NI 43 – 101 PRELIMINARY ECONOMIC ASSESSMENT UPDATE

COLES HILL URANIUM PROPERTY
PITTSYLVANIA COUNTY, VIRGINIA
UNITED STATES OF AMERICA

PREPARED FOR:

Virginia Energy Resources Inc.
Suite 611 - 675 West Hastings Street
Vancouver, BC Canada V6B 1N2

231 Woodlawn Heights – PO Box 399
Chatham, VA 24531

VIRGINIA URANIUM HOLDINGS, INC.

231 Woodlawn Heights – PO Box 399
Chatham, VA 24531

PREPARED BY:

John I. Kyle, PE
Lyntek Inc.
1550 Dover Street
Lakewood, CO 80215

Douglas Beahm, PE, PG
Principal Engineer
BRS Engineering
1225 Market Street
Riverton, WY 82501

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ITEM 1. SUMMARY

1.1 Property Description and Ownership

Virginia Energy Resources Inc. (TSX.V:VAE) currently holds approximately a 29% minority equity interest in VA Uranium Holdings, Inc. (Holdco), a British Columbia corporation. Holdco's 100% owned subsidiary, Virginia Uranium Inc., a Virginia corporation, controls the leasehold development and operating rights of the Coles Hill Uranium Property in Southside Virginia. Virginia Energy Resources, Inc. (VAE) acquired the indirect minority equity interest in the Coles Hill Uranium Property (CHUP) in 2009. The surface radiometric anomaly associated with the CHUP was found in the late 1970s by Marline Uranium Corporation (Marline), a wholly-owned subsidiary of Marline Oil Corporation.

The Coles Hill Property is located in Pittsylvania County, Virginia near the town of Chatham. Pursuant to an amended and restated arrangement agreement dated July 12, 2012 among Virginia Energy Resources Inc. ("**VAE**"), Virginia Uranium Ltd. ("**VUL**"), VA Uranium Holdings, Inc. ("**Holdco**") and 0942845 B.C. Ltd., the parties are proposing to complete a plan of arrangement (the "**Plan of Arrangement**") pursuant to which, among other things:

a) the name of VAE shall be changed from "Virginia Energy Resources Inc." to "Anthem Resources Incorporated";

b) VUL and Holdco will complete an amalgamation to form one corporation, the name of which will be "Virginia Energy Resources Inc." ("**Amalco**");

c) upon completion of the Plan of Arrangement::

   a. the holders of non-voting common shares of Holdco (other than VAE and VUL) shall be entitled to receive, in exchange for each non-voting common share of Holdco held, 0.1817 of a common share of Amalco (an "**Amalco Share**");

   b. shareholders of VAE shall be entitled to receive, in exchange for each common share of VAE held, (i) one-third (⅓) of one "new" common share of VAE; and (ii) one-tenth ($^1/_{10}$) of one common share of VUL (to be exchanged for Amalco Shares on a one-for-one basis);

   c. Amalco will indirectly hold the Otish property located in Québec and a 100% interest in the Coles Hill Uranium Property in Southside Virginia (the "**Coles Hill Property**" or the "**CHUP**");

   d. VAE will hold the Fir Island property and all other properties and assets currently held by VAE, other than the Otish property and the Coles Hill Property; and

   e. it is anticipated that Amalco will be a reporting issuer in the jurisdictions in which VAE is a reporting issuer (being British Columbia, Alberta and Québec) and it is the intention of Amalco to apply for listing of the Amalco Shares on the TSX Venture Exchange. VUL is currently a wholly owned subsidiary of VAE.

The transaction contemplated under the Plan of Arrangement will essentially result in, among other things: (i) VAE vending all of its interest in the Coles Hill Property (being an aggregate of approximately 29%) into VUL; (ii) VUL amalgamating with Holdco (the shareholders of Holdco hold the other indirect approximately 71% interest in the Coles Hill Property); and (iii) the resulting amalgamated company (i.e., Amalco) holding a 100% indirect interest in the Coles Hill Property.

The longitude and latitude of the center of the property is about 36º 52" north and 79º 18" west. Virginia Uranium Inc. leased or purchased the mineral rights to approximately 3,346 acres that includes associated surface rights to 2,702 acres covering the South Coles Hill Deposit (SCHD) and the North Coles Hill Deposit (NCHD), as well as areas for exploration, mining operations, milling, waste management areas, and set-backs. An area (the "Protected Area") of about 648 acres exists around the Coles Hill historical buildings and cultural areas where only underground mining is allowed. Figure 1.1 provides the location and layout of the property.

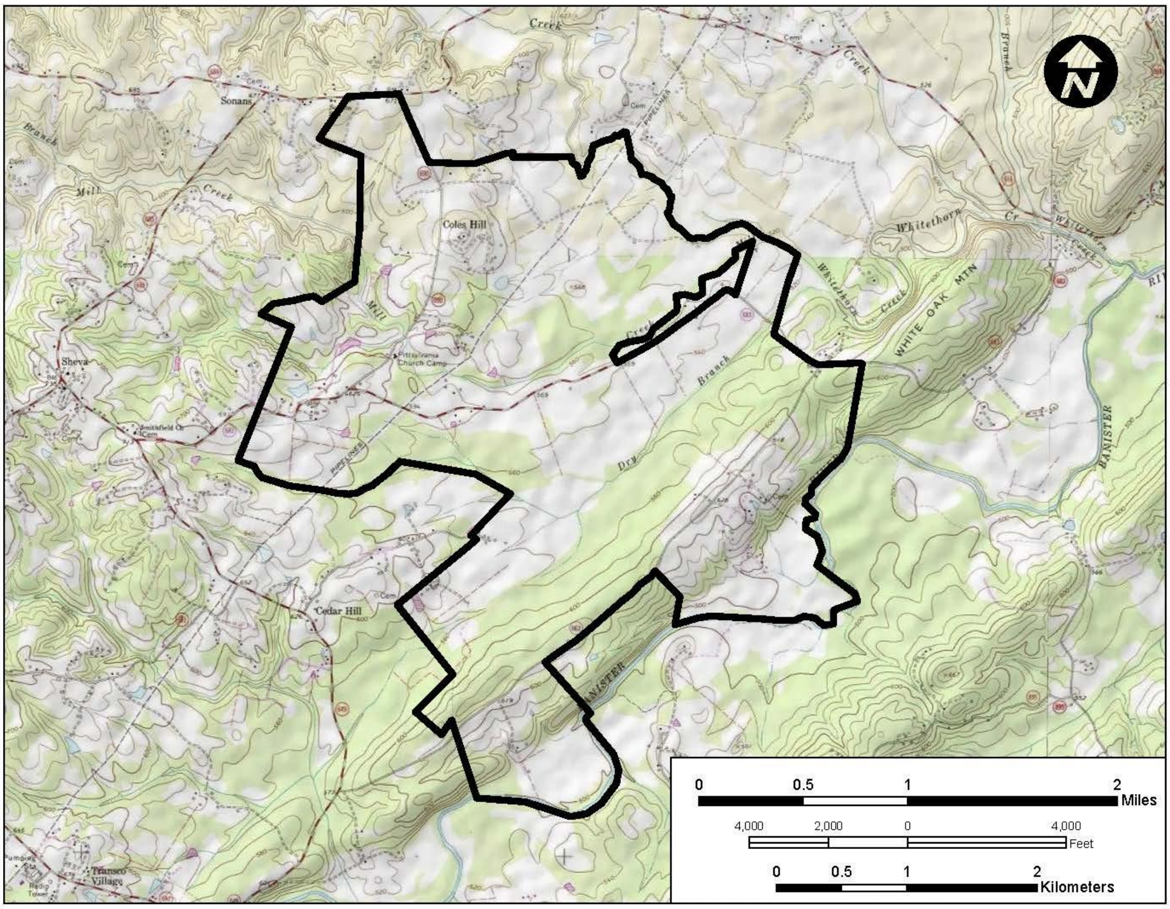


Figure 1.1: Topographic plan, and layout of property, Coles Hill Area, Pittsylvania County, Virginia

1.2 Geology and Mineralization

The CHUP, located in the Piedmont physiographic province of Virginia, consists of two approximately 350-meter-long by 250-meter-wide ellipsoidal mineral deposits (NCHD and SCHD). The character of the deposits is shaped by a complex combination of three major factors: faulting, fracture zones, and alteration.

The deposits are hosted within a fault-bounded wedge of Paleozoic mylonitic quartzo-feldspathic gneiss and some amphibolite exposed in the footwall of the Chatham fault zone, which forms the northwest limit of the Danville Triassic Basin. The host rocks are Leatherwood granite gneiss and amphibolite, which display hydrothermal alteration by sodium metasomatism, chloritization, argillization, hematization and uranium mineralization.

The CHUP contains a fracture-hosted hydrothermal deposit, with uranium situated in mylonite. The deposit has characteristics of hydrothermal fracture type uranium deposits. Hydrothermal solutions and associated uranium mineralization are presently hypothesized to have been mobilized by tectonic events.

The mechanism of uranium deposition at Coles Hill is similar to that in the Athabasca Basin, as indicated by the presence of alteration minerals hematite, epidote, and chlorite. A similar deposition mechanism in the Athabasca Basin has produced significant-grade uranium mineralization, which might also occur in the untested deeper parts of the Coles Hill Deposits (Jerden 2001).

1.3 Exploration Status

Virginia Uranium Inc. obtained permits for uranium exploration in November 2007 and started a core and rotary percussion drilling program in December 2007. By May 2008, the Company completed three core holes totaling 4,510 feet, seven rotary percussion holes totaling 8,758 feet, and the re-assaying of about 60 feet of core drilled by Marline.

Subsequent to the Behre Dolbear, 2008 technical report, Virginia Uranium, Inc. acquired the original drill hole data including geophysical and lithological logs, half foot uranium grade equivalent data, and chemical assay data both from core analysis and Delayed Neutron Logging (DNL). The original data was more complete and included 263 drill as compared to the 230 previously available. The data was transcribed from the original analog format to digital format and was used for the current resource estimate.

The current model was prepared under the direction of Douglas Beahm, PE, PG, President and Principal Engineer BRS Inc. by ExplorMine Consultants of South Africa (Northrop and Deiss, 2011). This study utilized the updated database as previously described. Geologic modeling and mineral resource modeling was completed using geostatistical methods rather than inverse distance cubed as in the 2008 estimate.

1.4 Proposed Development and Operations

No development activities in the field, other than exploration have occurred. Lyntek, in cooperation with BRS, Inc. evaluated the mineral resources, mining concepts, and processing of the ore from the Coles Hill Uranium Property. In summary, the preferable mining method is an underground stoping method. Although surface mining is a viable option, it appears at this point in time, that an all underground mine approach can be effectively employed.

Both acid and alkaline processing methods have been investigated. Due primarily to the consumption requirement and relative cost of sulfuric acid, an alkaline process is the most viable in spite of a slightly lower recovery rate. Adequate information is available for this level of study given the historical efforts and recent tests that have been conducted.

1.5 Mineral Resource Estimates

Mineral Resources

At the minimum grade cutoff criterion of 0.025 %eU_3O_8 estimated mineral resources are summarized in following tables for indicated and inferred mineral resource categories, respectively. Tonnages in this report are expressed in long tons.

TABLE 1.1: TOTAL INDICATED MINERAL RESOURCES

Total North and South Coles Hill				
Category	**Cutoff**	**Tons (million)***	**wt %eU_3O_8**	**lbs (million)**
Indicated	0.025	119.59	0.056	132.93

TABLE 1.2: TOTAL INFERRED MINERAL RESOURCES

Total North and South Coles Hill				
Category	**Cutoff**	**Tons (million)***	**wt %eU_3O_8**	**lbs (million)**
Inferred	0.025	36.28	0.042	30.41

*Long Tons

Mineral Reserves

This Preliminary Economic Assessment (PEA) focuses on underground mine extraction and utilized a cutoff grade of 0.06 %eU_3O_8 for the determination of mining limits and indicates that the portion of the mineral resource currently included in the underground mine design for the North and South Coles Hill areas are economic under current conditions within the present mine design limits. This portion of the Indicated Mineral Resource is considered in the Preliminary Economic Assessment as summarized in the following table.

TABLE 1.3: PORTION OF INDICATED MINERAL RESOURCE CONSIDERED IN THE PRELIMINARY ECONOMIC ASSESSMENT

Total North and South Coles Hill				
Category	**Cutoff**	**Tons (million)***	**wt %eU_3O_8**	**lbs (million)**
Indicated	0.06	32.9	0.098	64.2

*Long Tons

1.6 Preliminary Economic Assessment

A cash flow model was developed for a 3,000 tpd case that models annual periods of cash inflow and outflow, without the financing cost of capital. The project schedule, sequence of mining, mining rate and mining costs were used to develop the cash flow model. It is assumed that ore production commences one year after all mining permits and licenses have been received. The primary mining rates are 700,000 tons in year one, 1,050,000 tons from years two through four, 700,000 long tons per year for years five through twenty-five, and 467,000 tons per year from year twenty-six through year thirty-five. In addition to this production, mining pillars accounts for 350,000 tons per year for years five through twenty-five, and 233,000 tons per year from year twenty-six through year thirty-five. The predicted grade of production, which is based upon mine plans through the geologic model, appropriately diluted, show a range of grades from 0.079% to 0.126% U_3O_8, with an average of 0.0965% U_3O_8. Assuming a plant recovery of 85%, the total uranium production ranges from 1,225,000 lbs to 2,646,000 lbs. and averages 1,885,000 lbs. U_3O_8/year. The mill design and recovery rate is based upon prior metallurgic studies, which have been augmented by recent testing.

Including 25% contingency, the total capital investment prior to production is $147 million, however, the total capital through the 4th year of production of $200 million is a better estimate of initial capital due to the staging of the tailings cell construction. Including 25% contingency, the total capital spending over the life of the facility is $329 million. This cost estimate excludes any other specific non-project related costs that would be in addition to this project. For example, it would be reasonable to expect that further exploration and research programs could certainly range up to an additional $40 million in an effort to generate additional resources or address other non-project goals. Total direct and indirect operating costs are forecast under $31/lb. during the first 10 years with an average of $35/lb. U_3O_8 over the life of mine.

The economic analysis at a yellowcake price of $64/lb shows an internal rate of return of 36.3% before income taxes; at a discount rate of 7% the net present value is $427 million, including a 25% contingency. The economics indicate a project worthy of further evaluation.

1.7 Conclusions and Recommendations

1.7.1 Conclusions

Lyntek and BRS as a result of this updated study have arrived at the following conclusions:

The mine and mineral processing development alternatives presented herein demonstrate a potential for economically viable mineral resources, based on the cost and price estimates as discussed in this report. It must be noted that this evaluation is based upon mineral resources and not mineral reserves and mineral resources that are not mineral reserves do not have demonstrated economic viability. The preferred alternative for the development of the Coles Hill Uranium Project includes a Sub Level Open Stope (SLOS) underground conventional mine operation with on-site mineral processing via a conventional, alkaline mill. Surface mine alternatives were also evaluated and appear to have merit especially in light of the need for subsurface tailings disposal.

The technical risks related to the project are low as the mining and recovery methods are proven. The mining methods recommended have been employed successfully at similar projects in the past. The mineral processing methods employed are typical of those used in the industry for decades and are supported by metallurgical tests done to date and are available.

Primary risks related to permitting are rescinding the moratorium to allow mining in Virginia and gaining the confidence of the local community that the mining and milling can be safely conducted to protect human health and the environment. The remainder of the permitting issues is tied to obtaining the necessary permits to operate the mine and mill.

The authors are not aware of any other specific risks or uncertainties that might significantly affect the mineral resource estimates or the consequent economic analysis.

Estimation of costs and uranium price for the purposes of the economic analysis over the life of mine is by its nature forward-looking and subject to various risks and uncertainties. No forward-looking statement can be guaranteed and actual future results may vary materially.

The following conclusions have been made as a result of this study:

- The continuity of mineralization through to the surface in both the north and south deposits could support either open pit or underground mining, however underground mining is recommended (open pit is not discounted);
- Underground mining can be performed by sub-level open stoping (SLOS), a historically productive and a safe mining method;
- Surface mine options should be evaluated in light of the needs for subsurface tailings disposal and as a means of improving project economics and accelerating mine production;

- Additional drilling and specific data collection is recommended under Item 26 to better define mineral resource and increase the accuracy and reliability of the mine design and cost estimates herein;
- While acid leaching is expected to produce a higher uranium recovery, alkaline leaching is the more cost effective option;
- There is inadequate surface are for the tailings facilities, purchase additional surface area and/or consideration of sub-surface tailings disposal in combination with open pit mining is necessary;
- The overall conceptual economics are favorable for the Coles Hill project. The project shows an IRR of 36.3%; at a discount rate of 7% the net present value (NPV) is $427 million.
- The life of the mine is 35 years.
- Including 25% contingency, the initial capital investment prior to production is $147 million, however, the total capital through the 4th year of production is $204 million, while the total capital is estimated to be $329 million.
- The initial annual revenue ranges from $95 to $144 million.
- The direct and indirect economic benefits are on the order of $240 to $300 million.
- Total labor for both the mining and milling operations is forecast at 224 for the mine and 100 for the milling operations for a total of 324 employees. Of this, it is expected that 218 would be hourly workers and 106 would be staff.
- The annual payroll is forecast at $13 million for mining and $6 million for processing such that the total annual payroll would be $19 million. If the 25% contingency is attributed to this cost, the estimate would be $24 million.
- Salaries are expected to range from $35,000 to $250,000 per annum and hourly rates would range from $20 to $35 per hour.
- Annual material and supply costs are projected to be about $22 million during the primary mining phase such that total annual material and labor costs would roughly range from $41 to $46 million per year.
- It is envisioned during construction that 250 to 350 personnel would be necessary including employees and contractors.

1.7.2 Recommendations

Mineral Resource Related Recommendations

The extent of mineralization is not fully defined by current drilling. While additional drilling may or may not expand mineral resources, it is the author's interpretation and opinion that mineralization extends beyond the currently defined limits.

Mine Related Recommendations

- Detailed mine planning, both underground and surface, is recommended to optimize mine recovery and economics. These design efforts should also consider mine closure and reclamation requirements including provisions for subsurface tailings disposal. Budgetary estimate $500,000 US.
- Placement of tailings as paste backfill is contemplated in the current plan. Specific testing relative to admixtures is recommended. Testing should include geotechnical considerations relating to compressive strength, density and engineering properties. In addition, admixtures which are suitable from a geotechnical perspective should be tested for long term leaching characteristics. Specifically ASTM method C 1308, "Accelerated Leach Test for Diffusive Releases from Solidified Waste and a Computer Program to Model Diffusive Fractional Leaching from Cylindrical Waste Forms" is recommended. Fate transport of any constituents of concern from this testing should be evaluated. Budgetary estimate $100,000 US.
- The mining methods being considered are of a bulk nature and opportunities for selective mining are limited. Testing of methodologies such as radiometric ore sorting in the mining process is recommended to reject waste within the mine and improve run-of-mine grades. Radiometric ore

sorting could substantially reduce the volume of tailings. Testing should be phased with the initial evaluation by hand sorting and proof testing at a bench scale. Budgetary estimate for initial testing $50,000 US. Budgetary estimate for bench scale testing $200,000 US exclusive of sample collection costs.

- Core drilling is recommended at both North and South Coles to provide additional geologic, geotechnical, and hydrologic data, as well as representative samples for metallurgical testing and bench scale radiometric ore sorting. Single drill holes could be designed to provide data and samples for multiple purposes. This work could be phased and include geotechnical information acquisition and hydrologic data acquisition and modeling from about 15 sites. Budgetary estimate for this work depending on final scope the cost of contracted services and sample needs would be about $2,000,000 US.

Mineral Processing and Metallurgical Recommendations

- Allow access for the collection of a bulk metallurgical sample. This bulk sample would be tested at an off-site licensed facility to:
    - Determine the Bond Work Index (kWh/t) variability;
    - Determine the Work Index for semi-autogenous (SAG) mills;
    - Optimize leach conditions;
    - Evaluate the viability of processing paste tailings; and
    - Evaluate alternatives for tailings disposal.
- It is further recommended that test-work be conducted to determine the Bond Work Index (kWh/t) variability throughout the North and South ore bodies. The Work Index for SAG mills should also be determined. Cost Estimate $500,000 to $1,000,000.
- The tailings facilities have been designed for an in-place S.G. of 1.3, however further test-work is required to validate this. It is recommended that the tailings produced from alkaline leaching would be tested for physical properties such as bulk density and % solids post-filtration. Additionally, the option of paste processing all tailings (i.e. tailings and underground backfill) should be explored. Using paste tailings in the tailings impoundments is beneficial as it limits the infiltration of outside water and the remobilization of the tailings and potentially reduces the size, and therefore cost, of the impoundments themselves. Cost estimate for leach optimization, paste study, and filtering and settling tests is $150,000.
- Conduct further investigations into tailings disposal concepts to assess opportunities and optimize risk mitigation while assessing additional properties for tailings disposal. Cost estimate: $250,000.

Approach and Associated Costs

A phased approach is necessary to move the work forward efficiently. Phase I work will address: (1) more pressing considerations necessary to evaluate the current level of design; (2)investigate the tailings storage options to allow more in-depth evaluation of the mining and processing designs for the later phase work; (3) will serve to assess the underground tailings admixtures for leaching and structural characteristics. Phase II represents work to take the project to the next feasibility study level. This approach will promote the optimization potential for mining and processing designs for the next level of feasibility study via an organized, in-depth evaluation.

Costs associated with phase I are estimated at $550,000. Phase II associated costs are estimated at $ 2,625,000 for a total cost of $3,175,000. Results from phase I will be incorporated into the work design for phase II with regard to addressing the overall project feasibility evaluations.

The feasibility study objective is to provide the necessary information that would allow the project to be considered for economic development.

ITEM 2. INTRODUCTION

This report is prepared for Virginia Energy Resources, Inc. Suite 611 – 675 W. Hastings St. Vancouver, BC, Canada V6B 1N2. They are a public company traded on the TSX Venture Exchange under the symbol VAE, and VA Uranium Holdings, Inc and Virginia Uranium, Inc., both of 231 Woodlawn Heights – PO Box 399 Chatham, VA 24531. Virginia Energy Resources, Inc. (TSX.V:VAE), currently holds approximately a 29% minority interest in VA Uranium Holdings, Inc. (Holdco), a British Columbia corporation. Holdco's 100% owned subsidiary, Virginia Uranium Inc., a Virginia corporation, controls the leasehold development and operating rights of the Coles Hill Uranium Property in Southside Virginia. Mr. Walter Coles, Jr, is currently President and Chief Executive Officer of VAE and Executive Vice President of Holdco.

This report is prepared by John Kyle, PE of Lyntek, Inc. and Doug Beahm, PE, PG of BRS Engineering. Lyntek, Inc. is a mineral processing design company based in Lakewood, CO at 1550 Dover Street, 80215. Mr. Kyle, a Qualified Person and independent professional engineer, oversaw the development of the processing concepts and costs as well as the development of the economic analysis and the remainder of the report preparation. Mr. Kyle has extensive experience in uranium with evaluation of about three dozen projects. Mr. Kyle is responsible for sections 1-5, 13, and 17-27.

BRS Engineering is based in Riverton, WY at 1225 Market Street, 82501. Mr. Beahm, a Qualified Person and independent professional engineer with uranium resource estimation experience estimated the uranium resources and developed the mining concepts and costs for the Project, as documented in the Updated National Instrument 43-101 Technical Report, announced on June 30, 2011.

Mr. Beahm, is both a Professional Geologist and a Professional Engineer, and a Registered Member of the US Society of Mining Engineers (SME). He is independent of Virginia Energy Resources, Inc., using the test set out in Section 1.5 of National Instrument 43-101. Mr. Beahm is experienced with uranium exploration, development, and mining including past employment with the Homestake Mining Company, Union Carbide Mining and Metals Division, and AGIP Mining USA. As a consultant and principal engineer of BRS, Inc., Mr. Beahm has provided geological and engineering services relative to the development of mining and reclamation plans for uranium projects in Wyoming, Utah, Colorado, Arizona, and Oregon, as well as numerous mineral resource and economic feasibility evaluations. This experience spans a period of thirty-eight years dating back to 1974.

BRS was responsible for mineral resource estimation, mine design, and estimate of mine related capital and operating costs. Mr. Beahm's visits to the site include May 18 and 19, 2010, August 20, 2010, and April 4 through 6, 2011. On these occasions Mr. Beahm reviewed the original geologic data for the project including previous investigations, geophysical and lithologic logs data, chemical assay record, and physically inspected the available core from the project. Mr. Beahm is responsible for sections 6-12 and 14-16, with contributions to sections 3, 21, and 25-27.

2.1 Terms of Reference and Purpose of Report

The purpose of this Technical Report is to update the NI 43-101 Preliminary Economic Assessment – Coles Hill Property – December, 2010 and provide a brief review of the historical and current exploration activities conducted for the project, an independent audit and update of the most recent resource estimate, and a discussion of the scoping study conceptual design, including cost updates and a preliminary economic assessment of the project's potential economic viability. This study includes proposals and suggestions for additional delineation of the deposits to further define mineralization, metallurgical and process selection studies, and a pre-feasibility study to support development of the CHUP in Pittsylvania County, Virginia, USA. The report has been prepared in compliance with the requirements of National Instrument 43-101 and Form 43-101F1. The purpose of the report is to provide supporting documentation to be filed with the relevant securities commissions and the TSX Venture Exchange and to support the Listing Application ofAmalco (to be formed pursuant to the amalgamation of VUL and Holdco).

This report is not a Preliminary Feasibility Study and as such no Mineral Reserves are stated. Preliminary mine designs have been completed and the portions of the Indicated Mineral Resources within the preliminary mining limits were utilized in the Preliminary Economic Assessment, as discussed herein.

Items which are recommended to be addressed in order to upgrade the Preliminary Economic Assessment to a Preliminary Feasibility Study include but are not limited to:

- Incorporation of regulatory requirements in mine and mineral processing designs, resulting from the promulgation of uranium specific mining regulations and lifting of the current moratorium on uranium mining in the Commonwealth of Virginia.
- Optimization of tailings facility design with respect to operations, closure, and final reclamation.
- Evaluation of open pit mining options with consideration of utilizing the mined pits for sub-grade tailings disposal.
- Optimization of underground mine methods, designs and scheduling.
- Optimization of mineral processing and beneficiation methods.

2.2 Sources of Information

The information upon which this report includes the historical project archives contained within the Virginia Uranium, Inc. library. Data were collected from various reports and other data supplied by the Virginia Museum of Natural History, the Virginia Division of Mineral Resources, and extensive reporting of academic studies by faculty and students at Virginia Polytechnic Institute and State University, as well as from access to all known historical Marline data. The Marline data consist of drill logs, maps, process data, and reports. This information is relied upon and has been evaluated and verified where possible.

2.3 Qualified Person Inspection

Mr. Kyle conducted a site visit of the property on November 12, 2009 and May 18, and 19, 2010, and September 20 and 21, 2010. Mr. Beahm's visits to the site include May 18 and 19, 2010, August 20, 2010, and April 4 through 6, 2011.

ITEM 3. Reliance on Other Experts

Mr. Kyle received data and information consisting of deeds, leases, deed references, map references, legal opinions regarding standard practice of the transfer of surface and mineral rights, purchase agreements, sales closing documents, legal descriptions, property maps and property title insurance documents from Joe Aylor, the Chief Geologist of Virginia Uranium Inc., and the company's property attorney on and around August 30, 2012. Mr. Kyle has reviewed and relied upon this data and information as of August 30, 2012 to develop his opinion and the disclosure as set out under Item 4. Property Description and Location,

ITEM 4. Property Description and Location

The CHUP is situated in Pittsylvania County and the Piedmont physiographic province in southside Virginia (Figure 4.1, from Virginia Department of Mines, Minerals, and Energy). The property is 6 miles (10 kilometers) northeast of Chatham, the county seat, which has a population of about 1,500 people and is 30 miles (48 kilometers) north of Danville, an independent city bordering Pittsylvania County to the south, with a population of about 45,000 people. The Raleigh-Durham area, North Carolina, population about 1,200,000 people, is about 70 miles (110 kilometers) southeast of Danville, Virginia and 88 miles (140 kilometers) southeast of the CHUP.

The property was drilled from 1979 until 1984 with 182 rotary-percussion holes (totaling 124,799 feet) and 74 NQ core holes (totaling 65,082 feet), totaling 256 holes and 189,881 feet. In 2008, Virginia Uranium, Inc. completed the drilling of seven rotary-percussion (RP) holes (totaling 8,758 feet) and three NQ core holes (totaling 4,510 feet).

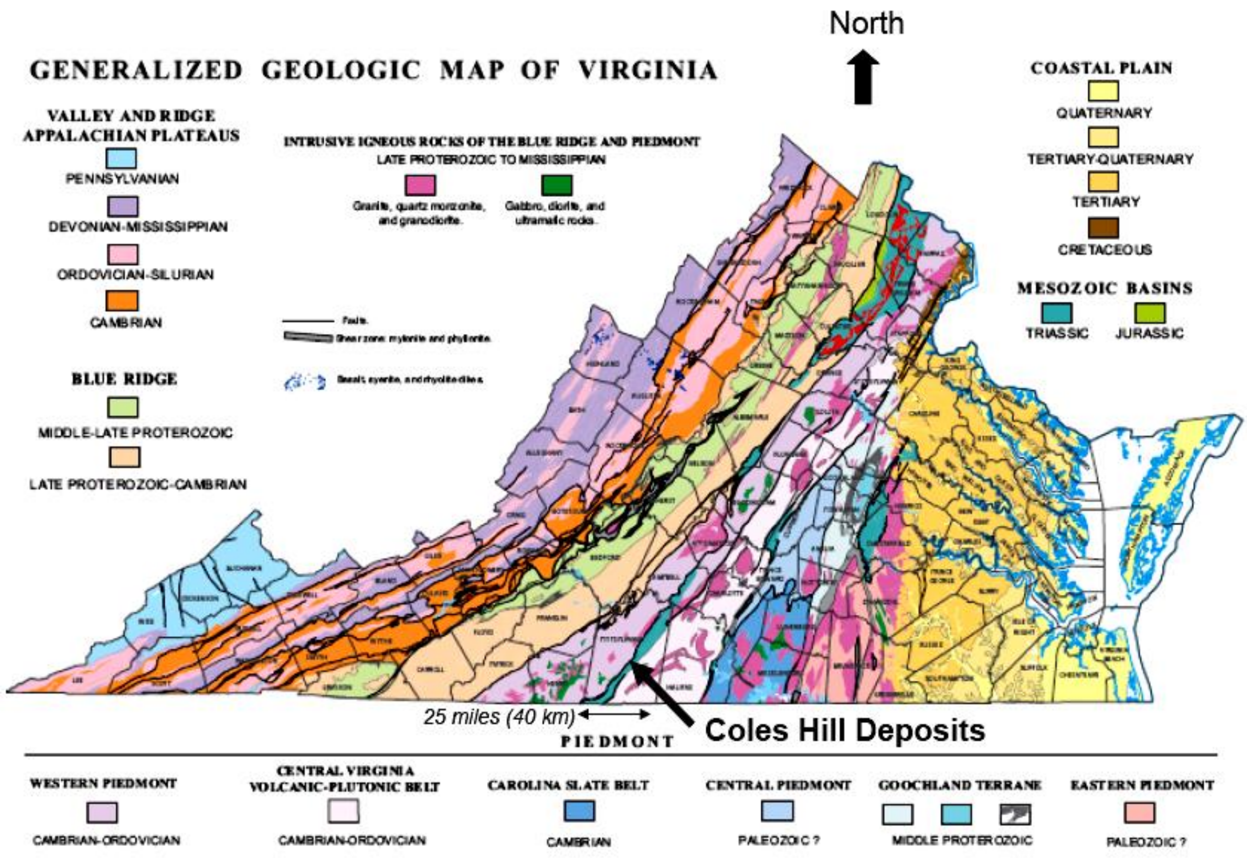


Figure 4.1: General location of Coles Hill Uranium Deposit in Pittsylvania County, Virginia

4.1 Mineral Titles

The Virginia Uranium, Inc. position on the CHUP consists of fee simple ownership and leasehold interests in the mineral and surface rights to a portion of the Coles property and the contiguous Bowen property (described in Item 4.0 of this report), as well as other properties. The total mineral rights and leases cover approximately 2,940 acres (1,190 hectares), including about 2,296 acres (929 hectares) in surface rights (Figure 4.2) (Table 4.1). Survey plots of the Coles farm lands and the Bowen farm lands are available from the land offices in Chatham, Virginia. The property plan was constructed from legal survey plots obtained from Pittsylvania County. Access to the property is provided by public-access roads to property under control by Virginia Uranium, Inc..

Two uranium deposits have been delineated on the CHUP. The SCHD is centered near latitude 36º52’18”N, longitude 79º18’00”W on the Spring Garden USGS 7.5-minute quadrangle topographic map, and the NCHD is centered near latitude 36º52’43”N, longitude 79º18’12”W on the Gretna USGS 7.5-minute quadrangle topographic map.

The CHUP consists of fee simple ownership and leasehold interests in the mineral and surface rights to a portion of the Coles property and the contiguous Bowen property, as well as other properties as shown in Table 4.1.

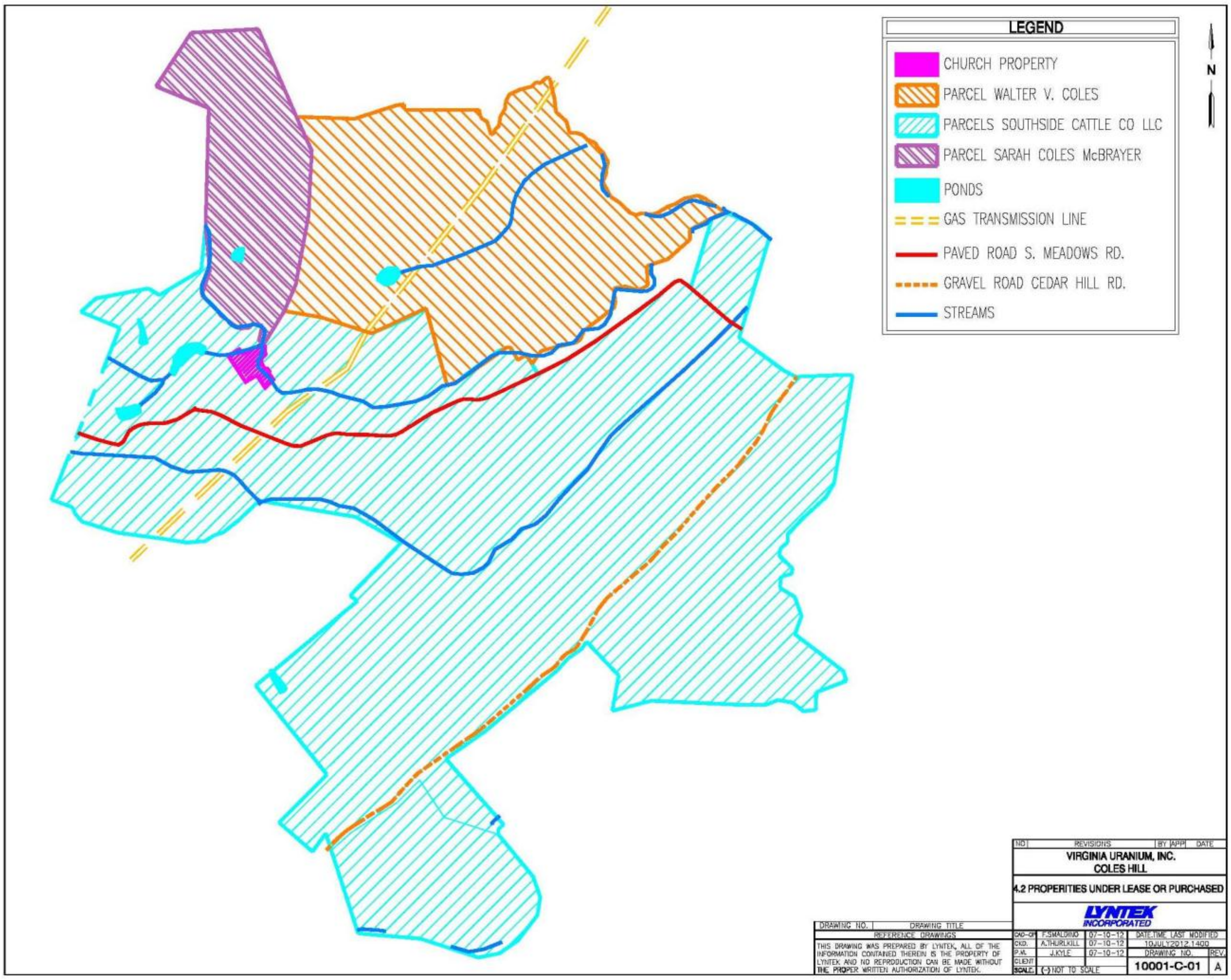


Figure 4.2: Virginia Uranium, Inc. Properties under lease or purchased

TABLE 4.1: PROPERTY OWNERSHIP AND LEASE				
Item		**Surface Rights (Acres)**	**Mineral Rights (Acres)**	**Term or Expiration Date**
A	Coles Hill, LLC Mineral Lease Dated 4/4/2007	252	904*	12/31/2045
B	Bowen Minerals, LLC Mineral Lease Dated 4/4/2007		112	12/31/2045
	Total of Parcels A & B	**252**	**1,016**	
C	Crider Option for Surface Rights Related to Bowen Lease Dated 5/29/2007	112		5/29/2037
D	Burt Land Purchase Dated 5/30/2007	0	767	2045
	Southside Cattle, LLC	767	0	
E	Coles Option Dated 5/31/2007 (Parcels 1 (148 acres) and 2 (206 acres))	354	354	12/31/2045
	Church Property	0	8	
F	Marline Purchase (Recorded 8/9/2007) Southside Cattle, LLC	8	0	None
G	Additional Land (Purchased 10/10/2007) Holmes	0	226	2045
	Southside Cattle, LLC	226	0	
H	Additional Land (Purchased 10/31/2007) Timberland	0	0	None
	Southside Cattle, LLC	410	410	
I	Additional Land (Purchased 11/6/2007) Martin	0	0	None
	Southside Cattle, LLC	167	167	
J	Jackson Land	0	0	**None**
	Southside Cattle, LLC	406	406	
	Total Contiguous Project Area	**2,702**	**3,346**	
*Protected Area in the Coles Hill, LLC Mineral Lease restricts surface rights but allows mineral rights by underground mining on about 648 acres (per 6/8/2007 Amendment)				

The use of surface rights has been restricted by the leases in about 652 acres near the historical Coles farm house (Protected Area). The type of mining that is allowed in the Protected Area is limited to underground mining by the terms of the lease with Coles Hill, LLC.

Surveyed land plots are available in Chatham, the county seat. After legal review, VAE is satisfied with ownership, title and agreements pertaining to the CHUP. The total mineral rights and leases cover approximately 2,940 acres (1,190 hectares), with about 2,296 acres (929 hectares) in surface rights. No obligations must be met until 2020 to retain any of the above interests.

4.2 Royalties, Agreements and Encumbrances

Surveyed land plots are available in Chatham, the county seat. The ownership and title to the surface and mineral rights for the CHUP are shown in (Table 4.1). John Kyle has reviewed deeds, leases, deed references, map references, legal opinions regarding standard practice of the transfer of surface and mineral rights, purchase agreements, sales closing documents, legal descriptions, maps of properties, and property title insurance documents regarding the land and mineral rights under control by Virginia Uranium, Inc and is of the opinion (not a legal opinion) that the company controls the land and mineral rights necessary to support the mining activities envisioned in this report. The control of the mineral and surface rights by Virginia Uranium, Inc. is not expected to be impacted in any way by the Plan of Arrangement.

4.2.1 Coles Lease and Bowen Lease

On April 4, 2007, Virginia Uranium, Inc. entered into a deed of mineral lease with Bowen Minerals, LLC (Bowen Lease) and a deed of mining lease with Coles Hill, LLC (Coles Lease and collectively with the Bowen Lease, “Leases”). Pursuant to the Leases, Virginia Uranium, Inc. was granted the sole and exclusive right to drill, quarry, mine, process, store, remove, and sell all of the uranium and all other fissionable source materials located on or under the lands of the two adjoining properties. The Leases expire on December 31, 2045, unless otherwise terminated or extended as agreed between the parties.

The Protected Area, as described in the Coles Hill Lease, contains historical sections for preservation on which surface activities are only allowed by written permission from the owners of the Protected Area. On June 8, 2007, Coles Hill, LLC amended their lease to allow for underground mining in the Protected Area as long as the mining does not disturb, harm, or damage the historic or other structures located within the Protected Area or restrict the enjoyment thereof. On November 6, 2007, the owners of the surface rights gave written permission for ten holes to be drilled in the Protected Area.

On each anniversary of the Effective Date after December 31, 2020, Virginia Uranium, Inc. has agreed to pay minimum annual rent (Anniversary Payment) to Bowen Minerals, LLC under the terms of the Bowen Lease and to Coles Hill, LLC under the terms of the Coles Lease. In addition, Virginia Uranium, Inc. has agreed to pay Coles Hill, LLC and Bowen Minerals, LLC, as applicable, an earned revenue royalty at a fixed percentage of the actual price per pound of U_3O_8 received by Virginia Uranium, Inc. for arms length sales to third parties. These costs have been included in the economic analysis.

4.2.2 Land Acquisition and Option Agreements

Uranium exploration is regulated by Virginia Department of Mines, Minerals, and Energy (DMME) and a permit must be obtained to conduct exploration activity. Virginia Uranium, Inc. applied for and obtained a permit on November 20, 2007 and updated on October 26, 2011 to conduct exploration activities on 194 acres and to drill 40 holes on the CHUP. This permit allows Virginia Uranium, Inc. to conduct drilling to a depth in excess of 50 feet for the purpose of determining the location, quantity, or quality of uranium ore.

Land acquired pursuant to the terms described in the agreement dated May 22, 2007 between Fred W. Burt and Shirley C. Burt (Burts) and Virginia Uranium, Inc. was assigned to Southside Cattle Company (SCC), a 100% subsidiary of Virginia Uranium, Inc.. SCC acquired approximately 767 acres of land contiguous to the South Coles Hill Deposit (Burt Lands), excluding any mineral rights associated with the Burt Lands (Reserved Minerals)(Table 6.1). SCC also acquired an option to lease the Reserved Minerals (Burt Option) from the Burts, which option may be exercised by SCC at any time prior to 2045. Upon exercise of the Burt Option, Virginia Uranium, Inc. through SCC shall have the right to remove and sever all such Reserved Minerals from the Burt Lands. Pursuant to an option agreement (Crider Option Agreement) dated June 1, 2007, between Roy Crider and Connie Crider (Criders) and Virginia Uranium, Inc., the Criders have granted to Virginia Uranium, Inc. an option to purchase approximately 112 acres of land which covers part of the surface rights of the South Coles Hill Deposit exercisable for a period of 30 years commencing on June 1, 2007. The minerals rights under these 112 acres are covered by the Bowen Minerals LLC Lease.

Pursuant to an option agreement (Coles Option Agreement) dated May 31, 2007, among Virginia Uranium, Inc., Walter Coles, Senior and Alice C. Coles, Virginia Uranium, Inc. acquired an option (Coles Option) to purchase approximately 354 acres of land, which covers the southern portion of the Project area (Table 4.1). The option was exercised by the company in the Fall of 2011.

4.2.3 Marline Property Purchase

Southside Cattle Company (SCC), on behalf of Virginia Uranium, Inc., purchased 8 acres of land that was owned by Marline and sold at auction for failure to pay taxes to Pittsylvania County in July 2007. The transfer of deed was recorded on August 9, 2007. This purchase conveyed only the surface rights to Virginia Uranium, Inc. and the mineral rights are retained by the churches that were part of Camp Pitt Church Camp.

4.2.4 Additional Property Purchase of October 10, 2007

SCC, on behalf of Virginia Uranium, Inc., purchased the surface rights to approximately 226 acres of then non-contiguous property for set-back purposes on October 10, 2007. The subsequent purchase of land on November 6, 2007 made this land contiguous to the Project site. The original landowner retains the mineral rights but has granted SCC the option to lease the mineral rights at any time prior to the Year 2045 for the same terms as the Coles and Bowen terms. Upon the exercise of the mineral rights option, Virginia Uranium, Inc. through SCC shall have the right, at any time, to remove and sever the mineral rights from this property for a period of 20 years.

4.2.5 Additional Property Purchase of October 31, 2007

SCC, on behalf of Virginia Uranium, Inc., purchased approximately 410 acres of contiguous property on October 31, 2007 for process and set-back purposes. No mineral lease payments are associated with this land purchase.

4.2.6 Additional Contiguous Property Purchase of November 6, 2007

SCC, on behalf of Virginia Uranium, Inc., purchased approximately 167 acres of contiguous property on November 6, 2007. At closing, a fee simple title was conveyed to SCC. This property purchase allowed the non-contiguous property purchase of October 10th to be contiguous to the Project site. The original landowner retains the mineral rights but has granted SCC the option to lease the mineral rights at any time prior to the Year 2045. Upon the exercise of the mineral rights option, Virginia Uranium, Inc. through SCC shall have the right to at any time to remove and sever the mineral rights from this property for a period of 20 years for the same terms as the Coles and Bowen mineral lease terms.

4.2.7 State Road 690 (Coles Road)

The state road that goes through the property is a prescriptive easement in favor of the Commonwealth of Virginia but the mineral rights remain with the landowners. Prior to the relocation or closure of this road, consent will be requested from the Commonwealth of Virginia and/or local authorities.

4.2.8 Santoy Resources Ltd. History

Santoy Resources Ltd. (TSX.V: **SAN**)(Virginia Energy Resources, Inc.), a former Canadian publicly traded firm (Santoy), announced on July 21, 2009 that it completed an acquisition of Virginia Uranium Ltd Santoy changed its name to Virginia Energy Resources, Inc. and as a result of the transaction initially acquired a 20.8% interest in VA Uranium Holdings, Inc., which was the parent company of Virginia Uranium, Inc.

4.3 Environmental Liabilities and Permitting

4.3.1 Residual Environmental Liabilities

No surface workings are present on the CHUP. No new residential, commercial, or industrial property development or construction was observed on or near the site.

Neither Lyntek or BRS is aware of any environmental liabilities related to the CHUP. Exploration holes drilled by Marline and Virginia Uranium, Inc. have been abandoned by cementing them from bottom to top as required by Virginia state regulations. Virginia Uranium, Inc. has abandoned most of its holes by cementing them from bottom to top, but one drill hole was abandoned by turning it into a monitoring well to study the fracture hydrology of the deposit.

NCHD and SCHD are mainly covered by a few meters of barren material, and drilling was the only invasive method used to explore the deposits. Other non-invasive methods, such as ground radiometry, magnetic, and gravity, have also been used. Prior to the conduct of new drilling, an exhaustive Uranium Exploration permit was submitted to and approved by DMME. This permit included an evaluation of the local environment, detailed operations and reclamations plans for drill sites, wetlands delineation, study of critical habitats for endangered or threatened species, and archeological, cultural, and historical resources. During operation and reclamation, the operations were and will continue to be regularly inspected by a DMME mine inspector.

4.3.2 Required Permits and Status

Applications for permits required for recent activities have been processed and received from necessary agencies as discussed above.

Mill licensing will be in accordance with existing US NRC and US EPA regulations for the mill life cycle. The license will be a Source Material License as required by Chapter 10 Code of Federal Regulations Part 40. The mill license is expected to require financial assurance for the cost of mill closure by a third party.

The mine will require a permit from the Virginia Division of Mineral Mining under Code of Virginia Title 45.1 – Mines and Mining Chapter 16 - Permits for Certain Mining Operations; Reclamation of Land and 4VAC Chapter 25- 31 Reclamation Regulations for Mineral Mining. Following lifting of the uranium moratorium, the Virginia General Assembly will consider whether to apply these mine permit requirements to uranium mining, to modify them for uranium, or to write separate mining rules specifically for uranium (see section 4.4).

For both the mine and the mill, existing and future State of Virginia, US EPA, and US NRC standards will apply to protection of radiological health and safety, water and air quality, ecological resources and cultural resources. VDEQ permits will probably be required for particulate and other airborne pollutant releases, for surface water impoundments, and for surface and ground water discharges, specifically:

- Virginia Water Protection Individual Permit
- Virginia Air Quality Permit
- Virginia Pollutant Discharge Elimination System Permit
- Virginia Dam Construction Permit
- Virginia Dam Low Hazard Potential Regular Operation and Maintenance Certificate.

Other than the exploration permit, no applications have been made for permits or licences.

4.4 Other Significant Factors

Currently, the Commonwealth of Virginia does not have uranium mining regulations and until such regulations are developed and allowed by statute, no applications for uranium mining permits can be accepted by state regulatory agencies. In November 2008, the Virginia Coal and Energy Commission created a sub-committee to evaluate uranium mining. The sub-committee engaged the National Academy of Sciences to undertake a study of potential impacts of uranium mining in Virginia. The NAS report, *Uranium Mining in Virginia*, was released on December 16, 2011. The sub-committee also commissioned another study performed by Chmura Economic and Analytics entitled *The Socioeconomic Impact of Uranium Mining and Milling in the Chatham Labor Shed, Virginia* that was released on November 11, 2011. Subsequently, in January 2012, Virginia Gov. McDonnell formed the Uranium Working Group, consisting of members from relevant departments of Virginia state government, to examine these and other studies to develop a draft regulatory framework for uranium mining rules, from which the Virginia Coal and Energy Commission will make a recommendation to the legislature. Virginia has a bicameral legislature so bills enabling uranium mining regulation development must be approved by both the House and the Senate. The Governor must then sign the bill for it to become enacted into law.

ITEM 5. ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

5.1 Physiography

The topography at the CHUP is subdued, with typical rolling hills of the Piedmont Province having elevations ranging from about 560 feet (170 meters) in Mill Creek to approximately 700 feet (213 meters) at Coles Hill. The area is drained by Mill and Whitethorn Creeks, with Mill Creek entering Whitethorn Creek 1.4 miles (2.2 kilometers) east of Coles Hill.

5.2 Access

The property is accessed by a major north-south highway, U.S. Highway 29, and is between the cities of Danville and Lynchburg. Danville, historically a textile mill and tobacco town, is about 30 miles (48 kilometers) to the south. Lynchburg, a city that has a significant nuclear industry presence, is about 50 miles (81 kilometers) to the north. The site can be accessed by driving through the towns of Chatham or Gretna, and then secondary roads. From Chatham, Virginia, secondary paved roads such as Chalk Level Road (State Road 685) intersect directly with the gravel Coles Road (State Road 690) that bisects the project area. A number of dirt roads and lanes provide access to the Coles farm land and Bowen farm lands that form the CHUP. The Company has also acquired rights to nearby lands for use in its operations. See Figure 5.1.

Figure 5.1: Virginia Resources, Inc. Property Location Map

5.3 Vegetation

The vegetation is characterized by an oak-pine forest normally classified as a temperate broadleaf deciduous forest. It is typically characterized by four layers; the canopy, dominated by mast-producing oaks and hickories, is 60 to 100 feet above the forest floor. Below it lies an understory of smaller trees such as dogwood and redbud; a shrub layer frequently dominated by heaths such as rhododendron, azalea, and mountain laurel; and an herb layer of diverse perennial forbs, mosses, lichens, and clubmosses. Woody vines are conspicuous in more moist habitats; most common are wild grape, Virginia creeper, and poison ivy.

5.4 Climate and Length of Operating Season

The climate in the region is characterized by warm temperatures during the summer months. Cooler temperatures in the winter months produce some freezing and snow, averaging about 10 inches annually from mid-November through mid-March. The area is warm, with average maximum temperatures in Chatham and Danville over 80°F from June through September and average maximum temperatures between 47°F and 52°F from December through February. There is no defined rainy season; however, severe storms accompanied by heavy rain may occur from June through September. Occasional rain and cooler temperatures occur in December and January. The average annual precipitation is about 42 inches, with monthly averages varying from about 3.0 inches to 4.6 inches, which must be considered within the design of the mine and plant facilities. The humidity averages about 80%, with higher average values near 90% occurring in August and September. A pleasant climate allows for the basis for an all-year mining operation schedule.

5.5 Sufficiency of Surface Rights and Location of Mining Facilities, Waste Dumps, Processing Plant, Tailings Facilities, and Waste Disposal Areas

It will be necessary to obtain suitable additional surface property to accommodate space for tailings cells.

5.6 Availability of Power, Water, and Manpower

The nearby Virginia power grid and Williams' Transco interstate gas pipeline provide a local source of natural gas and electrical power. There is adequate supply of natural gas and electricity nearby for the project that is within an economic distance to support the mining venture.

ITEM 6. HISTORY

6.1 History and Ownership

Since 1785, the Coles family has lived continuously on the Coles Hill farm in Pittsylvania County, Virginia. The land and mineral ownership at Coles Hill is private dating back to the original land grant.

In 1977 Marline targeted Danville Triassic Basin for uranium exploration. This area was targeted because of the nearby possible uranium source rock, stratigraphic traps possibly in the area, and indications of uranium from airborne surveys (Dribus 1978). Geologists of Marline Uranium Corporation first discovered the deposit in 1978.

A history of exploration and development in the project area is highlighted below:

1977
June: Marline initiated ground radiometric reconnaissance surveys.

1978
Mid-1978: Ground surveys led to discovery of uranium-bearing rocks, and a lease acquisition program was started.

September: First mineral leases acquired by Marline

December: Coles Hill lease was acquired

March 1979 surface sampling yielded a grab sample which assayed 0.50 weight % U_3O_8

1981
April: Virginia Coal and Energy Commission (CEC) undertook a study on uranium development in the Virginia Commonwealth, and created a Uranium Subcommittee in late summer.

October 1: Uranium mineral deposits at Coles Hill voted by Marline board of directors to be called Swanson Uranium Project.

1982

Legislation passed in the spring of 1982 in the Virginia Senate that prevents any Virginia agency from accepting permit applications for uranium mining before July 1, 1984 or until program for permitting uranium mining is established by statute.

December 1: Marline and Union Carbide Corporation entered into an agreement to complete a feasibility study by June 1984.

1983
February 7: Senate Bill 155 established the Uranium Administration Group, to examine uranium development "at specific sites in Pittsylvania County."

1984
July 13: UMETCO submitted studies for Marline and Union Carbide related to development of Coles Hill Deposit.

1985

The Uranium Subcommittee of the CEC and the UAG reported, "We now conclude that the moratorium on uranium development can be lifted if essential specific recommendations derived from the work of the Task Force are enacted into law."

1990
The Swanson Uranium Project is abandoned by Marline and the mineral leases reverted to the original owners.

With the resurgence of the demand for uranium and increasing prices the Coles family was approached by numerous mining companies seeking to lease the uranium mineralization at Cole Hill. Instead of leasing their property the Coles family created their own company.

2006

The Coles Family reached an agreement with the neighboring Bowen Family whose farm land encompasses a portion of the ore body. As a result, the two families agreed to form their own company. On January 16, 2007 Virginia Uranium, Inc. was formed.

6.2 Historic Drilling

A drilling program was initiated in 1979 by Marline. The first set of drill holes consisted of 256 drill holes (178 rotary percussion (RP) holes and 74 diamond and 11 partial drill holes drilled by Boart Longyear Contracting Services) drilled between 1979 to 1982. Of these, 24 RP holes were drilled outside the current project boundary. From January to March 1984, 3 NQ holes were drilled. Data from these holes are not available. Cores from drilling were placed in boxes and stored in a facility west of State Route 690.

Subsequently, the historic core samples were donated to the Virginia Museum of Natural History and are housed on site.

6.3 Historic Resource and Reserve Estimates

In 1982 Marline retained PAH to estimate resources for the Coles Hill Uranium Deposit. **The results of the historical mineral resource estimates were summarized in two reports titled "Geologic Reserves Coles Hill North Deposit, Pittsylvania County, Virginia" and "Geologic Reserves Coles Hill South Deposit, Pittsylvania County, Virginia" both dated August, 1982.** Using the acceptable method of the time, reserve estimates were summarized; the results are seen in Table 6.1. The method used by PAH to estimate "reserves" are as follows:

"The measured reserves include material which is within 50 feet of a drill hole or is between holes showing continuous mineralization of similar grade up to a distance of 120 feet. The indicated reserves include the balance of the material within the mineralized outlines. To complete the grade information on some sections where drilling is widely spaced, hole information was projected from adjacent sections. The grade information projected was done as discreet grade ranges or as the average of the mineralized column being projected. The selection of the projections was based on a judgment of how well the projections correlated with adjacent material."

TABLE 6.1: HISTORIC ESTIMATES

%U_3O_8 Cutoff	Measured		Indicated		Total		Million Pounds
	Million Tons	% U_3O_8	Million Tons	% U_3O_8	Million Tons	% U_3O_8	
South Coles Hill Deposit (SCHD)							
0.150	3.28	0.232	1.46	0.209	4.74	0.225	21.3
0.125	3.89	0.217	1.78	0.196	5.67	0.210	23.9
0.100	4.76	0.198	3.69	0.154	8.45	0.179	30.2
0.075	5.62	0.180	8.33	0.116	14.0	0.142	39.6
0.050	8.73	0.137	13.0	0.097	21.7	0.113	49.1
0.025	13.3	0.103	16.4	0.085	29.7	0.093	55.2
North Coles Hill Deposit							
0.150	0.557	0.204	0.376	0.225	0.933	0.212	3.96
0.125	1.07	0.170	0.869	0.172	1.94	0.170	6.58
0.100	2.66	0.133	2.97	0.127	5.63	0.130	14.6
0.075	5.30	0.109	6.44	0.104	11.7	0.106	24.9
0.050	11.0	0.085	14.9	0.080	25.9	0.082	42.5
0.025	17.2	0.068	24.7	0.063	41.9	0.065	54.5

Estimates shown in Table 6.1 were reported by PAH as “Geologic Reserves”. This categorization is not recognized by CIM or other foreign codes. Using CIM guidelines, this historical estimate equates, in the author’s opinion, to measured and indicated mineral resources. However, neither the author nor another qualified person has performed sufficient work to classify the historical estimate as a current mineral resource and Virginia Energy Resources Inc. is not treating the historical estimate as a current mineral resource. Current mineral resource estimates are provided in Section 14 of this report.The historical estimates do not comply with current CIM standards, are not 43-101 compliant, and should not be relied upon.

It is the opinion of the author that the historic mineral exploration practices by Marline and Union Carbide was to the industry’s best practice standards at the time.

6.4 Prior Property Production

There has been no prior production from the property.

ITEM 7. GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geological Setting

The regional geologic setting of the CHUP is derived from Jerden (2001) and Henika's *Geologic Map of the Virginia Portion of the Danville 30 by 60 Minute Quadrangle* (2002). The project area is situated along the northwestern margin of the Chatham Fault Zone, which separates the Danville Triassic Basin (Mesozoic Basin) on the east from structurally deformed and metamorphosed crystalline rocks of the Piedmont physiographic province to the west (Figure 7.1 Hibbard, et. al. 2003). The Coles Hill uranium deposit is hosted in mylonitic quartzo-feldspathic gneiss of the Leatherwood Granite (Tappa et. al., 440 Ma, Kish et al., 1979), part of the Martinsville Intrusive Suite. In general, gneisses and mica schists of the Fork Mountain Formation and Martinsville Intrusive Suite are mapped as part of the Smith River Allochthon (475 Ma), a thrust-faulted nape which has transported these formations from their place of origin (Conley and Henika, 1973).

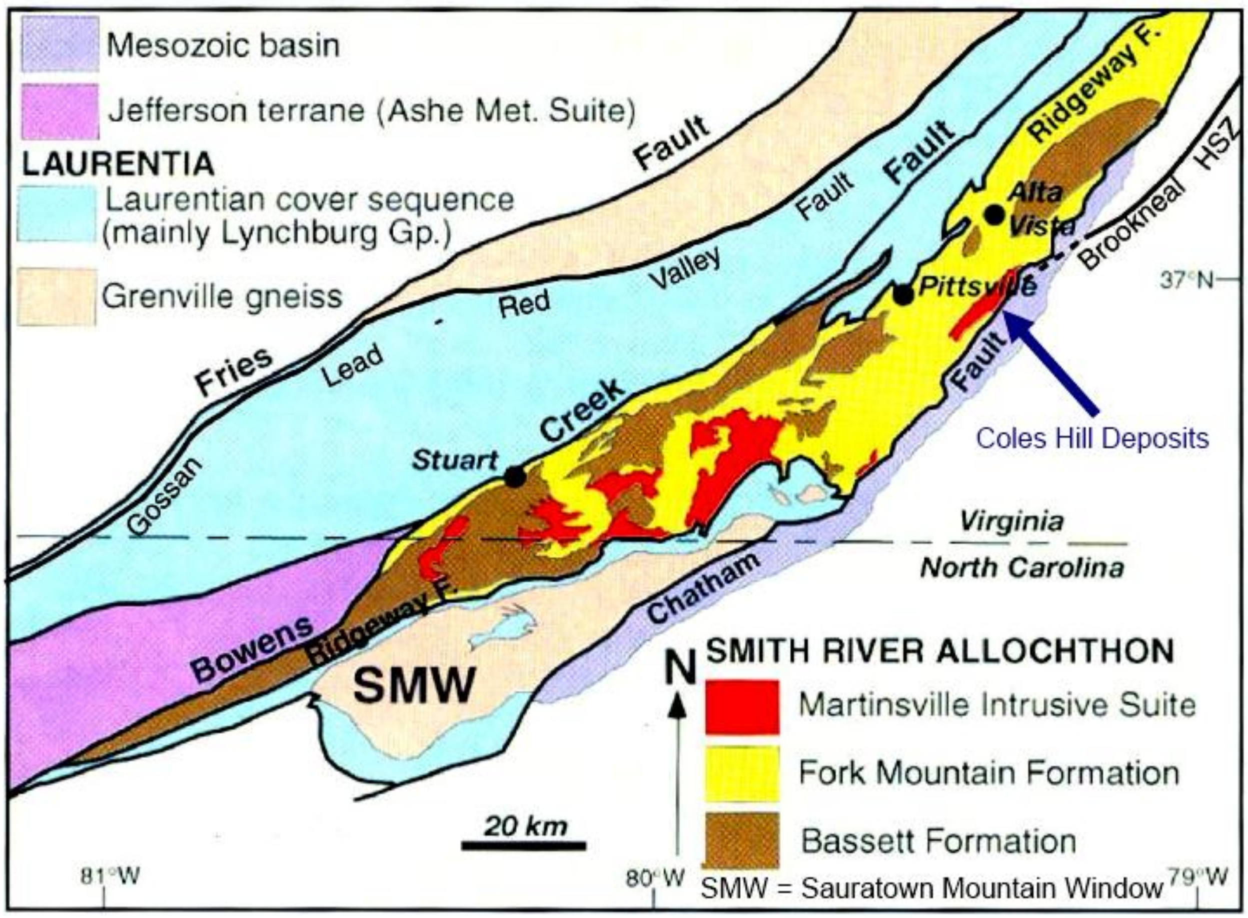


Figure 7.1: Regional Geologic Setting

The geology of the CHUP deposit was mapped by Marline and Union Carbide geologists, as well as Henika and Thayer (1983) and modified by Jerden (2001). The Chatham Fault, was named by Meyerston in 1963 and mapped and studied by Lineberger (1983), delineates the normally-faulted northwest margin of the Danville Triassic Basin. According to Jerden, the uranium deposits are hosted within a fault-bounded wedge of late Precambrian- early Paleozoic mylonitic quartzo-feldspathic gneiss with lesser amphibolite, which is found along the northwest side of the Chatham Fault.

7.2 Local Geology

The Coles Hill uranium deposit is comprised of two known areas of significant uranium mineralization referred to as the North and South Coles. The South Coles contains approximately 60% of the current known resource, depending on cutoff and is slightly higher in average grade than North Coles. In both areas, mineralization continues to the surface. Mineralized zones are generally over 100 feet thick vertically and occasionally up to 200 feet thick. Figure 7.2 shows the geologic setting and general limits of mineralization, as indicated by drilling and surface expression, in plan view.

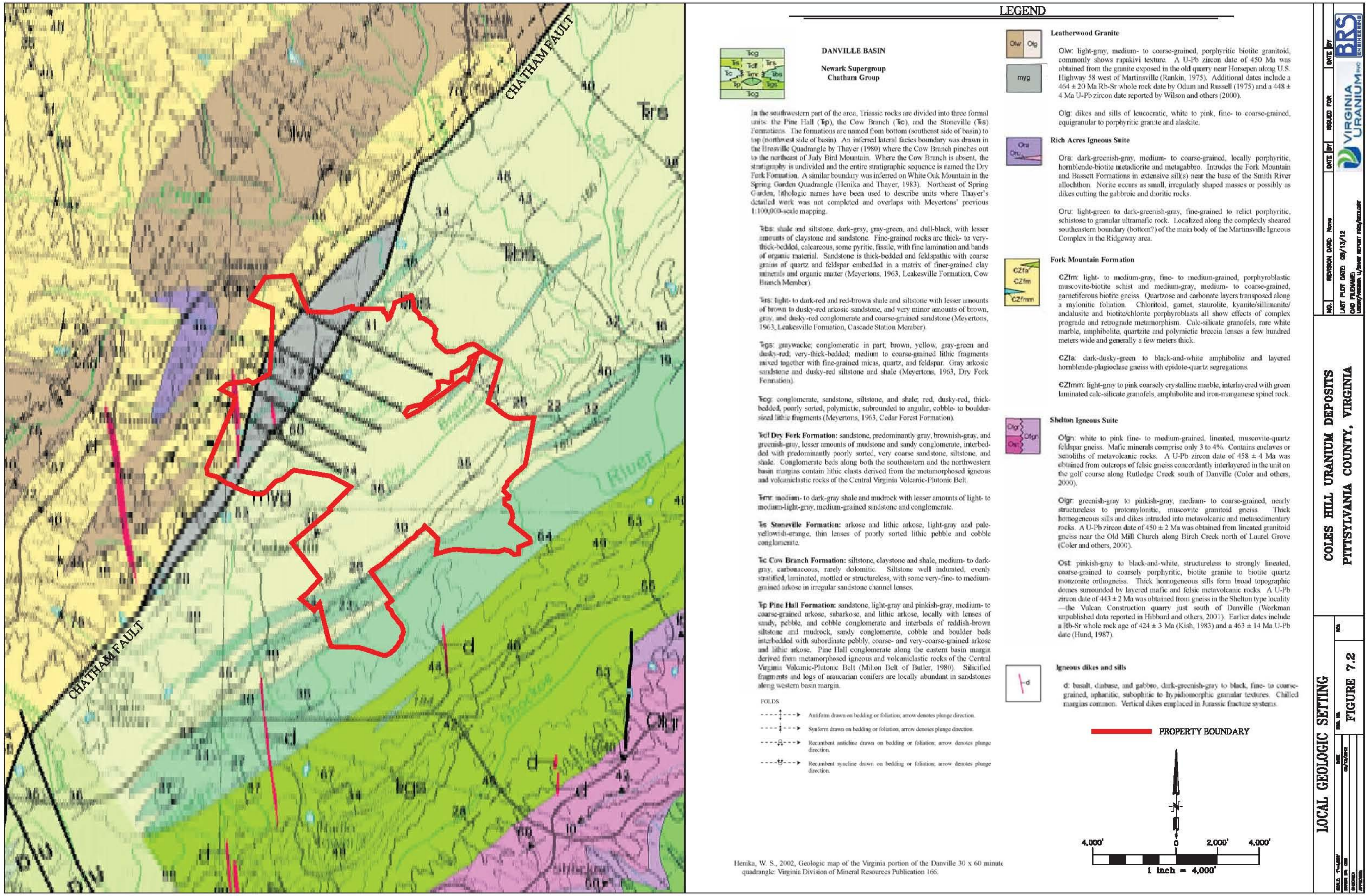


Figure 7.2: Local Geologic Setting

7.3 Geologic Controls

The Coles Hill deposits are structurally controlled, with ore primarily concentrated in within the Leatherwood Granite. Mineralization is ten to several tens of feet thick, whereas along plunge to the south, the ore is continuous over tens to hundreds of feet (UMETCO, July 13, 1984). Both South Coles and North Coles strike northeast-southwest. South Coles plunges approximately 30 degrees to the southwest and the North Coles plunges about 20 degrees to the northeast. The general plunge of the ore bodies to the south is 40 degrees and the general plunge of the bodies to the north is 20 degrees. Uranium mineralization at Coles Hill is hosted by the Leatherwood Granite which is bounded by the Chatham fault to the west and to the east and at depth by the Fork Mountain Schist.

7.3.1 South Coles Hill

Cross sectional views of the South Coles Hill area, displaying the major geologic units and uranium mineralization, are shown on Figure 7.3 and Figure 7.4, respectively. Figure 7.3 also shows the depth of surficial oxidation, which is typically less than 40 feet over the Leatherwood granite and up to 100 feet over the Paleozoic sediments. Cross section locations are shown on Figure 10.1, Drill Hole Location Map.

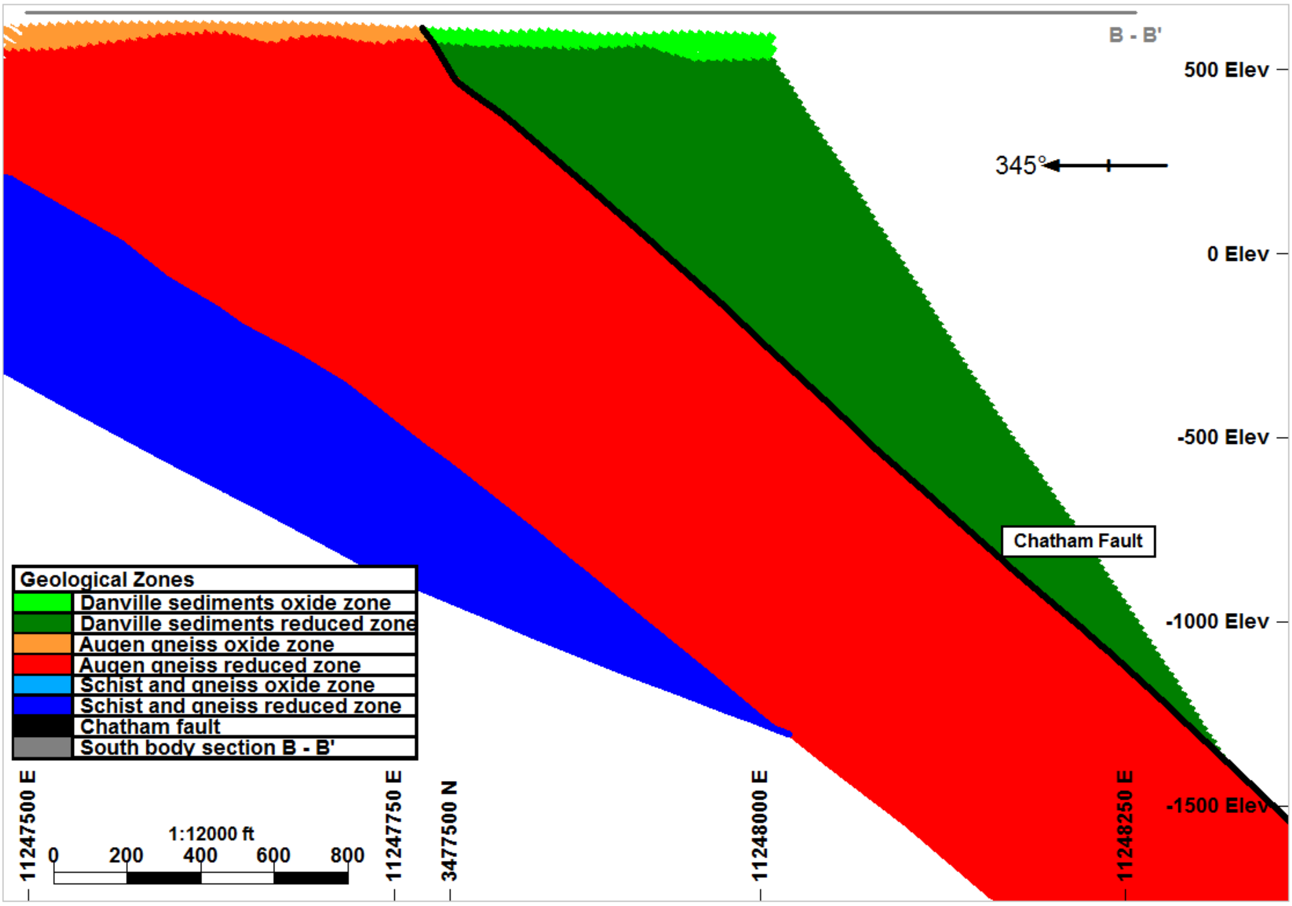


Figure 7.3: Sectional View South Coles Hill Geology

Figure 7.4 shows the mineralized blocks as interpreted from the drill data for the South Coles area. The upper bounding surface shown in black is the Chatham Fault. The lower bounding surface in red is the base of the Leatherwood Granite. At South Coles the majority of the mineralization is bounded by the Chatham fault with Triassic (Mesozoic) sediments to the east.

Note that mineralization, as shown in Figure 7.4, is apparently limited by the extent of drilling to the south and east not by any geologic controls or conditions. It is the author's opinion, based on available drill data, that mineralization is likely to extend at depth to the south and east on the currently defined mineralization. The reader is cautioned however that additional drilling in this area may or may not extend the known mineralization.

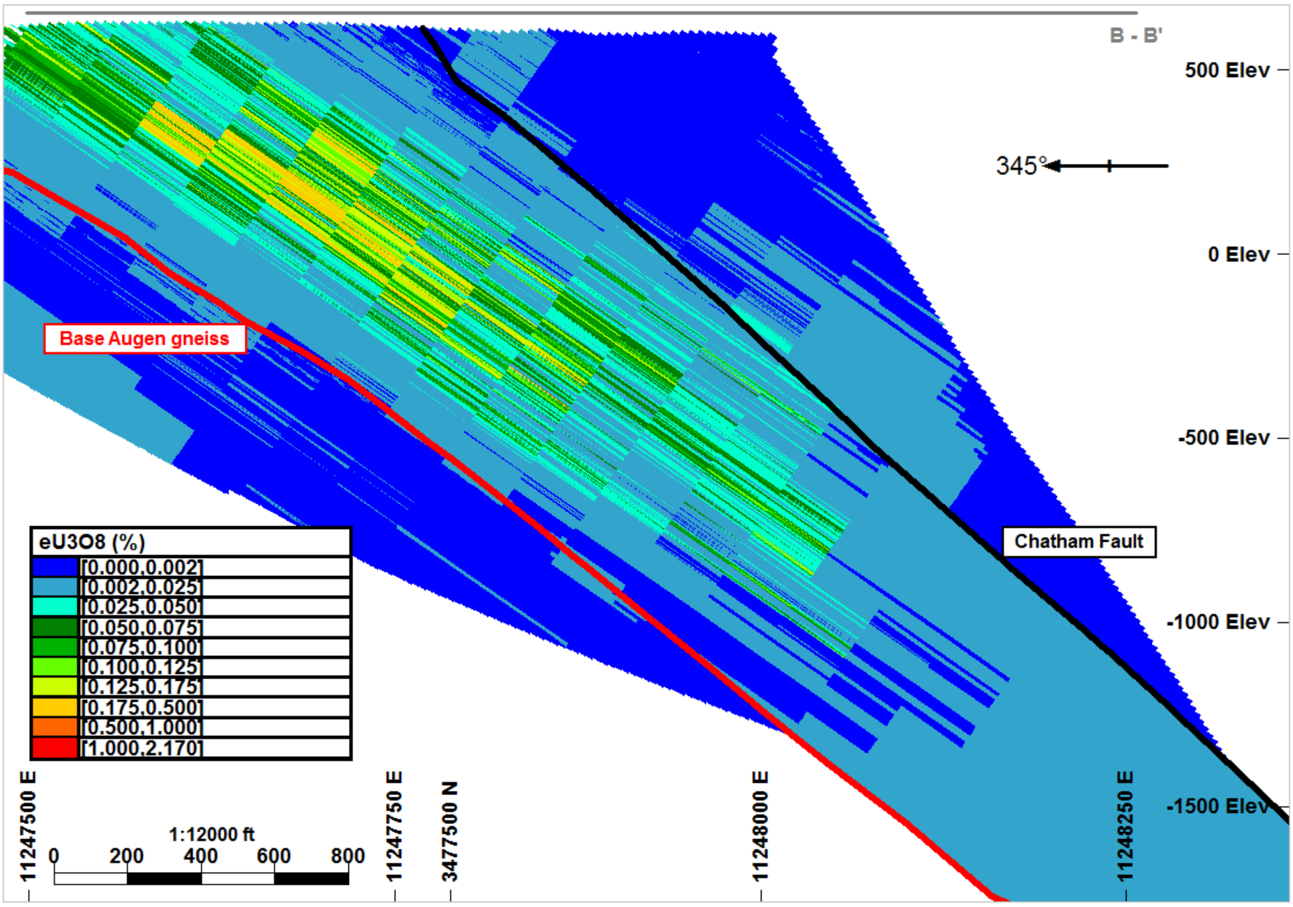


Figure 7.4: Sectional View South Coles Hill Mineralization

7.3.2 North Coles Hill

Cross sectional views of the North Coles Hill area, displaying the major geologic units and uranium mineralization, are shown on Figure 7.5 and Figure 7.6, respectively. Figure 7.5 also shows the depth of surficial oxidation, which is typically less than 40 feet over the Leatherwood granite and up to 100 feet over the Triassic meta-sediments. Note the thickening of the Leatherwood Granite at the North Coles as compared to South Coles. Cross section locations are shown on Figure 10.1, Drill Hole Location Map.

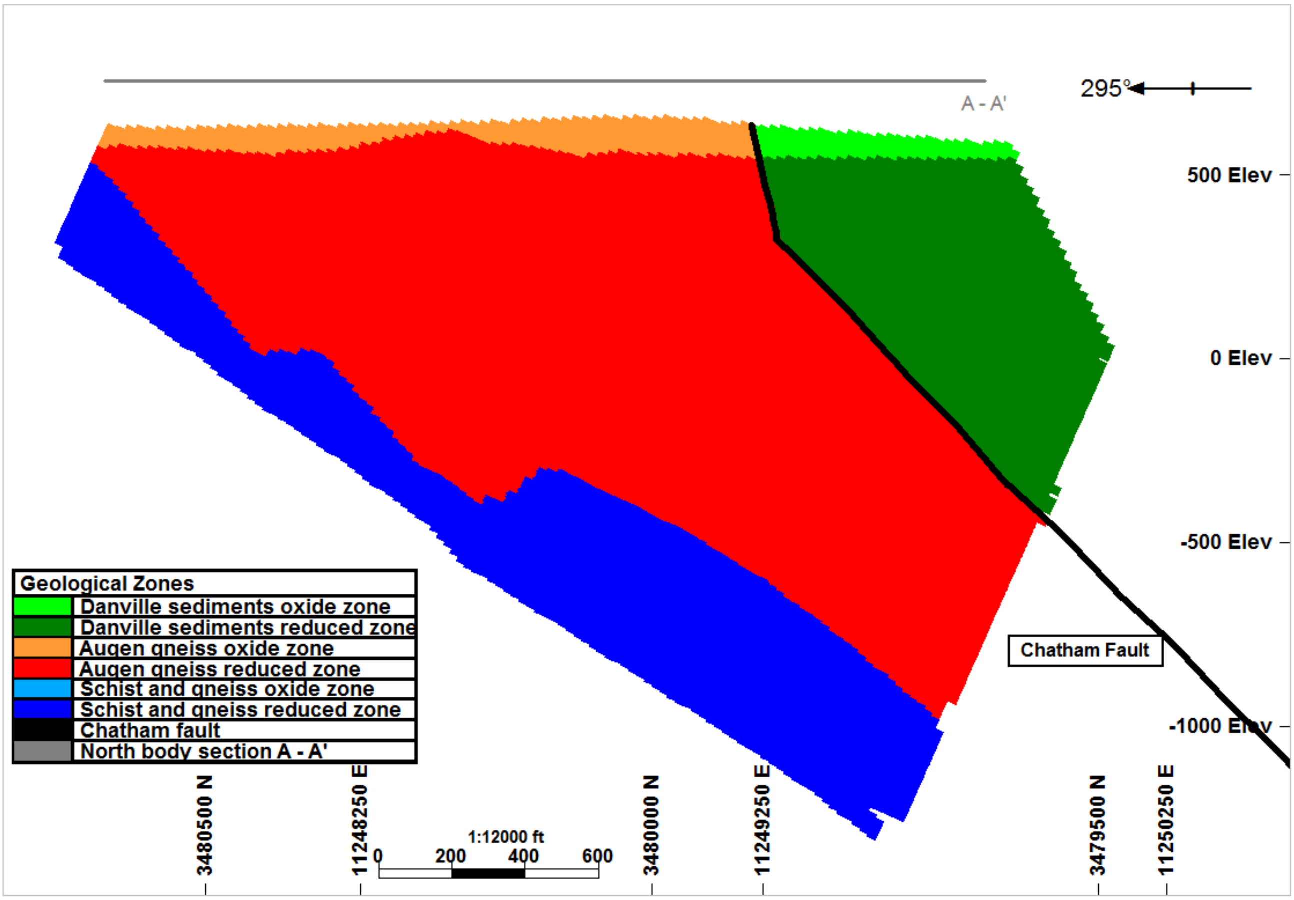


Figure 7.5: Sectional View North Coles Hill Geology

Figure 7.6 shows the mineralized blocks as interpreted from the drill data for the North Coles area. The upper bounding surface shown in black is the Chatham Fault. The lower bounding surface in red is the base of the Leatherwood Granite. At South Coles the majority of the mineralization is overlain by Paleozoic sediments.

Note at North Coles, as compared to South Coles, most of the mineralization is exposed at the surface and greater percentage of the mineralization is consequently not overlain by Triassic sediments. As with South Coles there are areas where the limits of mineralization are not defined by current drilling. It is the author's opinion, based on available drill data, that mineralization is likely to extend beyond the currently defined limits mineralization. The reader is cautioned however that additional drilling in this area may or may not extend the known mineralization.

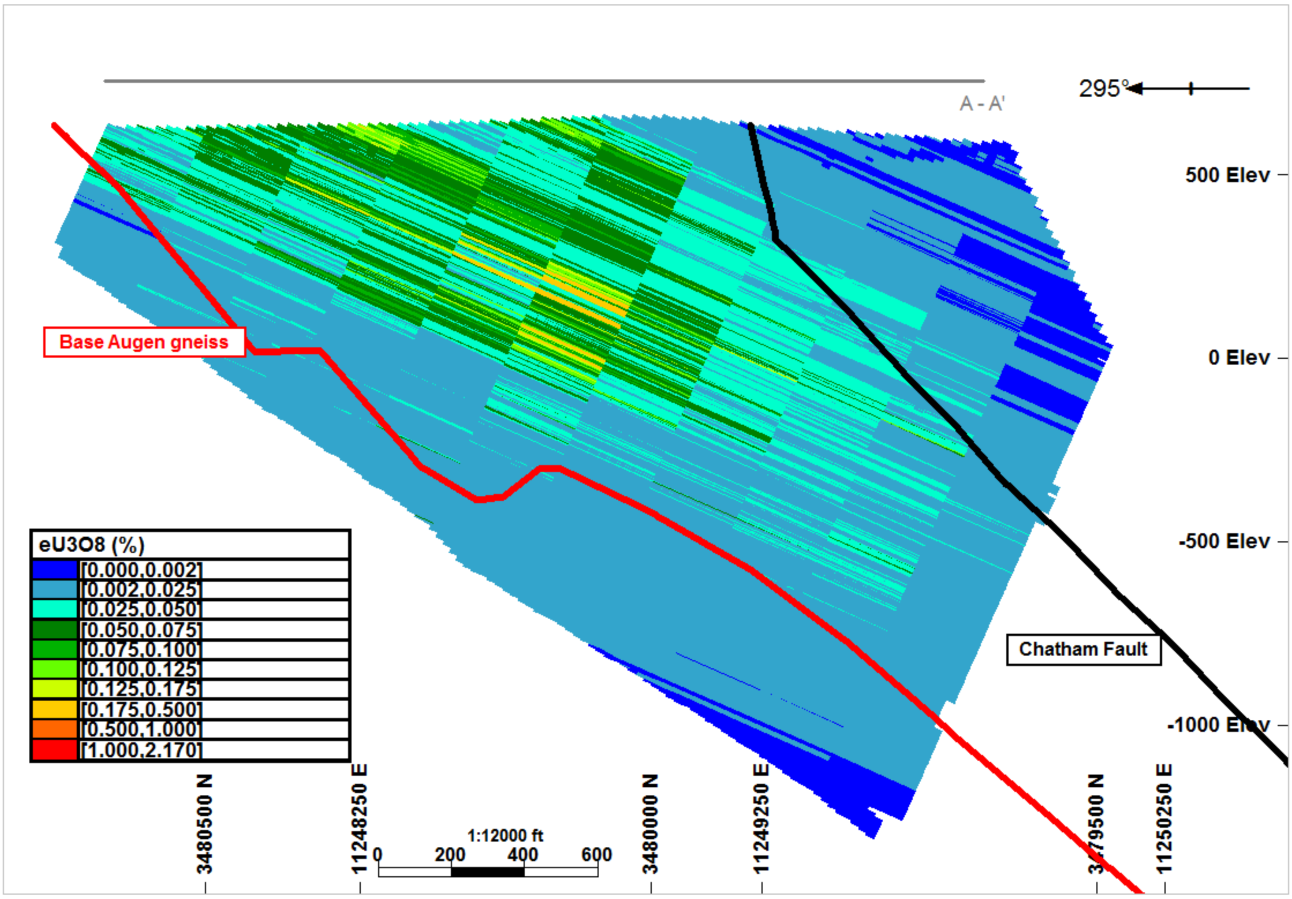


Figure 7.6: Sectional View North Coles Hill Mineralization

7.4 Mineralization

Uranium mineralization in the Coles Hill Uranium Property is hosted by three separate textural rock types. These three textural types are: (a) hematitic and mylonitic Leatherwood Granite (Figure 7.7), (b) hematitic amphibolite intrusive into Leatherwood Granite (Figure 7.8), and (c) densely fracture-filled Leatherwood Granite. All of the hosted rock types are found west of the Chatham Fault Zone.

Figure 7.7: Hematitic and mylonitic Leatherwood Granite

Figure 7.8: Hematitic amphibolites

At Coles Hill, one of the host rocks is mylonitized orthogneiss with depleted quartz and structurally controlled Na-metasomatism as albitization along the Chatham fault zone. Uranium is associated with hydrothermally filled fractures and veinlets as rims on and/or rimmed by apatite, chlorite, barite, titanium oxide, hematite, calcite, and pyrite. These are well represented as rims and veinlets in two examples as noted in photomicrographs such as in Figure 7.9, pyrite, titanium oxide, and uranium association and Figure 7.10 apatite, titanium oxide, chlorite, and uranium association.

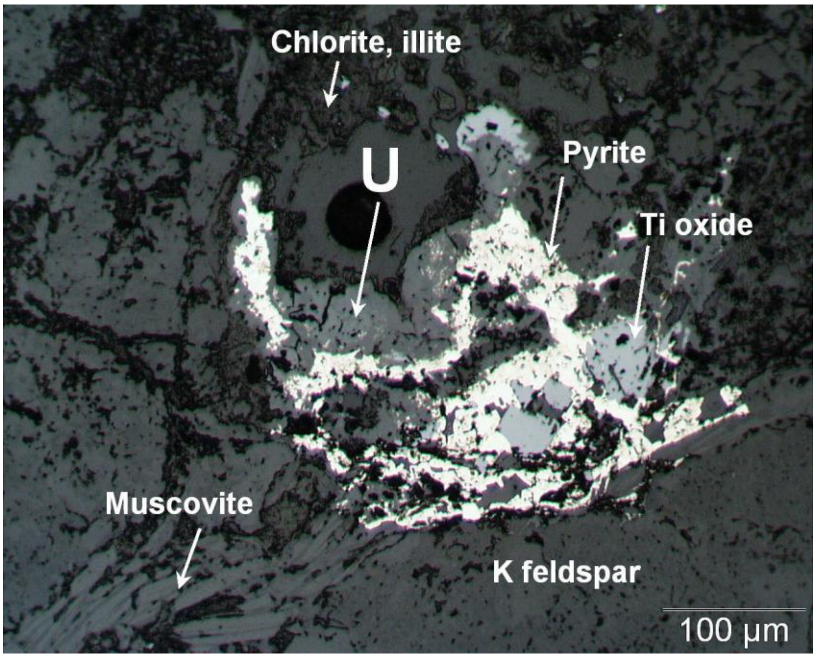


Figure 7.9: Reflected light: Uranium minerals are developed on pyrite and titanium oxide, and associated with chlorite.(S-602, 505 to 508 feet)

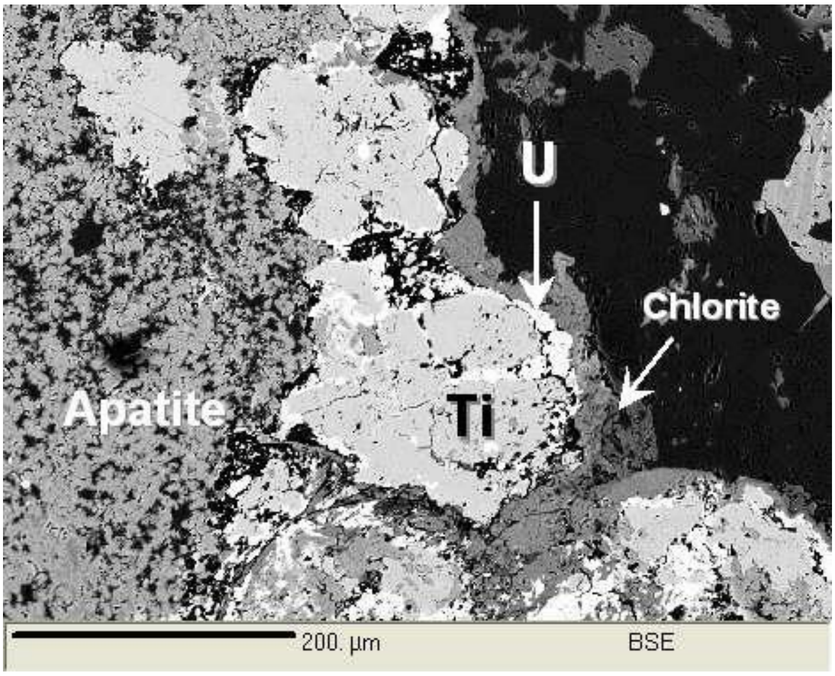


Figure 7.10: Microprobe, backscatter electrons: Titanium oxides are rimmed and impregnated by Uranium and associated with apatite and chlorite (S-603, 282.3 to 283.0 feet)

Uranium minerals identified by Klemm and Wagner (1980) are pitchblende, coffinite, and uraninite. The uranium high-grade shell is concentrated in an ellipsoidal pod or lense, that plunges south at 45° and occurs from the surface to at least 1,500 feet. As determined by downhole mapping, the regional foliation of the Leatherwood Granite gneiss is approximately N30°E, dipping at 30°SE, subparallel to the Chatham Fault.

ITEM 8. DEPOSIT TYPES

8.1 Deposit Types

The Coles Hill Uranium Deposit is a hydrothermal, fracture-hosted deposit, with tectonic events related to the Chatham Fault Zone allowing transport of hydrothermal solutions, alteration, and associated uranium mineralization. Chemical and mineralogical changes in the host gneiss and amphibolite resulted in areas of apatite enrichment, chloritization, and hematization. Oxygen depletion and resultant redox reactions from hematization of iron in magnetite and other mafic minerals neutralized uranium-transporting solutions, allowing for deposition of uranium-bearing minerals. The increased hematite and lowered magnetite content is reflected in anomalously low magnetic signatures for the Coles Hill Deposits.

In Dahlkamp (1993), the Coles Hill deposit is classified as Type 3, Class 3.1.1, which is an intragranitic vein deposit, having veins formed within the intrusion. Host rock criteria include highly differentiated leucogranitic rocks of crustal origin, and structural vein control by commonly one or more parallel oriented dilational fracture systems.

Previous reports state that the uranium deposition mechanism at Coles Hill is similar to that in the Athabasca Basin, as indicated by the presence of alteration minerals hematite, epidote, and chlorite. The deposition mechanism in the Athabasca Basin has produced significant-grade uranium mineralization, which might also occur in the untested deeper parts of the Coles Hill Deposits (Behre Dolbear, 2008). The author cannot independently verify this information and this information is not necessarily indicative of the mineralization on the property that is the subject of this technical report.

ITEM 9. EXPLORATION

The discovery of Coles Hill resulted from regional exploration conducted by Marline Uranium. In 1977, Marline targeted Danville Triassic Basin for uranium exploration based upon regional geologic models being developed in Canada related to the exploration of unconformity vein mineralization. Regional exploration conducted by Marline included airborne radiometric surveys. Geologists of Marline Uranium Corporation first discovered the deposit in March, 1979 when following up with ground surveys to test airborne anomalies. In addition to verifying radiometric anomalies on the ground outcrop samples were reported to contain significant uranium grades.

Since this early exploration effort, which led to the discovery of Coles Hill, exploration has been done almost exclusively by drilling. As described under Item 10, Drilling, from the period of 1979 through 1984, Marline Uranium and Union Carbide Corporation, in joint a venture, completed both rotary and core drilling programs.

In addition, the discovery outcrop was sampled as part of the preparation of the initial technical report on the project (Behre Dolbear, 2008). The soil sample referenced as #0364 from the discovery outcrop on Coles Hill Road was collected in a north-south direction and consisted of 2 kilograms of chips of saprolitic altered gneissic rock. The sample was located at GPS station with UTM coordinates of 17S 0651224N and 4082143E in the westerly drainage ditch along State Route 690, near the main entrance to the Coles Hill Manor house. The sample was placed in a plastic sack, secured and delivered by PAC to ACME Laboratory in Vancouver, British Columbia, an ISO 9001 Accredited Laboratory for Inductivity Coupled Plasma – Mass Spectrometry (ICP-MS) and uranium analysis. The soil sample contained 1,516.8 ppm (0.152 % U_3O_8) uranium by ICP-MS.

ITEM 10. DRILLING

The discovery outcrop was found in March 1979 and drilling on the site came after that point in time. Drilling on the project consists of some 263 total rotary percussion drill holes of which 74 holes were core drill holes with an additional 11 drill holes that were at least partially cored. All drill holes, rotary and core, were logged geophysically by commercial vendors.

Of the 263 drill holes, 258 are of a historic nature completed during the period of 1979 through 1984 by former operators Marline Uranium and Union Carbide Corporation (as UMETCO). Three core holes and two rotary percussion drill holes were completed by Virginia Uranium, Inc. in 2008. In addition, five of the historic rotary percussion drill holes that were re-opened and logged geophysically in 2008 to verify the historic data as discussed in section 12 of this report. **Error! Reference source not found.** shows the location of drilling on the property. Drill holes included both vertical and angle drilling.

The mineral deposits plunge approximately 40 degrees south at South Coles and 20 degrees north at North Coles. This magnitude of dip will affect the true thickness of the mineralization as observed in almost all drill holes, however, this was compensated for in the resource modeling which utilized block rotated to the strike and dip of the deposit in three dimensional space.

As described under Item 7 of this report, mineralization can be several 10's to 300 feet thick. Most of the mineralization is moderate in grade, less than 0.20 %eU_3O_8. Some high grade intercept in excess of 1% are observed in the data. Grade distribution of the deposit is lognormal as shown on Figures 14.4 and 14.5. Figure 10.2 shows the downhole distribution of grade, both chemical and radiometric, for hole S-603 completed in the South Coles area (Figure 10.1) in 2008. Drill hole S-603 contains significantly higher grade intervals within lower grade intersections which is typically of the more highly mineralized portions of the deposit.

Figures 7.4 and 7.6 show the overall distribution of grade, as projected from the block model of the deposit, in cross section for the South and North Coles Hill deposits, respectively.

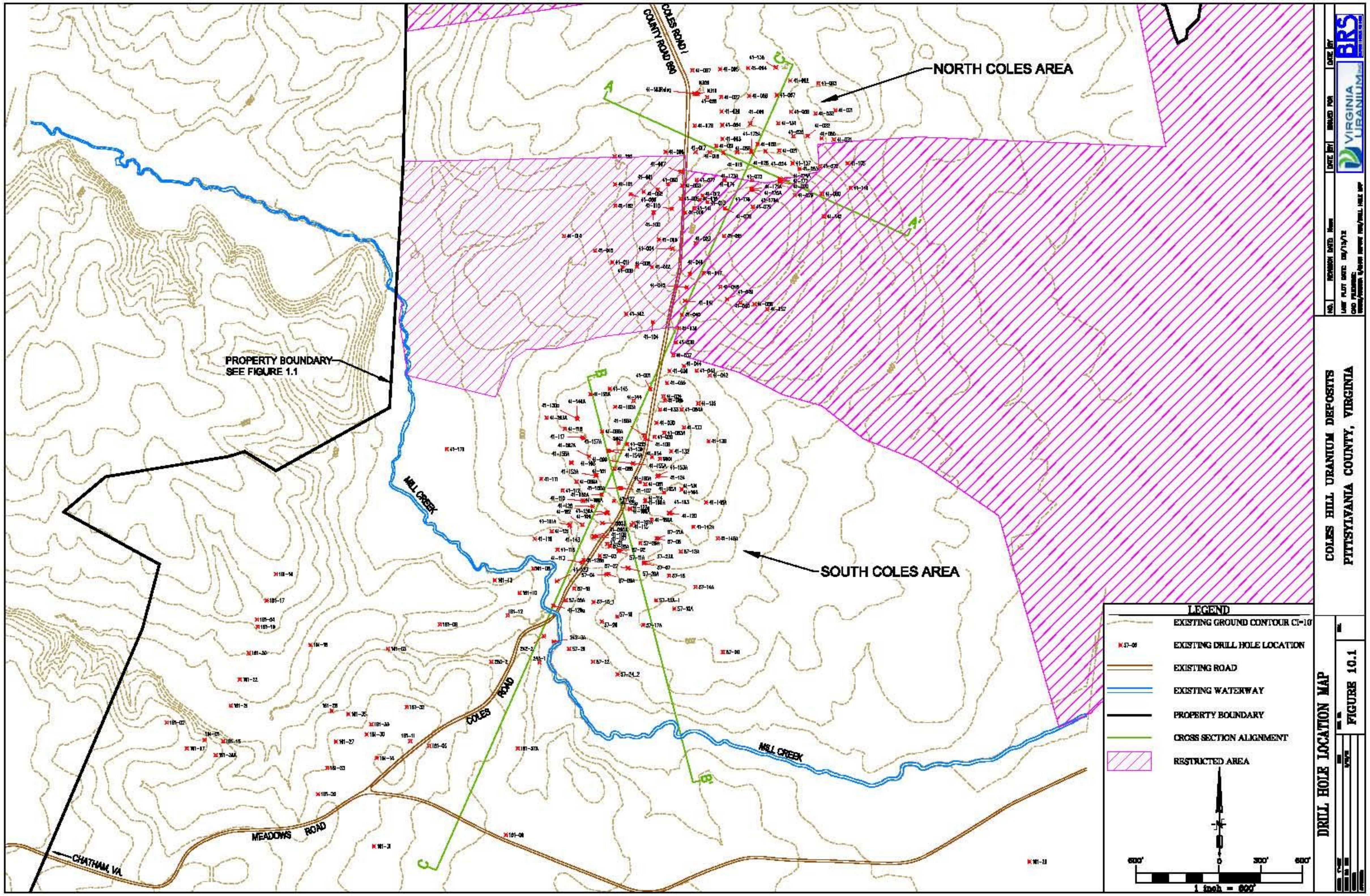


Figure 10.1: Drill Hole Location Map

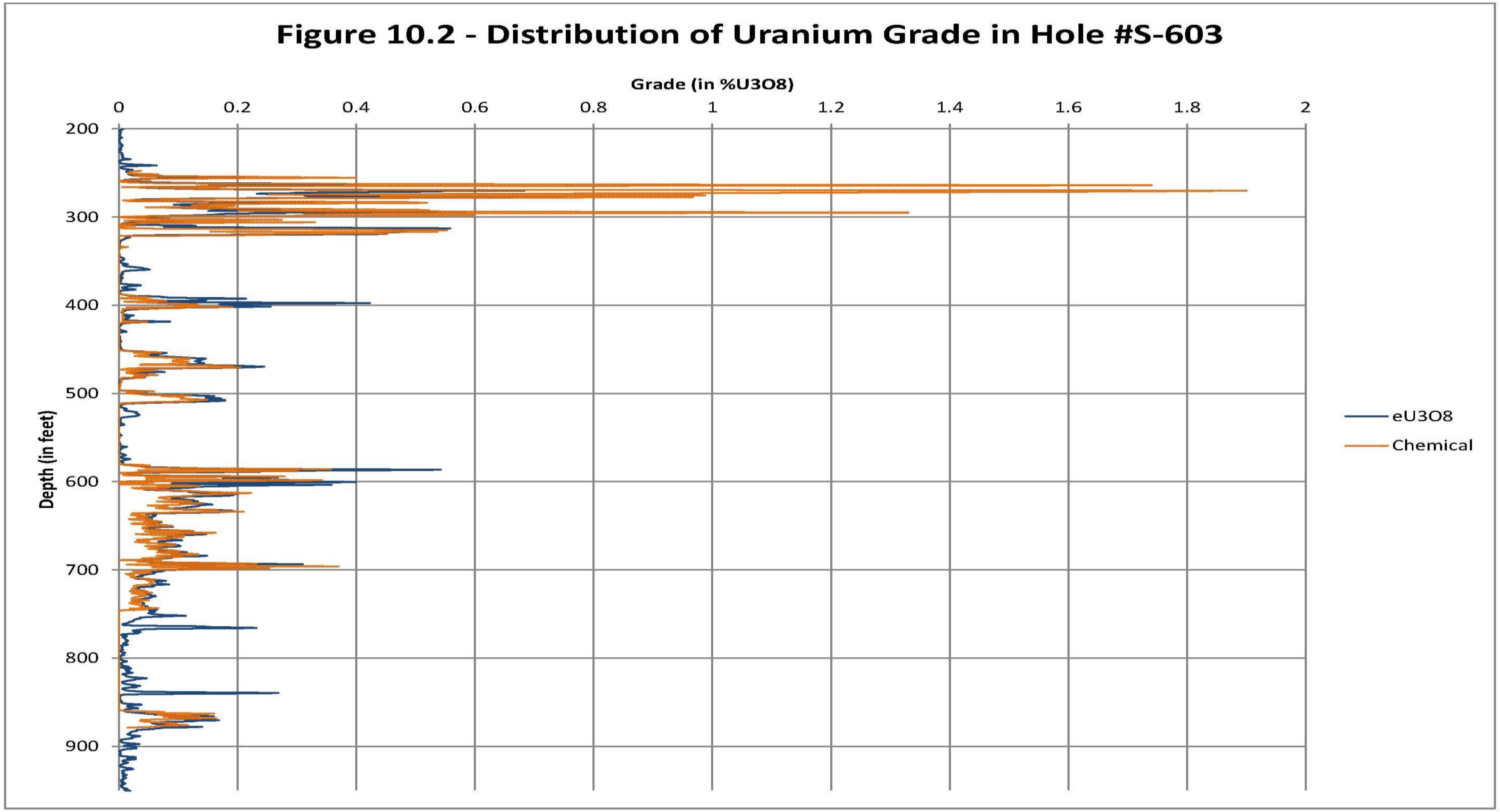


Figure 10.2: Down-hole distribution of grade, both chemical and radiometric, for hole S-60

ITEM 11. SAMPLING PREPARATION, ANALYSES, AND SECURITY

This procedure has been reviewed by the author and is both adequate and complete.

A formal sampling and materials handling system was established by Virginia Uranium, Inc. with input from Behre Dolbear in 2008. The sampling and security was strictly organized and maintained because of the radioactive nature of the commodity.

The general procedural guidelines include:

- Maintain full time chain of custody for all samples;
- Emphasize consistency between samples, geologist, technicians and contractors:
    1) Use standard forms for Driller's Daily Log, Core Box and Assay Checklists; and
    2) Standard Rock Classifications, Lithologic Log Sheets, Chain of Custody, and Training Sign-off Sheets;
- Double check recording and entry of data; and
- Use electronic data entry procedures.

All rotary and core holes in 2008 were drilled in a vertical orientation. All cores from the core holes were collected in 10 ft intervals from a 10 ft core barrel. Downhole geophysical gamma measurements were continuous and indicated a continuous apparent thickness of mineralization. Core was sampled in 1ft intervals for chemical assay, and handheld scintillometer readings were compared to downhole geophysical depths.

Core drilling recovery factors exceeded 95% and did not materially impact the accuracy and reliability of the results. Sample quality was excellent due to extremely high core recovery and they are believed to be representative since handheld scintillometer results compared favorably to downhole geophysics and chemical assay results. The one foot sample intervals were found to be representative for chemical assay results. A two-foot maximum deviation between core and geophysics was found and could be easily adjusted for depth. No other factors are believed to have resulted in sample biases.

Mineralization was found in granite and amphibolite rocks. The one-foot sampling interval was chosen to differentiate between the rock types and mineralized sections that were detected with a handheld scintillometer. The widths of mineralized zones varied. No mineralization was found in the Triassic rocks or Fork Mountain Schist.

A compendium of operating procedures titled "Virginia Uranium, Inc. Standard Operating Procedures for Uranium Exploration Program in Pittsylvania County, Virginia" was prepared on December 7, 2007 with updates and was followed by Virginia Uranium, Inc.. A copy of the procedures is available on request from the Virginia Uranium, Inc. office. The procedures have been reviewed and approved by Behre Dolbear.

The sample preparation, analysis, and security procedures are defined in the Standard Operating Procedures document and include the following elements:

- General Procedural Guidelines include physical sample and data handling standards that provide an audit trail from point of collection through laboratory analysis and storage.
- Roles and responsibilities are detailed for the Qualified Person/geologist, driller, Virginia Uranium, Inc. geologist, and the laboratory technician.
- Chain of custody methods and documentation for sample security are specified.
- Site radiological survey to determine background radiation is defined.

During the 2008 drilling campaign, all aspects of the sample preparation was conducted by an employee, officer, director, or associate of Virginia Uranium, Inc. using the standard operating procedures.

For the 2008 drilling campaign, one foot lengths of core were split, bagged and labeled. Samples were always kept secured in a locked area or under direct supervision of Virginia Uranium, Inc. employees. They were then shipped, via express delivery, with a chain of custody to Energy Laboratory Inc. (Energy Labs) of Casper, Wyoming. The Energy Labs comprehensive QA/QC program meets or exceeds the rigorous criteria established by the United States Environmental Protection Agency (EPA) and State Agencies (where applicable). Energy Laboratories is certified under the Safe Drinking Water Act by Region VIII EPA, and the States of Idaho, Montana, Nevada, North Dakota, South Dakota, Washington, and Wyoming. Samples received at Energy Laboratories are under a strict monitoring and tracking system from log-in to completion. Samples are logged in immediately upon receipt and are carefully checked for any special handling that may be needed. All analytical procedures, sample handling, and preservation techniques are EPA approved (where applicable) and strictly adhered to. Energy Labs duplicates every tenth sample to measure and control the precision of work. Where applicable, Energy Labs also spikes every tenth sample to test accuracy. Reference samples from the EPA or from private sources are tested by the laboratory with every set of samples to provide a third measure of the performance of equipment and personnel.

Information on Energy Labs accreditations and certifications can be found on the Energy Labs website. Where possible, Energy Labs uses EPA, ASTM, APHA, NIOSH, OSHA, or published analytical methods and follows the procedures with strict adherence to described protocol and recommended QA/QC parameters. Actual method operating procedures are described in the Standard Operating Procedures Manual, and are available for review at the laboratory. Details can be found at: http://www.energylab.com/QualityControlList.asp?branch=Casper. The Energy Labs Quality Manual and related quality documentation meets requirements of the National Environmental Laboratory Accreditation Program (NELAP) and American Association of Laboratory Accreditation (A2LA) standards.

The detailed uranium and closed can gamma procedures used by Energy Labs are as follows: both closed can gamma and chemical analysis splits require drying @ ~105°C for >16 hours in a convection oven followed by grinding via a plate pulverizer to -100 mesh. Approximately 200 grams are required for the closed-can gamma analysis and this mass is placed in a 3” diameter × 1” tall soils tin, which is then sealed with electrical tape. A minimum 15 day in growth interval is employed to establish secular equilibrium between 226Radium and the gamma emitting daughter of interest, 214Bismuth. As radon emanation studies have repeatedly demonstrated that a maximum of only 30% of 222Rn can be removed from a soils matrix using somewhat extreme techniques, the 15-day period ensures at least a 98% complete ingrowth of 214Bi. “Closed Can” uranium analysis works on the premise that, in a particular ore body, the activities of 238U and 226Ra will be in secular equilibrium being that the half-life of uranium is much greater than that of 226Ra. Once the can is sealed with the sample contained, conditions are ideal for attaining secular equilibrium between 226Ra, 222Rn, and 214Bi, which is quantified using a 2 inch NaI detector at the 214Bi 609 Kev energy region. Since 238U is the only possible source of 226Ra, the specific activity of 238U is applied to the tested activity of 226Ra to determine the total uranium concentration. The efficiency of the counting system is determined using certified 226Ra standards in the same geometry and density as the canned core samples. The official method identification used in data reporting, is EPA-901.1.

Chemical analysis preparation is conducted on a strong mineral acid digest of the dried and ground core using preparation technique SW3050. After drying, grinding, and blending, a 1-gram subsample is taken and delivered to a digestion vessel. Fifty percent nitric acid is added to the vessel (50 ml centrifuge tube) and the vessel is loosely sealed and heated in a water bath @ 95°C for >16 hours. Following the heating period, the volume is adjusted to a known level, typically 50 ml. Uranium analysis (and other metals) is performed on the solution by Inductively Coupled Argon Plasma (ICP) emission spectroscopy against certified commercial standards (such as EPA Method -200.7/200.8/SW6010).

Quality control measures were employed to check assay and check other analytical and testing procedures as required by certification requirements and company procedures. Select sample duplicates were sent directly from Energy Labs to Saskatchewan Research Council (SRC) in Saskatoon, Saskatchewan, Canada for external confirmation of the results. SRC has the following certifications: ISO/IEC 17025:2005 accredited by the Standards Council of Canada (scope of accreditation #537). The laboratory also participates in regular inter-laboratory tests. Details related to the SRC laboratory can be found at: http://www.src.sk.ca/html/labs_facilities/geo_labs/uranium/index.cfm.

No corrective actions were reported to have been taken by either Energy Labs or SRC related to the 2008 drilling campaign sample analyses. The uranium values obtained by Energy Labs were confirmed by sending, via Chain of Custody, a random control group of samples that covered three grade ranges of interest to SRC for check assays. The results of these confirmatory tests are shown in Table 16.1 and Figure 16.4. “Blind” standards (spikes), blanks, and duplicates performed by Energy Labs were spot checked by Behre Dolbear. Behre Dolbear believes the sample preparation, security, analytical procedures, and results during the 2008 drilling campaign were adequate and properly documented.

In addition, to the forgoing procedures which apply only to the 2008 drilling, the author reviewed the procedures followed by the previous operators, Marline Uranium and Union Carbide Corporation, during the period of 1978 through 1984. While the procedures were not as explicit as those developed in 2008, the geophysical logging was completed by a commercial vendor, Century Geophysical Corporation, who was then and is now one of the leading vendors for geophysical logging for uranium and other minerals such as coal. The log header information for the geophysical logs recorded instrument calibration data and borehole conditions such as casing and fluids which would affect the accuracy of the log data. With respect to chemical assay, the historical core analysis and metallurgical testing relied primarily on Hazen Research who, like Century, was then and is now a leading vendor in this regard. In addition, to assays completed by Hazen there were comparative assay from at least 3 other vendors as well as in-house assay equipment.

The author concludes that the sample handling and data collection procedures followed during the period of 1978 through 1984 met industry best practices of the day. The data has been well preserved and provided a secure and reliable source of data for the project.

ITEM 12. DATA VERIFICATION

Data verification was completed as part of the previous technical report (Behre Dolbear, 2008) and by the author. The author concludes that the data is adequate for the purposes of mineral resource estimation and preliminary mine planning.

Data verification included:

- Verification of surface drill hole locations
- Verification of radiometric or eU_3O_8 data from geophysical logs.
- Verification of lithologic descriptions of core and subsurface geologic contacts.
- Verification of core data.
- Verification of rock density
- Verification and determination of radiometric equilibrium conditions.
- Verification of down hole drift surveys

12.1 Drill Hole locations

Surface locations of drill holes were plotted using coordinates in the electronic database developed by Behre Dolbear, 2008. These location were compared to scans of original drill hole maps. As discussed under Item 14 some variance in elevation was observed. These were corrected to Digital elevation Model (DEM) developed for the project.

12.2 Radiometric Data

Virginia Uranium, Inc. obtained historical geophysical data related to 251 holes (171 RP and 80 core of which six are cross-over holes – both RP and core for the same hole) from the Virginia Museum of Natural History and the Virginia Department of Mines, Minerals and Energy, as well as from Marline's private data files. The historical Marline geophysical data was confirmed by re-drilling five historical holes. The RP holes selected for re-drilling were located in the North Deposit (41-19, 41-21, 41-138, and 41-183) with one in the South Deposit (41-145) and drilled with rotary percussion. The holes were re-logged with modern geophysical probes using Century Geophysical equipment. Three of the holes were also re-logged using Schlumberger geophysical probes that included a spectral gamma probe with a reconfirmation of MM&A results. (Behre Dolbear, 2008).

Subsequent to the 2008 report, Virginia Uranium, Inc. acquired the original data for the project including geophysical logs, lithologic logs, chemical assay records and various maps and reports. This data was scanned by Virginia Uranium, Inc. and provided to the author for use on this project. As the historic data was more complete that the data copies located at the Virginia Museum of Natural History and the Virginia Department of Mines, Minerals and Energy and included the original ½ grade determinations from the commercial geophysical logging vendor, Century Geophysical, the ½ foot data was transcribed into digital format. This data was compared to the 2008 database. Although some errors were found in both the data transcription and the 2008 database, once these were corrected the GT for comparable intervals were within 12%, in favor of the original data.

12.3 Lithology and Geologic Contacts

The drill core located at the Virginia Museum of Natural History and the Virginia Department of Mines, Minerals and Energy was examined by a BRS associate and the author. Specifically the location of the Chatham Fault and the base of the Leatherwood Granite were determined where intersected by core. This data was used to develop the geologic model as described under Item 14.

12.4 Core Assays

In 2008 six Marline holes core was re-assayed, and sixty samples were sent to Energy Labs that represented 1-foot intervals. When comparing historical chemical data to the new chemical data, there was a high correlation factor (R2=0.99) and high slope (0.92). This provided high confidence in using the historical data. In addition, 3 new core holes were completed and assayed. BRS reviewed the assay data and data handling and found they followed best practices.

12.5 Rock Density

Rock density of samples from S-601 and S-602 were measured, and it was shown that the UMETCO factor (about 2.3% less dense than the results of Virginia Uranium, Inc. analyses) provides a slightly more conservative resource estimate than would the use of recent analyses. The importance of the Virginia Uranium, Inc. analysis is that it validates the use of the UMETCO factor, 2.56 g/cc (Behre Dolbear, 2008).

12.6 Radiometric Equilibrium

The evaluation of radiometric equilibrium conditions is discussed under Item 14 of this report. The author concludes that for the purposes of mineral resource estimation the assumption of radiometric equilibrium is valid and supported by available data.

12.7 Downhole Surveys

Drift surveys were available for slightly less than ½ of the drill holes. In the drill holes, 50 feet of deviation is not experienced on average until around 500 feet of depth. The maximum deviation of 50 feet occurs at 700 feet to 800 feet; at three standard deviations (sigma) at 400 feet; and at six sigma at 200 feet to 300 feet. The data used in this analysis came from 46 core drill holes, 40 from the southern deposit and six from the northern deposit. The rotary percussion drill holes do not go beyond 50 feet deviation until around 500 feet, and the distance at which core and percussion holes usually reached a 50-foot deviation would be around 600 feet (Behre Dolbear, 2008).

Drill hole drift was considered during the evaluation of mineral resource described under Item 14 of this report. The observed drift was determined to be random in nature so that no preferential drift could be applied to drill holes without drift data. In 2 instances drill hole without drift data indicated mineralization outside the geologic boundaries. As this drift could not be accurately resolved, the data was removed from the database.

ITEM 13. MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Summary of Mineral Processing and Metallurgical Testing

Considerable test-work and an evaluation study have been performed in the past on the VUI ore. Dravo Engineer and Constructors in 1981 and Pincock, Allen and Holt in January 1982 conducted feasibility studies. Hazen Research conducted acid and alkaline leach studies in 1982. On November 12, 1982, Colorado School of Mines Research Institute (CSMRI) then performed a comprehensive study of the processing characteristics of composite samples from both the North and the South areas. UMETCO (formerly Union Carbide) and Marline Uranium further evaluated the Hazen and CSMRI work in a report titled "*Swanson Project Geology, Mine and Mill Design and Environmental Studies"* (July 13, 1984). Lyntek's evaluation draws primarily on the UMETCO summary, as well as some data from the CSMRI study as well as the recent studies by Resource Development, Inc. (RDi). In the opinion of Lyntek, this work was performed by credible organizations whose work was respected and is worthy of this preliminary economic analysis.

13.2 Basis for Assumptions Regarding Recovery Estimates

13.2.1 UMETCO MINERALS CORPORATION Metallurgical Studies - 1984

These studies are summarized in the Swanson Project Geology, Mine and Mill Design and Environmental Studies, July 13, 1984. The following was extracted from Volume 2, Section 1.

13.2.1.1 Ore Composites

Ore from the Coles Hill South ore body, obtained from CSMRI, consisted of about 100 pounds of a composite made up of 170 samples of core from 34 holes. Twenty samples from 11 holes were above the 200 foot level. The remaining portion from each of the samples, plus forty smaller samples not included in the CSMRI composite, were combined to provide another composite (206 pounds) of similar composition for the metallurgical tests. Listed below are comparative assays of the major constituents of each composite.

TABLE 13.1: C1-1 SOUTH COLES HILL ORE COMPOSITES

	CSMRI Lab	*Grand Junction Lab*
% U308	*0.106 ± 0. 004*	*0.102 ± 0.006*
% P04	*2.3*	*2.5*
% Ca	*2.8*	*2.8*
% CO2	*1.0*	*0.9*
% Fe	*2.6*	*3.0*

These composites represent the entire known ore body and are satisfactory to determine the average processing parameters for the project.

13.2.1.2 Alkaline Carbonate Leach Process

Based on metallurgical studies covered in this report and on environmental and economic considerations, the alkaline process has been selected for the proposed Swanson uranium mill.

13.2.2 DESIGN BASIS

13.2.2.1 Alkaline Leaching Design Basis

In CSMRI's report titled Process Development Studies for the Swanson Uranium Project, November 12, 1982 they summarize the ambient alkaline tests that were conducted. These results show on Table 13.2:

13.2.2.2 Fixed Conditions

Ore Grind, mesh: (99% passing)		65	Lixiviant: Na2CO3, gpl	50
	NaHCO3, gpl	20	Air flow, cc/min	100
Leaching Temperature, ºC		90	Leaching Time, Hr:	24
Head Analysis	(North Composite U_3O_8%)	0.076		
	(North Composite U_3O_8%)	0.104		

Alkaline Leach Test No.	Composite Tested	Pressure psig	Consumption Equiv Na2CO3 lb/ton	Residue U_3O_8 %	Extraction U_3O_8 %
8	South	0	4.7	0.0095	84.7
7	North	0	1.4	0.0085	86.6

Additional tests were conducted to demonstrate the impact of leach time on extraction (recovery) rates. These tests showed on Page 51 that "*Approximately 89% of the U_3O_8 extraction was obtained for the north composite and 87.5% U_3O_8 extraction for the south composite in 32 hours of leaching. These data also show that higher U_3O_8 extractions, equivalent to approximately a 1% increase, could be obtained by extending the leaching time to 52 hour.*" Therefore, under the tested conditions, the recoveries would be 88.5% and 90%.

13.2.2.3 Resource Development Inc. Metallurgical Studies - Spring 2012

A series of tests were conducted to investigate the impact that grind size and Na2CO3:NaHCO3 ratio, had upon recoveries for 24-hour tests (primarily) under ambient pressure conditions at 90ºC, without air addition. These results provided a range of data with results across the spectrum as would be expected. However, these studies provide additional insight into the processing characteristics, which is valuable.
One sample of drill core rejects, designated as high grade was provided to Resource Development Inc. (RDi) for the most recent studies. Overall the samples studied by CSMRI and RDi were very similar. The RDi sample analyzed 0.096% U_3O_8, which was slightly lower than the sample studied by CSMRI in the past, which was reported at approximately 0.104% U_3O_8 (see table above). The RDi sample analyzed 3.41% CaO (CSMRI Sample at 2.8% Ca = 3.92% CaO), 1.42% MgO and 3.93% Fe2O3 (CSMRI Sample at 2.6 Fe and 3.0%, Fe = 3.72% Fe_2O_3 and 4.29% Fe_2O_3).

The general leaching study process conditions were set forth in memo from Dr. Terry McNulty to Dr. Deepak Malhotra dated November 3, 2011. The starting conditions were set at a grind with a p80=150 mesh, 90°C at atmospheric pressure, 24-hours of leaching with kinetic samples collected at 6 and 12 hours, ratios of sodium carbonate to sodium bicarbonate from 3.75:1 to 5:1, 55% solids and slight vortexing agitation to allow oxygen excess during leaching.

Twelve alkaline leaching tests were conducted on the sample of high grade drill core rejects. The leaching tests investigated grind sizes with p80's from as coarse as 1% coarser than 65-mesh to as fine as 80% passing 200-mesh. The leaching times studied were primarily up to 24-hours, with only two tests continuing to 48-hours.

The following Table 13.2 summarizes the data provided by RDi for the most recent alkaline leaching studies.

TABLE 13.2: RDI ALKALINE LEACHING RESULTS						
RDi	**Grind, Mesh**	**eH**	**% U_3O_8 Extraction**			
Test No	**P_{80}**	**End**	**6**	**12**	**24**	**48**
6	100	-230	34.6	69.1	71.7	--
8	100	-46.4	67.6	73.1	72.9	--
2A	115	--	--	>100	>100	89.5
2B	115	--	--	87.6	89.2	90.1
3	150	-221.5	57.0	69.0	82.2	--
4	150	-221.8	59.3	77.2	86.7	--
5	150	-230.3	56.9	73.3	82.0	--
9	150	-56.4	84.7	89.8	76.7	--
7	200	-236.5	34.4	40.4	52.6	--
10	200	-80.2	77.6	78.6	85.3	--
11	200	-251.9	1.3	16.6	66.6	--
12	200	-242.4	3.1	3.7	46.0	--
13	200	-150.2	58.0	77.0	79.6	--

Additional leaching studies should be undertaken with closer control of the oxidizing conditions during leaching. In the most current leaching studies Lyntek is of the opinion that there was not sufficient oxygen in the slurry, and that when the pH was allowed to exceed 10.5, the oxidizing conditions necessary to place and keep the uranium in solution were not optimized.

13.3 Representativeness of the Metallurgical Samples

As noted above, a comprehensive program was developed to prepare a composite of the ore to represent the deposit for the CSMRI studies. These samples were selected from many core holes selected to represent the entire deposit, such that this is acceptable for a preliminary economic assessment. The samples tested by RDi were characteristically very similar to the ore samples tested by CSMRI, so it is inferred that these samples are very well suited to representing the ore deposit for a preliminary economic assessment.

13.4 Mineral Processing Factors and Economic Extraction

Based on the CSMRI test results and the current leaching results from RDi studies, it is reasonable to expect the extraction of U_3O_8 (from material representative of the sample tested), would be between 85% and 90% after mill losses (1-1.5%) are considered, hence, it is assumed the average recovery will be 85% for the preliminary economic assessment. It is recommended that confirmatory tests be developed and conducted during the pre-feasibility study. There are also questions that have arisen from the historical metallurgical tests results, which are to be expected, that also need to be addressed.

13.5 Potential Economic Recovery Factors

Given the test work that has been completed, there have been no results demonstrating issues that might affect economic recovery by a significant amount. There are factors that require additional testing to define test anomalies that are not yet understood such as the difference in recoveries between the north and south ore bodies or particular test results that suggest confirmation or retesting to better understand the meaning of the results. It is necessary to conduct focused tests to better estimate the recovery that can be expected from the processing operations. For example on the same sample, there are several indications that longer leach times will produce higher recovery rates, but there is also an example showing longer leaching time and lower recovery. More recent tests didn’t include air introduction to the leach tanks and these results had higher recovery than historical tests, under similar conditions, that had lower overall recovery rates. Additional tests will be necessary to understand the ore and the behavior of current processing techniques.

ITEM 14. MINERAL RESOURCE ESTIMATES

14.1 Previous Technical Reports and Mineral Resource Estimates

14.1.1 Available Data (Behre Dolbear, 2008)

A technical report was initially completed for the Coles Hill project in 2008 for Virginia Uranium Inc.(Behre Dolbear, 2008). Although this estimate was completed subsequent to the implementation of National Instrument 43-101 ("NI 43-101") and is compliant with current accepted reserve and resource classifications as set forth by the Canadian Institute of Mining and Metallurgy (CIM), it was not completed on behalf of the current owner and should be considered of a historic nature and as such should not be relied upon. At the time this report was prepared historical data was available from about 230 drill holes. The drill hole data consisted of copies of geological logs and cross-sections, down hole analog geophysical logs, laboratory assay data, and down hole orientation survey drift data available through the local museum. Analog geophysical logs were digitized and converted to equivalent radiometric assay data utilizing appropriate calibration factors and methods. In addition, three core holes were drilled by Virginia Uranium, Inc. and four historic drill holes were logged using commercial geophysical logging equipment.

Laboratory assay data was available for 80 historical core holes and 1 new Virginia Uranium, Inc. core hole (S-603 in the South Deposit with the results of S-601 and S-602 arriving after the modeling), totaling 55,311 sampled feet and 20,863 samples. This data used to determine radiometric equilibrium conditions.

Finally, available down hole orientation (drift) survey data was incorporated in the model for 96 total drill holes. An analysis of the down hole surveys indicated that the variance from ideal was moderate as there was less than 50 feet difference from ideal down to a depth of 300 feet. (Behre Dolbear, 2008)

14.1.2 Historical Mineral Resource Model (Behre Dolbear, 2008)

For the Behre Dolbear estimate, the Coles Hill geologic model was generated. A block modeling method was used to estimate the amount of resources. Two geologic contacts were entered that bound the main host rock, which comprises mylonitic feldspar augen gneiss and amphibolite of the Late Ordovician Leatherwood granite. The upper bounding structure is the Triassic Coles fault, and the lower contact is between augen gneiss and an underlying Fork Mountain schist unit. A block model was produced based on 20 feet × 20 feet horizontal and 10 feet vertical block sizes within the boundaries of the main ore host lithology. Larger block sizes were used outside the two bounding contacts, and the model was estimated fully in all directions.

Two mineralized domains, north and south, were established based on drill hole data and historical modeling. The nominal spacing of drill holes is about 100 feet in both North-South and East-West directions. The nominal sample spacing was 0.5 feet for radiometric equivalent data and 1 to 2 feet for assay data. For both data types, samples were composited over a 3-foot interval measured along the drill hole. A variographic analysis was performed, and search parameters were chosen based on the directions of continuity and ranges of influence from this analysis, as well as from visual inspection of the data. One search ellipsoid was used for both deposit domains with three orthogonal axes as follows: a horizontal axis oriented N30°E (strike direction) with a search radius of 250 feet, an axis plunging 40° in a S60°E direction (dip direction) with a search radius of 250 feet and a sub-vertical axis orthogonal to the other two, with a search radius of 50 feet. The selected orientation of the search ellipsoid mimics the strike and dip of compositional layering and flattening foliation in the host rock. Principal radii of 250 feet in strike and dip directions were chosen to be 2.5 times the nominal drill hole spacing. A smaller search radius of 50 feet sub-vertical was chosen because of the greater number of samples in this direction.

Searching with declustering of data was performed in octants using a maximum of eight sample points and a minimum of two sample points per octant. Grade estimation was performed with an inverse distance cubed algorithm. Grade shells of 0.20, 0.10, 0.05, and 0.025 wt % U_3O_8 were used to constrain the grade estimation to accept only composites within the boundaries of that grade shell. Grade estimation was performed on a mass (weight percent) basis. A density value of 0.080 tons/cubic foot (2.56 g/cc) was employed for all rock types, which is typical of a granitoid rock with fracture porosity. This density value was also determined and used by Union Carbide in a July 13, 1984 evaluation of the Coles Hill Deposit (Behre Dolbear, 2008).

14.1.3 Historical Mineral Resource Estimate (Behre Dolbear, 2008)

Measured and indicated resource estimations from the aforementioned model are reported in**Error! Reference source not found.** (Behre Dolbear, 2008).

TABLE 14.1: HISTORICAL RESOURCE ESTIMATES – JUNE 4, 2008 (MILLIONS OF TONS AND POUNDS IN-PLACE)

Cutoff % eU_3O_8	Measured[1]			Indicated[1]			Total[1]		
	Tons[2]	% eU_3O_8[3]	Pounds U_3O_8	Tons[2]	% eU_3O_8[3]	Pounds U_3O_8	Tons[2]	% eU_3O_8[3]	Pounds U_3O_8
South Coles Hill Deposit (SCHD)									
0.200	0.397	0.301	2.39	2.35	0.264	12.4	2.75	0.270	14.9
0.150	0.562	0.264	2.97	4.56	0.221	20.1	5.12	0.225	23.1
0.125	0.654	0.246	3.22	5.24	0.210	22.0	5.90	0.214	25.2
0.100	0.755	0.228	3.45	5.31	0.209	22.2	6.07	0.211	25.6
0.075	1.35	0.164	4.44	16.7	0.122	40.9	18.1	0.125	45.3
0.050	2.28	0.124	5.65	22.3	0.109	48.7	24.5	0.111	54.3
0.025	6.62	0.064	8.42	44.6	0.071	63.5	51.2	0.070	71.9
North Coles Hill Deposit (NCHD)									
0.200	-	-	-	0.519	0.320	3.32	0.519	0.320	3.32
0.150	-	-	-	0.851	0.262	4.46	0.851	0.262	4.46
0.125	-	-	-	0.927	0.252	4.67	0.927	0.252	4.67
0.100	-	-	-	0.959	0.247	4.74	0.959	0.247	4.74
0.075	-	-	-	7.31	0.103	15.1	7.31	0.103	15.1
0.050	-	-	-	13.2	0.088	23.1	13.2	0.088	23.1
0.025	-	-	-	47.5	0.050	47.1	47.5	0.050	47.1
CHUP Project Total (South and North Coles Hill Deposits)									
0.200	0.397	0.301	2.39	2.87	0.274	15.7	3.26	0.278	18.1
0.150	0.562	0.264	2.97	5.41	0.227	24.6	5.97	0.231	27.6
0.125	0.654	0.246	3.22	6.17	0.216	26.7	6.82	0.219	29.9
0.100	0.755	0.228	3.45	6.27	0.215	26.9	7.03	0.216	30.4
0.075	1.35	0.164	4.44	24.0	0.116	55.9	25.4	0.119	60.4
0.050	2.28	0.124	5.65	35.4	0.101	71.7	37.7	0.103	77.4
0.025	6.62	0.064	8.42	92.1	0.060	111	98.7	0.060	119

[1]Total tonnage above cutoff grade and average weight % U_3O_8 of that tonnage

[2]Short tons based on a rock density of 2.56 g/cc

[3]Weight %

For the Behre Dolbear estimate the following definitions we used:

Measured – within a radius of 50 feet of a drill sample composite. This distance is half the nominal drill spacing of 100 feet.

Indicated – within a radius of 50 to 200 feet of a drill sample composite. This distance is twice the nominal drill spacing of 100 feet.

The Behre Dolbear estimate did not include an estimate of inferred mineral resources. The mineral resource estimate was provided at a range of cut off grades but did not specify a recommended cutoff grade or minimum thickness. The author reviewed the data, methodology, and results of the Behre Dolbear study and concludes that it provides a reasonable estimate of the mineral resources present at the project subject cutoff criteria subsequently discussed. The Behre Dolbear estimate was completed in 2008 for Virginia Uranium Inc. Although this estimate was completed subsequent to the implementation of National Instrument 43-101 ("NI 43-101") and is compliant with current accepted reserve and resource classifications as set forth by the Canadian Institute of Mining and Metallurgy (CIM), it was not completed on behalf of the current owner and should be considered of a historic nature and as such should not be relied upon.

14.2 Current Mineral Resource Estimate

Subsequent to the Behre Dolbear, 2008 technical report, Virginia Uranium, Inc. acquired the original drill hole data including geophysical and lithological logs, half foot uranium grade equivalent data, and chemical assay data both from core analysis and Delayed Neutron Logging (DNL). The original data was more complete and included 264 drill holes as compared to the 230 previously available. The data was transcribed from the original to digital format and was used for the current estimate.

In addition, while the Behre Dolbear estimate was considered reasonable for the purposes of estimating mineral resources, for mine planning purposes a more detailed block model was preferred. The current model was prepared under the direction of Douglas Beahm, PE, PG, President and Principal Engineer BRS Inc. by ExplorMine Consultants of South Africa (Northrop and Deiss, 2011). This study utilized the updated database as previously described. Geologic modeling and mineral resource modeling was completed using geostatistical methods rather than inverse distance cubed is the 2008 estimate.

14.2.1 Available Data

A new topographic surface was generated from computer assisted drawings (CAD) supplied by BRS staff and acceptable borehole collar positions. If borehole collar elevations were within 2 feet of the actual 2ft surface contoured elevations, they were utilized in conjunction with the contours to create a digital terrain model (DTM). The remainder of the borehole collars were then projected vertically onto this newly generated surface. The cause of the elevation differences was due to a number of different historical data sources and measurements. All data was rectified to a common datum prior to resource estimation.

The radiometric borehole results were combined and checked. Any anomalies were reported to BRS staff. In turn they referred back to the original hardcopies to resolve these issues. Originally BRS re-captured all radiometric results above a 0.02% threshold as the original dataset supplied by VUI was captured at different scales requiring validation. This 0.02% threshold posed a potential issue with respect to the estimate as some holes had results for the entire trace and others did not. It was then decided to use the original background data below 0.01% and merge it with the clean validated radiometric data above 0.02% threshold. Any other missing values were replaced with 0.0025% in the augen gneiss and 0.001% in the gneiss and schist footwall as well as the Danville sediments hanging wall.

A number of drillholes do not have drift information available. Of the 264 holes available for the Resource estimate, 126 holes had drift information available. All holes without drift results were treated as vertical holes. A constant drift amount could not be applied to these holes, as the holes with drift results demonstrate varied characteristics (Figure 14.1). Cross section locations are shown on Figure 10.1, Drill Hole Location Map.

The borehole data was compiled in Datamine™ mining and exploration software. All drill holes were decomposited to 0.5ft as that is the dominant sampling interval.

BRS provided ExplorMine with a re-interpreted Coles Fault surface produced in CAD, which bounds the North and South uranium mineralization to the east. BRS also provided 17 basal contact point positions for the schist and gneiss unit, as well as 34 basal contact positions for the augen gneiss unit from borehole logs and borehole core. The augen gneiss unit is the host rock for the north and south uranium bodies. The above mentioned points were utilized to create a schist unit and an augen gneiss unit basal surface. Where high value holes protruded through the augen gneiss surface, additional points from those intersections were incorporated into the surface creation.

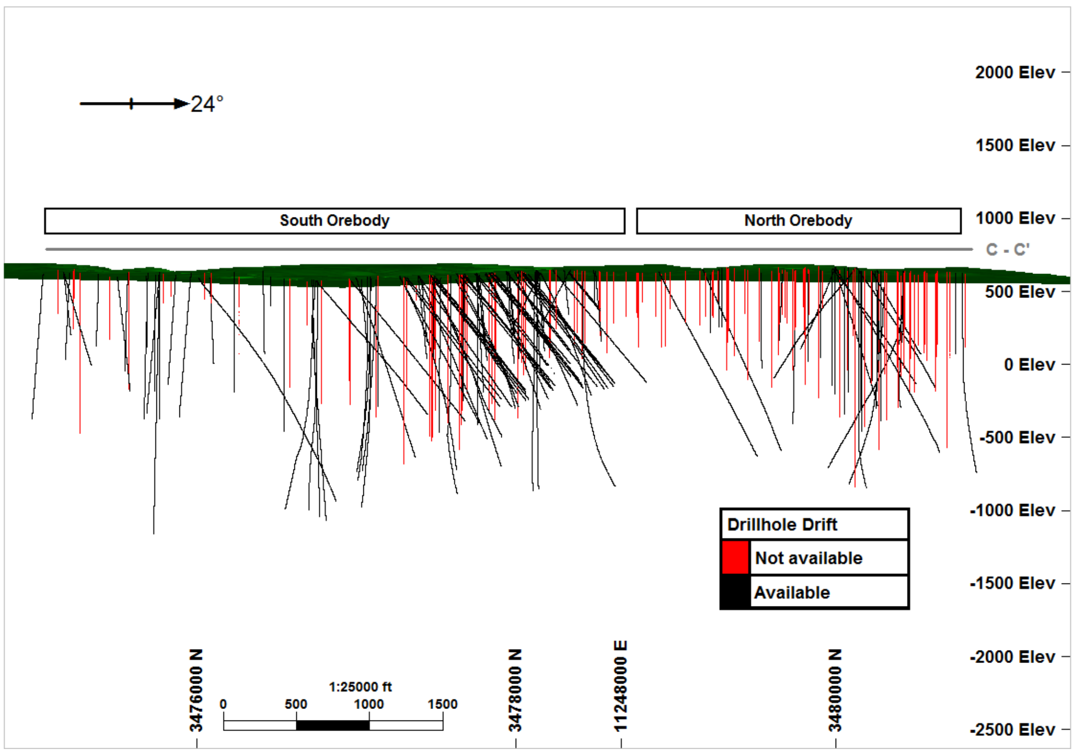


Figure 14.1: Drill Hole Orientation and Drift

14.2.2 Mineral Resource Model

In the first instance the data was demarcated in a 0.02% eU_3O_8envelope, which singled out country rock from host rock. This limit is supported by a break in the value distribution plot at 0.02% eU_3O_8 in Figure 14.2 and Figure 14.2. On examining the data it became evident that the ore body was three dimensionally distributed in space, and the upper and lower surfaces fairly even in relation to each other, and to the rotated XY best fit plane. The spatial geostatistics would therefore have to be done in two dimensions in the XY plane, but extending upwards and downwards as a series of layers with the same total dimensional extent as the total extent of mineralized zones (0.02% eU_3O_8 cut off mentioned above). Because of the uneven vertical total thickness of the ore body, total thickness accumulation composites would have a random distribution and would not produce a reasonable estimate to the method of geostatistics. Therefore it was decided to practice two dimensional spatial geostatistics on the composited grade values of the uranium within the individual successive layers, within the ore envelope. The size of composite was set to give the lowest variance as deduced from doing a series of vertical variograms on composites from 0.5 up to 2.0 feet. The 1.0ft vertical composite width was found to fit the case best when taking both North and South Coles mineralization.

The general shape of the Coles and the separate geographical locations dictated that it should be divided up into 2 separate areas on the basis of their locations. The confinement of the Coles into two separate locations is thought to be structurally controlled. These were designated as North Coles and South Coles. Naive statistics were done on the schist and gneiss oxidized (ZONEO=1) and reduced zones (ZONEO=11); augen gneiss oxidized (ZONEO=2) and reduced (ZONEO=22) portions and the Danville sediments oxidized (ZONEO=3) and reduced (ZONEO=33) zones.

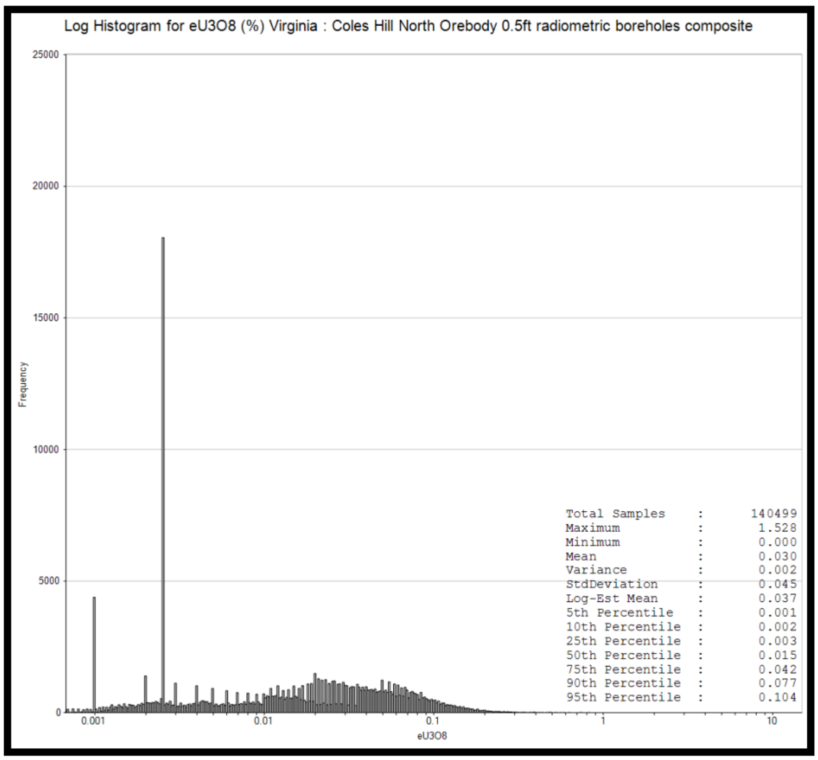


Figure 14.2: Log histogram of North Coles 0.5ft borehole composites

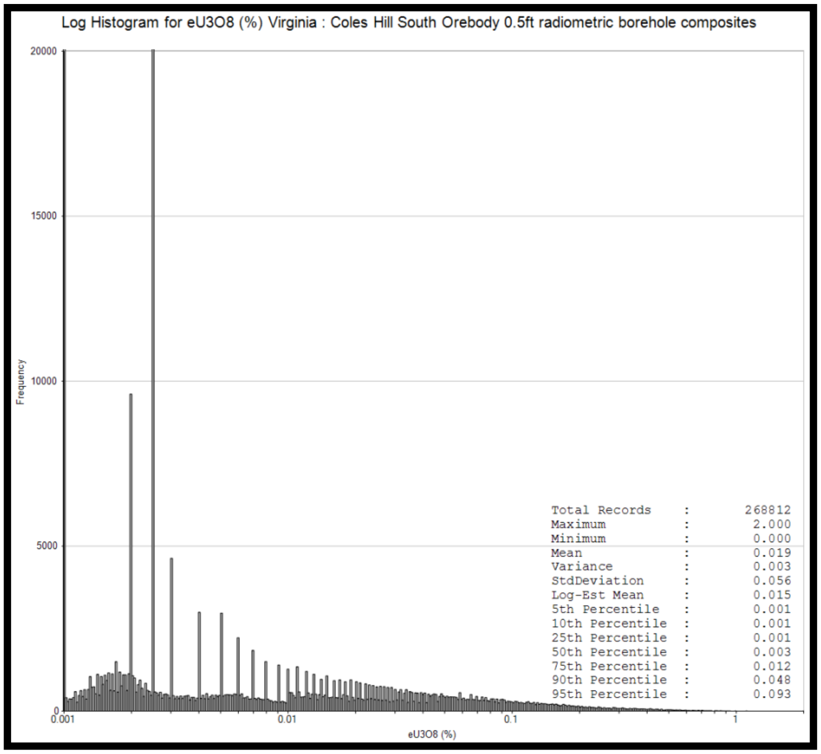


Figure 14.3: Log histogram of South Coles 0.5ft borehole composites

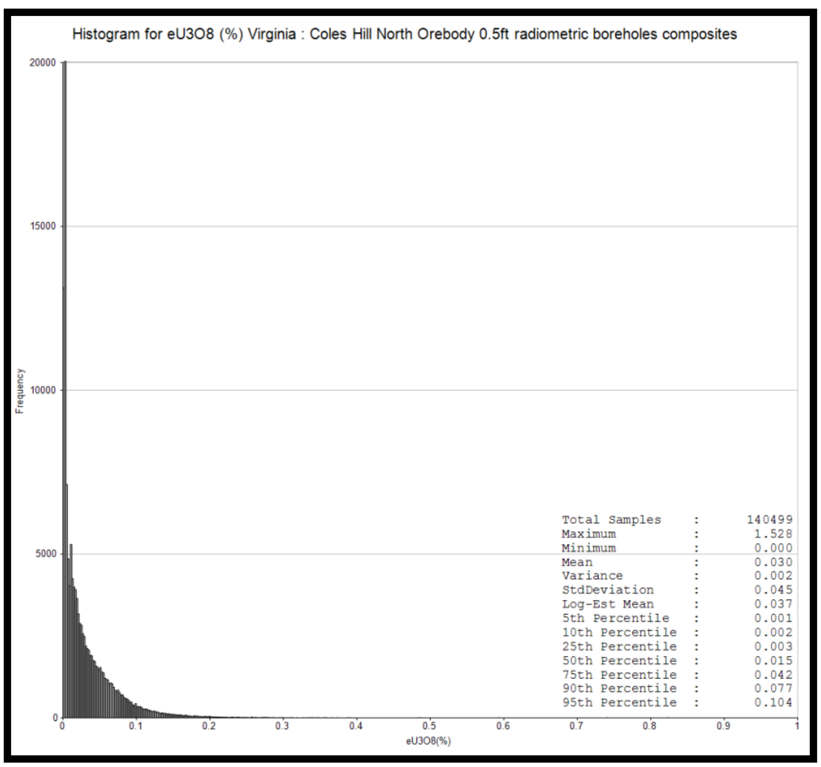


Figure 14.4: Histogram of North Coles 0.5ft borehole composites.

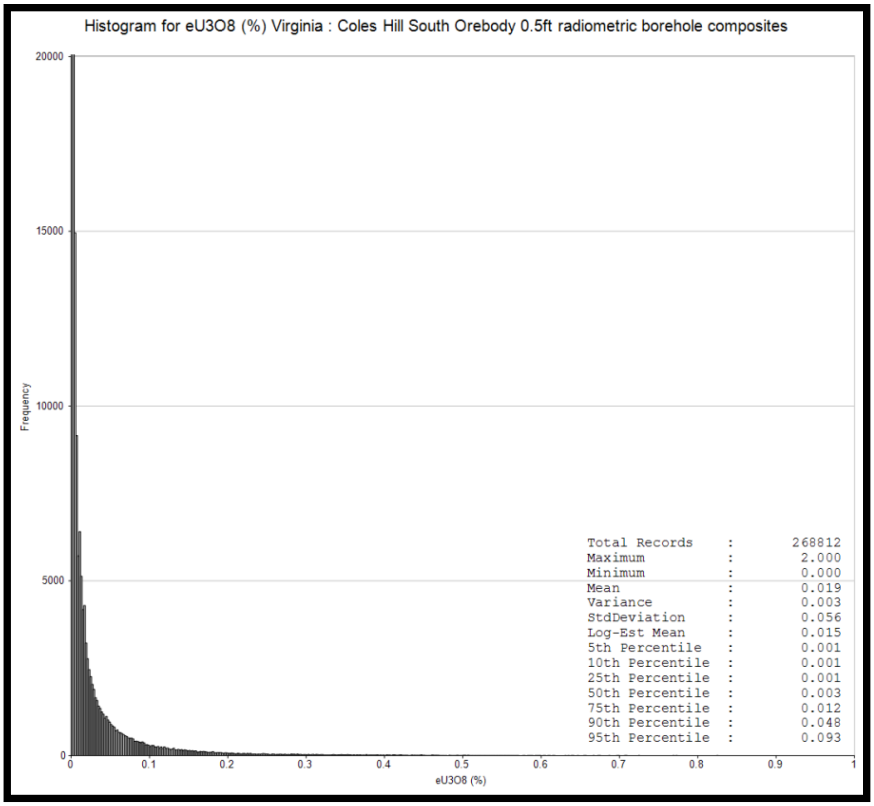
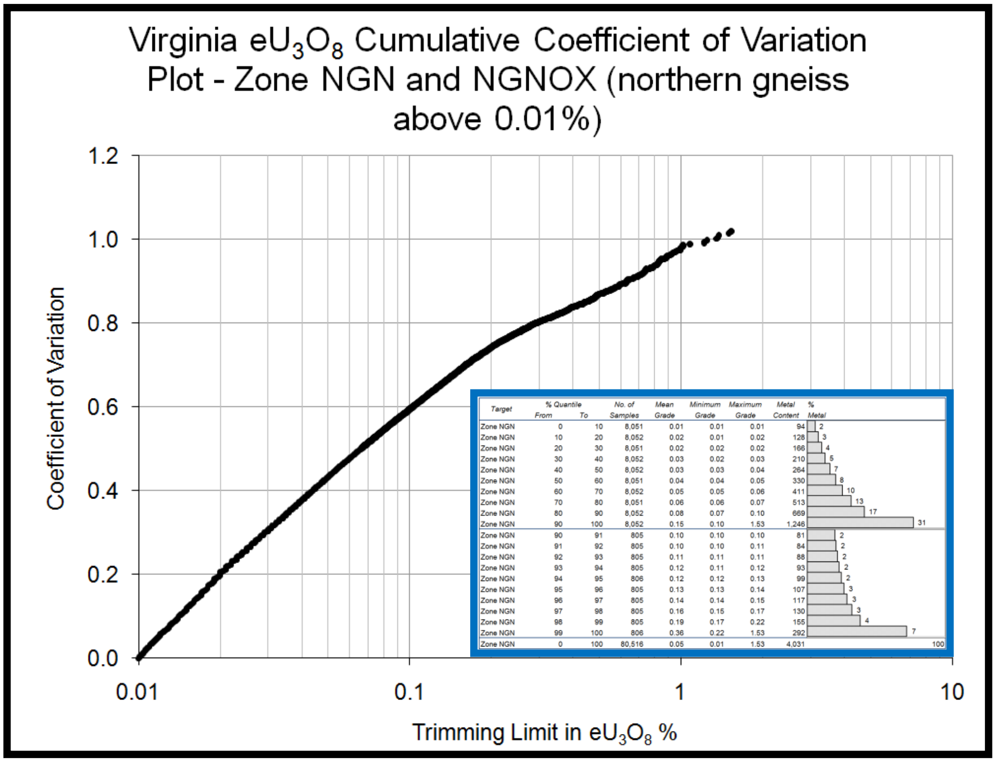


Figure 14.5: Histogram of South Coles 0.5ft borehole composites.

Classical statistics and histograms of the two different areas of mineralization total dataset are presented above in Figure 14.2 through Figure 14.5 in both log and normal space respectively. The dissimilarity between the two areas is evident, and because of this the variography was approached separately. Insufficient samples occurred in the schist for variography and so variogram models from the augen gneiss only data were utilized. The oxidized zones had insufficient sample coverage for variography. The log plots were slightly negatively skewed, showing that for classical methods of mean calculation a third parameter would have to be added. The log histograms indicate a number of populations, which could not be separated geologically, however these populations may be due to different eras of radiometric analysis instruments with differing degrees of significance with respect to their decimal reporting.

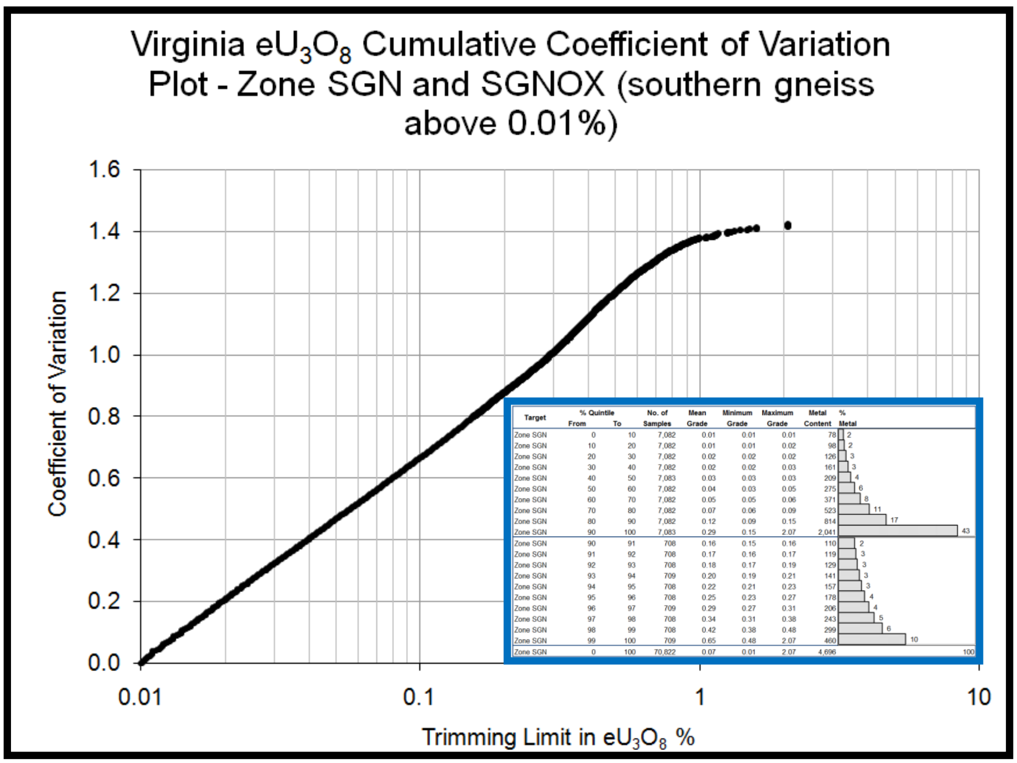
Cutting of Outliers

Initially a lower cut of 0.02% was applied due to the data recapture focusing on values above 0.02%. Later the limit was lowered to 0.01% as there seemed to be significant number of values in the range of 0.01-0.02% that were valid and facilitated better structured variograms. A number of composites had to have detection limits applied below 0.01% as they were not available. These limits are discussed under geological modeling. Upper cutting was unnecessary in both North and South areas as there were no outliers existing in the database as demonstrated by the cumulative coefficient of variation graphs presented in Figure 14.6 for the North Coles Figure 14.7 for the South Coles.



Target	% Quantile From	To	No. of Samples	Mean Grade	Minimum Grade	Maximum Grade	Metal Content	% Metal
Zone NGN	0	10	8,051	0.01	0.01	0.01	94	2
Zone NGN	10	20	8,052	0.02	0.01	0.02	128	3
Zone NGN	20	30	8,051	0.02	0.02	0.02	166	4
Zone NGN	30	40	8,052	0.03	0.02	0.03	210	5
Zone NGN	40	50	8,052	0.03	0.03	0.04	264	7
Zone NGN	50	60	8,051	0.04	0.04	0.05	330	8
Zone NGN	60	70	8,052	0.05	0.05	0.06	411	10
Zone NGN	70	80	8,051	0.06	0.06	0.07	513	13
Zone NGN	80	90	8,052	0.08	0.07	0.10	669	17
Zone NGN	90	100	8,052	0.15	0.10	1.53	1,246	31
Zone NGN	90	91	805	0.10	0.10	0.10	81	2
Zone NGN	91	92	805	0.10	0.10	0.11	84	2
Zone NGN	92	93	805	0.11	0.11	0.11	88	2
Zone NGN	93	94	805	0.12	0.11	0.12	93	2
Zone NGN	94	95	806	0.12	0.12	0.13	99	2
Zone NGN	95	96	805	0.13	0.13	0.14	107	3
Zone NGN	96	97	805	0.14	0.14	0.15	117	3
Zone NGN	97	98	805	0.16	0.15	0.17	130	3
Zone NGN	98	99	805	0.19	0.17	0.22	155	4
Zone NGN	99	100	806	0.36	0.22	1.53	292	7
Zone NGN	0	100	80,516	0.05	0.01	1.53	4,031	100

Figure 14.6: North Coles coefficient of variation at 0.01% lower cut-off. A quantile analysis is contained in the insert.



Target	% Quintile From	To	No. of Samples	Mean Grade	Minimum Grade	Maximum Grade	Metal Content	% Metal
Zone SGN	0	10	7,082	0.01	0.01	0.01	78	2
Zone SGN	10	20	7,082	0.01	0.01	0.02	98	2
Zone SGN	20	30	7,082	0.02	0.02	0.02	126	3
Zone SGN	30	40	7,082	0.02	0.02	0.03	161	3
Zone SGN	40	50	7,083	0.03	0.03	0.03	209	4
Zone SGN	50	60	7,082	0.04	0.03	0.05	275	6
Zone SGN	60	70	7,082	0.05	0.05	0.06	371	8
Zone SGN	70	80	7,082	0.07	0.06	0.09	523	11
Zone SGN	80	90	7,082	0.12	0.09	0.15	814	17
Zone SGN	90	100	7,083	0.29	0.15	2.07	2,041	43
Zone SGN	90	91	708	0.16	0.15	0.16	110	2
Zone SGN	91	92	708	0.17	0.16	0.17	119	3
Zone SGN	92	93	708	0.18	0.17	0.19	129	3
Zone SGN	93	94	709	0.20	0.19	0.21	141	3
Zone SGN	94	95	708	0.22	0.21	0.23	157	3
Zone SGN	95	96	708	0.25	0.23	0.27	178	4
Zone SGN	96	97	709	0.29	0.27	0.31	206	4
Zone SGN	97	98	708	0.34	0.31	0.38	243	5
Zone SGN	98	99	708	0.42	0.38	0.48	299	6
Zone SGN	99	100	709	0.65	0.48	2.07	460	10
Zone SGN	0	100	70,822	0.07	0.01	2.07	4,696	100

Figure 14.7: South Coles coefficient of variation at 0.01% lower cut-off.

The experimental variograms were orientated in a plane parallel to the plunge of the deposit as visually examined from mineralized borehole intersections. Experimental variograms were analyzed for all geological zones as discussed previously. Due to data scarcity in the hanging wall and footwall rocks respective to the augen gneiss it was decided to utilize the oxidized and reduced augen gneiss host rock composites to produce representative variograms. The idea was to estimate with a soft boundary due to the fact that the geological boundaries have limited information informing them.

Variography was performed in both the XY plane orientated with respect to the plunge direction and in the vertical direction (Z) perpendicular to XY plane. Very poor correspondence between the sill of the vertical variograms and the sill in the XY 2D plane was obtained. Also very poorly structured variogram models were obtained in the 2D plain despite resorting to both log and pairwise relative transforms. Maximum vertical geostatistical continuity shown by the variogram models in the vertical could not be demonstrated, probably due to missing data below detection limit. This affected variography in the XY plane also probably due to the zonal effect (see Northrop,WD,2003). Therefore in order to obtain definitive variogram model structures in the XY plane, layered variography was performed parallel to the most tabular expression of the mineralized zone. This must correspond very closely to the plane of brecciation exhibited by the host rock, as a result of the first tectonic event. This produced the voids necessary for the emplacement of the disseminated Uranium mineralization (Behre Dolbear, 2008). From this set of layered data the average variograms in the XY plane were analyzed for the full package in the augen gneiss. Distinctive anisotropic structures were obtained for the North Coles, which were elongated in the direction of shearing and brecciation down the structural dip of the mineralization. The anisotropy of the variogram model in the South Coles did not correspond to the direction of this primary tectonic event, but mimicked the overprint direction of cross-cutting planes of weakness in that area attributed to a second tectonic event as described in the geological account.

It must be noted that the prominence of the secondary direction can be attributed to the closer proximity of the Coles fault to the South Coles as compared to the North Coles. Therefore the background down structural dip anisotropic direction in the South Coles was probably not distinguished because it is obscured by the direction of weakness produced by the second tectonic event depicted in the experimental variogram contours displayed in Figure 14.8. This is anisotropic at almost right angles to that direction. This relationship between anisotropic directions and the directional features produced by the two tectonic events for the two areas can be seen depicted in the

Figure 14.9 below. The horizontal variograms are displayed graphically Figure 14.10 and Figure 14.11 below for each of the respective mineralized areas.

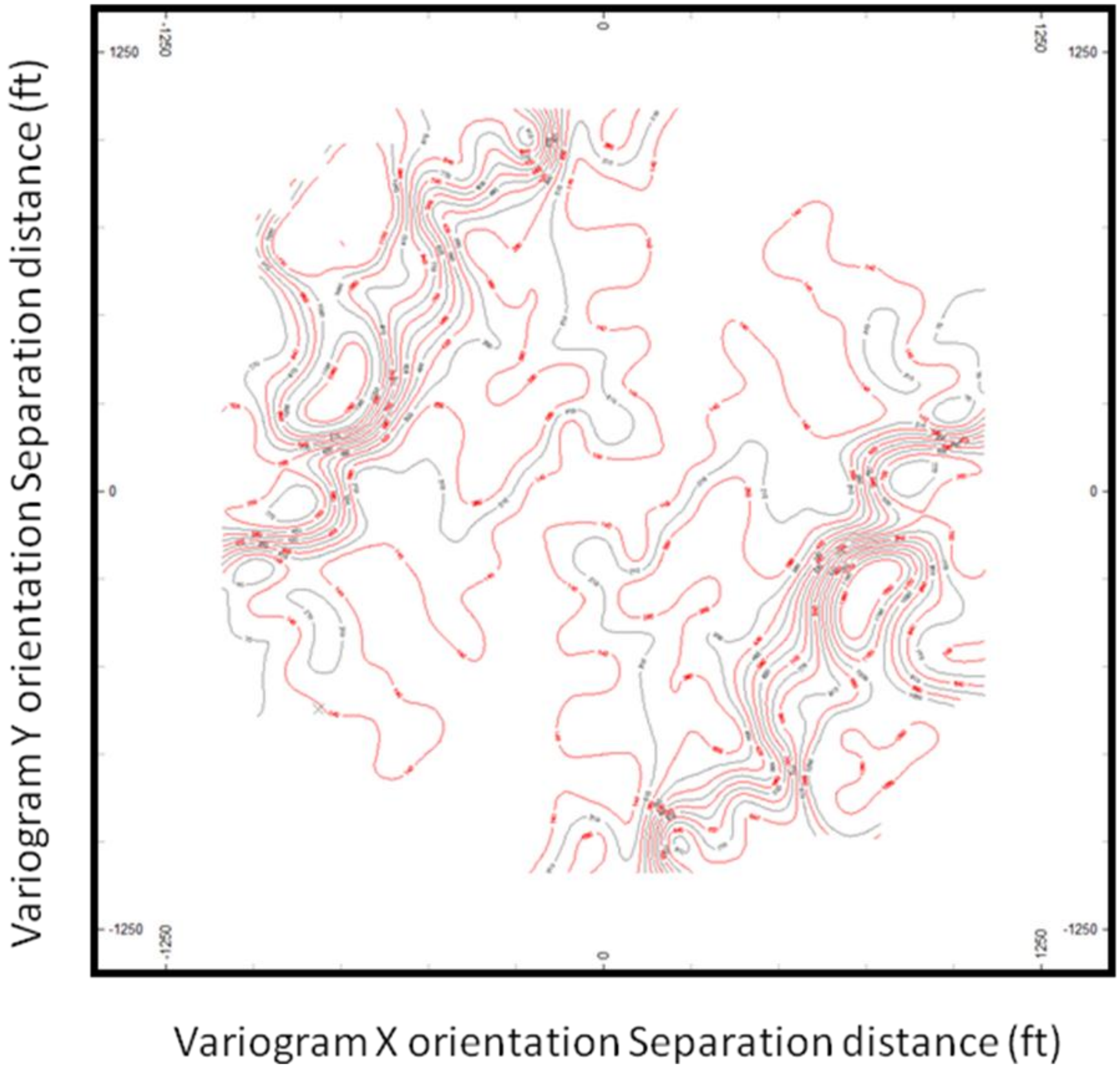


Figure 14.8: Experimental variogram contours (variance between sample pairs) for South Coles Hill showing directional interference (X and Y rotated into the mineralization orientation).

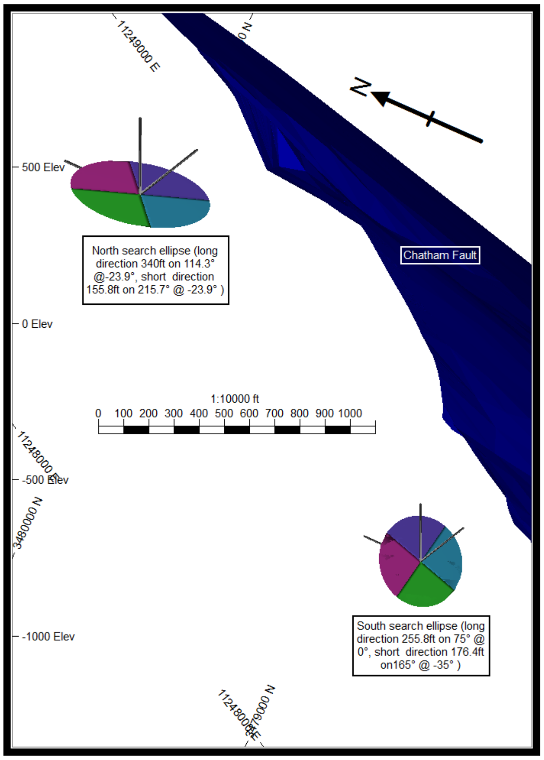


Figure 14.9: Modeled experimental variogram orientations for the North and South Coles areas respectively in relationship to the Chatham Fault.

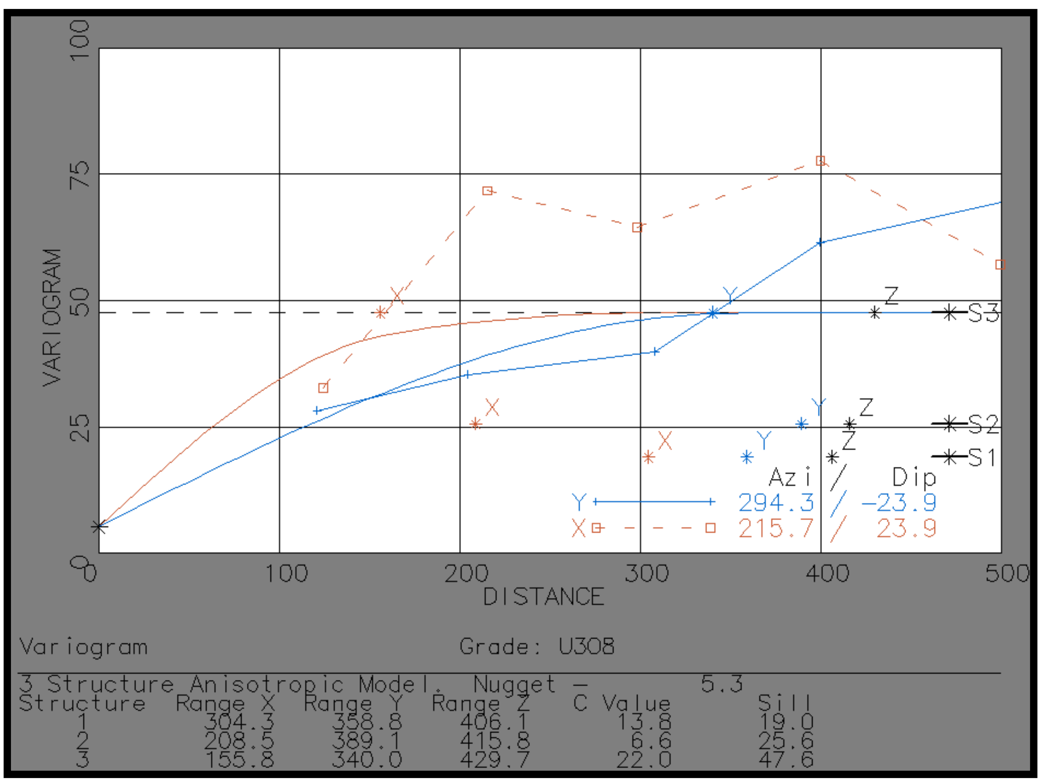


Figure 14.10: Modeled horizontal 1 ft drill hole composite average layered semi-variogram for the eU3O8 (%) value for the North Coles.

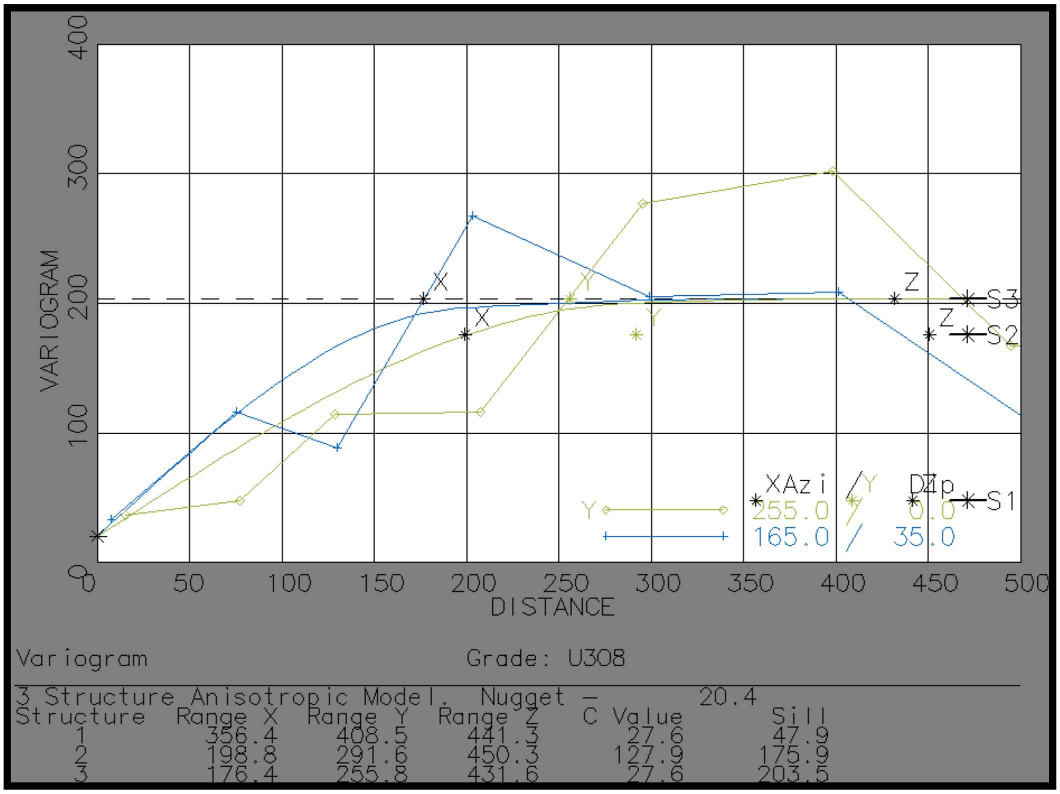


Figure 14.11: Modeled horizontal 1 ft drill hole composite average layered semi-variogram for the eU3O8 (%) value for the South Coles.

Kriging Methodology

Optimum block sizes and the optimum number of samples to access in the estimate were determined by a number of test runs on strategically placed blocks in high density and low density areas in each area. The optimum parameters were determined by seeing what produced the best regression slope (R) and kriging efficiency, and the lowest spread in 90% confidence limits, but still retaining the smallest block size relating to the probable future smallest mining unit ("SMU"). These three parameters actually produce good correlations amongst themselves. Plot outs of the graphs are presented in Figure 14.12 to Figure 14.13 for the North Coles and Figure 14.14 to Figure 14.15 for the South Coles.

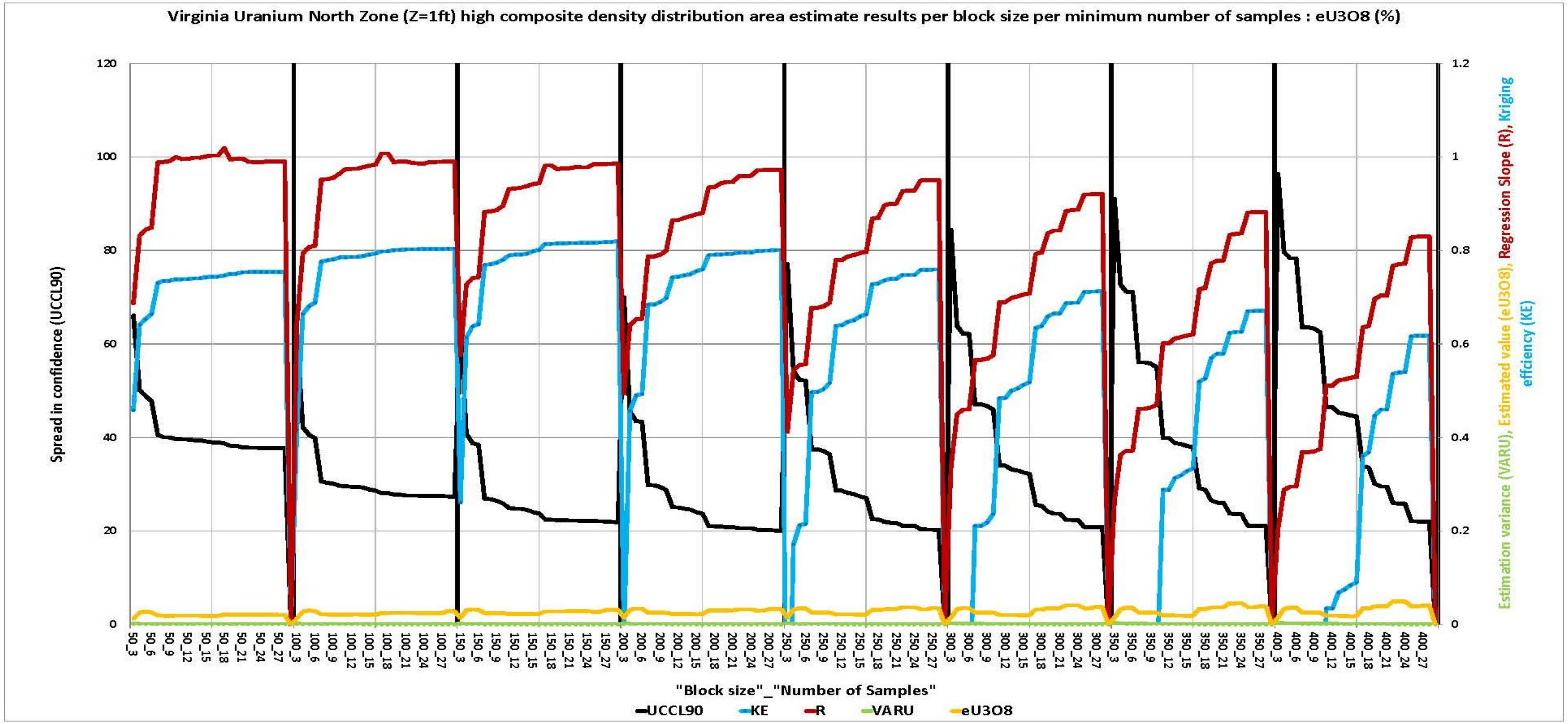


Figure 14.12: Example of geostatistical determination of optimum block size for Mineral Resource estimation for the North Coles.

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Figure 14.12: Example of geostatistical determination of optimum block size for Mineral Resource estimation for the North Coles.

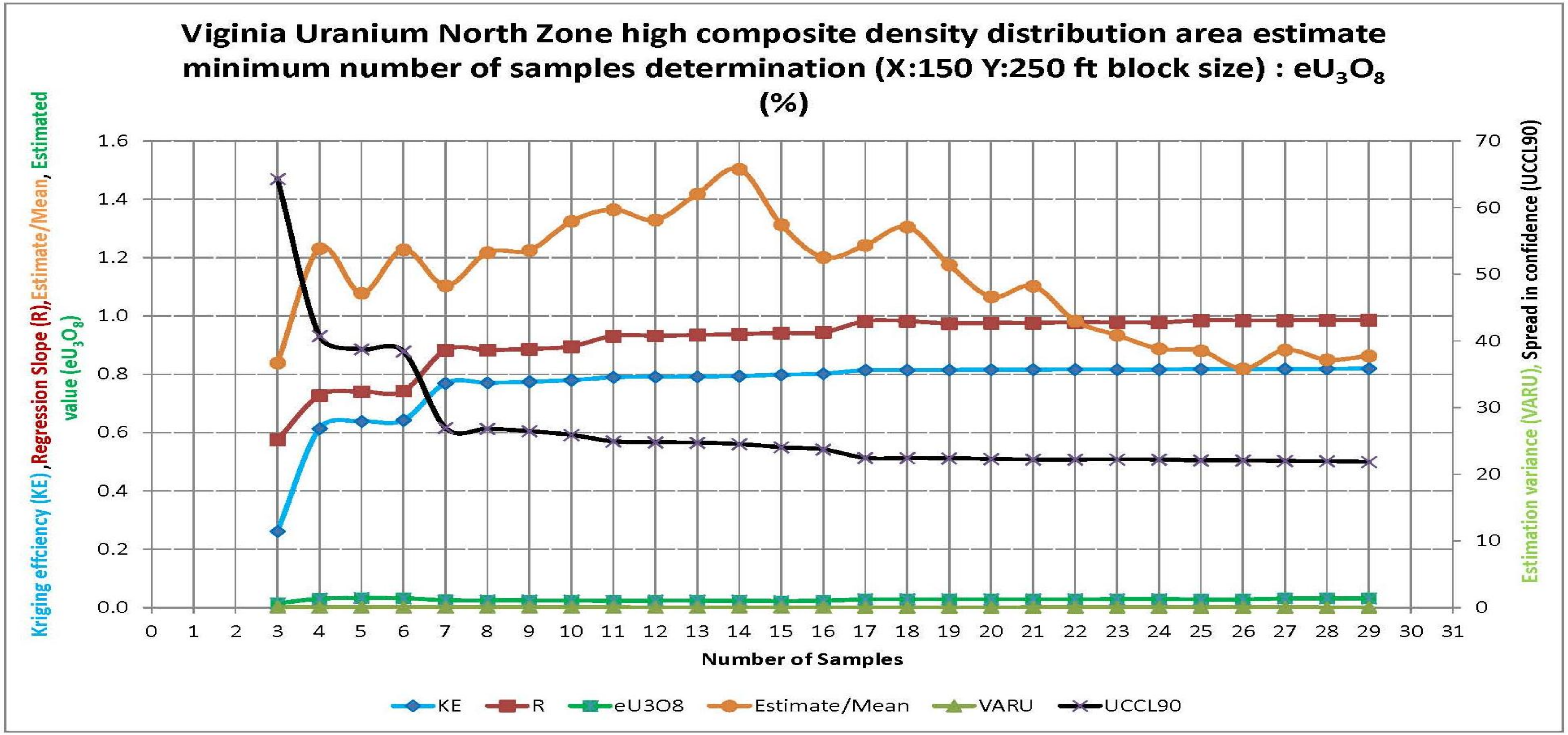

Figure 14.13: Example of geostatistical determination of optimum minimum and maximum number of samples for mineral resource estimation for the North Coles.

Figure 14.13: Example of geostatistical determination of optimum minimum and maximum number of samples for mineral resource estimation for the North Coles.

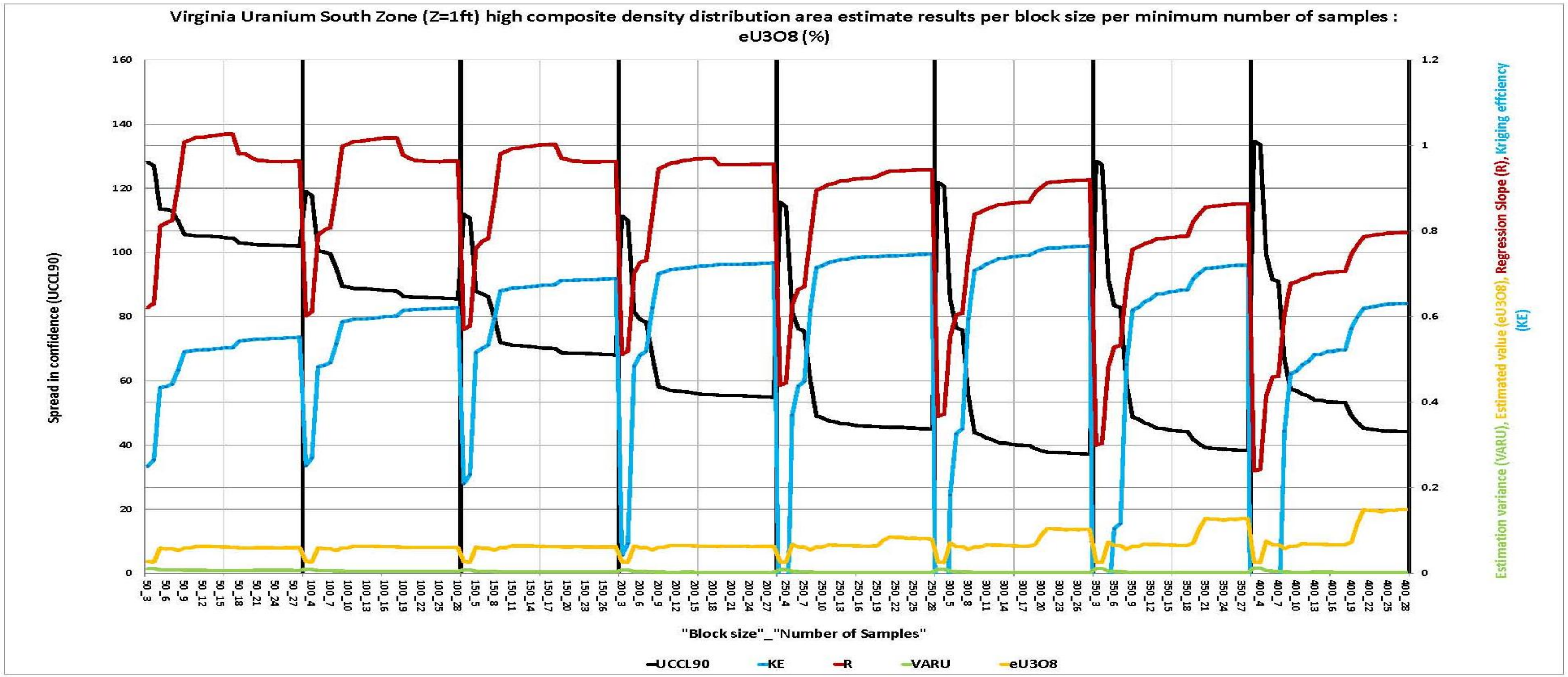

Figure 14.14: Example of geostatistical determination of optimum block size for Mineral Resource estimation for the South Coles.

Figure 14.14: Example of geostatistical determination of optimum block size for Mineral Resource estimation for the South Coles.

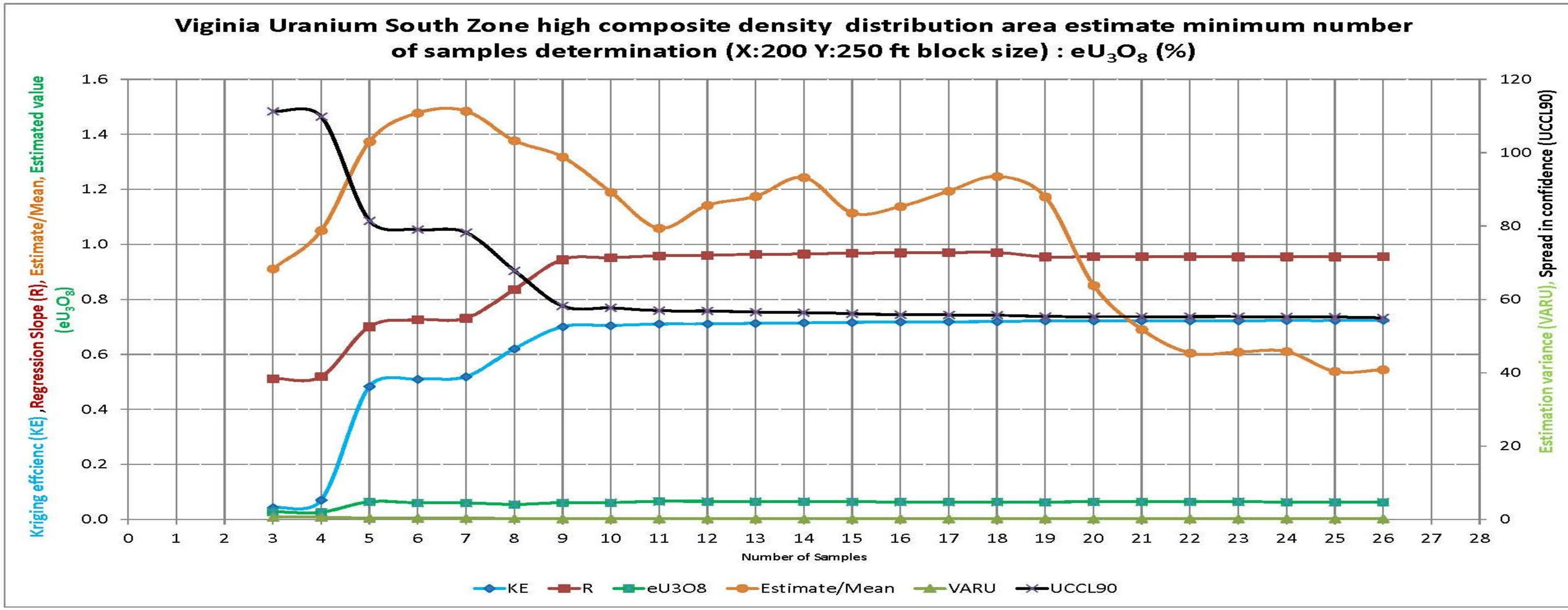

Figure 14.15: Example of geostatistical determination of optimum maximum and minimum number of samples for Mineral Resource estimation for the South Coles.

Figure 14.15: Example of geostatistical determination of optimum maximum and minimum number of samples for Mineral Resource estimation for the South Coles.

Three separate searches were effected namely 1 times the range of the variogram, 1.5 times the range of the variogram and 2 times the range of the variogram. For the North Coles a minimum of 17 samples and a maximum of 25 samples were deemed to be geostatistically appropriate for the first two searches. The 3rd search was set at a minimum of 8 samples and a maximum of 25 samples. For the South Coles a minimum of 10 samples and a maximum of 20 samples for the first two searches and for the 3rd a minimum of 8 samples and a maximum of 20 samples were deemed to be appropriate (Figures 19 and 21). The search was confined to a specific layer due to the vertical variability of the mineralization and therefore the octant search method was deemed unnecessary.

14.2.3 Mineral Resource Categories

The 2 dimensionally kriged layered stacked block models were categorized into Indicated and Inferred on the basis of global industrial wide accepted limits of the kriging efficiency refer to Figure 14.16 and Figure 14.7. Cross section locations are shown on Figure 10.1, Drill Hole Location Map. The over-riding factor was that no Geostatistical Indicated Resources was allowed beyond a search volume, equal to the 1.5 times the range of the variograms, as pairs of samples further than that apart show no correlation (Snowden, 1996). The kriging efficiency limits were utilized in the Resource categorization as follows:

North Ore body 150 X 250 blocks >= 0.3 Kriging efficiency and within 1.5 times range of the variogram : Indicated Mineral Resources
South Ore body 200 X 250 blocks >= 0.3 Kriging efficiency and within 1.5 times range of the variogram: Indicated Mineral Resources.

Areas around existing intersections were demarcated to 1.5 times the range of the variogram. These were then estimated to a maximum of 2 times the range of the variogram with a minimum of 8 samples. If they were estimated then they were classed as Indicated Mineral Resources. The author reviewed the resource projection and concludes, based on geological continuity, that the mineral resource meets CIM criteria as an Indicate Mineral Resource.

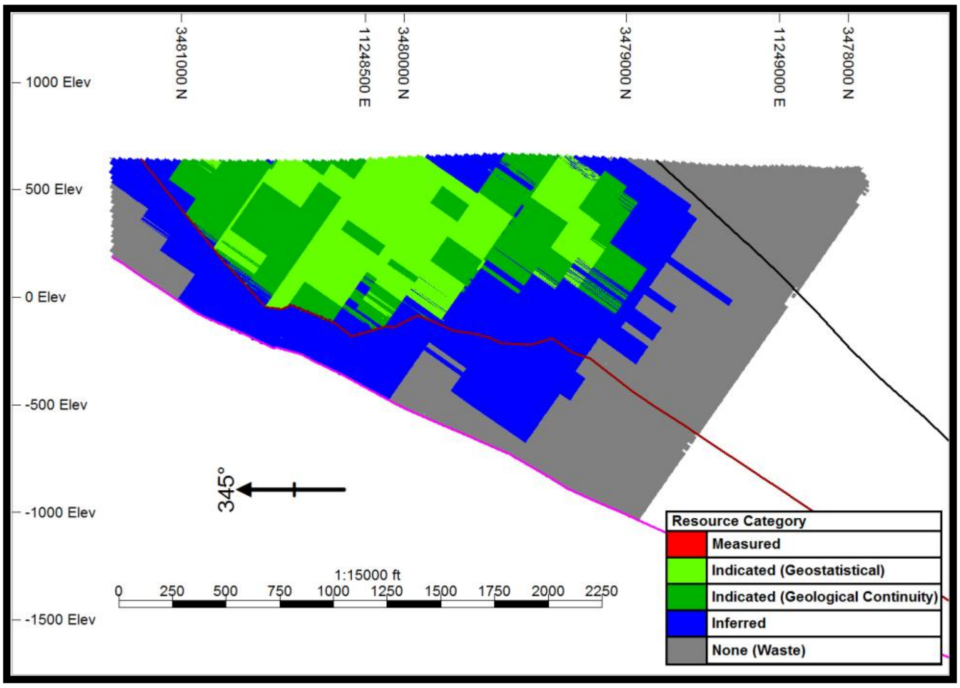

Figure 14.16: Northerly-southerly section through the North Coles demonstrating the Resource categorization.

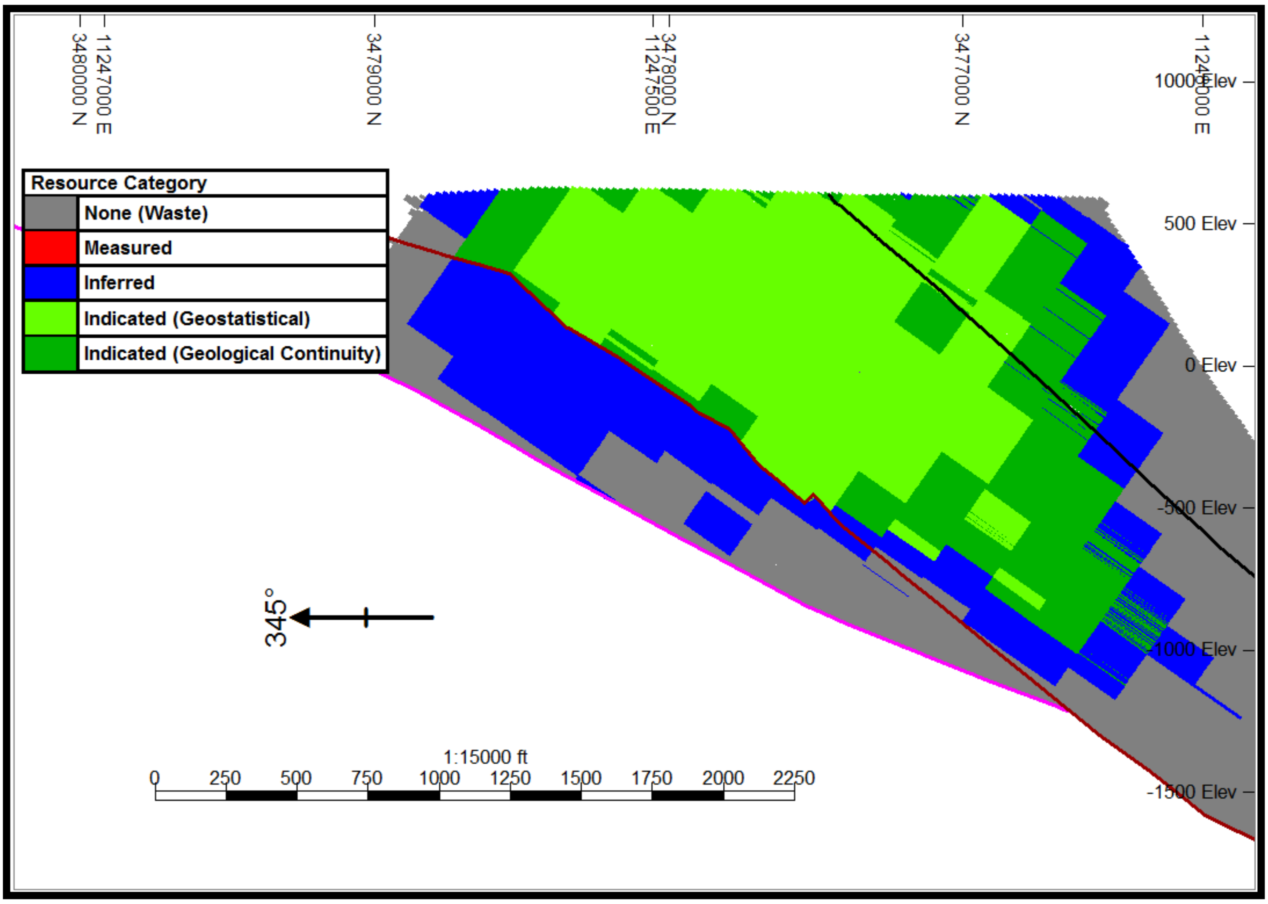

Figure 14.17: Northerly-southerly section through the South Coles demonstrating the Resource categorization.

A value less than 61% spread in confidence limits at the 90th percent confidence level is categorized as an Indicated Resource in the North Coles on the basis of geostatistical confidence, which is well within industry norms. However this is limited to twice the relevant variogram range on the basis of geological confidence.

In the South Coles the spread in confidence limits by regression that was equivalent 0.3 kriging efficiency was >> 61%. However larger block sizes (300 X 350 meter) would have given a spread in confidence limits below 61% also as demonstrated in the optimization runs. These block sizes were not used for practical reasons of planning because of their over large block size. On this basis the limit of the >= 0.3 kriging efficiency on the chosen block size was the expected limit for Geostatistical Indicated Resources in the South and North areas. Indicated Resources were extended to one and a half time the range of the variogram in distance and estimated at a maximum of twice the range of the variogram model on the basis of geological confidence on the continuity of the South and North mineralization.

Grade-tonnage curves for each respective mineralized area are given in Figure 14.18 and Figure 14.19.

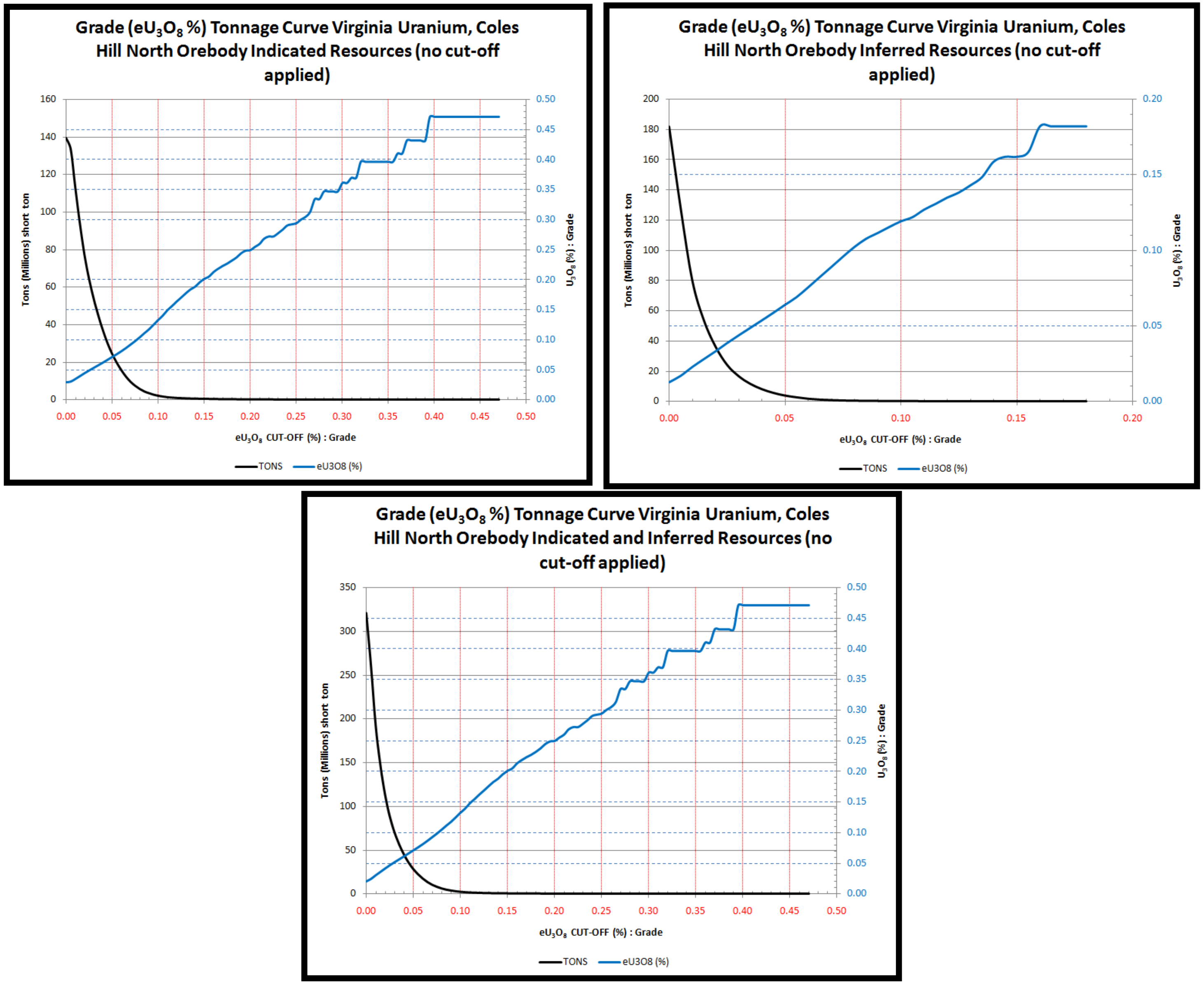

Figure 14.18: Grade tonnage curves for North Coles Mineral Resource estimate, Coles Hill Project.

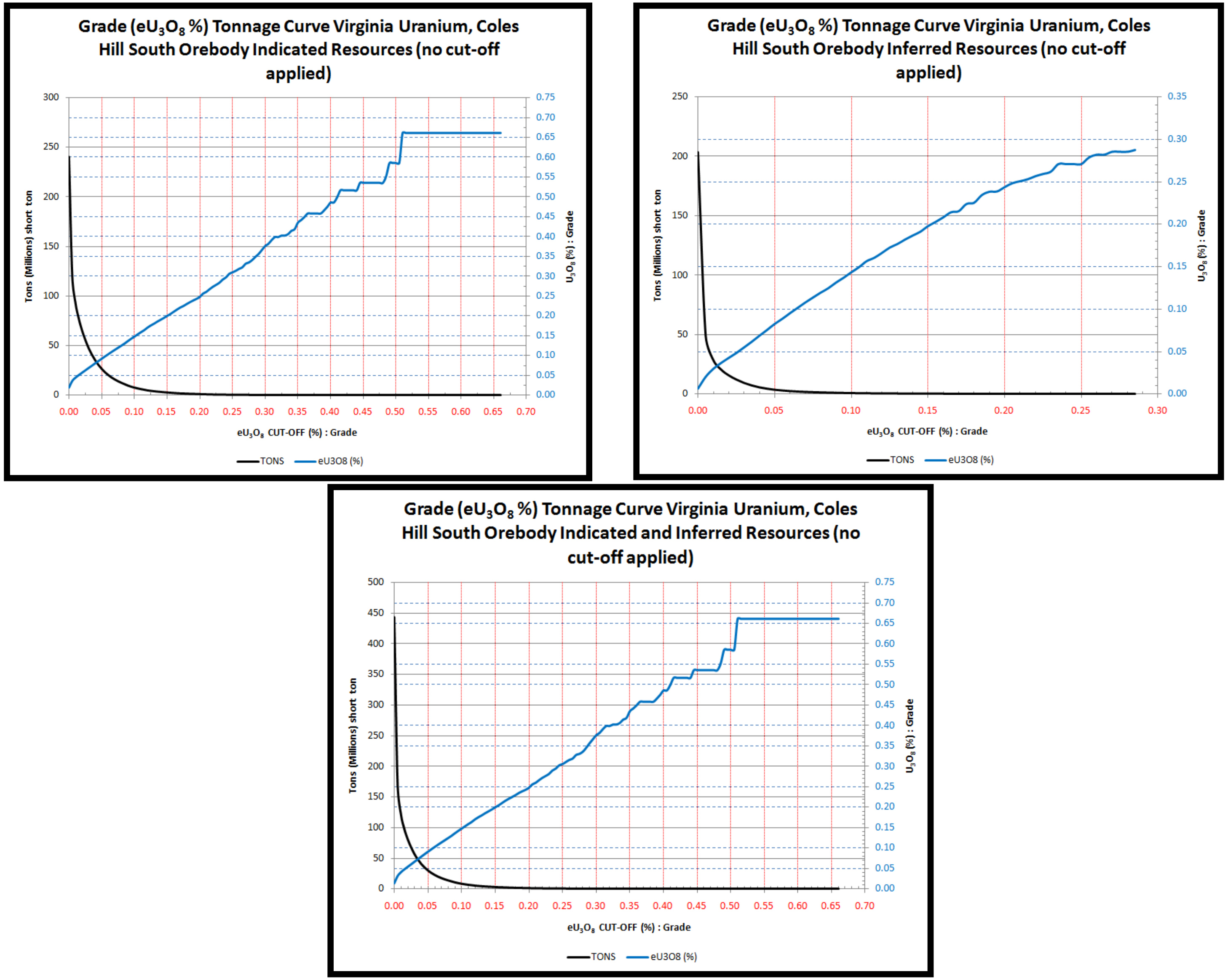

Figure 14.19: Grade tonnage curves for South Coles Mineral Resource estimate, Coles Hill Project.

14.3 Current Mineral Resource Estimate

The current mineral resource estimate represents and an approximate 11 % increase in the total Indicated Mineral Resource estimate with respect to total pounds and a 17% increase with respect to total tons which results in a 7% decrease in the estimated average grade. In addition, the current estimate includes inferred mineral resources not calculated in the previous estimate. It is the author's opinion that these variances are due to the additional data available for the current estimate and the resource methodology relating to block size, search distance, search algorithm.

Mineral resource estimates for both Indicated Mineral Resources and Inferred Mineral Resources follow as summarized in Table14.2 and Table14.3, respectively. Recommended cutoff grade for reporting is highlighted. Cutoff criterion is subsequently discussed.

TABLE14.2: INDICATED MINERAL RESOURCES

North Coles Hill				
Category	**Cutoff**	**Tons (million)**	**wt %eU_3O_8**	**lbs (million)**
INDICATED	0.025	64.16	0.050	63.73
INDICATED	0.050	24.39	0.072	35.14
INDICATED	0.075	7.40	0.099	14.57
INDICATED	0.100	2.08	0.134	5.56
INDICATED	0.125	0.82	0.171	2.79
INDICATED	0.150	0.44	0.202	1.76
South Coles Hill				
Category	**Cutoff**	**Tons (million)**	**wt %eU_3O_8**	**lbs (million)**
INDICATED	0.025	55.43	0.062	69.20
INDICATED	0.050	26.43	0.092	48.50
INDICATED	0.075	13.95	0.119	33.33
INDICATED	0.100	7.54	0.148	22.27
INDICATED	0.125	4.26	0.176	14.98
INDICATED	0.150	2.63	0.200	10.51
Total North and South Coles Hill				
Category	**Cutoff**	**Tons (million)**	**wt %eU_3O_8**	**lbs (million)**
INDICATED	0.025	119.59	0.056	132.93
INDICATED	0.050	50.81	0.082	83.64
INDICATED	0.075	21.35	0.112	47.90
INDICATED	0.100	9.62	0.145	27.83
INDICATED	0.125	5.08	0.175	17.77
INDICATED	0.150	3.07	0.200	12.27

TABLE14.3: INFERRED MINERAL RESOURCES

North Coles Hill				
Category	**Cutoff**	**Tons (million)**	**wt %eU_3O_8**	**lbs (million)**
INFERRED	0.025	24.17	0.039	18.68
INFERRED	0.050	3.82	0.064	4.89
INFERRED	0.075	0.56	0.096	1.08
INFERRED	0.100	0.19	0.119	0.46
INFERRED	0.125	0.06	0.138	0.16
INFERRED	0.150	0.01	0.162	0.04
South Coles Hill				
Category	**Cutoff**	**Tons (million)**	**wt %eU_3O_8**	**lbs (million)**
INFERRED	0.025	12.12	0.048	11.72
INFERRED	0.050	3.49	0.083	5.76
INFERRED	0.075	1.46	0.114	3.32
INFERRED	0.100	0.70	0.143	2.01
INFERRED	0.125	0.37	0.172	1.28
INFERRED	0.150	0.22	0.198	0.85
Total North and South Coles Hill				
Category	**Cutoff**	**Tons (million)**	**wt %eU_3O_8**	**lbs (million)**
INFERRED	0.025	36.28	0.042	30.41
INFERRED	0.050	7.31	0.073	10.65
INFERRED	0.075	2.02	0.109	4.40
INFERRED	0.100	0.89	0.138	2.47
INFERRED	0.125	0.43	0.168	1.44
INFERRED	0.150	0.23	0.196	0.89

14.4 Cutoff Criterion

Cutoff criterion is based on grade and/or a combination of thickness and grade. Cutoff criterion is represents the breakeven point of costs compared to revenue. As such the cutoff criterion varies over the life of the project with variations of costs and revenues. The following table provides a calculation of breakeven cutoff grades for direct operating costs (OPEX) and fully loaded costs at a sales price of $65 per pound. The calculation of breakeven cutoff grade allows for a mineral processing recovery of 85%.

TABLE14.4: MINIMUM CUTOFF GRADE			
	Operating Cost $/Ton	Gross Value at 85% and $65/lb	Approximate Breakeven Grade % eU_3O_8
Underground Mine and Mineral Processing OPEX only (Marginal Cost)	$27/ton*	$27.60	0.025 % eU_3O_8
Underground Mine and Mineral Processing with capital (Fully Loaded)	$ 51.50**	$50.83	0.046 % eU_3O_8

*(Lyntek, 2010) average OPEX without contingency.
**Add capital $6/ton, 30% contingency, and 20% margin
Based on the foregoing the recommended minimum grade cutoff criterion for the reporting of total mineral resources is 0.025 eU_3O_8. However, for mine planning a minimum grade cutoff of 0.046% is recommended along with a minimum mining thickness depending on the mining method selected.

14.5 Radiometric Equilibrium

By definition radioactive isotopes decay until they reach a stable non-radioactive state. The radioactive decay chain isotopes are referred to as daughters. When all the decay products are maintained in close association with the primary uranium isotope U_{238} for the order of a million years or more, the daughter isotopes will be in equilibrium with the parent isotope. Disequilibrium occurs when one or more decay products are dispersed as a result of differences in solubility between uranium and its daughters. In addition, both the primary isotope of uranium U_{238} and it daughters emit different forms of radiation as they decay. The primary field instruments for the indirect measurement of uranium, either surface or down-hole probes, measure gamma radiation. Within the uranium decay the gamma emitting elements are primarily $Radium_{226}$, $Bismuth_{214}$, and Uranium with $Radium_{226}$ being the dominant source of gamma radiation.

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

Recent sample assay and core data (2008) is available from 6 core holes, 50 samples, within the project, three each in North and South Coles. This data was collected by Behre Dolbear and is included in their technical report. This data included check assay and comparisons of the 2008 data to historic data. This comparison was possible as the cores were available for re-sampling and the drill holes were re-entered and logged geophysically (Behre Dolbear, 2008).

All of the samples tested were from reduced portions of the deposits at depths from 244 to 861 feet from the ground surface. The author reviewed this data and concludes that the chemical data verifies the radiometric equivalent data and indicates a slightly positive disequilibrium factor of 1.06 to 1.

In addition to reviewing this data the author reviewed historic radiometric and chemical assay data from 84 core holes. This data showed more variance that the 2008 data in part due to the fact that some of the samples were from shallower, oxidized portions of the deposit.

When the current mineral resource estimation was completed it was recognized that with portions of the mineralization exposed at the surface, it was possible if not likely that uranium mineralization in the oxidized zone would not be in radiometric equilibrium. When the mineral resource estimate was completed the estimate was subdivided by geologic zones for oxidized and reduced portions of the mineralization. Overall 5.01% of the indicated mineral resource and 3.50% of the inferred mineral resource is oxidized. This situation is more prevalent at North Coles representing 8.96% and 4.64% of the indicated and inferred mineral resources, respectively, as compared to South Coles representing 1.67% and 1.64% of the indicated and inferred mineral resources, respectively. In the oxidized portions of the deposit DEF factors as low as 0.5 to 1 were observed.

The author concludes that fully discounting the oxidized portions of the deposits and thereby reducing the mineral resource estimates by 5%, would be offset by the enrichment factor of 6% observed for the reduced portion of the deposit. No adjustment of the mineral resource estimate is thus recommended. However, it is recommended that future mine planning and mineral reserve estimation properly account for the negative disequilibrium conditions in the oxidized portions of the mineralization and that proper mine grade control procedures be established. This will not affect underground mine planning but would affect surface open pit mine planning due to the limited depth of surface oxidation.

14.6 Other Material Conditions

The author is not aware of other conditions which would materially affect the mineral resource estimates other than the environmental and permitting challenges as discussed under Item 20 of this report.

14.7 Mineral Resource Summary

At the minimum grade cutoff criterion the total mineral resources based of 0.025 %eU_3O_8 estimated mineral resources are summarized in Table 14.5 and Table 14.6 for indicated and inferred mineral resource categories, respectively.

TABLE 14.5: TOTAL INDICATED MINERAL RESOURCES

Total North and South Coles Hill				
Category	**Cutoff**	**Tons (million)**	**wt %eU_3O_8**	**lbs (million)**
INDICATED	0.025	119.59	0.056	132.93

TABLE 14.6: TOTAL INFERRED MINERAL RESOURCES

Total North and South Coles Hill				
Category	**Cutoff**	**Tons (million)**	**wt %eU_3O_8**	**lbs (million)**
INFERRED	0.025	36.28	0.042	30.41

ITEM 15. MINERAL RESERVE ESTIMATES

15.1 Mineral Reserves

No Mineral Reserves have been defined at the current level study for the Coles Hill Project. Indicated Mineral Resources are tabulated at various cutoff grades in Table14.2 under Item 14 of this report. By CIM standards a Mineral Reserve is defined as the economically mineable part of a Measured or Indicated Mineral Resource demonstrated by at least a Preliminary Feasibility Study. The current study is a Preliminary Economic Assessment rather than a Preliminary Feasibility Study.

For the purposes of the Preliminary Economic Assessment, preliminary mine designs for the extraction of a portion of the indicated mineral resource were completed as discussed under Item 16 of this report. Both surface and underground mining methods were considered for this project. This study focuses on underground mine extraction and utilized a cutoff grade on 0.06 %eU_3O_8 for the determination of mining limits. Mineralization is near surface in both the North and South Coles areas and it is recommended that further mine design and economic analyses consider a combination of open pit and underground mining.

The Preliminary Economic Assessment, as discussed herein, indicates that the portion of the mineral resource currently included in the preliminary underground mine design for the North and South Coles Hill areas is economic under current conditions. This portion of the Indicated Mineral Resource is considered in the Preliminary Economic Assessment as summarized in Table15.1.

TABLE15.1: PORTION OF INDICATED MINERAL RESOURCE CONSIDERED IN THE PRELIMINARY ECONOMIC ASSESSMENT

Total North and South Coles Hill				
Category	**Cutoff**	**Tons (million)**	**wt %eU_3O_8**	**lbs (million)**
Indicated	0.06	32.9	0.098	64.2

15.2 Other Material Conditions

The author is not aware of other conditions which would materially affect the mineral resource estimates other than the environmental and permitting challenges as discussed under Item 20 of this report.

ITEM 16. MINING METHODS

This section addresses the potential mining development of the Coles Hill uranium deposit. This section of the report also addresses the mine plan; estimates mine related costs, and scheduling for the overall economic analysis of this project. Mining by open pit and underground methods were evaluated as part of this study. Options considered ranged from all open pit mining to all underground mining, with several scenarios which combined open pit and underground methods. This analysis suggests that an all underground mining concept could be employed, but this does not preclude open pit mining as an acceptable option for further consideration. Based on a variety of factors, the preferred alternative for this study is underground mining utilizing Sub Level Open Stope (SLOS) mining. It is estimated that this method will recover approximately 85% of the mineralization above the determined grade cutoff when combined with limited secondary pillar recovery by room and pillar methods.

16.1 Underground Mining

The underground mining method recommended in this study consists of Sub Level Open Stope Mining (SLOS) for primary extraction and room and pillar and/or drift mining for partial recovery of the pillars. SLOS is recommended as the preferred method of extraction for a variety of reasons including:

- Mineralization is well suited to this mining method.
  - Mineralized zones are quite thick,
  - continuously mineralized, and
  - are steeply dipping which facilitates movement of the broken rock in the stopes by gravity to the draw points
- The method is bulk mining with high mining rates.
- The method has relatively low mining costs as compared to other underground mining methods.
- Worker safety and exposure is minimized.
  - Personnel do not need to enter the active stopes.
  - Remote loading equipment is utilized for clearing the stopes.
  - Workers are not exposed to unstable ground conditions.
  - Exposure to mine gases and dust is minimized.

Typical stope dimensions are shown on Figures 16.1 and 16.2 for the North and South Coles deposits, respectively. The general layout and dimensions are the same but due to the dip of the mineralization the draw point layout will vary.

Typical dimension of the stopes up to a maximum of 200 feet in height (based on mineralization); 150 feet along strike; and 50 feet perpendicular to strike.

Key assumptions used for an underground plan in this study include:

- Cutoff grade 0.06% U_3O_8.
- Production rate target at 3,000 tpd for 350 days/year.
- Primary stoping will extract approximately 70% of the total resource above the cutoff.
- Pillar retreat will extract an additional 15% of the total resource above the cutoff, (50% of the 30% not incorporated in the primary stopes).
- Based on the geostatistical model of mineralization the dimensions of the stopes will generally be limited by geotechnical considerations rather than the limits of mineralization. As such the stopes will generally end in mineralized material and thus no additional dilution was added beyond the waste taken within the stope limits which is estimate by the resource model.

- Similarly secondary recovery from the pillars will generally be within mineralized material rather than near boundaries on mineralization and no additional dilution was added.

Figures 16.3 and 16.4 show the general layout of the SLOS for the North and South Coles areas, respectively.

16.2 Geohydrology and Geotechnical Considerations

Data relative to geohydrology and geotechnical conditions at the site are of historical nature. From the historic data, the lower limit of rock compressive strength of the Leatherwood Granite is 7,500 psi. If testing demonstrates higher rock strength within the mining zones then the dimensions of the stopes could be safely increased. The current preliminary layout is conservatively based on the lowest reported rock strength. With respect to groundwater conditions, historic data indicates inflow from groundwater will be less than 200 gpm. While this data is considered by the author to be valid based on site observation, review of core samples, and general inferences from the geologic setting. For the purpose of preliminary mine design and preliminary feasibility studies, it is recommended that additional data be gathered with respect to both the geohydrology of the site and geotechnical conditions of the host formation and overlying and underlying strata.

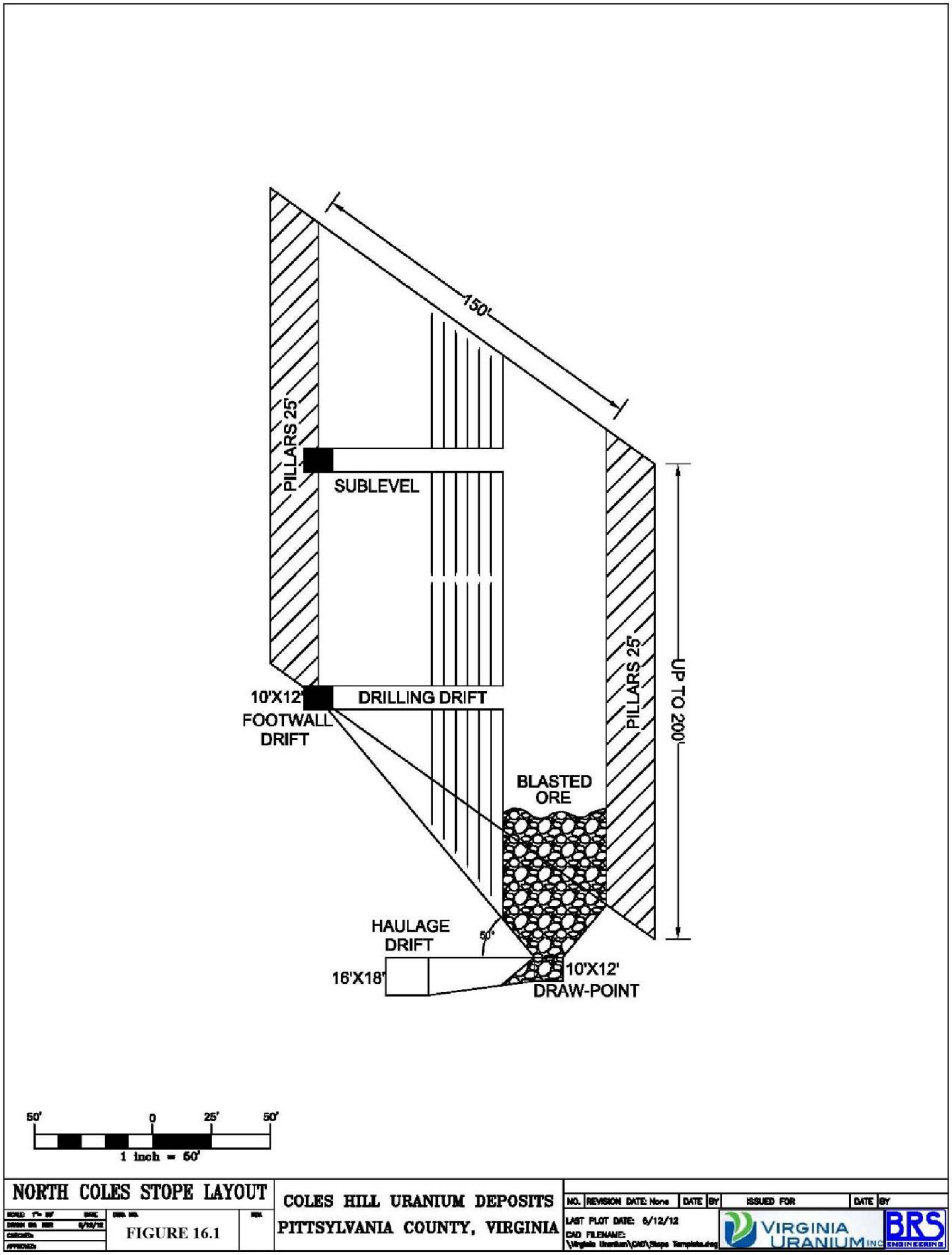

Figure 16.1: North Coles Stope Layout

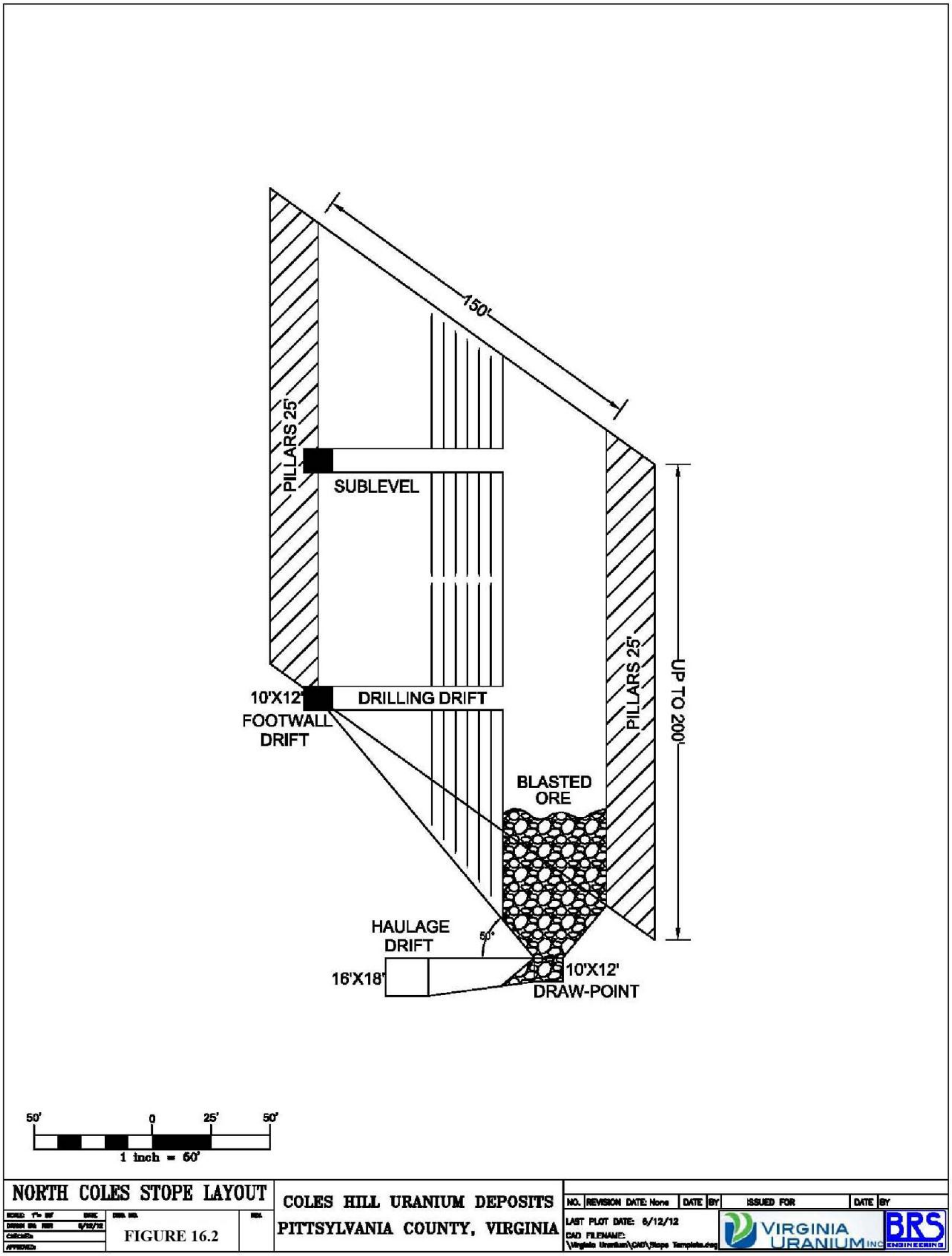

Figure 16.2: South Coles Stope Layout

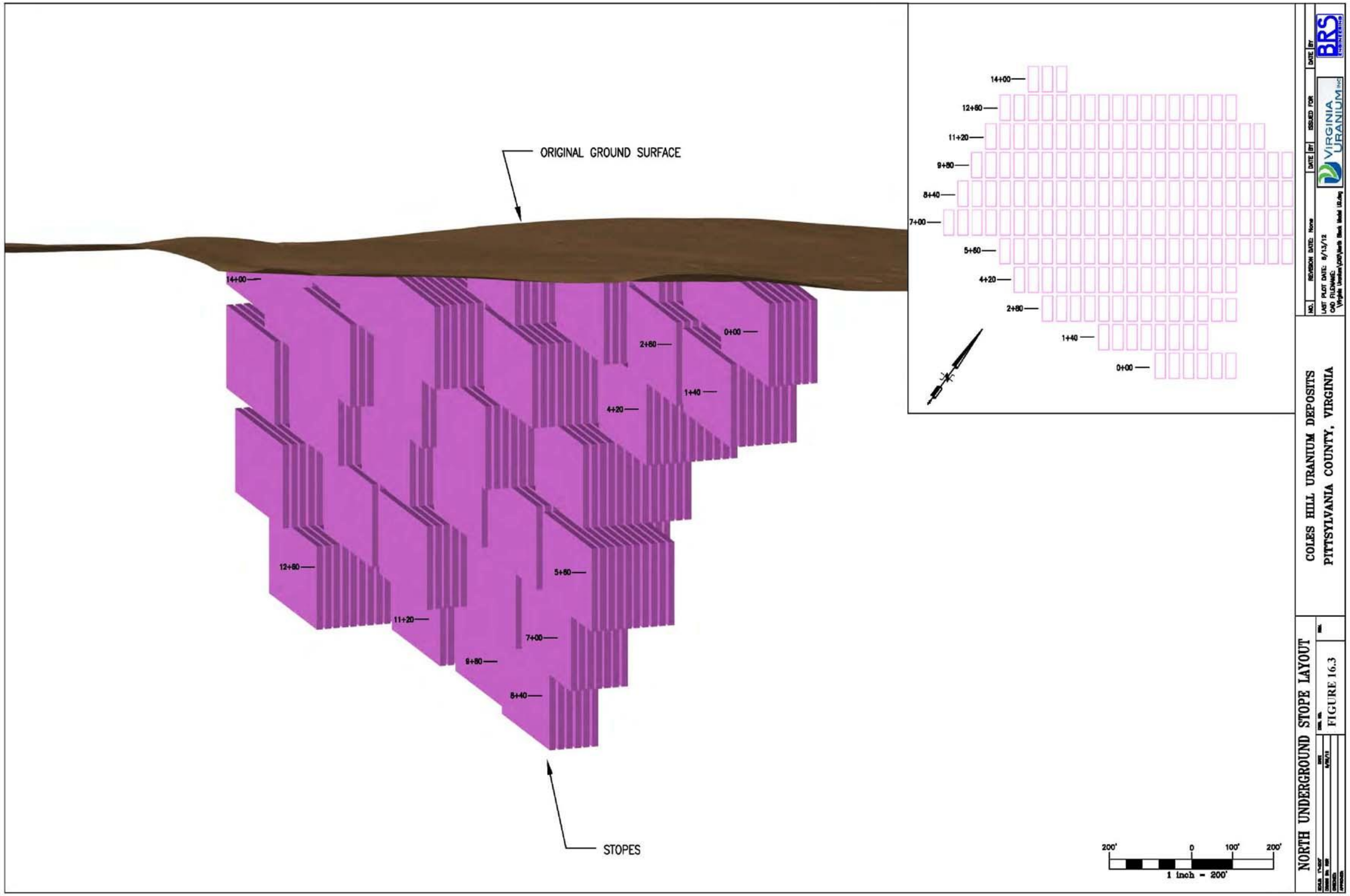

Figure 16.3: North Stope Layout

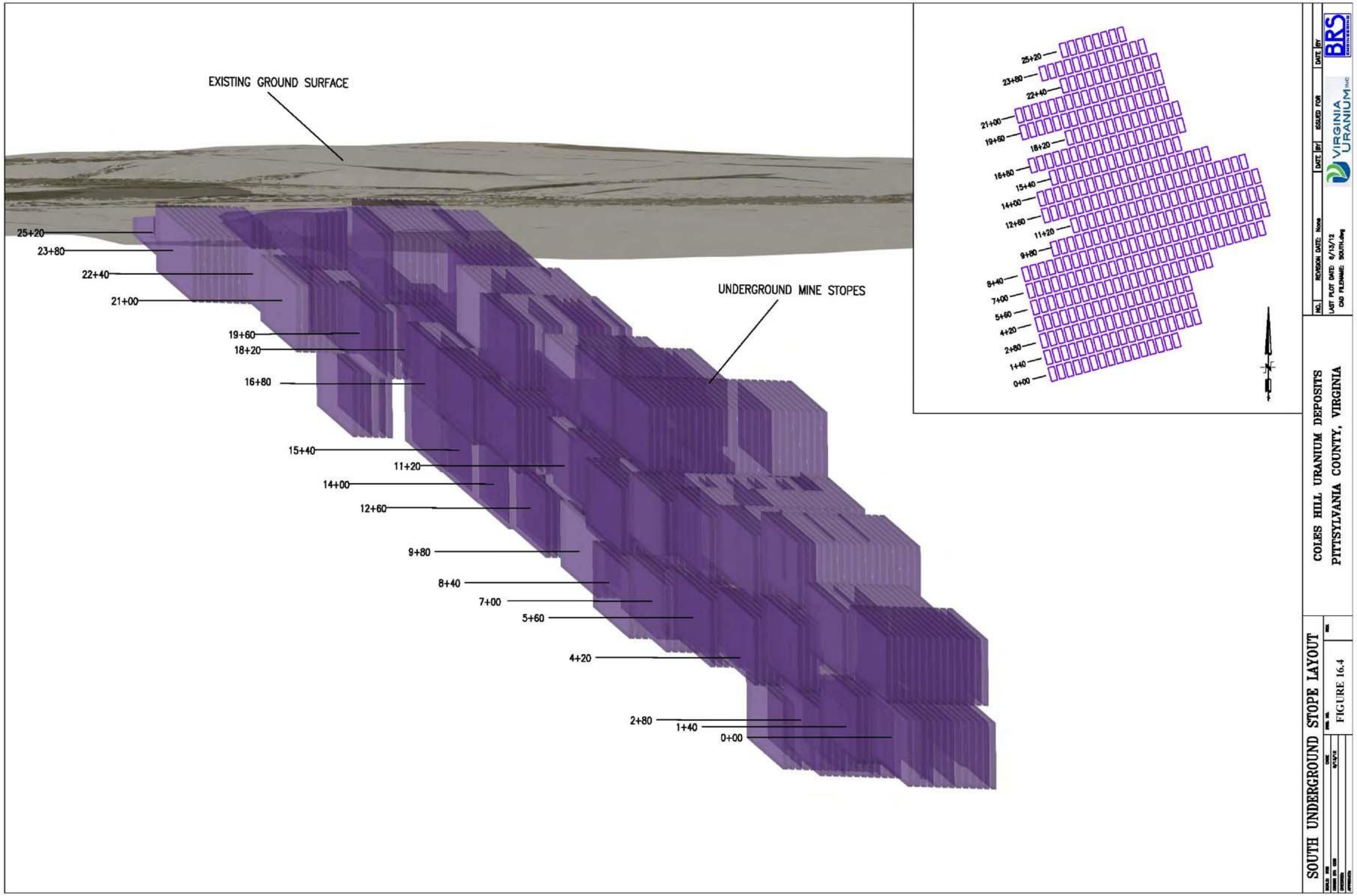

Figure 16.4: South Stope Layout

16.3 Mining Rate and Mine Life

The mining rate for primary stope extraction is estimated at 3,000 tons per day with four mining crews working ten-hour shifts for a total of 350 days per year. Without increasing the work force or scheduled hours the mining rate for pillar extraction is estimated at one third the stoping rate or 1,000 tons per day. The mining schedule assumes that work is only occurring in one mine at a time. The general schedule shown on Table 16.1 follows the general sequence South Coles Primary Stoping; North Coles Primary Stoping; followed by pillar removal at both North and South Coles. The current mine schedule meets the overall production target. Scheduling alternatives could be considered which accelerate production. Such alternatives could include the inclusion of partial open pit mining and/or the additional of underground mining crews and equipment such that North and South Coles could be mined simultaneously.

TABLE 16.1: MINE SCHEDULE

	Startup	Primary Stoping South Coles	Primary Stoping North Coles	Pillar Extraction North and South Coles
	Year 1	Years 2 through 15	Years 15 through 25	Years 25 through 35
Rate tons per year (tpy)	700,000 tpy	1050,000 tpy	1050,000 tpy	700,000 tpy
Total Tons x 1,000	700,000	14,700,000	10,500,000	7,000,000
grade %eU_3O_8	0.126	0.103	0.088	0.098
Total Lbs x 1,000	1,764,000	30,177,000	18,564,000	13,720,000

16.4 Development Requirements

The mine will be accessed via declines. Equipment selection and manpower, as subsequently discussed, includes specific allowance for the development of declines, haulage drifts and cross cuts, raises and vents to be bored. Main haulages will be 16 x 18 feet in cross section. Declines will be 12% or less in grade to accommodate 40 tons haulage trucks. Pre-development activities will commence 2 year prior to initial production in order to establish mine access and ventilation.

16.5 Mine Equipment

The following table lists the major mine equipment and is subdivided by mine development, mining, and support equipment.

TABLE 16.2: MINE EQUIPMENT

Mine Development		
	Type	Number
	2 Boom Drilling Jumbo	1
	Rock Bolters	1
	4 yd LHD	2

	40 t Haul Truck	2
	Shotcrete Truck	1
	Concrete Truck	1
	Scissor Lift Truck	1
	Incidentals	1
Underground Mining		
	Stope Drilling Jumbo	2
	Rock Bolters	1
	8 yd Remote LHD	3
	40 t Haul Truck	3
	Shotcrete Truck	1
	Scissor Lift Truck	1
	Incidentals	1
Support Equipment		
	Stores Delivery Truck	1
	Man Carrier	2
	Explosives Truck	2
	Water Truck	2
	Raise Borer	4
	Lube Truck	1
	Batch Plant	2
	U/G Grader	1
	Light Vehicles	10
	Trucks Flatbed	1
	Misc. Safety - Refuge chamber	2
	Communications System	1
	Incidentals	1

16.6 Mine Labor

Manpower estimates for the mine operation, including direct supervision, operating labor, and maintenance is provided in Table 16.3. This does not include manpower related to the operation and maintenance of the processing facility or general administrative staff, nor does the manpower estimate make any allowances for ancillary or off-site manpower related to equipment and materials suppliers.

TABLE 16.3: MINING LABOR REQUIREMENTS		
Hourly Labor		
	Stope Miners/Drillers/Blasters	16
	Development Miners	16
	Equipment Operators	8
	Support Miners	8
	Diamond Drillers	2
	Crusher/Backfill Operators	8
	Electricians	12
	Mechanics/Electricians	16
	Maintenance Workers	20
	Helpers	8
	UG Laborers	22
	Surface Laborers	12
Total Hourly		**148**
Supervision (Salaried)		
	Manager	1
	Superintendent	3
	Foremen	12
	Engineer	6
	Geologist	6
	Shift Supervisors	8
	Technician	8
	Accountant	4
	Purchasing	4
	Personnel	4
	Administrative Assistant	8
	Clerks	12
Total Salaried		**76**
	Total Manpower	**224**

16.7 Mine Operating Costs

TABLE 16.4: SUMMARY OF UNDERGROUND MINING COST ESTIMATE		
Sub-level Open Stope	**Primary Stopes**	**Pillar Extraction**
Operating Costs/Ton		

Equipment Operation	$ 1.36	$ 2.51
Supplies	$ 4.91	$ 5.05
Hourly Labor	$ 7.76	$ 9.97
Administration	$ 5.00	$ 5.61
Sundries	$ 1.85	$ 2.21
Total per Ton	**$ 20.88**	**$ 25.35**
Per Ton of Material	**$ 18.98**	**$ 23.05**
Additional allowances		
Increase Labor	$ 5.10	$ 6.58
Total Operating Cost/ton	**$ 24.08**	**$ 29.63**

Other Considerations

16.7.1 Cutoff Criterion

Minimum cutoff criterion is discussed for mineral resource determination under Item 14. This addresses the minimum breakeven grade that can be considered but does not fully address the average grade necessary to support overall project economics. In this sense, a selection of a mining cutoff grade is an iterative process. For this preliminary study, a run-of-mine grade of 0.10%eU_3O_8 was targeted. Iterations at various cutoff grades were completed. For this study a nominal cutoff grade of 0.06%eU_3O_8 was selected which resulted in an overall average run-of-mine grade of 0.098%eU_3O_8.

Detailed design efforts should continue to optimize grade and resource extraction and, where possible, extract higher grade zones early in the mine life. The design of mine stopes and pillars will be need to be optimized to maximize mine recovery and profitability.

16.7.2 Mine Ventilation

The main intake ventilation will be through the decline, while exhaust air will be drawn off through a series of raise-bore ventilation raises. Some raises will go through to surface and some from level to level. At full production, ventilation volumes are expected to be around 300,000 cfm, a quantity primarily driven by the underground diesel fleet emissions, within regulatory requirements. The velocity of the airflow in a 16’ x 18’ decline for 300,000 cfm is around 12 to 18 ft/sec.

Although radon gas is typically of concern in underground uranium mines, mineralization at Coles Hill is of moderate grade, and, as such, has far lower radon levels than higher grade mines such as the major uranium mines in Canada. When the ventilation is adequate to meet diesel emission standards, radon levels will be well below applicable standards. Regulation will require monitoring of mine gases and emission and protection of workers form potential exposures. As previously stated SLOS mining has advantages with respect to worker protection as workers do not need to enter the active stopes and the mineralized material is drawn using remote equipment.

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ITEM 17. RECOVERY METHODS

17.1 RESOURCES

17.1.1 Ore Resources

The mineral resources available for mining are discussed in Section 16, Mining Methods. The diluted ore grade values were used as the basis for the processing plant design and cost development.

17.1.2 Water Resources

The mill process water system will collect precipitation and site run-off collection, mine dewatering and recycle from the tailings system. The alkaline processing method designed maximizes internal plant process water recycling in order to conserve reagent and water consumption. Based on the predicted material balance the required water supply to the plant during standard operations is 270 gpm (0.6 cfs); during startup the required supply is 1000 gpm (2.2 cfs). The water treatment system is located near the mill and will process approximately 300 gpm water.

17.1.3 Other Resources

Other resources necessary for the mine and mill facility operations include natural gas, electricity, mine and mill consumables, and labor. As the area is highly developed, both locally and regionally, adequate supplies of the necessary resources are well within the economic reach of the project. The mine and mill power requirement is expected to be approximately 5-7 megawatt and the work force need will be about 325 - 350.

17.2 PRODUCTION AND MINE LIFE

The life of the project as described is projected to be 35 years, based upon the economics of the current uranium price and the ore grade calculation. Table 17.1 shows the expected production schedule, grade, and recovery assumptions used for the initial production years in the processing plant design and economic model.

TABLE 17.1: PRIMARY PROCESSING PRODUCTION ASSUMPTIONS

Operating Hours per Day	24
Scheduled Shifts per day	3
Scheduled Day per week	7
Operating Days per year average	350
Average Head Grade U_3O_8	0.098%
Mill Leach Recovery U_3O_8	85%

As the mining progresses later in the production stages, when only pillar extraction mining is being executed, the production rate will drop to about 1/3 of the primary mining rate. Processing operations will be modified to 6 ten-hour or 7 eight-hour operating shifts per week rather than 21 eight-hour shifts.

17.3 Process Description

This Preliminary Economic Assessment considers a mill and yellowcake facility that includes standard typical ore processing equipment to recover uranium. In this preliminary design, leaching of uranium will be accomplished using carbonate reagents.

17.3.1 Crushing and Grinding

Run-of-Mine (ROM) material will be crushed through a primary jaw crusher before feeding the grinding stage. It is assumed that the crushing plant will operate 2 ten-hour shifts per day at 90% availability, which results in an average operating time of 20 hours per day.

The ROM material will be fed to a vibratory grizzly feeder to route fine material past the jaw crusher to the reclaim stockpile. The coarse material will be crushed through the jaw crusher before feeding the reclaim stockpile. A covered overland conveyor system will transport the crushed material from the crushing circuit to the reclaim stockpile that is adjacent to the processing plant. Design will focus upon minimizing dust and noise impacts to the environment.

Primary grinding will be performed in a semi-autogenous grinding (SAG) mill. Material will be fed from the ore stockpile via belt feeders and conveyors. The SAG mill discharge will be coarse screened to remove pebbles, if any, and this oversize material will be re-circulated back to the SAG mill. Space has been allowed for retro-fitting of a pebble crusher if required.

The secondary grinding circuit is a typical cyclone-ball mill configuration. The primary SAG mill discharge is pumped directly to a cyclone cluster to remove the majority of fine material. The cyclone oversized material is fed to a ball mill in closed circuit. The target cyclone overflow product size is P_{80} 65-mesh (Tyler).

17.3.2 Alkaline Leach

The slurry is pumped from the grinding circuit to a densification thickener. The thickener overflow is re-circulated back to the SAG mill feed as process water. The thickener underflow is pumped to the leaching circuit.

The leach process occurs at atmospheric (ambient) pressure in eight (8) agitated tanks. The slurry flows by gravity from tank to tank through the leach circuit. The total retention time for the slurry in the leach circuit is 44 hours. The leach temperature is elevated to 194 ºF and maintained by adding steam to each leach tank. Soda ash (Na_2CO_3) and sodium bicarbonate ($NaHCO_3$) are added to each leach tank. The leach circuit discharge is then pumped to the counter current decantation (CCD) circuit.

Air is the oxidizing agent in the alkaline leach circuit and is added under low pressure through submerged spargers located in each leach tank.

17.3.3 Counter Current Decantation (CCD)

The CCD system is designed to recover the dissolved uranium values from the leached solids, which, after exhaustion, are subsequently disposed of in the tailings impoundment. Countercurrent washing of the leached slurry is carried out in eight (8) high capacity type 70 ft diameter thickeners. The thickeners are arranged at the same elevation, such that both the underflows and the overflows require pumping. The pregnant liquor and slurry solids are pumped from the leach system to the first CCD thickener. The solids settle to the bottom of the thickener and are pumped to the second CCD thickener while the relatively solid-free liquid overflows from the first CCD thickener and is pumped to a ClariCone. The ClariCone underflow is returned to the first CCD thickener while the overflow pregnant liquor is fed to the uranium precipitation stage. The underflow from each thickener is mixed with the overflow from the next unit in line. Wash water is mixed with the feed to the eighth thickener; wash water will be a mixture of fresh water and re-carbonated recycle. The underflow from the eighth thickener is pumped to the tailings facility.

17.3.4 Tailings Disposal

Tailings from a uranium milling facility are classified as 11(e)2 "tailings or waste material" as defined by NRC regulations. To this purpose, the NRC must approve the design of tailings disposal plans, as part of the overall license to operate. Adequate areas have been identified to place the tailings facilities and work is ongoing to finalize the location of the tailings cells.

Based on the current mine plan, slurry from the CCD plant will either be pumped to the surface tailings impoundment cells or to the paste plant. In the process, plant waste or tailings will be mixed with approximately 5% cement to produce a paste, which is estimated to contain about 30% moisture. As permitted by regulatory agencies, this paste will be returned to the underground workings as backfill using positive displacement pumps. The paste tailings will solidify, limiting the infiltration of outside water, limiting the remobilization of the tailings and adding structural integrity to allow pillar extraction and thus maximizing the uranium resource recovery. Additional work will need to be conducted to optimize the paste tailings design.

Tailings planned for surface disposal, employing regulatory guidelines, shows that there is currently inadequate surface disposal acreage within the current surface land control area. More in-depth evaluation of the surface tailings storage is required in order to optimize the design and reduce local and regional concerns regarding the safety considerations of the tailings facilities relative to the wet environment at the mine site. The company is considering the sub-surface disposal of all tailings, and expects to include that priority in the next project feasibility study.

17.3.5 Uranium Precipitation, Drying and Packaging

The pregnant (uranium bearing) liquor from the ClariCone overflow is transferred to the first of three precipitation tanks in the first precipitation circuit. Caustic soda (NaOH) is added to maintain a constant pH in the precipitation stage. Uranium is precipitated as uranium peroxide ($UO_4 \cdot 2H_2O$). The uranium precipitate slurry is pumped from the third precipitation tank to the first yellowcake thickener, where most of the solution is separated from the uranium oxide solids. The thickener overflow is sent to a re-carbonation stage for recycling to the grinding and leaching circuits.

A second stage of precipitation is designed to remove impurities entrained in the first precipitate. The first thickener underflow is fed to the yellowcake re-dissolve tank, where the solids are contacted with sulfuric acid. The uranium is then re-precipitated in a series of three precipitation tanks. The uranium precipitate slurry is pumped from the third precipitation tank to the second yellowcake thickener, where most of the solution is separated from the uranium oxide solids. The thickener overflow is sent to a re-carbonation stage for recycling.

The thickener underflow is fed to the yellowcake filter press. In the yellowcake filter press, most of the solution in the uranium precipitate slurry is removed. The resulting filter cake is then washed with water to remove all the remaining dissolved salts so they do not appear in the final product after drying.

The dewatered yellowcake falls into a shaftless conveyor where it is re-slurred with clean water then is pumped to rotary paddle vacuum dryer. The off-gasses, mainly water vapor, are drawn through a "sock" type filter to trap dust. The gas system has a condenser to recover the water, which is recycled. Vacuum is provided by a liquid ring vacuum pump, which will act as a scrubber to capture any dust escaping the dryer. Seal water from the vacuum pump will be used elsewhere in the plant, so any uranium reaching the pump will be recycled.

The dried yellowcake discharges from the dryer by gravity via a rotary valve and a drum filling system into 55-gal steel drums. The drum filling station includes a weigh scale; a vibrator and controls that fill each drum to a pre-assigned net weight. The yellowcake product drums are transferred by forklift truck to product storage.

ITEM 18. PROJECT INFRASTRUCTURE

18.1 Existing Infrastructure

18.1.1 Access by Air

Access by air is typical of eastern U.S. coastal regions. Major air carriers provide service at national airports with a two hour drive from the property. Typically access can be gained by travel into Raleigh Durham in North Carolina or Richmond in Virginia. Smaller aircraft use Danville and Lynchburg Virginia airports.

18.1.2 Roads

The property is accessed by a major north-south highway, U.S. Highway 29, and is between the cities of Danville and Lynchburg. Danville, historically a mill and tobacco town, is about 30 miles (48 kilometers) to the south. The site can be accessed by driving through the towns of Chatham or Gretna, and then secondary roads. From Chatham, Virginia, secondary paved roads such as Chalk Level Road (State Road 685) intersect directly with the gravel Coles Road (State Road 690) that bisects the project area. The roads in the region and the vicinity provide very good access to the property for construction and daily mine traffic. It will be necessary to specifically review the local roads and bridges in the region relative to heavy loads that may be required during construction activities.

18.1.3 Rail

Virginia has excellent coverage for rail transportation. The closest rail would be the Norfolk Southern rail line that runs north and south between Lynchburg and Charlottesville, Virginia and ties into major international rail hubs. A second Norfolk Southern line runs east and west through Roanoke, Altavista and Burkeville. These railroads will provide regional rail service for the project.

18.1.4 Local Labor, Towns, and Villages

There are many local towns and villages within a 100 mile radius. These towns will provide the necessary support businesses for the mining operation as well as housing and infrastructure for the mining personnel that will work at the mine and processing plant. All the necessary schools, hospitals, emergency services, police and fire personnel and facilities, etc. are all well established. The region is attractive and can easily attract the necessary skills required to operate the mine and plant. In addition, coal mining is active within the region such that there are many mine supporting businesses that will be able to quickly adapt to the needs of the mine and plant.

18.1.5 Power, Gas, and Water

The Virginia power grid and Williams' Transco interstate gas pipeline provide a local source of natural gas and electrical power. Only short connections will be necessary to tie into these regional power and gas lines. Water supply will be provided from local sources as the rainfall in the region provides adequate water resources for the project.

18.1.6 Required Infrastructure

Required infrastructure for the project includes site access roads, power lines, natural gas lines, and surface land for waste stockpiles, mining and processing plant and facilities, and tailings storage areas.
The CHUP consists of leases on the mineral and surface rights to a portion of the Coles property and the contiguous Bowen property, as well as other properties. The total surface rights and leases cover approximately 2,296 acres (929 hectares). The use of surface rights has been restricted by the leases

covering about 648 acres near the historical Coles farm house (Protected Area). Surveyed land plots are available in Chatham, the county seat.

Mining personnel can reasonably be recruited from the local area, as the skill sets needed for miners exist already among people and companies who are comfortable with mining, farming, and heavy equipment. Nearby Virginia Tech, in Blacksburg, Virginia, have a large mining engineering department and a significant geology department that can provide high-quality employees. The Commonwealth of Virginia has a strong mining heritage that is active in the western portion of the state, with many companies providing specialized mining services and veteran mining personnel who can assist in this project. This project site has significant advantages over remote areas for the recruitment and retention of mining personnel as well as local and regional infrastructure.

The Virginia Uranium, Inc. controlled surface area will be used for the mining of the North and South Coles Hill Uranium Deposits. A tailings storage facility is planned along the southeastern portion of the property along a ridge that parallels a small local drainage and on other surface lands surrounding the uranium deposit. Recent third-party work has been conducted that provides new information regarding flood plain designations. It will be necessary to collect this information, analyze the meaning of the data, and plan accordingly relative to the existing tailings storage plan. Virginia Uranium, Inc. has about 1,508 acres (609 hectares) of contiguous land under control for potential mine, mill, waste, and tailing management areas, as well as set-back provisions. A number of areas outside the leased area provide suitable sites for tailings management areas and plant sites. Some of the tailings could be returned to mining areas. Additional land may be required to ensure efficient site operations. Development of the NCHD and SCHD as a mine may require management of local surface drainages, wetlands and groundwater, and relocation or closure of part of State Road 690, a gravel country road called Coles Road.

ITEM 19. MARKET STUDIES AND CONTRACTS

Due to the existing stage of development, there are no contracts in place for perceived sales of uranium from the project. The uranium market is difficult to assess in detail due to the confidentiality of agreements between buyers and sellers working in a closed market. The market is best defined by uranium price forecasters who provide professional market reports on uranium sales that assist in predicting market behavior. The prices employed in this report are based upon the three-year rolling average for the long-term uranium price published by UxC and Trade Tech. The average 3-year rolling average price is $63.35 through May of 2012, so an average rounded price of $64 per pound of U_3O_8 has been used in the economic analysis.

ITEM 20. ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Results and Issues

Some database updating activities have taken place, however, some data collection activities will be initiated when appropriate to provide the most recent environmental information. The information below provides the Project Master Plan environmental baseline studies that include– current updates to Marline databases.

- Surface watershed hydrology and floodplain – final draft report – results available
- Surface water quality – draft in preparation
- Ground water hydrology – future program
- Ground water quality – future program
- Meteorology – scoping study in progress, five temporary towers have been constructed
- Air quality – future program
- Ecology –aquatic, terrestrial, avian, vegetation – work plans in draft, some field data collected
- Cultural resources – completed – results available for exploration areas
- Radiological background – update in progress – new study of previous results is available
- Soils – work plan in draft, have Zipper and Donovan 2011 report available for 20 sites
- Socioeconomics – completed by others – results available

20.1.1 Issues

- No scientifically supported issues.

20.2 Requirements and plans for waste and tailings disposal, site monitoring, and water management

Plans for waste and tailings disposal - Prime option is disposal of tailings in underground workings, as cementitious backfills. The current mining concept employs surface tailing cells located across the property. Cells do not have a size larger than 40 acres surface area, are located above floodplains, and are lined as required by federal standards. Overburden will be used for grade adjustment, closure cover, and backfill, with excess being spoiled in designed berms. Saprolite materials containing clay are present on the property and are well suited and can be used for lining the basins of the surface tailings cells as well as capping the cells for final reclamation. The company anticipates that the next feasibility study will include a mine plan that prioritize the placement of all tailings below grade.

Water management – Plans will avoid disturbance in the active major (perennial) drainage channels and flood plains wherein these drainages will not be impacted or diverted. Small local intermittent drainages around the mine facilities will be managed and controlled to minimize surface runoff that comes into contact with mining areas or materials. Surface water that is impacted by mining and milling activities will be impounded in lined containments and treated as needed to meet water quality standards before release from the permit/license area. All mill process water will be double-contained and recycled through the mill circuit to the extent possible. The water source will be on-site wells drawing water from the ore zone, from other wells, and from newly constructed local surface water impoundments.

Site monitoring – Water quality will be monitored at stream sampling stations and monitoring wells located both up-gradient and down-gradient from the site. Air quality will be monitored in primary up-wind and down-wind directions and the nearest residence using stations equipped with particulate and radiological sampling devices. Water and air quality measurements will be reported to regulatory agencies quarterly. After mine closure, air and water quality will continue to be monitored at the same sampling locations until mill decommissioning and tailing stabilization are complete. Biological parameters will be monitored seasonally as required by state agencies. Both ground water quality and tailing closure performance will be monitored until permit/license termination and full release of financial assurance, probably at least 10 years after decommissioning.

20.3 Project permitting requirements, status of permit applications, and known requirements to post performance or reclamation bonds

Mine permitting – VA has an existing hard rock mine permitting process in place, and VA Governor Bob McDonnell has directed a Uranium Working Group to "establish a draft statutory and conceptual regulatory framework" for uranium mining due in report by December 2012. Legislation will be introduced to VA General Assembly in 2013 to lift the uranium moratorium and enable preparation of uranium-specific mining rules, expected to be an addition to Reclamation Regulations for Mineral Mining (4 VAC 25 – 31) . The mine permit is expected to require financial assurance for the cost of mine closure by a third party. No mine permit application will be submitted until after these rules have been implemented.

Mill licensing – Initiation of mill licensing process through the U.S. Nuclear Regulatory Commission (US NRC) will begin after the uranium moratorium has been lifted, probably in 2013. Licensing will be in accordance with existing US NRC and US EPA regulations for the mill life cycle. The license will be a Source Material License as required by Chapter 10 Code of Federal Regulations Part 40. The mill license is expected to require financial assurance for the cost of mill closure by a third party.

Other permits – Existing VA and US EPA standards will apply to protection of radiological health and safety, water and air quality, ecological resources and cultural resources. VDEQ permits will probably be required for particulate and other airborne pollutant releases, for surface water impoundments, and for surface and ground water discharges, specifically:

- Virginia Water Protection Individual Permit
- Virginia Air Quality Permit
- Virginia Pollutant Discharge Elimination System Permit
- Virginia Dam Construction Permit
- Virginia Dam Low Hazard Potential Regular Operation and Maintenance Certificate.

20.4 Potential social and community-related requirements and plans.

Public Participation Program

Since 2008, Virginia Uranium, Inc. Inc has been conducting public outreach and information activities, both formal and informal. Formal activities have included presentations to Virginia state and local officials, communities, business groups and written response to inquiries from these parties. Informal public outreach has included site tours, sponsored trips to uranium facilities elsewhere, interviews with news media, and one-on-one meetings.

A Public Participation Plan (PPP) for the Coles Hill Project was prepared in April 2011 and has been incrementally implemented since then. The PPP provides additional structure for public participation by linking VUI's efforts to the federal and state public participation requirements (4VAC25-11), identifying responsibilities of VUI and regulatory agencies, and outlining the required elements of a Public Involvement Plan (PIP) that will be followed throughout the project life cycle. The PIP builds on VUI's on-going outreach activities and adds to them as appropriate for successive stages of the project.

20.5 Mine Closure Requirements

Mine closure – The mine will be required to submit a closure plan as part of the mine permit under Virginia's Reclamation Regulations for Mineral Mining [4 VAC 25 - 31] as administered by the Department of Mines, Minerals, and Energy. Initial fees are $1050/ disturbed acre, and yearly updates of the bond are required under 4VAC25-31-220, with the current rate of $1000/ acre disturbed in the succeeding year. Closure must establish an approved post-mining land use.

Mill closure – The mill must be decommissioned and the mill facilities, including tailings, closed in accordance with federal requirements (10 CFR 40, 40 CFR 192). These include demolition and removal or on-site disposal of all radiologically contaminated equipment and materials, site soil cleanup to residual radium levels not exceeding 5 pCi/g above background min the top 0.5 foot of soil, and stabilization and covering of tailings to protect against release for 1000 years and limitation of radon flux from cover surface of not more than 20 pCi/m^2 s.

ITEM 21. CAPITAL AND OPERATING COSTS

21.1 Mining Capital Costs

Major mine related initial capital costs include costs for mine equipment, development work (declines etc), and support facilities are summarized in Table 21.1.

TABLE 21.1: SUMMARY OF MINING INITIAL CAPITAL COSTS (IN $000)

Type	Number	Price	Total
Mining Equipment			
2 Boom Drilling Jumbo	1	$ 655	$ 655
Rock Bolters	1	$ 674	$ 974
4 yd LHD	2	$ 565	$ 1,130
40 t Haul Truck	2	$ 614	$ 1,229
Shotcrete Truck	1	$ 461	$ 461
Concrete Truck	1	$ 274	$ 274
Scissor Lift Truck	1	$ 284	$ 284
Other	1	$ 651	$ 651
Sub Total			**$ 5,357**
Decline Equipment			
Stope Drilling Jumbo	2	$ 655	$ 1,310
Rock Bolters	1	$ 674	$ 674
8 yd Remote LHD	3	$ 840	$ 2,520
40 t Haul Truck	3	$ 614	$ 1,843
Shotcrete Truck	1	$ 461	$ 461
Scissor Lift Truck	1	$ 284	$ 284
Other	1	$ 1,071	$ 1,071
Sub Total			$ **8,163**
Support Equipment			
Stores Delivery Truck	1	$ 101	$ 101
Man Carrier	2	$ 101	$ 202
Explosives Truck	2	$ 151	$ 302
Water Truck	2	$ 137	$ 273
Raise Borer	4	$ 310	$ 1,240
Lube Truck	1	$ 189	$ 189
Batch Plant	2	$ 38	$ 76
U/G Grader	1	$ 282	$ 282
Light Vehicles	10	$ 40	$ 400
Trucks Flatbed	1	$ 400	$ 400
Misc – Refuge Chamber	2	$ 100	$ 200
Communications System	1	$ 315	$ 315
Other	1	$ 642	$ 642
Sub Total			**$ 4,622**
TOTAL INITIAL CAPITAL			**$ 18,142**

The initial mine capital costs of US$26M will be incurred over a three year period during the pre-production and startup phase of the project. Ongoing mine capital costs for capital equipment replacement and capitalized mine development expenses (declines, development drifts, and ventilation shafts) will continue to a few years prior to completion of mining. These costs over an approximate 30 year period total in about US$42 million bring the total life of mine mining capital cost to approximately US$68 million. Note that the capital cost estimates do not include surface facilities and access incorporated in the plant site costs, nor is any contingency included in these estimates. Contingencies have been incorporated in the costs model as separate line items.

21.2 Mining Operating Costs

Underground mine operating costs were estimated for Sub-Level Open Stope mining for the primary stopes at a production rate of 3,000 tons per day. Operating costs for pillar extraction were estimated based on cut and fill methods at a mining rate of 1,000 tons per day. The estimated mining operating costs on a per ton basis is summarized in Table 21.2.

TABLE 21.2: SUMMARY OF MINING OPERATING COSTS FOR 3,000 TPD

Sub-level Open Stope	**Primary Stopes**	**Pillar Extraction**
Operating Costs/Metric Tonne		
Equipment Operation	$ 1.36	$ 2.61
Supplies	$ 4.91	$ 5.05
Hourly Labor	$ 7.76	$ 9.97
Administration	$ 5.00	$ 5.61
Sundries	$ 1.85	$ 2.21
Total per Metric Tonne	$ 20.88	$ 25.35
Per Ton of Material	**$ 18.98**	**$ 23.05**
Additional allowances		
Increase Labor	$ 5.12	$ 6.58
Total Operating Cost/Ton	**$ 24.08**	**$ 29.63**

Additional costs for paste backfill of mined areas are included as a portion of the tailings operating costs.

21.3 Processing Capital

The processing capital cost for a 3,000 tpd alkaline leach plant is summarized in Table 21.3.

TABLE 21.3: ALKALINE PROCESSING CAPITAL SUMMARY

Direct Costs	**$US (000)**	
Material Handling	$ 5,258	
Grinding	$ 6,697	
Leaching	$ 5,964	
CCD and Filtration	$ 7,438	
First Stage Precipitation	$ 1,631	
Precipitation, Filtering and Packaging	$ 4,523	
Reagent System	$ 444	
Utilities	$ 3,265	
Concrete	$ 5,203	
Structural Steel	$ 5,466	
Buildings	$ 6,153	
Electrical	$ 2,905	
Power Substation (5MW)	$ 3,000	
Instrumentation and Control	$ 2,047	
Piping	$ 1,562	
Tailings (reported below)	$ -	
Other	$ 1,707	
Subtotal Direct Costs		**$ 63,243**
Indirect Costs		
Engineering	$ 6,280	
Construction Management	$ 1,884	
Freight	$ 1,972	
Contractor Small Tools and Consumables	$ 1,262	
Subtotal Indirect Costs		**$ 11,398**
Subtotal Capital Costs		**$ 74,641**
Contingency		
Contingency at 25%	$ 18,660	
Subtotal Contingency		**$ 18,660**
TOTAL CAPITAL		**$ 93,301**

21.4 Tailings Capital Costs

The tailings impoundment cell costs are outlined in Table 21.4. The cost to build includes all material and labor. The reclamation cost includes material and labor for five feet of cover, six inches of topsoil and re-vegetation. The capital for the cells are included on a schedule based on mining rate, impoundment construction taking place the year prior to need, and no more than two 40 acre cells being disturbed at any given time.

TABLE 21.4: TAILINGS IMPOUNDMENT COST SUMMARY

	Cell Capacity	**Cost/Cell ($000)**	
	KTons/Cell	**Build**	**Reclaim**
Cell 1A – Bottom	2,769	$ 10,232	$ 696
Cell 1A – Top	-	$ 5,370	
Cell 1B	2,769	$ 9,474	$ 673
Cell 1C	2,769	$ 10,228	$ 554
Cell 2A	920	$ 3,366	$ 760
Cell 2B	831	$ 3,991	$ 754
Cell 6	811	$ 5,568	$ 307
Cell 3A	750	$ 3,314	$ 180
Cell 3B	412	$ 3,079	$ 167
Cell 4A	675	$ 1,156	$ 82
Cell 4B	1745	$ 2,052	$ 111
Cell 7A	1653	$ 5,426	$ 294
Cell 7B	1592	$ 5,182	$ 281
Cell 7C	1508	$ 5,193	$ 281
Cell TDD	4,026	$ 25,076	$1,359
Total	**19,269**	**$ 99,162**	**$ 6,499**

21.5 Processing Operating Costs

The processing operating costs are summarized in Table 21.5.

TABLE 21.5: SUMMARY OF PROCESSING OPERATING COSTS FOR 3,000 TPD ALKALINE PROCESS

Area	**Annual**	**$/ton of ore**	**$/lb** U_3O_8
Raw Materials	6,070,461	$5.07	$2.93
Labor (All inclusive)	5,839,325	$5.56	$3.20
Power	1,697,205	$1.62	$0.93
Water	207,900	$0.20	$0.11
Spare Parts	1,563,677	$1.49	$0.86
Office and Lab Supplies	500,000	$0.48	$0.27
General and Administrative	850,000	$0.81	$0.47
Yellowcake Transportation.	218,725	$0.21	$0.12
Total Operating Costs w/o Cont.	**$16,947,292**	**$15.43**	**$8.89**

The manpower for the processing plant is shown below in Table 21.6.

TABLE 21.6: PROCESSING LABOR REQUIREMENT

Position	Persons
Salaried	
Mill Superintendent	1
Asst Mill Superintendent	1
Mill General Foreman	1
Shift Forman	4
Maint Gen Forman	1
Maintenance Forman	4
Instrument Technician	4
Employee Relations Mgr	1
Secretaries	2
Radiation Safety Officer	1
Safety Supervisor	1
Environmental Officer	1
Purchasing Agent	1
Warehouseman	2
Metallurgist	1
Chief Chemist	1
Controller	1
Clerks	2
Total Salaried	**30**
Hourly Labor	
Maintenance:	
Electricians	2
Electrician Helpers	2
Mechanics	10
Mechanics Helpers	10
Subtotal	**24**
Plant Operation:	
Plant Technician	2
Site Security	4
Safety and Environmental Tech	3
Laboratory Analysts	3
Loader Operator	4
Crusher Operator	4
Grind/Leach Operator	4
CCD Operator	4
Precip Operator	4
Tailing Operator	4
Plant Helper	4
Utility	2
General Labor	4
Subtotal	**46**
Total Hourly	**70**
Total Salaried	**30**
Grand Total Labor	**100**

21.6 Tailings Operating Costs

The operating cost for the transport of tails from the CCD circuit to the surface tailings impoundments is included in the processing cost for the plant. The operating costs for the tails paste processing and transport of paste for backfill is $2.11/ton of ore mined. These costs include transport, power, fly ash, fuel and expendable items. Labor is accounted for elsewhere. The costs for deposition of the tailings material into the tailings cells are included in the plant operating costs.

ITEM 22. ECONOMIC ANALYSIS

22.1 Lease and Royalty Agreements

The lease and royalty costs total approximately \$140 million dollars (or \$2.56/lb U_3O_8) over the life of the mine.

22.2 Virginia Tax Information

Virginia's tax system segregates the local and state sources of revenue by allowing the local governments to collect taxes on real estate, tangible personal property, machinery and tolls used in a mining or manufacturing business and merchants' capital. The major taxes paid by manufacturers and mining companies are real estate, machinery and tools taxes.

The plant equipment in this estimate has been subjected to tax as Business Personal Property. The rate is assessed at 30% of original purchase price in Year 1 and is reduced by 2.5% per year thereafter, until which time the assessed rate reaches 5% and then remains constant. The tax rate is \$8.50 per \$100.00 assessed value. The maintenance vehicles in this estimate have been subjected to tax by the same methodology as Business Personal Property Tax and are based on the Department of Motor Vehicles original sales price of the vehicle. All mining equipment has been taxed as Machinery and Tools. Tax is assessed at 10% of original capitalized cost and the tax rate is \$4.50 per \$100 assessed value. The property tax is estimated at \$12M over the life of the mine with a cost of \$0.22/lb. of U_3O_8.

22.3 Cash Flow Model for 3, 000 TPD

A cash flow model was developed for the alkaline case at 3,000 tpd that models annual periods of cash inflow and outflow without financing cost of capital. This model was based on the design criteria for 3,000 tpd, a 3,000 tpd conceptual mass balance and capital and operating costs for 3,000 tpd. Any apparent rounding differences contained in the tables and values related to the cash flow are due to the number of significant figures contained within the model in Microsoft Excel. The key assumptions for the economic model are discussed below.

The project schedule, sequence of mining, mining rate and mining costs were also used to develop the cash flow model. It is assumed that ore production commences one year after all mining permits and licenses have been received. The primary mining rates are 700,000 tons in year one, 1,050,000 tons from years two through four, 700,000 tons per year for years five through twenty-five, and 467,000 tons per year from year twenty-six through year thirty-five. In addition to this production, mining pillars accounts for 350,000 tons per year for years five through twenty-five, and 233,000 tons per year from year twenty-six through year thirty-five. The predicted grade of production, which is based upon mine plans through the geologic model, appropriately diluted, show a range of grades from 0.079% to 0.126% U_3O_8, with an average of 0.0965% U_3O_8. The grades are based upon the geologic model discussed above. Assuming a plant recovery of 85%, the total uranium production ranges from 1,225,000 lbs to 2,646,000 lbs. and averages 1,885,000 lbs. U_3O_8/year. The mill design and recovery rate is based upon prior metallurgic studies which have been augmented by recent testing as described above.

The primary mining cost is forecast at \$24.08/ton and forecast at \$29.23/ton for the pillar recovery operations.

The processing costs have been estimated based upon the process flow sheet for the operation, the manpower requirement, power and water requirements, and necessary reagents. The tailings costs are included in the capital for the Paste Fill Plant, Piping to Impoundment Cells and Tailings Impoundment Cells, and for operating the tailings cells, the paste fill processing, and tailings cell cover and topsoil and re-vegetation.

The costs are included as a schedule based on the mining rate, impoundment construction taking place the year prior to need, and no more than two 40 acre cells being disturbed at any given time.

Including 25% contingency, the total capital investment prior to production is $147 million, however, the total capital through the 4th year of production of $200 million is a better estimate of initial capital due to the staging of the tailings cell construction. Including 25% contingency, the total capital spending over the life of the facility is $329 million. This cost estimate excludes any other specific non-project related costs that would be in addition to this project. For example, it would be reasonable to expect that further exploration and research programs could certainly range up to an additional $40 million in an effort to generate additional resources or address other non-project goals.

The cash flow model is summarized in Table 22.1. The economic analysis at a yellowcake price of $64/lb shows an internal rate of return of 36.3% before income taxes; at a discount rate of 7% the net present value is $427 million, including a 25% contingency.

TABLE 22.1: SUMMARY OF ALKALINE 3,000 TPD ECONOMIC MODEL

		Initial Capital	**Total Capital**
Capital Expenditures ($000)		Yr -2 to 1	**LOM**
Permitting/bonding		$ 10,000	$ 10,000
Development (preproduction)		$ 5,000	$ 5,000
Mine		$ 26,400	$ 67,757
Mill		$ 74,641	$ 74,641
Tailings		$	$ -
Paste Fill Plant and Equipment		$ 3,948	$ 3,948
Pipe to Impoundment Cells		$ 19	$ 2,429
Tailings Impoundment Cells		$ 15,649	$ 99,161
Contingency at 25%		$ 33,9146	$ 65,734
Initial Capex		$ 169,571	
Total Capex			**$ 328,670**
* Weighted Average	* Primary	* Pillar	**LIFE OF MINE**
Operating Costs, $/lb U_3O_8	Yr 1 to 25	Yr 26 to 36	**TOTALS**
Production (Mlbs U_3O_8)	37.04	17.55	54.59
Underground Mining- Primary	$ 14.53	$ -	$11.42
Underground Mining- Pillars	$ -	$ 17.55	$ 3.75
Processing	$ 9.31	$ 9.26	$ 9.30
Tailings			
Paste to Underground Backfill	$ 0.36	$ 0.42	$ 0.37
Reclamation			
Impoundment Cell Cover and Topsoil	$ 0.09	$ 0.21	$ 0.12
Revegetation	$ 0.01	$ 0.02	$ 0.01
Closure costs	$ 0.18	$ 0.18	$ 0.18

Administration	$ 0.58	$ 0.88	$ 0.65
Contingency at 25%	$ 6.27	$ 7.13	$ 6.45
Simple fee royalties	$ 2.56	$ 2.56	$ 2.56
Property tax	$ 0.21	$ 0.16	$ 0.22
Average Total by Mining Method	*$ 34.11*	*$ 38.37*	
Total Opex			**$ 35.04**

Operating costs can be further broken down as summarized in Table 22.2. Annual expense includes operating costs for mining, milling and reclamation; all estimates for operating expense include a 25% contingency.

TABLE 22.2: ANNUAL OPERATING EXPENSES

Production Period	Annual Expense ($/lb)	25% Contingency ($/lb)	Tax and Royalties ($/lb)	Total Operating ($/lb)
Years 1-10	22.23	5.56	2.94	30.72
Years 11-20	26.57	6.64	2.67	35.88
Years 21-35	28.71	7.18	2.71	38.60
Life of Mine	25.81	6.45	2.78	35.04

The potential overall project economics at various discount rates and uranium contract prices are summarized in Table 22.3.

TABLE 22.3: NPV AND DCFROR MATRIX

	Discount Rate							
Uranium $/lb	**5%**		**7%**		**8%**		**10%**	
	NPV M$	**DCFROR**	**NPV**	**DCFROR**	**NPV**	**DCFROR**	**NPV**	**DCFROR**
$55	$334	19.1%	$246	16.8%	$212	15.7%	$158	13.6%
$64	$561	29.8%	$427	27.4%	$375	26.2%	$293	23.9%
$75	$840	42.3%	$648	39.7%	$574	38.4%	$458	35.9%
$85	$1,093	53.4%	$850	50.5%	$576	49.1%	$608	46.4%

Total labor for both the mining and milling operations is forecast at 224 for the mine and 100 for the milling operations for a total of 324 employees. Of this, it is expected that 218 would be hourly workers and 106 would be staff. It is envisioned during construction that 250 to 350 personnel would be necessary including employees and contractors.

The annual payroll is forecast at $13 million for mining and $6 million for processing such that the total annual payroll would be $19 million. If the 25% contingency is attributed to this cost, the estimate would be $24 million. Salaries are expected to range from $35,000 to $250,000 per annum and hourly rates would range from $20 to $35 per hour. Annual material and supply costs are projected to be about $22 million during the primary mining phase such that total annual material and labor costs would roughly range from $41 to $46 million per year. The direct and indirect economic benefits are on the order of $240 to $300 million.

A sensitivity analysis has been conducted to determine how sensitive the project economics are to changes in operating cost, capital costs, and uranium prices and has been summarized in Table 22.4.

TABLE 22.4: SENSITIVITY ANALYSIS

Financial	Operating Cost		Capital Cost		Price	
Indicator	-10%	+10%	-10%	+10%	-10%	+10%
IRR	40.0%	32.5%	41.2%	32.3%	28.3%	44.0%
DCFROR	30.8%	23.9%	32.0%	23.6%	19.9%	34.6%
NPV $(000)	$496	$358	$450	$404	$298	$556

The project is most sensitive to the uranium price as it can be seen that a 10% reduction in price results in a reduction in the net present value (NPV) from $427M to $298M, while an increase of 10% results in an NPV of $556M. Meanwhile, a 10% reduction in operating cost results in a NPV increase from $427M to $496M, while a 10% increase in operating cost results in a reduction in the NPV TO $358M. The project is least sensitive to capital cost changes; with about a $23M swing in the NPV.

ITEM 23. ADJACENT PROPERTIES

The indications are that there are no material mining properties adjacent to the Coles Hill Project.

ITEM 24. OTHER RELEVANT DATA AND INFORMATION

In 1985, a subcommittee of the Virginia Coal and Energy Commission, the Uranium Administrative Group (UAG) made the following recommendation: "Based on all these efforts, we can now conclude that the moratorium on uranium development can be lifted if essential specific recommendations derived from the work of the task force are enacted into law." Sixteen members of the UAG supported the recommendation with two dissents. The moratorium was not lifted because specific legislation was not introduced due to the drop in uranium prices that eliminated the economic viability of any venture trying to mine uranium in Virginia. Leased mineral rights were eventually returned to the land owners.

In September 2007, the executive branch of the state government published the Virginia Energy Plan, which provided a comprehensive analysis for how the state might become more energy independent. The report highlighted that approximately 35% of electricity generation in Virginia comes from nuclear power plants, while all the nuclear fuel (uranium) is currently imported into the state. Due to the presence of substantial uranium resources in Southside Virginia the report recommended that serious consideration be given to the development of a local uranium mining initiative. The following direct quotes from the Virginia Energy Plan refer to the Coles Hill uranium deposit in Pittsylvania County:

> "*There are sufficient resources to support a uranium mining industry in Pittsylvania County with enough to meet the fuel needs of Virginia's current generation... Virginia should assess the potential value of and regulatory needs for uranium production in Pittsylvania County*."

During the 2008 General Assembly in Virginia, Virginia Uranium, Inc. supported legislation that proposed a study of uranium development in the state that followed the recommendations of the Virginia Energy Plan that was published in September 2007. While the Virginia Senate approved a uranium study bill on February 12, 2008 by a vote of 36 in favor and 4 opposed, the legislation was subsequently tabled during a hearing of the Rules Committee in the Virginia House of Delegates. Thus, the uranium study bill was not approved. In November 2008, the Virginia Coal and Energy Commission created a sub-committee to evaluate uranium mining. The sub-committee engaged the National Academy of Sciences (NAS) to undertake a study with an anticipated duration of 18 months. The Virginia Coal and Energy Commission will make suggestions to the legislation based on the outcome of the NAS study. Since Virginia has a bicameral legislature, bills must be approved by both the House and the Senate. The Governor must then sign the bill for it to become enacted into law.

To balance the picture, in addition to a relying on the generation of electricity from reactors sited at Surry and North Anna, Virginia has a strong history in other phases of the nuclear fuel industry. Norfolk is home to the U. S. Navy's, nuclear powered aircraft carriers and submarines. Portsmouth is home to the Northrop Grumman's shipyard that recently built and commissioned the nuclear aircraft carrier, the USS George H. W. Bush. On October 23, 2008, Northrop Grumman and AREVA announced that they have joined forces to build a new manufacturing and engineering facility in Newport News to establish a world-class facility to manufacture heavy components for the U.S. Evolutionary Power Reactor (EPR), AREVA's Generation III + nuclear reactor. Lynchburg is home to AREVA and Babcock & Wilcox employing 5,000 people. Nor is mining new to Virginia which, ranked nationally, is tenth in the production of coal and fifth in the production of crushed stone.

Mine permitting is an issue that has yet to be finalized in Virgina. Virginia has an existing hard rock mine permitting process in place, and VA Governor Bob McDonnell has directed a Uranium Working Group to develop a "regulatory framework" for uranium mining due in report by December 2012. Legislation will likely be introduced to VA General Assembly in 2013 to lift the uranium moratorium and enable preparation of uranium-specific mining rules, expected to be an addition to Reclamation

Regulations for Mineral Mining (4 VAC 25 – 31) . The mine permit is expected to require financial assurance for the cost of mine closure by a third party. No mine permit application will be submitted until after these rules have been implemented.

Mill licensing – Initiation of mill licensing process through the U.S. Nuclear Regulatory Commission (US NRC) will begin after the uranium moratorium has been lifted, probably in 2013. Licensing will be in accordance with existing US NRC and US EPA regulations for the mill life cycle. The license will be a Source Material License as required by Chapter 10 Code of Federal Regulations Part 40. The mill license is expected to require financial assurance for the cost of mill closure by a third party.

ITEM 25. INTERPRETATION AND CONCLUSIONS

The mine and mineral processing development alternatives presented herein demonstrate a potential for economically viable mineral resources, based on the cost and price estimates as discussed in this report. It must be noted that this evaluation is based upon mineral resources and not mineral reserves and mineral resources that are not mineral reserves do not have demonstrated economic viability. The preferred alternative for the development of the Coles Hill Uranium Project includes a Sub Level Open Stope (SLOS) underground conventional mine operation with on-site mineral processing via a conventional, alkaline mill. Surface mine alternatives were also evaluated and appear to have merit especially in light of the desire for subsurface tailings disposal.

The technical risks related to the project are low as the mining and recovery methods are proven. The mining methods recommended have been employed successfully at similar projects in the past. The mineral processing methods employed are typical of those used in the industry for decades and are supported by metallurgical tests done to date.

Risks related to permitting are relative to primarily having the moratorium rescinded to allow mining in Virginia and gaining the confidence of the local community that the mining and milling can be safely conducted. The remainder of the permitting issues is tied to obtaining the necessary permits to operate the mine and mill.

The authors are not aware of any other specific risks or uncertainties that might significantly affect the mineral resource estimates or the consequent economic analysis.

Estimation of costs and uranium price for the purposes of the economic analysis over the life of mine is by its nature forward-looking and subject to various risks and uncertainties. No forward-looking statement can be guaranteed and actual future results may vary materially.

The following conclusions have been made as a result of this study:

- The continuity of mineralization through to the surface in both the north and south deposits could support either open pit or underground mining, however underground mining is recommended (open pit is not discounted);
- Underground mining can be performed by sub-level open stoping (SLOS), a historically productive and a safe mining method;
- Surface mine options should be evaluated in light of the desire for subsurface tailings disposal and as a means of improving project economics and accelerating mine production;
- Additional drilling and specific data collection is recommended under Item 26 to better define mineral resource and increase the accuracy and reliability of the mine design and cost estimates herein;
- While acid leaching is expected to produce a higher uranium recovery, alkaline leaching is the more cost effective option;
- There is inadequate surface are for the tailings facilities, additional surface area and /or consideration of sub-grade disposal in combination with open pit mining is necessary;
- The overall conceptual economics are favorable for the Coles Hill project. The project shows an IRR of 36.3%; at a discount rate of 7% the net present value (NPV) is $427 million.
- The life of the mine is 35 years.
- Including 25% contingency, the initial capital investment prior to production is $147 million, however, the total capital through the 4th year of production is $204 million, while the total capital cost is $329 million.
- The initial annual revenue ranges from $95 to $144 million.

- The direct and indirect economic benefits are on the order of $240 to $300 million.
- Total labor for both the mining and milling operations is forecast at 224 for the mine and 100 for the milling operations for a total of 324 employees. Of this, it is expected that 218 would be hourly workers and 106 would be staff.
- The annual payroll is forecast at $13 million for mining and $6 million for processing such that the total annual payroll would be $19 million. If the 25% contingency is attributed to this cost, the estimate would be $24 million.
- Salaries are expected to range from $35,000 to $250,000 per annum and hourly rates would range from $20 to $35 per hour.
- Annual material and supply costs are projected to be about $22 million during the primary mining phase such that total annual material and labor costs would roughly range from $41 to $46 million per year.
- It is envisioned during construction that 250 to 350 personnel would be necessary including employees and contractors.

ITEM 26. RECOMMENDATIONS

Mineral Resource Related Recommendations

The extent of mineralization is not fully defined by current drilling. While additional drilling may or may not expand mineral resources, it is the author's interpretation and opinion that mineralization extends beyond the currently defined limits.

Mine Related Recommendations

- Detailed mine planning, both underground and surface, is recommended to optimize mine recovery and economics. These design efforts should also consider mine closure and reclamation requirements including provisions for subsurface tailings disposal. Budgetary estimate $500,000 US.
- Placement of tailings as paste backfill is contemplated in the current plan. Specific testing relative to admixtures is recommended. Testing should include geotechnical considerations relating to compressive strength, density and or engineering properties. In addition, admixtures which are suitable from a geotechnical perspective should be tested for long term leaching characteristics. Specifically ASTM method C 1308, "Accelerated Leach Test for Diffusive Releases from Solidified Waste and a Computer Program to Model Diffusive Fractional Leaching from Cylindrical Waste Forms" is recommended. Fate transport of any constituents of concern from this testing should be evaluated. Budgetary estimate $100,000 US.
- The mining methods being considered are of a bulk nature and opportunities for selective mining are limited. Testing of methodologies such as radiometric ore sorting in the mining process is recommended to reject waste within the mine and improve run-of-mine grades. This testing should be phased with the initial testing by hand sorting and proof testing at a bench scale. Budgetary estimate for initial testing $50,000 US. Budgetary estimate for bench scale testing $200,000 US exclusive of sample collection costs.
- Core drilling is recommended at both North and South Coles to provide additional geologic, geotechnical, and hydrologic data, as well as representative samples for metallurgical testing and bench scale radiometric ore sorting. Single drill holes could be designed to provide data and samples for multiple purposes. This work could be phased and include geotechnical information acquisition and hydrologic data acquisition and modeling from about 15 sites. Budgetary estimate for this work depending on final scope the cost of contracted services and sample needs would be about $2,000,000 US.

Mineral Processing and Metallurgical Recommendations

- Allow access for the collection of a bulk metallurgical sample. This bulk sample would be tested at an off-site licensed facility to:
    - Determine the Bond Work Index (kWh/t) variability;
    - Determine the Work Index for semi-autogenous (SAG) mills;
    - Optimize leach conditions;
    - Evaluate the viability of processing paste tailings; and
    - Evaluate alternatives for tailings disposal.

  It is further recommended that test-work be conducted to determine the Bond Work Index (kWh/t) variability throughout the North and South ore bodies. The Work Index for SAG mills should also be determined; Cost Estimate $125,000. The detail of the costs is shown in Table 26-1.
- The tailings facilities have been designed for an in-place S.G. of 1.3, however further test-work is required to validate this. It is recommended that the tailings produced from alkaline leaching are

tested for physical properties such as bulk density and % solids post-filtration. Additionally, the option of paste processing all tailings (i.e. surface tailings and underground backfill) should be explored. Using paste tailings in the surface tailings impoundments is beneficial as it limits the infiltration of outside water and the remobilization of the tailings and potentially reduces the size, and therefore cost, of the impoundments themselves.

- Conduct further investigations into tailings disposal concepts to assess opportunities and optimize risk mitigation while assessing additional properties for tailings disposal. Cost estimate for leach optimization, paste study and filtering and settling tests is $250,000. The detail of the costs is shown in Table 26-1.

Table 26.1: Recommended Studies Cost Estimate

Comprehensive Grinding Studies and Evaluation

Test	Samples	$/Sample	Total $
Composite Sample Preparation	15	$ 750	**$ 11,250**
Head Analyses	15	$ 400	**$ 6,000**
Grind Study	15	$ 1,000	**$ 15,000**
SAG Mill Work Index	15	$ 1,000	**$ 15,000**
Bond Ball Mill Work Index	15	$ 800	**$ 12,000**
Thickening Tests	15	$ 2,000	**$ 30,000**
Subtotal			**$ 89,250**
Engineering Support, Report Writing, and Lab Oversight			**$ 20,000**
Sample Collection and Transport			**$ 3,000**
Subtotal			**$ 23,000**
Subtotal			**$ 112,250**
Contingency	10%		**$ 11,225**
Subtotal			**$ 123,475**
Total		estimate	**$ 125,000**

Tailings Disposal Investigation

Test	Samples	$/Sample	Total $
Composite Sample Preparation	3	$ 750	**$ 2,250**
Thickening Tests	3	$ 2,000	**$ 6,000**
Bulk Density Determination	12	$ 25	**$ 300**
Filtration Tests	3	$ 2,000	**$ 6,000**
Paste Tails Tests	7	$ 5,000	**$ 35,000**
Alternate Site Evaluation	1	$ 40,000	**$ 40,000**
Cost and Risk Analysis and Mitigation	1	$ 35,000	**$ 35,000**
Paste Tails Concrete/Fly Ash Tests	10	$ 5,000	**$ 50,000**
Meteoric Water Mobility Tests	7	$ 3,000	**$ 21,000**
Structural Engineering Tests	5	$ 3,000	**$ 15,000**

Engineering Analysis and Reporting	1	$ 15,000	$	15,000
Subtotal			**$**	**225,550**
Contingency	10%		**$**	**22,555**
Subtotal			**$**	**248,105**
Total		estimate	**$**	**250,000**

This work needs to be conducted in two Phases as Phase II represents optimization work that is of value to take the project to the next feasibility study level, while Phase I work will address more pressing work necessary for the current level of design. Phase I work would include the work to investigate the tailings storage options that will be of value in assisting to evaluate mining and processing designs for the later stage work. In Phase I, it will also be of value to determine the characteristics of the underground tailings admixtures for leaching and structural characteristics, and evaluate sorting techniques. This work would allow modification of or further mining and processing optimization potential for the next level of feasibility study. Phase I work would consist of $550,000 and Phase II work would consist of $2,625,000 in costs as shown in Table 26-2 for a total of $3,175,000. At the end of Phase I, a decision as to the preferred tailings placement and potential ore sorting concepts would be generated such that the following Phase II and subsequent feasibility study could be modified to incorporate the results of the Phase I studies. The impact of the proposed cost of new tailings management concept would be evaluated to assess overall project economics. A decision would be made at the end of Phase I as to the overall design of the tailings management plan and the employment of any sorting technologies, which would be incorporated into Phase II work plans and the subsequent feasibility study. Decisions on data from the Phase II studies would be incorporated into the feasibility study for overall project evaluation.

Table 26-2
Work Phase I and Work Phase II
Recommended Studies Cost

Phase	Study	Cost $	
I	Paste Backfill Studies	$	100,000
I	Ore Hand Sorting Analysis	$	50,000
I	Radiometric Sorting Analysis	$	200,000
I	Tailings Storage Study	$	200,000
Subtotal		**$**	**550,000**
II	Detailed Mine Planning	$	500,000
II	Core Drilling Expansion	$	2,000,000
II	Grinding Analysis	$	125,000
Subtotal		**$**	**2,625,000**
Total		**$**	**3,175,000**

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ITEM 28. DATE AND SIGNATURE

CERTIFICATE OF AUTHOR

John I. Kyle, P.E.

I, John I. Kyle, am a Mining Engineer Registered (#15882) in Colorado, USA, do hereby certify that:

1) I am a Vice President of Lyntek, Inc. with offices at 1550 Dover Street, Lakewood, CO 80215;

2) I hold a B.Sc. Mining (1974) from the Colorado School of Mines in Golden, CO, USA and a Masters in Business Administration from Denver University in Denver, CO, USA (1986);

3) I am a Professional Engineer Registered (#15882) in Colorado, USA since 1978 and am a member of the Society of Mining Engineers in the US and Canadian Institute of Mining in Canada;

4) I have been practicing my profession as an Engineer for over 37 years and have been employed as a Consulting Engineer since 1988. I have worked on over 3 dozen uranium projects on a global scale.

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

6) I am responsible for sections 1-5, 13, and 17-27 of the report entitled "Preliminary Economic Assessment – Update Coles Hill Uranium Property Pittsylvania County, Virginia" dated September 6, 2012 (the Technical Report) relating to Virginia Energy Resources, Inc.'s Coles Hill Uranium Project. I visited the project site most recently onSeptember 20 and 21, 2010 and did not have any involvement with the property prior to requests by Virginia Energy Resources to work upon the project other than the preparation of the precursor report: Preliminary Economic Assessment – Coles Hill Property – December, 2010.

7) I am independent of Virginia Energy Resources, Inc., Virginia Uranium Inc., and Virginia Uranium Holdings, Inc. applying all of the tests in Section1-5 of National Instrument 43-101.I am independent of the Coles Hill Property and the property vendor as required under Section 3.2 of Appendix 3F *Mining Standards* of the TSX Venture Exchange Corporate Finance Manual.

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

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

10) As of the date of my certificate, to the best of my knowledge, information and belief, the Sections 1-5, 13, and 17-27 contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 6th day of September 2012.

("Signed") ("Sealed")

John I. Kyle, P.E.

CERTIFICATE OF AUTHOR

DOUGLAS L. BEAHM

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

1. I am responsible for sections 6-12 and14-16 and a contributor with respect to sections 3, 21, and 25-27. of the report entitled "Preliminary Economic Assessment – Update Coles Hill Uranium Property Pittsylvania County, Virginia" dated September 6, 2012 (the Technical Report) relating to Virginia Energy Resources, Inc.'s Coles Hill Uranium Project.
2. I am the Principal Engineer and President of BRS, Inc., 1130 Major Avenue, Riverton, Wyoming 82501.
3. I graduated with a Bachelor of Science degree in Geological Engineering from the Colorado School of Mines in 1974.
4. I am a licensed Professional Engineer in Wyoming, Colorado, Utah, and Oregon; a licensed Professional Geologist in Wyoming; and Registered Member of the Society for Mining, Metallurgy and Exploration, Inc. ("SME")
5. I have worked as an engineer and a geologist for over 37 years. My work experience includes uranium exploration, mine production, and mine/mill decommissioning.
6. I have limited prior working experience on the project through a former employer.
7. My recent visits to the site include May 18 and 19, 2010, August 20, 2010, and April 4 through 6, 2011. On these occasions I reviewed the original geologic data for the project including previous investigations, geophysical and lithologic logs data, chemical assay records, and physically inspected the site and the available core from the project.
8. I am independent of Virginia Energy Resources, Inc., Virginia Uranium Inc., and Virginia Uranium Holdings, Inc. applying all of the tests in Section1-5 of National Instrument 43-101.I am independent of the Coles Hill Property and the property vendor as required under Section 3.2 of Appendix 3F Mining Standards of the TSX Venture Exchange Corporate Finance Manual.
9. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with same.
10. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.
11. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

September 6th, 2012
Signed and Sealed

Douglas L. Beahm, PE, PG
Principal Engineer, BRS Inc.