 1997, Geology in the Resource and Reserve Estimation of Narrow Vein Deposits; Exploration and Mining Geology, Vol. 6, #4, October 1997, pp. 317-333.

Esterhuizen, G.S. (2006) Evaluation of the Strength of Slender Pillars. 2006 SME Annual Meeting and Exhibit, March 27-29, St. Louis, Missouri, preprint 06-003. Littleton, CO: Society for Mining, Metallurgy, and Exploration, Inc., March, pp.1-7.

eur-lex.europa.eu/LexUriServ/LexUriServ.do?uri=OJ:L:1998:330:0032:0054:EN:PDF (1998) Council Directive 98/83/EC of 3 November 1998 on the Quality of Water Intended for Human Consumption 23 p.

European Integrated Pollution Prevention and Control Bureau (EIPPCB) (2009) Best Available Techniques Reference Document for the Management of Tailings and Waste-Rock in Mining Activities. ftp://ftp.jrc.es/pub/eippcb/doc/mmr_adopted_0109.pdf

European Parliament and of the Council (2006) Mining Waste Directive. Directive 2006/21/EC of 15 March 2006 on the Management of Waste from Extractive Industries and Amending Directive 2004/35/EC.

Freeze, R. A., Cherry, J. A. (1979) Groundwater. Prentice-Hall, Englewood Cliffs, NJ. 604 pp.

Gherghel, C., De Souza, E., Ventilation Requirements for Uranium Mines, 12th U.S./North American Mine Ventilation Symposium 2008, Wallace (ed), 2008.

Hartman, H. L., 1982, Mine Ventilation and Air Conditioning, Wiley Interscience, 2nd Ed.

Hazen Research Inc. (2011) Additional Batch Alkaline Pressure Oxidation Experiments on Kuriskova Samples. December 19. Prepared for Tournigan Energy Ltd.

Hazen Research Inc. (2011) Metallurgical Evaluation of the Kuriskova Uranium Project. July 26. Prepared for Tournigan Energy Ltd.

Hazen Research Inc. (2012) Recovery of Uranium and Molybdenum from Alkaline Pressure Oxidation Leach Liquors. January 18. Prepared for Tournigan Energy Ltd.

HES-COMGEO, 2011. Report and Evaluation of Hydrodynamic (Aquifer) Tests. December 2011. 16 pp.

Howell, R., Mayer, X. 2009. VBW Piezometer Installation. Memorandum to Al Kuestermeyer, Rich Eliason, Jozef Cisovsky, and Boris Bartalsky, Tournigan Energy. February 25, 2009.

INGEO-ighp, 2012. Soil Survey of the Tournigan Kuriskova Project. INGEO-ighp, s. r.o., Bytčická 16, 010 01 Žilina, January 2012.

International Atomic Energy Agency (IAEA). 2002a. Monitoring and Surveillance of the Residues from the Mining and Milling of Uranium and Thorium. Safety Reports Series No. 27. International Atomic Energy Agency, Vienna.

International Atomic Energy Agency (IAEA). 2002b Management of Radioactive Waste from the Mining and Milling of Ores. Safety Guide No. WS-G-1.2. International Atomic Energy Agency, Vienna.

Janis, Stanislav. "Results from Laboratory Testing Rock Mechanics". Ingeo Envilab, s.r.o., Žilina Slovakia, internal report, October 20, 2011.

Johnson, T., "Global Uranium Supply and Demand", Counsel on Foreign Relations, Inc., updated January 14, 2010.

Jurčová, O., 2011. Pedological Survey of the Exploratory Territory (Search Report).

Kopecký, J., October 1993, MEGA, Join-stock company, Stráž pod Ralskem, Czech Republic, Technological Research of U-Mo ore from Jahodná Site.

Kuestermeyer, A.L., (1984) Capital and operating costs estimation for milling of uranium ores in the United States.

Merritt, Robert C. (1971) The Extractive Metallurgy of Uranium, Prepared for United States Atomic Energy Commission.

Méryová, E., Frličková, M., Urbaník, J., Méry, V., 2005: Hydrogeological map of southern part of the Spissko-gemerske rudohorie Mts. In scale 1:50 000. Manuscript. Archive - Geofond, SGUDS Bratislava.

Miesfera Consult, 2011. Community Legislation in the Field of Water System, Applicable Limits in Relation to the Discharge of Sewage into the Environment from Mining the Minerals. Prepared for Ludvika Energy by Miesfera Consult. May 2011.

Mitchell, G. W., Greeff, P. W., 1981, Underground Environmental Considerations in Planning Large Uranium Mine Employing Sublevel Stoping Techniques; Design and Operation of Caving Sublevel Stoping Mines, Society of Mining Engineers.

MKSP, 2000. The Morphogenetical Soil System in Slovak Republic, Soil Science and Conservation Research (SSCRI) , Bratislava, 2000, Sobotská, J. (editor).

Moore, S. C., Kolb, S. G., McLean, C., Erickson, A. J., 1992, SME, Applied Mining Geology; General Studies, Problems of Sampling & Grade Control in Ore Reserve Estimation, pp. 129-145.

Moran, A., Daviess, F., Sharma, R., April 16, 2009, Updated NI 43-101 Technical Report on Resources, Kuriskova Uranium Project, Eastern Slovakia.

Moran, A., Daviess, F., Bowell, R., More, S., Sharma, R., Bartalsky, B., Novotny, L., Cisovsky, J., and Pomorsky, F., SRK Consulting, August 29, 2008, NI 43-101 Technical Report on Resources, Kuriskova Uranium Project, Eastern Slovakia.

PAH, July 23, 2009, Preliminary Assessment – Kuriskova Uranium Project, Eastern Slovakia.

Pelham, D., White, G., Bennett, J., and Balakrishnan, R., Technical Report for Preliminary Assessment of the Jahodna Uranium Project, Slovakia, A.C.A. Howe International Ltd., May 11, 2006.

Resource Development Inc. (2009) Scoping metallurgical testing of Kuriskova Uranium Samples From Eastern Slovakia. November 2. Prepared for Tournigan Energy Ltd. Sitasova, Eva, 2009. Kuriskova Project Botanical Survey. Prepared for Tournigan Energy Ltd. Undated.

Schimid, S., et al, The Alpine-Carpathians-Dinaridic orogenic system: correlation and evolution of tectonic units, Swiss J. Geosci. 101 pp 139-186, 2008

Sobek, A.A., W.A. Schuller, J.R. Freeman, and R.M. Smith, 1978. Field and laboratory methods applicable to overburden and mine soils. EPA. Report EPA-600/2-78-054

Souza, E.M.de and Archibald, J.F. Rock Classification as an influence in Mine Design Operations. Mining Science and Technology, Vol. 6, 8 pp.

SRK memo to Mike Mracke, et al., Site Visit Report and Geotechnical Logging Manual, August 2008.

State Geological Institute of Dionyz Stur (SGUDS) (2011) Basic Evaluation of Hydrogeology at U-Mo Deposit Košice I (Kuriskova). December 2011. 37 pp.

Tatum, C., member SME, October 2001, Production-Rate Selection for Steeply Dipping Tabular Deposits.

Tetra Tech, 2010a. Kuriskova Geochemical Characterization Program-Interim Report. Prepared for Tournigan Energy Ltd. by Tetra Tech, September 2010.

Tetra Tech, 2010b. Kuriskova Geochemical Characterization Program-Quarry Rock. Prepared for Tournigan Energy Ltd. by Tetra Tech, December 2010.

Tetra Tech, 2011a. Technical Memorandum-Hydrogeologic Characterization: Data Recommendations. Prepared for Tournigan Energy Ltd. January 21, 2011.

Tetra Tech, 2011b. Kuriskova Geochemical Technical Memorandum-Overview of Cemented Tailings. Prepared for Tournigan Energy Ltd. by Tetra Tech, October 2011.

Tetra Tech, 2012. Tournigan Kuriskova Project Preliminary Feasibility Study. Prepared for Tournigan Energy Ltd. by Tetra Tech, February 2012.

Unconsulting, 2011. “Uranium Market Outlook, Quarterly Market Report”, Q3 2011.

Uranium Market Outlook, 2009. Quarterly Market Report Q2 2009 – Executive Summary, published.

U.S. Energy Information Administration, 2011. “Today in Energy”, July 15, 2011.

U.S. Energy Information Administration, 2011. “International energy Outlook 2011”, September 19, 2011.

U.S. Energy Information Administration, 2012. 4th Quarter 2011 Domestic Uranium Production Report, February 2012.

U.S. Army Corps of Engineers (USACE), 2010. Hydraulic Engineering Center’s – Hydrologic Modeling System (HEC-HMS). Version 3.5. June 30, 2010.

U.S. Code of Federal Regulations. Title 40 Protection of the Environment - Part 190 - Environmental Radiation Protection Standards for Nuclear Power Operations.

U.S. Code of Federal Regulations. Title 10 Energy- Part 2- Standards for Protection Against Radiation.

U.S. Nuclear Regulatory Commission (NRC), 1980. Radiological Effluent and Environmental Monitoring at Uranium Mills. Regulatory Guide 4.14. Office of Standard Development. NRC. Rockville, MD.

NRC, 1987. Methods for Estimating Radioactive and Toxic Airborne Source Terms for Uranium Milling Operations. Regulatory Guide - 3.59. NRC Division of Fuel Cycle Safety and Safeguards, Office of Nuclear Material Safety and Safeguards, Washington, DC 20555-0001 Division of Fuel Cycle Safety and Safeguards.

VODS, Uranpress, and Environment, 2008. Data for Environmental Assessment of the Location Košice I. Prepared for Ludovika Energy s.r.o. Report dated October 2008. 322 pp.

Wolkersdorfer, C., and Bowell, R., 2005. Contemporary Reviews of Mine Water Studies in Europe, Mine Water and the Environment 24: Supplementary Material c Springer Verlag 2005, specifically the section entitled “Mine Water Issues in Slovakia”, pp. 34-37, by Peter Bajotos.

World Nuclear Association, 2011. “Nuclear Power in the World Today”, updated February 2011.

World Nuclear Association, 2011. “Uranium Production Figures, 2000-2010, April 2011.

World Nuclear Association, 2011. “Plans for New Reactors Worldwide”, updated August 2011.

World Nuclear Association, 2011. “Supply of Uranium”, updated September 2011.

World Nuclear Association, 2011. “World Uranium Mining”, updated December 2011.

World Nuclear Association, 2011. “The Economics of Nuclear Power”, updated December 2011.

www.accuval.net/insights/industryinsights, 2008. Uranium supply and Demand Expected to Increase.

www.cameco.com/investors/uranium_prices_and_spot_price/spot_price_5yr history/, 2011. Uranium Price. February 2011.

www.eia.gov/uranium/production/annual/html/ussummary.html (1993-2010) Summary Production Statistics of the U.S. Uranium Industry, 1993-2010."

www.uraniumblog.com (2012) "U.S. EIA Forecasts Nuclear Demand to Outpace Growth."

www.windpowerengineering.com/policy/environmental World Nuclear Power Output Slowing.

www.who.int/water_sanitation_health/dwq/qdwq3rev/en (2006) Guidelines for Drinking Water Quality. Third Edition.

www.world-nuclear.org/info/inf91.html (2011) Nuclear Power in Slovakia.

www.world-nuclear.org/info/inf22.html (2010) Uranium Markets.

TETRA TECH

CERTIFICATE OF QUALIFIED PERSON

I, Rex Clair Bryan, Ph.D., of Golden, Colorado do hereby certify:

- I am a Senior Geostatistician with Tetra Tech MM, Inc. with a business address at 350 Indiana Street, Suite 350, Golden, Colorado 80401.
- This certificate applies to the technical report entitled Preliminary Feasibility Study of Kuriskova Uranium Project - East Central Slovakia, dated March 13, 2012 (the "Technical Report").
- I am a graduate of Michigan State University - East Lansing, (BS in Engineering degree with honors, 1971 and an MBA degree, 1973), Brown University in Providence, Rhode Island (MS degree in Geology, 1977) and Colorado School of Mines in Golden, Colorado (Ph.D. in Mineral Economics, 1980). I am a Registered Member in good standing (#411340) of the Society for Mining, Metallurgy, and Exploration, Inc. (SME). My relevant experience is that I have worked as a resource estimator and geostatistician for a total of thirty-one years since my graduation from university; I have been an employee of a leading geostatistical consulting company (Geostat Systems, Inc. USA), with large engineering companies such as Dames and Moore, URS, and Tetra Tech and as a consultant for more than 30 years. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was August 22nd through August 24th, 2011.
- I am responsible for Sections 4.0 through 12.0, Section 14.0 an Section 23.0 of the Technical Report. I share responsibility with others for Sections 1.0 through 3.0 and Sections 25.0 through 27.0.
- I am independent of Tournigan Energy Ltd as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the parts of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the parts of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this eighth day of March, 2012 at Golden, Colorado.

Rex Clair Bryan, Ph.D.
Senior Geostatistician
Tetra Tech MM, Inc.

TETRA TECH

CERTIFICATE OF QUALIFIED PERSON

I, Richard W. Jolk, P.E., PhD, of Golden, Colorado, do hereby certify:

- I am a Principal Mine and Metallurgical Engineer, Certified Minerals Appraiser with Tetra Tech MM, Inc. with a business address at 350 Indiana Street, Suite 350, Golden, Colorado 80401, USA.
- This certificate applies to the technical report entitled Preliminary Feasibility Study of Kuriskova Uranium Project - East Central Slovakia, dated March 13, 2012 (the "Technical Report").
- I am a graduate of the Colorado School of Mines, (Bachelor of Science degree in Metallurgical Engineering, 1978 – Master of Science degree in Mine Engineering, 1986 – Master of Science degree in Environmental Engineering , 1993 – Doctorate in Mine Engineering specializing in Process Engineering Optimization, 2007). I am a licensed Professional Engineer in good standing in the State of Colorado, license number 24448. My relevant experience includes working for over 10 years with operators in mine and mineral processing operations, 12 years with engineering firms in project valuation, design, engineering, construction and commissioning, and 11 years as an independent minerals industry consultant. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- I did not personally inspect the Tournigan Kuriskova Project Site.
- I am responsible for Sections 13.0, 17.0, 20.0, and Section 22.0 of the Technical Report. I share responsibility with others for Sections 1.0 through 3.0, 21.0, and Sections 25.0 through 27.0.
- I am independent of Tournigan Energy Ltd as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the parts of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the parts of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this eighth day of March, 2012 at Golden, Colorado.

Richard W. Jolk, P.E., PhD
Principal Mine and Metallurgical Engineer, Certified Minerals Appraiser
Tetra Tech MM, Inc.

TETRA TECH

CERTIFICATE OF QUALIFIED PERSON

I, Andrew P. Schissler, P.E., PhD of Golden, Colorado, do hereby certify:

- I am a Principal Mine Engineer with Tetra Tech MM, Inc. with a business address at 350 Indiana Street, Suite 350, Golden, Colorado 80401, USA.
- This certificate applies to the technical report entitled Preliminary Feasibility Study of Kuriskova Uranium Project - East Central Slovakia, dated March 13, 2012 (the "Technical Report").
- I am a graduate of the Colorado School of Mines, (Bachelor of Science degree in Mining Engineering, 1975 – Doctorate in Mining and Earth Systems Engineering, 2002). I am a Founding Registered Member in good standing of the Society of Mining Engineers (#2849600). I am a licensed Professional Engineer in good standing in the States of Colorado, license number 17362 and Utah, license number 158682-2202. My relevant experience is that I have worked as a Mining Engineer for thirty four years, a Senior Lecturer at Colorado School of Mines for 1 year in Mining Engineering, and an Assistant Professor, Member Graduate Faculty for 3 years at Pennsylvania State University in Mining Engineering. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was on July 8 to 13, 2012 for 6 days.
- I am responsible for Sections 15.0, 16.0 and 24.0 of the Technical Report. I share responsibility with others for Sections 1.0 through 3.0 and Sections 25.0 through 27.0.
- I am independent of Tournigan Energy Ltd as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the parts of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the parts of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this eighth day of March, 2012 at Golden, Colorado.

Andrew P. Schissler, P.E., PhD
Principal Mine Engineer
Tetra Tech MM, Inc.

CERTIFICATE OF QUALIFIED PERSON

I, Landy Stinnett, PE, ASA, of Lakewood, Colorado, do hereby certify:

- I am a Principal with FGM Consulting Group, Inc. with a business address at P.O. Box 1438, Golden, Colorado.
- This certificate applies to the technical report entitled Preliminary Feasibility Study of Kuriskova Uranium Project - East Central Slovakia, dated March 13, 2012 (the "Technical Report").
- I am a graduate of the South Dakota School of Mines and Technology with a Bachelor of Science degree in Geological Engineering in 1959, and with a Master of Science degree in Geological Engineering in 1963; and I graduated from the University of Minnesota with a Master of Science degree in Mining engineering in 1967.
- I am a registered professional engineer in Wyoming (License #4502), and am a Registered Member of SME (#3104700). I am an Accredited Senior Appraiser (Mines & Quarries) with the American Society of Appraisers (#9274). My relevant experience is largely in the areas of reserve estimation, mineral appraisals, and mine economic analysis. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- I have not been to the Kuriskova site.
- I am responsible for Section 19.0 of the Technical Report.
- I am independent of Tournigan Energy Ltd. as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the portion of the Technical Report for which I am responsible has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the portion 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.

Signed and dated this thirteenth day of March, 2012 at Golden, Colorado.

Landy Stinnett

Landy Stinnett, PE, ASA
Principal
FGM Consulting Group, Inc.

TETRA TECH

CERTIFICATE OF QUALIFIED PERSON

I, Scott Voltura, P.E., of Golden, Colorado, do hereby certify:

- I am a Project Engineer with Tetra Tech MM, Inc. with a business address at 350 Indiana Street, Suite 350, Golden, Colorado 80401, USA.
- This certificate applies to the technical report entitled Preliminary Feasibility Study of Kuriskova Uranium Project - East Central Slovakia, dated March 13, 2012 (the "Technical Report").
- I am a graduate of the Colorado School of Mines, (Bs Engineering, 1991). I am a Professional Engineer in good standing in the states of Colorado, license number 32667 and Arizona, license number 31734. My relevant experience is that I have worked in the mining industry for 20 years as a Project Engineer, Project Manager, and Engineering Manager. In addition, I have held positions maintenance and operations management. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- I did not personally inspect the Tournigan Kuriskova Project Site.
- I am responsible for Section 18.0, of the Technical Report. I share responsibility with others for Sections 1.0 through 3.0 and Sections 25.0 through 27.0.
- I am independent of Tournigan Energy Ltd as defined by Section 1.5 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the parts of the Technical Report that I am responsible for have been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the parts of the Technical Report that I am responsible for contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this eighth day of March, 2012 at Golden, Colorado.

Scott A. V

Scott A Voltura, P.E.
Project Engineer
Tetra Tech MM, Inc.

APPENDIX A
CLASSIFIED RESOURCES ESTIMATES

Table A.1. Summary of Classified Resources

Geology Domain	Sub-Domain	Model Zone	% U	Tonnes ('000)	% U_3O_8	U_3O_8 ('000 lbs)	% Mo	Tonnes ('000)	Mo ('000 lbs)	Current Resource Update (Year)	Previous Resource Update (Year)
Indicated Resources											
Main Zone	ZONE1N (Main Zone North)	1	0.507	1,790	0.598	23,601	0.056	1,790	2,210	2011	2010
	UP MAIN ZONE	1.2	0.211	54	0.248	296	0.033	54	39	2010	2008
	ZONE1S (Main Zone South)	1.1	0.339	207	0.400	1,824	0.073	207	333	2011	2009
Hanging Wall North	ZONE2N(43) (HW North)	2	0.279	109	0.329	791	0.016	82	29	2011	2010
	ZONE3N(44) (HW North)	3	0.403	99	0.475	1,037	0.025	99	55	2011	2010
Zone 45	ZONE45 (NEW ZONE)	5	0.523	69	0.617	938	0.425	69	647	2011	2010
Main Zone total indicated		1+1.1+1.2	0.482	2,051	0.569	25,721	0.057	2,051	2,582		
Zone 45 total indicated		5	0.523	69	0.617	938	0.425	69	647		
HW north total indicated		2+3	0.338	208	0.399	1,828	0.021	181	83		
Total Indicated (All Domains)			0.471	2,328	0.555	28,487	0.065	2,301	3,312		
Inferred Resources											
Main	ZONE1N Main Zone North)	1	0.194	490	0.229	2,471	0.017	490	184	2011	2010
	UP MAIN ZONE	1.2	0	0						2010	2008
	ZONE1S (Main Zone South)	1.1	0.156	1,641	0.184	6,655	0.024	1,612	853	2011	2009
H.W. Andesite	ZONE2N(43) (HW North)	2	0.215	130	0.254	727	0.024	110	58	2011	2010
	ZONE3N(44) (HW North)	3	0.153	230	0.180	915	0.047	185	192	2011	2010
	ZONE 4 (HW North)	4	0.095	52	0.112	128	0.071	52	81	2010	2008
	ZONE2S (HW South)	2.1	0.087	181	0.103	409	0.003	181	12	2008	2008
	ZONE3S (HW South)	3.1	0.106	336	0.125	926	0.024	288	155	2008	2008
Zone 45	ZONE 45 (NEW ZONE)	5	0.426	39	0.502	432	0.378	39	325	2011	2010
Main Zone Total Inferred		1+1.1+1.2	0.165	2,131	0.194	9,127	0.022	2,102	1,037		
H.W. Zone Total Inferred		2+3+4+2.1+3.1	0.129	929	0.152	3,105	0.044	855	823		
Zone 45 Total Inferred		5	0.426	39	0.502	432	0.378	39	325		
Total Inferred (All Domains)			0.157	3,099	0.185	12,664	0.033	2,996	2,185		

1. In situ uranium resources refers to global in-place resources to which a mine design has not yet been applied; although, the above stated resources meet the definition of having the "potential for economic extraction" at the cutoff provided.
2. CIM compliant resource classification using industry standard block modeling techniques by EUU and validated by Tetra Tech.
3. Bulk density of 2.75 used for all rock types.

APPENDIX B
DRILLING TABLES

Table B.1. Drill Hole Collar Data 1990 to June 2008
European Uranium Resources Ltd. – Kuriskova Uranium Project

Hole ID	Northing	Easting	Elevation (amsl)	Dip (deg)	Azimuth (deg)	Depth (m)	Year
992	-1234199.06	-270390.03	590.5	89.3	25	478	1990
1179	-1234432.75	-270395.3	589.97	85.5	25	559.9	1990
1180	-1234142.74	-270593.21	571.38	90	29	577	1990
1181	-1234113.37	-270426.76	576.91	79	75	391.5	1990
1182	-1234049.28	-270463.45	568.07	76.6	53	403	1990
1215	-1234114.44	-270430.49	576.51	86.5	45	449.2	1990
1216	-1234051.86	-271030.12	610.48	87.3	59	278.9	1990
1217	-1234370.15	-271028.49	557.03	86.2	58	396.1	1990
1218	-1234081.17	-270494.99	566.28	90	68	417	1990
1219	-1234427.53	-270391.27	589.4	90	134	306.4	1990
1220	-1234360.28	-270263.07	594.2	75	336	455	1990
1221	-1233729.08	-270668.19	504.32	86.4	37	354.6	1990
1222	-1234084.32	-270496.03	566.95	78.9	50	382.3	1990
1223	-1234144.29	-270590.99	571.57	87.8	148	578	1990
1225	-1234382.09	-270202.03	593.03	89.5	68	444	1990
1226	-1234462.23	-270135.47	563.29	90	60	244	1990
1227	-1234531.38	-270197.53	569.84	89.6	254	466.6	1990
1233	-1234404.37	-270573.31	610.87	87.9	186	791.6	1990
1234	-1234330.74	-270719	619.03	88.6	144	792	1990
1235	-1234125.78	-270294.13	608.72	77.5	19	246.9	1990
1239	-1233976.8	-270527.65	554.19	77.5	68	352.2	1990
1242	-1234051.33	-270460.58	567.94	72.6	44	319	1990
1245	-1234176.89	-270239.75	605.14	82.4	50	378.3	1990
1246	-1234474.18	-270760.11	616.95	89.3	198	956	1990
1247	-1234356.92	-270260.29	594.79	74.1	338	447.7	1990
1248	-1234114.82	-270429.25	576.77	85.9	296	412.1	1990
1215-1	-1234114.44	-270430.49	576.51	86.5	330	368.5	1990
KG-J-1	-1234093.73	-270513.97	565.57	85	67	440.4	2005
KG-J-1A	-1234092.02	-270512.46	565.67	88.5	46	444.1	2005
KG-J-2	-1234165.13	-270473.3	575.41	88.17	40	480.4	2005
KG-J-4	-1234161.87	-270572.11	571.55	90	85	596.3	2005
KG-J-10	-1234343.06	-270270.77	595.62	89	75	411.5	2006
KG-J-11	-1234000.26	-270702.46	561	88	68	474.4	2006
KG-J-12	-1234475.87	-270259.97	577.99	88	100	429.5	2006
KG-J-13	-1234324.99	-270171.62	597.45	88	75	275	2006
KG-J-14	-1234262.72	-270211.95	608.6	90	60	330	2006
KG-J-15	-1234574.69	-270132.06	540.07	88	57	286	2006

Hole ID	Northing	Easting	Elevation (amsl)	Dip (deg)	Azimuth (deg)	Depth (m)	Year
KG-J-15A	-1234579.29	-270133.51	539.95	87	135	153	2006
KG-J-17	-1233449.47	-270793.77	562.23	88.09	70	298.2	2006
KG-J-3	-1234297.37	-270321.04	598.82	88.17	75	426.3	2006
KG-J-5	-1234104.76	-270660.14	567.1	88	57	513.1	2006
KG-J-6	-1234041.31	-270606.1	555.4	87.83	86	433	2006
KG-J-7	-1234219.3	-270525.65	578.46	88.5	65	556.9	2006
KG-J-8	-1234291.95	-270489.73	586.68	90	123	525	2006
KG-J-9	-1234353.17	-270410.09	590.61	88.17	42	522.3	2006
KG-J-19A	-1234268.36	-270218.63	609.92	82	65	300.8	2007
KG-J-19B	-1234268.11	-270217.45	609.86	75	65	228.3	2007
KG-J-20A	-1234237.59	-270166.81	595.55	89.17	72	225	2007
KG-J-20B	-1234227.09	-270164.21	594.85	80	60	171	2007
KG-J-21A	-1234198.63	-270261.52	611.91	88	65	372.7	2007
KG-J-21B	-1234198.16	-270260.39	611.79	82	65	337	2007
KG-J-21C	-1234198.26	-270262.75	611.87	85	90	352.7	2007
KG-J-21D	-1234197.59	-270260.88	611.87	85	0	349	2007
KG-J-22A	-1234185.3	-270190.66	600.9	90	320	175	2007
KG-J-23A	-1234170.45	-270221.25	604.44	88	65	195.5	2007
KG-J-24A	-1234221.93	-270316.48	599.06	75	65	355.2	2007
KG-J-24B	-1234222.62	-270317.6	598.87	85	65	367	2007
KG-J-25A	-1234127.3	-270299.76	609.21	88	55	343.7	2007
KG-J-25B	-1234127.51	-270300.09	609.22	82	55	337	2007
KG-J-25C	-1234126.91	-270303.71	609.46	85.2	335	375.7	2007
KG-J-26A	-1234325.39	-270184.86	600.14	88	65	304	2007
KG-J-26B	-1234325.29	-270183.9	600.03	78.7	65.3	246.4	2007
KG-J-26C	-1234325.02	-270183.77	600.02	68.3	66	207.3	2007
KG-J-27A	-1234166.98	-270279.02	612.12	86.2	52.8	341.3	2007
KG-J-27B	-1234167.95	-270279.32	612.2	89.4	261	389	2007
KG-J-28A	-1234295.67	-270205.36	608.44	89.7	40.2	374.6	2007
KG-J-28B	-1234295.56	-270204.8	608.28	85.7	65.6	305	2007
KG-J-28C	-1234295.62	-270204.34	608.32	81.6	72.4	257.2	2007
KG-J-29A	-1234345.85	-270548.24	610.35	89.7	109.2	505	2007
KG-J-29B	-1234343.03	-270548.88	610.21	87	251	282	2007
KG-J-30A	-1234381.2	-270478.87	596.02	89.6	67.6	492.3	2007
KG-J-30B	-1234392.55	-270458.49	593.87	89.2	11.3	650.8	2007
KG-J-31A	-1234156.37	-270451.67	577.21	80	58.2	432	2007
KG-J-31B	-1234163.3	-270453.42	577.51	75	91	436.3	2007
KG-J-32A	-1234253.25	-270293.37	601.63	88.4	88.8	421.5	2007
LH-K-1A	-1234280.99	-270265.34	603.61	85	67	386.1	2008

Hole ID	Northing	Easting	Elevation (amsl)	Dip (deg)	Azimuth (deg)	Depth (m)	Year
LH-K-1B	-1234281.44	-270264.72	603.64	75	67	306.3	2008
LH-K-2A	-1234252.94	-270292.28	601.63	84	65	401	2008
LH-K-2B	-1234253.2	-270292.05	601.62	78	64	315	2008
LH-K-2C	-1234253.08	-270291.81	601.66	72	60	312	2008
LH-K-3A	-1234295.56	-270198.52	607.35	56	64	223	2008
LH-K-3B	-1234296.36	-270200.28	607.66	68	65	226.1	2008
LH-K-4A	-1234272.41	-270326.47	598.48	87	60	414	2008

Table B.2. Significant Kuriskova Mineralized Intercepts 1990 to June 2008
European Uranium Resources Ltd. – Kuriskova Uranium Project

Hole ID	From	To	Interval	% U	% U_3O_8	Year
LH-K-1A	360.1	363.3	3.2	0.228	0.269	2008
LH-K-1B	278.3	281.4	3.1	0.772	0.910	2008
LH-K-2A	377.7	378.2	0.5	0.065	0.076	2008
LH-K-2B	287.2	289.5	1.6	0.390	0.460	2008
LH-K-2C	271.7	273.7	2	0.066	0.078	2008
LH-K-3B	200.6	201	0.4	0.132	0.156	2008
LH-K-4A	397.6	399.1	1.5	0.243	0.287	2008
KG-J-19A	232.6	241	5.2	0.442	0.521	2007
KG-J-19B	214.2	217	2	0.085	0.100	2007
KG-J-20A	210	213.8	2.8	0.070	0.082	2007
KG-J-21A	313.15	327	11.35	0.328	0.387	2007
KG-J-2 IB	246.5	247	0.5	0.653	0.770	2007
KG-J-21C	291	291.7	0.7	0.205	0.242	2007
KG-J-2 ID	305.5	311.5	6	0.498	0.587	2007
KG-J-24A	311.5	312.8	1.3	0.071	0.083	2007
KG-J-24B	346	353	4	0.644	0.759	2007
KG-J-25A	301.5	307.7	6.2	0.688	0.811	2007
KG-J-25B	268	268.7	0.7	0.184	0.217	2007
KG-J-25C	317	328	9	0.188	0.222	2007
KG-J-26A	263	266	3	0.167	0.197	2007
KG-J-26B	217.5	220.4	1.4	0.256	0.302	2007
KG-J-26C	192	194	2	0.172	0.203	2007
KG-J-27A	286.4	287	0.6	0.194	0.229	2007
KG-J-27B	352.5	354	1.5	0.076	0.090	2007
KG-J-28A	332	338	6	0.121	0.142	2007
KG-J-28B	267.5	271	3.5	0.966	1.140	2007
KG-J-28C	248.5	254	5.5	0.472	0.557	2007
KG-J-31A	380	384.2	4.2	0.121	0.143	2007
KG-J-3 IB	410.7	414.9	4.2	0.246	0.290	2007
KG-J-30B	588	592.5	4.5	0.177	0.209	2007
KG-J-13	250.7	253	0.8	0.562	0.662	2006
KG-J-14	299	304	5	0.563	0.664	2006
KG-J-6	411.5	412	0.5	0.512	0.604	2006
KG-J-7	510.7	514.3	2.2	0.144	0.170	2006
KG-J-10	375.8	376.8	1	0.189	0.223	2006
KG-J-12	389	389.8	0.8	0.044	0.052	2006
KG-J-3	399.6	400.4	0.8	0.071	0.084	2006
KG-J-8	502	506.5	4.5	0.406	0.478	2006
KG-J-9	489.8	493.5	3.7	0.280	0.330	2006
KG-J-1	406.9	409.3	2.4	0.214	0.253	2005
KG-J-1 A	420.5	425.2	2	4.132	4.873	2005
KG-J-2	449.6	454	3.2	0.399	0.471	2005
KG-J-4	545.2	546.7	1.5	0.138	0.163	2005
*1180	501.1	517.9	12.5	0.324	0.382	1990
*1181	316.8	321.5	3.6	0.121	0.143	1990

Hole ID	From	To	Interval	% U	% U_3O_8	Year
*1182	300.1	304.9	4.8	1.232	1.453	1990
*1215	393.9	395.6	1.7	1.064	1.255	1990
*1218	393.4	398.1	3.1	0.425	0.502	1990
*1220	404.1	409.1	5	0.286	0.337	1990
*1222	361.5	363	1.5	0.821	0.968	1990
*1223	552.3	552.7	0.4	0.040	0.047	1990
*1234	676.8	677	0.2	0.030	0.036	1990
*1245	227	228.4	1.4	0.061	0.071	1990
*1247	429.2	429.7	0.5	0.066	0.077	1990
*1248	400.3	401.4	1.1	0.676	0.797	1990
*992	439.9	444.2	2.4	0.035	0.042	1990
*1179	513.1	513.5	0.4	0.149	0.176	1990
*1233	757.1	758.2	1.1	0.142	0.168	1990

Table B.3. 2009 to 2010 Drill Hole Listing
Tournigan Energy, Ltd. – Kuriskova Uranium Project

Hole ID	Northing	Easting	Elevation amsl	Dip (deg)	Azimuth (deg)	Depth (M)	Year
LE-K-21	-1234079	-270344	604.46	79	65	316.4	2009
LE-K-22	-1234077	-270342	604.49	75.7	49	287	2009
LE-K-23	-1234055	-270347	601.97	85.3	37	298	2009
LE-K-24	-1233981	-270404	586.22	84	77	285.9	2009
LE-K-25	-1234167	-270275	611.92	83.5	21	317.4	2009
LE-K-26	-1233978	-270402	586.22	86.9	60	297	2009
LE-K-27	-1234167	-270274	611.98	79.8	47	296	2009
LE-K-28	-1233977	-270401	586.34	87.4	10	48	2009
LE-K-29	-1233919	-270442	576.58	89.1	97	309.4	2009
LE-K-30	-1234173	-270226	605.19	84.2	66	242.2	2009
LE-K-31	-1234176	-270229	605.61	86.5	35	285	2009
LE-K-32	-1233918	-270441	576.55	79.1	29	160.4	2009
LE-K-33	-1234055	-270348	601.97	81.8	16	317.6	2009
LE-K-34	-1234225	-270319	598.76	89.7	286	447	2009
LE-K-35	-1234222	-270320	598.81	78.9	347	306.2	2009
LE-K-36	-1234223	-270320	598.63	85.2	336	458	2009
LE-K-37	-1233972	-270399	586.27	85.1	30	32.2	2009
LE-K-38	-1233969	-270394	586.43	80.1	334	287.6	2009
LE-K-39	-1234055	-270349	601.99	78.6	353	307	2010
LE-K-40	-1233920	-270442	576.63	84.4	104	283	2010
LE-K-41	-1233918	-270443	576.46	83.9	20	288.2	2010
LE-K-42	-1233921	-270444	576.52	83.7	343	306	2010
LE-K-43	-1233922	-270444	576.54	83.2	135	185	2010
LE-K-44	-1233918	-270442	576.48	66.9	58	144	2010
LE-K-45	-1233919	-270443	576.45	85.2	239	224.2	2010
LE-K-46	-1234224	-270318	598.84	70	52	346.5	2010
LE-K-47	-1234225	-270317	599.03	70	77	344.2	2010
LE-K-48	-1233918	-270443	576.4	65	13	129	2010

Table B.4. 2009 to 2010 Significant Drillhole Intercepts
European Uranium Resources Ltd. – Kuriskova Uranium Project

Hole	Intercept Zone	From (m)	To (m)	Length (m)	% U	% U_3O_8
LE-K-21	Hanging Wall	134	134.8	0.8	0.043	0.051
	Hanging Wall	194.3	194.6	0.3	0.032	0.038
	Hanging Wall	197	197.3	0.3	0.039	0.046
	Hanging Wall	208.9	209.3	0.4	0.373	0.44
	Hanging Wall	216.6	217.35	0.75	0.067	0.079
	Hanging Wall	252	252.4	0.4	0.062	0.073
	Hanging Wall	254.4	254.7	0.3	0.113	0.133
	Hanging Wall	255.5	256	0.5	0.031	0.037
	Main	266.1	270.9	4.8	0.729	0.86
	including	269	270.55	1.55	1.911	2.254
	Foot Wall	283.4	284.6	1.2	0.282	0.333
	Foot Wall	290.2	290.65	0.45	0.114	0.134
	Foot Wall	293	293.5	0.5	0.205	0.242
LE-K-22	Hanging Wall	93	93.3	0.3	0.061	0.072
	Upper Main	181.6	182.4	0.8	0.076	0.09
	Main Zone	238.3	239.4	1.1	0.31	0.366
LE-K-23	Upper Main	210.7	212.5	1.8	0.049	0.058
LE-K-24	Hanging Wall	138.2	138.5	0.3	0.198	0.233
	Upper Main	233.6	233.9	0.3	0.078	0.092
	Fault 614	253.3	253.6	0.3	0.067	0.079
	Main	268	269.3	1.3	0.507	0.598
	including	268.9	269.3	0.4	1.61	1.899
LE-K-25	Hanging Wall	211.9	212.6	0.7	0.076	0.09
	Upper Main	216.5	223.4	6.9	0.532	0.627
	including	217.5	219.5	2	1.632	1.925
	Fault 614	229.7	230.25	0.55	0.682	0.804
	Fault 614	231.8	232.6	0.8	1.56	1.84
	Hanging Wall	241	247	6	0.374	0.441
	including	241.35	242.75	1.4	1.173	1.384
	Main	264.5	265.7	1.2	0.162	0.191
	Foot Wall	280.6	280.9	0.3	0.04	0.047
	Foot Wall	287	287.5	0.5	0.032	0.037

Grade higher than 0.4 % eU3o8 is highlighted as bold
Conversion factor for eU% to eU308 is 1.17924
Significant intersection is considered at cutoff of .03 eU%
*Holes with eU% (radiometric data)

Hole	Intercept Zone	From (m)	To (m)	Length (m)	% U	% U_3O_8
LE-K-26	Hanging Wall	141	141.5	0.5	0.036	0.043
	Hanging Wall	152.5	152.8	0.3	0.03	0.036
	Upper Main	244	244.3	0.3	0.407	0.48
	Fault 614	257.3	257.6	0.3	0.069	0.081
	Main	274.5	279	4.5	0.318	0.375
	including	276.3	277	0.7	1.39	1.639
LE-K-27	Fault 614	227.8	228.3	0.5	0.032	0.038
	Fault 614	232	232.8	0.8	0.61	0.719
	including	232	232.4	0.4	1.01	1.191
	Main	244.8	245.6	0.8	0.076	0.09
	Foot Wall	249.4	249.8	0.4	0.044	0.051
	Foot Wall	260.3	262.4	2.1	0.581	0.685
	including	261.5	262	0.5	1.66	1.958
	Foot Wall	265.5	267.35	1.85	0.753	0.888
	including	266.5	267	0.5	2.1	2.476
	Foot Wall	270.5	271.7	1.2	0.049	0.058
LE-K-29	Zone 45	148.9	151	2.1	0.439	0.517
	Zone 45	152.6	153.6	1	0.073	0.086
	Zone 45	166.5	167	0.5	0.088	0.103
	Hanging Wall	244	244.3	0.3	0.033	0.038
LE-K-30	Fault 614	204.5	205.5	1	0.05	0.059
	Foot Wall	211.85	212.3	0.45	0.026	0.031
LE-K-31	Fault 614	207.7	208.7	1	0.341	0.402
	Fault 614	211.1	211.4	0.3	0.053	0.063
	Hanging Wall	224	225	1	0.037	0.043
	Hanging Wall	226	227	1	0.071	0.083
	Main	241	241.8	0.8	0.117	0.138
	Foot Wall	251.7	253.5	1.8	0.145	0.171
	Foot Wall	268	269.8	1.8	0.209	0.247
LE-K-32	Zone 45	97.4	98	0.6	0.223	0.263
	Zone 45	101.5	102	0.5	0.03	0.036
	Zone 45	103.5	105.5	2	0.612	0.722
	including	104	104.5	0.5	1.55	1.828
	Hanging Wall	149	149.3	0.3	0.034	0.04
LE-K-33	Hanging Wall	215	215.4	0.4	0.023	0.028
	Hanging Wall	254.5	255.6	1.1	0.189	0.222
	Hanging Wall	259.2	259.5	0.3	0.059	0.07
	Main	276.5	278.3	1.8	1.501	1.77
	including	276.5	278	1.5	1.79	2.111

Hole	Intercept Zone	From (m)	To (m)	Length (m)	% U	% U_3O_8
LE-K-34	Hanging Wall	221.7	222	0.3	0.064	0.075
	Hanging Wall	232	232.6	0.6	0.026	0.031
	Hanging Wall	245.5	247	1.5	0.025	0.03
	Hanging Wall	267	269	2	0.229	0.269
	Fault 614	280.7	281	0.3	0.053	0.062
	Hanging Wall	295.5	295.9	0.4	0.101	0.119
	Hanging Wall	296.6	297	0.4	0.02	0.024
	Hanging Wall	304.4	304.7	0.3	0.059	0.07
	Main	402.1	402.5	0.4	0.032	0.038
LE-K-35	Hanging Wall	218.1	218.6	0.5	0.088	0.103
	Hanging Wall	220.15	220.55	0.4	0.024	0.029
	Hanging Wall	235	235.5	0.5	0.068	0.08
	Hanging Wall	263	264	1	0.226	0.267
	Hanging Wall	265.5	266	0.5	0.097	0.114
	Hanging Wall	268.6	269.4	0.8	0.37	0.436
LE-K-36	Hanging Wall	272.3	273.6	1.3	0.11	0.129
	Hanging Wall	274.6	278.3	3.7	0.262	0.309
	Hanging Wall	286.7	287.1	0.4	0.13	0.153
	Hanging Wall	290.7	292.6	1.9	0.107	0.126
	Hanging Wall	348	348.3	0.3	0.034	0.04
	Hanging Wall	349	350	1	0.025	0.029
	Hanging Wall	356.4	356.7	0.3	0.051	0.06
	Hanging Wall	360.7	361	0.3	0.03	0.036
	Main Zone	415.5	415.8	0.3	0.145	0.171
LE-K-39	Main	263.8	268.5	4.7	1.427	1.682
	including	263.8	265	1.2	2.46	2.901
	including	266.2	267.8	1.6	2.035	2.4
LE-K-40	Zone 45	135.1	138	2.9	0.35	0.412
LE-K-41	Zone 45	128.3	130.6	2.3	1.934	2.28
	including	128.6	130.2	1.6	2.678	3.158
	Zone 45	137	137.3	0.3	0.037	0.044
	Main Zone	265.5	266.1	0.6	0.409	0.482
LE-K-42	Zone 45	140.7	143.6	2.9	2.127	2.508
	including	141	143.3	2.3	2.666	3.144
	Zone 45	148	149	1	0.045	0.053
	Zone 45	165.7	166	0.3	0.051	0.06
*LE-K-43	Zone 45	147.15	147.6	0.45	0.053	0.062
*LE-K-44	Zone 45	86.5	88	1.5	0.285	0.336
	Zone 45	107	110.2	3.2	0.064	0.075

Hole	Intercept Zone	From (m)	To (m)	Length (m)	% U	% U_3O_8
	Zone 45	112	113	1	0.025	0.029
	Zone 45	117.5	119.5	2	0.062	0.073
*LE-K-45	Zone 45	178.4	178.8	0.4	0.064	0.075
* LE-K-48	Zone 45	97.1	97.4	0.3	0.025	0.029
*LE-K-46	Hanging Wall	155.4	155.9	0.5	0.039	0.046
	Hanging Wall	156.7	157.3	0.6	0.139	0.164
	Hanging Wall	162.3	163.8	1.5	0.214	0.253
	Hanging Wall	168	169.9	1.9	0.148	0.175
	Hanging Wall	182.1	182.7	0.6	0.036	0.042
	Hanging Wall	184.9	190.5	5.6	0.121	0.143
	Hanging Wall	192	192.4	0.4	0.098	0.116
	Hanging Wall	194.8	196.4	1.6	0.153	0.181
	Hanging Wall	200.3	200.9	0.6	0.175	0.206
	Hanging Wall	201.6	201.9	0.3	0.083	0.098
	Hanging Wall	206.2	207.3	1.1	0.112	0.132
	Hanging Wall	208	212.4	4.4	0.296	0.349
	including	211	211.3	0.3	1.91	2.252
	Hanging Wall	213.9	214.5	0.6	0.268	0.316
	Hanging Wall	216.7	217	0.3	0.104	0.123
	Fault 614	230.6	231	0.4	0.042	0.049
	Hanging Wall	265.4	266.8	1.4	0.046	0.054
	Hanging Wall	269	272.1	3.1	0.079	0.094
	Main	279.1	280.1	1	0.116	0.137
	Foot Wall	293.1	295.5	2.4	0.578	0.681
	including	294.7	294.9	0.2	1.878	2.215
	Foot Wall	297.7	300	2.3	0.75	0.884
	including	298.7	298.9	0.2	1.853	2.185
	including	299.3	299.5	0.2	1.896	2.236
* LE-K-47	Hanging Wall	154.7	155.8	1.1	0.198	0.233
	Hanging Wall	164.4	165.3	0.9	0.263	0.311
	Hanging Wall	171.3	172.1	0.8	0.065	0.077
	Hanging Wall	191.6	196	4.4	0.123	0.145
	Hanging Wall	199.3	201	1.7	0.082	0.097
	Hanging Wall	202.2	203.8	1.6	0.158	0.186
	Hanging Wall	205.7	212.7	7	0.238	0.281
	including	207.7	208.4	0.7	1.316	1.552
	Main	284.2	286.9	2.7	0.191	0.226

Table B.5. Kuriskova Uranium Project Drilling to Date
European Uranium Resources Ltd. – Kuriskova Uranium Project

Year	No. of Holes Drilled	Drill holes within the Resource	Total Meters Drilled
Pre-1990	53	27	17,000
2005	3	3	7,595
2006	15	15	
2007	30	29	12,712
2008	8	8	9,267
2008 (infill)	23	23	
2009-2010	28	26	7,548
2010-2011	18	18	4,548
TOTAL	160	149	58,670

Table B.6. 2010-2011 Drill Hole Listing
European Uranium Resources Ltd. – Kuriskova Uranium Project

Hole Id	Easting	Northing	Elevation (amsl)	Dip (deg)	Azimuth (Deg)	Depth (m)	Year
LE-K-49	-270466.96	-1233875.6	569.78	86.7	34.7	297.3	2010
LE-K-50	-270468.71	-1233879.2	569.89	85.9	175	206.6	2010
LE-K-51	-270468.27	-1233877.2	569.77	82.4	175.4	214.8	2010
LE-K-52	-270468.26	-1233878.4	569.99	84.3	206.7	231.1	2010
LE-K-53	-270496.66	-1233814.3	558.85	80.6	208.9	212.6	2010
LE-K-54	-270497.36	-1233813.8	558.76	77.7	239.1	241.5	2010
LE-K-55	-270496.21	-1233814.9	558.95	77.4	207.9	239.5	2010
LE-K-56	-270494.7	-1233815.1	559.16	75.8	173.2	200.2	2010
LE-K-57	-270467.48	-1233879.3	570.13	82.7	135.8	191	2010
LE-K-58	-270408.78	-1233954.0	583.94	81.9	113.3	174.7	2010
LE-K-59	-270407.6	-1233951.6	583.75	73.2	48	127.6	2010
LE-K-60	-270465.82	-1233878.2	570.22	74.8	88.9	155.5	2010
LE-K-61	-270436.45	-1233913.6	576.48	43.9	58.3	120.7	2010
LE-K-62	-270466.75	-1233877.4	569.95	76.8	234.6	250	2010
LE-K-63	-270131.5	-1234462.5	563.14	89.9	149.8	214	2010
LE-K-64	-270461.62	-1234324	589.09	78.3	68	481.5	2011
LE-K-65	-270462.5	-1234325.4	589.32	75.2	86	476.1	2011
LE-K-66	-270461.68	-1234325.8	589.4	86.9	72.8	512.8	2011

Table B.7. 2010 to 2011 Significant Drill Hole Intercepts
European Uranium Resource Ltd. – Kuriskova Uranium Project

Hole	Intercept Zone	FROM (m)	TO (m)	Length (m)	% U	% U_3O_8
LE-K-49	Zone 45	154.7	155	0.3	0.126	0.148
	Zone 45	167.4	167.7	0.3	0.084	0.099
LE-K-50	Zone 45	158.35	158.65	0.3	0.053	0.062
	Zone 45	165.7	168.6	2.9	0.662	0.781
	including	167	167.7	0.7	0.533	0.628
	including	168.15	168.6	0.45	2.400	2.830
	Zone 45	170.55	171.7	1.15	0.139	0.164
LE-K-51	Zone 45	177	180	3	0.170	0.201
	including	177.3	177.8	0.5	0.433	0.510
	including	178.9	179.3	0.4	0.486	0.574
	Zone 45	183.3	183.75	0.45	0.111	0.130
LE-K-52	Zone 45	184.8	189.5	4.7	0.129	0.152
	including	186.6	187.1	0.5	0.597	0.704
	including	187.6	187.9	0.3	0.365	0.430
	Zone 45	199.8	200.15	0.35	0.046	0.054
LE-K-53	Zone 45	178.5	179	0.5	0.138	0.163
LE-K-56	Zone 45	178	178.3	0.3	0.183	0.215
LE-K-57	Zone 45	149.5	152.8	3.3	0.693	0.817
	including	150.8	151.15	0.35	1.150	1.356
	including	152	152.8	0.8	1.880	2.217
	Zone 45	156.8	158.15	1.35	0.278	0.328
	including	157.15	157.5	0.35	0.726	0.856
	Zone 45	170.2	170.5	0.3	0.113	0.133
LE-K-58	Zone 45	134.1	134.8	0.7	0.172	0.203
LE-K-59	Zone 45	96	96.4	0.4	0.197	0.232
LE-K-60	Zone 45	96	96.45	0.45	0.168	0.198
	Zone 45	136.3	136.7	0.4	0.031	0.036
LE-K-64	Hanging Wall	371	371.7	0.7	0.070	0.083
	Hanging Wall	374.2	374.7	0.5	0.055	0.065
	Hanging Wall	396.4	396.85	0.45	0.117	0.138
	Hanging Wall	415.4	415.7	0.3	0.096	0.113
	Main	451.7	454.7	3	0.064	0.076
	Main	456.7	457	0.3	0.073	0.085
	Main	458	461.3	3.3	0.410	0.483
	including	459.7	460.35	0.65	1.246	1.469
	including	460.7	461	0.3	0.980	1.156
LE-K-65	Hanging Wall	344	344.3	0.3	0.034	0.040
	Hanging Wall	363.5	364.2	0.7	0.088	0.103

Hole	Intercept Zone	FROM (m)	TO (m)	Length (m)	% U	% U_3O_8
	Hanging Wall	366	366.35	0.35	0.085	0.100
	Main	454.9	455.7	0.8	0.194	0.228
LE-K-66	Hanging Wall	402.1	404.2	2.1	0.059	0.069
	Hanging Wall	405.5	407	1.5	0.042	0.050
	Hanging Wall	446.35	446.65	0.3	0.216	0.255
	Hanging Wall	458.4	458.7	0.3	0.032	0.038
	Main	481.3	483	1.7	0.604	0.712
	including	482	482.4	0.4	1.463	1.725
	including	482.7	483	0.3	1.262	1.488
	Main	484.3	492.9	8.6	0.156	0.184
	including	488.35	489.05	0.7	0.540	0.637
	including	490.6	491	0.4	0.689	0.812

Grade higher than 0.4 percent e U_3O_8 is highlighted as bold
Conversion factor for eU% to e U_3O_8 is 1.17924
Significant intersection is considered at cutoff of 0.03 eU%