Macusani Project

Macusani, Peru

NI 43-101 Report - Preliminary Economic Assessment

Prepared For:

Plateau Uranium Inc.

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IMPORTANT NOTICE:

This report was prepared as a National Instrument 43-101 Technical Report, in accordance with Form 43-101F1, for Plateau Uranium Inc. by GBM Minerals Engineering Consultants Limited. The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in GBM's services, based on: i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended to be filed as a Technical Report with Canadian securities regulatory authorities pursuant to National Instrument 43-101, Standards of Disclosure for Mineral Projects. Except for the purposes legislated under provincial securities law, any other use of this report by any third party is at that party's sole risk.

CERTIFICATE OF QUALIFIED PERSON

MICHAEL JOHN SHORT

As the individual who has co-authored or supervised the preparation of sections of the Technical Report prepared for Plateau Uranium Inc. (the "Issuer") entitled "NI 43-101 Report Preliminary Economic Assessment" dated effective 12 January 2016 (the "Technical Report"), I hereby certify that:

1. I am a Chartered Engineer and the Managing Director of GBM Minerals Engineering Consultants Limited ("GBM") of Regal House 70 London Road, Twickenham, Middlesex, TW1 3QS, England.
2. I graduated with a Bachelor of Engineering in Civil Engineering from the University of New South Wales, Australia in 1975. I am a Chartered Engineer (CEng) registered with the Engineering Council UK and a Fellow of the Institute of Materials, Minerals and Mining (FIMMM). I am a Chartered Professional (CP) and a Fellow of the Australasian Institute of Mining and Metallurgy (FAusIMM). I am a Chartered Professional Engineer (CPEng) and Fellow of Engineers Australia (FIEAust). I have been working in the engineering profession continuously since 1975 and have had experience in metallurgy, process design and engineering, plant operations and management.
3. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 ("NI-43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for the purposes of NI 43-101.
4. I have personally visited the Macusani Yellowcake Project site in October 2009 and January 2011. Additionally GBM professionals visited the site in November 2013.
5. I have authored or supervised the work carried out by other GBM professionals for GBM's contribution to the Technical Report, and take responsibility for Sections 1, 2, 3, 4, 5, 6, 18, 19, 20, 21, 22, 24, 25, 26 and 27.
6. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.
7. I, and GBM, have had prior involvement with the project on behalf of Macusani Yellowcake Inc. as described in the following report:
    - Technical Report entitled "Preliminary Economic Assessment for the Colibri II and III Properties" dated 2010, No. 0387-PR-001.
    - Technical Report entitled "Preliminary Economic Assessment" dated 2014, No. 0501-RPT-001

8. I have read NI 43-101, and the sections of the Technical Report for which I am responsible, as stated above, have been prepared in compliance with NI 43-101 and Form 43-101F1.

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

Dated this 12 January 2016, London, United Kingdom.

Michael John Short, BE (Civil Eng), CEng FIMMM, FAusIMM(CP), FIEAust CPEng

Qualified Person

CERTIFICATE OF QUALIFIED PERSON

THOMAS APELT

As the individual who has co-authored or supervised the preparation of sections of the Technical Report prepared for Plateau Uranium Inc. (the "Issuer") entitled "NI 43-101 Report - Preliminary Economic Assessment" dated effective 12 January 2016 (the "Technical Report"), I hereby certify that:

1. I am a Process Engineer with GBM Minerals Engineering Consultants Limited ("GBM") of Regal House 70 London Road, Twickenham, Middlesex, TW1 3QS, England.

2. I graduated with a Bachelor of Engineering in Chemical Engineering from the University of Queensland, Australia in 1992. I graduated with a Doctor of Philosophy (Chemical Engineering) from the University of Sydney, Australia in 2007. I am a Chartered Engineer (CEng) registered with the Engineering Council UK and am a Member of the Institute of Chemical Engineers (MIChemE). I am a Chartered Professional – Metallurgy (CPMet) and a Member of the Australasian Institute of Mining and Metallurgy (MAusIMM). I have been engaged in the engineering profession since 1993 and have had experience in metallurgy and process design and optimisation, including experience in heap leach design.

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

4. GBM professionals have visited the Macusani Yellowcake Project site in October 2009, January 2011 and November 2013.

5. I have authored or supervised the work carried out by other professionals for GBM's contribution to the Technical Report, and take responsibility for Sections 13 and 17.

6. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.

7. I, and GBM, have had prior involvement with the project as described in the following reports:

    - Technical Report entitled "Preliminary Economic Assessment for the Colibri II and III Properties" dated 2010, No. 0387-PR-001.
    - Technical Report entitled "Preliminary Economic Assessment" dated 2014, No. 0501-RPT-001

8. I have read NI 43-101, and the sections of the Technical Report for which I am responsible, as stated above, have been prepared in compliance with NI 43-101 and Form 43-101F1.

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

Dated this 12 January 2016, London, United Kingdom.

Thomas Apelt

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Thomas Apelt, BEng, PhD (Chem Eng), CEng MIChemE, MAusIMM(CP)

Qualified Person

CERTIFICATE OF QUALIFIED PERSON

DAVID ROBERT YOUNG

As the individual who has co-authored or supervised the preparation of sections of the Technical Report prepared for Plateau Uranium Inc. (the "Issuer") entitled "NI 43-101 Report - Preliminary Economic Assessment" dated effective 12 January 2016 (the "Technical Report"), I hereby certify that:

1. I am an independent Professional Geologist and have contracted my services to The Mineral Corporation ("TMC") of Block B, Homestead Office Park, 65 Homestead Avenue, Bryanston, Johannesburg, South Africa.

2. I graduated with a Bachelor of Science Honours in Geology from the London University, Chelsea College, United Kingdom in 1974. I am a practising geologist registered with the South African Council for Natural Scientific Professions (No. 400989/83), a Fellow of the South African Institute of Mining and Metallurgy (FSAIMM), a Fellow of the Australasian Institute of Mining and Metallurgy (FAusIMM), and a Fellow the Geological Society of South Africa (FGSSA). I have practiced geology continuously since 1974 and have had experience reporting uranium mineral resources to the South African Nuclear Fuels Regulators, the mapping of acid volcanics and estimation of Mineral Resources for three dimensional ore bodies.

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

4. I have personally visited the Macusani Yellowcake Project site in August 2008, July 2009, January 2011 and May 2013 including undertaking data verification activities.

5. I have authored or supervised the work carried out by other professionals for TMC's contribution to the Technical Report, and take responsibility for Sections 6, 7, 8, 9, 10, 11, 12, 14 and 23.

6. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.

7. I, and The Mineral Corporation have had prior involvement with the project as described in the following reports:

    - Technical Report entitled "Mineral Resource Estimates of the Colibri Project held by Global Gold S.A.C. in the Puno District of Peru" dated December 2008, No. C-MYI-COL-731-506.

- Technical Report entitled "Update to Mineral Resource Estimates of the Colibri Project held by Global Gold S.A.C. in the Puno District of Peru" dated April 2010, No. C-MYI-COL-731-592.
- Technical Report entitled "Update to Mineral Resource Estimates of the Colibri Project held by Global Gold S.A.C. in the Puno District of Peru" dated September 2010, No. C-MYI-COL-731-637.
- Technical Report entitled "Mineral Resource Estimates of the Corachapi Project held by Global Gold S.A.C. in the Puno District of Peru" dated October 2010, No. C-MYI-CON-881-644.
- Technical Report entitled "Mineral Resource Estimates of the Chilcuno Chico deposit held by Global Gold S.A.C. in the Puno District of Peru" dated August 2012, No. C-MYI-CHI-1170-802.
- Technical Report entitled "Mineral Resource Estimates for the Colibri 2 & 3 / Tupuramani, Kihitian and Triunfador Uranium Projects, held by Global Gold S.A.C. in the Puno District of Peru" dated 20 September 2013, No. C-MYI-COL-731-872.
- Technical Report entitled "Consolidated Mineral Resource estimates for the Kihitian, Isivilla and Corani Uranium Complexes controlled by Plateau Uranium Inc., in the Puno District of Peru" dated June 2015, No. C-MYI-MRU-1568-960.
- Technical Report entitled "Preliminary Economic Assessment" dated 2014, No. 0501-RPT-001.

8. I have read NI 43-101, and the sections of the Technical Report for which I am responsible, as stated above, have been prepared in compliance with NI 43-101 and Form 43-101F1.

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

Dated this 12 January 2016, Hout Bay, South Africa.

D.R. Young

David Robert Young, BSc (Hons), FGSSA, FSAIMM, FAusIMM, Pr Sci Nat

Qualified Person

CERTIFICATE OF QUALIFIED PERSON

MARK MOUNDE

As the individual who has co-authored or supervised the preparation of sections of the Technical Report prepared for Plateau Uranium Inc. (the "Issuer") entitled "NI 43-101 Report Preliminary Economic Assessment" dated effective 12 January 2016 (the "Technical Report"), I hereby certify that:

1. I am a Chartered Mining Engineer and the Technical Director with Wardell Armstrong International Limited ("WAI") of Wheal Jane, Baldhu, Truro, Cornwall, TR3 6EH, England.
2. I graduated with a Bachelor of Engineering degree in Mining Engineering from the University of Exeter, United Kingdom in 1993. I am a Chartered Engineer (C.Eng.) registered with the Engineering Council UK (Registration number 569902) and a Member of the Institute of Materials, Minerals and Mining (MIMMM). I have been working in the engineering profession continuously since 1993 and have had experience in mine design, operations, planning and management.
3. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 ("NI-43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for the purposes of NI 43-101.
4. I have personally visited the Macusani Yellowcake Project site in November 2013.
5. I have authored or supervised the work carried out by other professionals for WAI's contribution to the Technical Report, and take responsibility for Sections 15, 16 and 21.
6. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.
7. WAI has had prior involvement with the project on behalf of Macusani Yellowcake Inc. as described in the following report:
    - Technical Report entitled "Preliminary Economic Assessment" dated 2014, No. 0501-RPT-001
8. I have read NI 43-101, and the sections of the Technical Report for which I am responsible, as stated above, have been prepared in compliance with NI 43-101 and Form 43-101F1.
9. To the best of my knowledge, information, and belief, as of the date of this certificate, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 12 January 2016, Truro, United Kingdom.

M. R.S. Mounde

Mark Mounde, B.Eng., CEng MIMMM

Qualified Person

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STANDARD TERMS AND NOMENCLATURE

The following abbreviations are used throughout this PEA:

Abbreviation	Description
AACE	AACE International (previously known as American Association of Cost Engineering and Association for the Advancement of Cost Engineering).
ANSI	American National Standards Institute
Azincourt	Azincourt Uranium Inc.
BLS	Barren Leach Solution
BRT	Bottle Roll Test
Cameco	Cameco Corporation
CAPEX	Capital Expenditure
CIMM	CIMM Perú S.A.
CIT	Corporation Income Tax
Contact Uranium	Contact Uranium Peru S.A.C.
CWENGA	Chemical and Water Engineering Associates Pty Ltd
DCF	Discounted Cash Flow
EIA	Environmental Impact Assessment (Estudio de Impacto Ambiental)
EMP	Environmental Management Plan (Programa de Adecuacion y Manejo Ambiental)
EPC	Engineering Procurement Construction
EPCM	Engineering Procurement Construction Management
FoS	Factor of Safety
GBM	GBM Minerals Engineering Consultants Limited
Global Gold	Global Gold S.A.C.
GWe	Gigawatt electrical
HDPE	High Density Polyethylene
HEU	Highly Enriched Uranium
HV	High Voltage
IAEA	International Atomic Energy Agency
ICP-MS	Inductively Coupled Plasma Mass Spectrometry
ICP-OES	Inductively Coupled Plasma Optical Emission Spectrometry
ILS	Intermediate Leach Solution
IPEN	Instituto Peruano de Energia Nuclear
IRR	Internal Rate of Return
IUREP	International Uranium Resources Evaluation Project Mission
IX	Ion Exchange
KE	Kriging Efficiency
LOM	Life of Mine
LV	Low Voltage

Abbreviation	Description
Ma	Million years before present
MCC	Motor Control Centre
MEM	Ministry of Energy and Mines (Ministerio de Energía y Minas República del Perú)
MIK	Multiple Indicator Kriging
Minergia	Minergia S.A.C.
MRMR	Modified Rock Mass Rating
MV	Medium Voltage
n /a	Not applicable
NATO	North Atlantic Treaty Organisation
NPSH	Net Positive Suction Head
NPS	Nominal Pipe Size
NPV	Net Present Value
NPVS	NPV Scheduler^{TM}
Osinergmin	Organismo Supervisor de la Inversión en Energía y Minería
OPEX	Operating Expenditure
PEA	Preliminary Economic Assessment
PEM	Potentially Economic Material
PFS	Pre-Feasibility Study
PLC	Programmable Logic Controller
PLS	Pregnant Leach Solution
Plateau Uranium	Plateau Uranium Inc. (formerly Macusani Yellowcake Inc.)
PSE	Process Systems Enterprise
PMC	Project Management Contractor
RMR	Rock Mass Rating
ROM	Run of Mine
SMC	SAG Mill Comminution
SX	Solvent Extraction
TMC	The Mineral Corporation
UCS	Ultimate Compressive Strength
UTS	Ultimate Tensile Strength
UNDP	United Nations Development Programme
USD	United States Dollar
Vena	Vena Resources Inc.
VSD	Variable Speed Drive
WAI	Wardell Armstrong International Limited

The following terms and definitions are used throughout this PEA:

Term	Definition
Bulk density	A macroscopic density of a material and includes a weighted average of the particle apparent density and the void occupying fluid.
Colibri Complex	The Colibri Complex includes the Colibri II, Colibri III and Tupuramani mining concessions, which together contain deposits referred to as the Colibri II &III and Tupuramani deposits. Collectively these will be referred to as Complex 2 or the Colibri Complex throughout this report.
Corachapi Complex	The Corachapi Complex includes the Corachapi, Taititira and Taypicorani mining concessions, which together contain a combined deposit referred to as the Corachapi deposit. Collectively these will be referred to as Complex 1 or the Corachapi Complex throughout this report.
Corani Complex	The Corani Complex includes the Lincoln XXIX, Lincoln XXX, Calvario II and Calvario III mining concessions, which together contain deposits referred to as the Calvario II, Colvario III and Nueva Corani deposits. Collectively these will be referred to as Complex 5 or the Corani Complex throughout this report.
Effective Date (1)	The date of the most recent scientific or technical information included in the technical report.
Inferred Mineral Resources (2)	An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes.
Indicated Mineral Resources (2)	An Indicated Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics can be estimated with sufficient confidence to allow the application of modifying factors (such as mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social and government factors) in sufficient detail to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation.
Interoceanica Highway	The Interoceanica Highway is a system of sealed roads that link the ports of San Juan and Ilo on the western coast of Peru, over the Andes, to the western side of Brazil.
Isivilla Complex	The Isivilla Complex includes the Lincoln XXVI and Triunfador I mining concessions, which together contain deposits referred to as the Calvario I, Puncopata, Calvario Real and Isivilla deposits. Collectively these will be referred to as Complex 4 or the Isivilla Complex throughout this report.
ISO tank	A tank container built of stainless steel surrounded by an insulation and protective layer of usually polyurethane and aluminium. The vessel is in the middle of a steel frame. The frame is 6.05 m long, 2.40 m wide and 2.40 m or 2.55 m high.
Kihitian Complex	The Kihitian Complex includes the Kihitian, Lincoln XXVII and Tantamaco 3 mining concessions, which together contain deposits referred to as the Chilcuno Chico, Tuturumani, Tantamaco and Quebrada Blanca deposits. Collectively these will be referred to as Complex 3 or the Kihitian Complex throughout this report.
Measured Mineral Resources (2)	A Measured Mineral Resource is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of modifying factors (such as mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social and government factors) to support detailed mine planning and final evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation.

Term	Definition
Meta-autunite and Autunite	Meta-autunite is a dehydration product of its close cousin autunite. When the mineral autunite loses water it converts to meta-autunite. The project is based on deposits containing these minerals.
Mineral Reserves (2)	A Mineral Reserve is the economically mineable part of a Measured and/or Indicated Mineral Resource. It includes diluting materials and allowances for losses, which may occur when the material is mined or extracted and is defined by studies at Pre-Feasibility or Feasibility level as appropriate that include application of modifying factors (such as mining, processing, metallurgical, infrastructure, economic, marketing, legal, environmental, social and government factors). Such studies demonstrate that, at the time of reporting, extraction could reasonably be justified.
Mineral Resources (2)	A Mineral Resource is a concentration or occurrence of diamonds, natural solid inorganic material, or natural solid fossilised organic material including base or precious metals, coal and industrial minerals in or on the Earth's crust in such form, grade or quality and quantity that there arereasonable prospects for eventual economic extraction. The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling.
Pre-Feasibility Study (2)	A Preliminary Feasibility Study is a comprehensive study of a range of options for the technical and economic viability of a mineral project that has advanced to a stage where a preferred mining method, in the case of underground mining, or the pit configuration, in the case of an open pit, is established and an effective method of mineral processing is determined. It includes a financial analysis based on reasonable assumptions on mining, processing, metallurgical, economic, marketing, legal, environmental, social and governmental considerations and the evaluation of any other relevant factors which are sufficient for a Qualified Person, acting reasonably, to determine if all or part of the Mineral Resource may be classified as a Mineral Reserve.
Qualified Person (1)	A Qualified Person means an individual who is an engineer or geoscientist with at least five years of experience in mineral exploration, mine development or operation or mineral project assessment, or any combination of these; has experience relevant to the subject matter of the mineral project and the Technical Report; and is a member or licensee in good standing of an approved professional association.
Raffinate	The liquid left after a solute has been extracted by solvent extraction.
Sayaña Complex	The Sayaña Complex includes the Samilio I, Triunfador 4 and Triunfador 5 mining concessions, which together contain the Agaton, Sayaña West and Sayaña Central deposits. Collectively these will be referred to as Complex 6 or the Sayaña Complex throughout this report.
Sulfur, Sulfide, Sulfuric, Sulfate	Note the spelling adopted for this project.
the project	The project comprises the mining and processing of yellowcake on the Macusani Plateau area with mines being located within the Colibri Complex, Kihitian Complex and Isivilla Complex and processing via a dynamic heap leach being completed on the plateau adjacent to the Colibri Complex.
Yellowcake	U_3O_8

SECTION 1 SUMMARY

1.1 SUMMARY

Plateau Uranium Inc. (Plateau Uranium) subsidiary companies hold various uranium mineral rights on the Macusani Plateau in Peru. Plateau Uranium predecessor companies have been dedicated to the exploration and extraction of uranium in Peru since 2007. The main properties considered in this report are the Colibri, Kihitian and Isivilla Complexes which are located in south eastern Peru approximately 650 km south east of Lima and 20 km northwest of the nearby town of Macusani. The project proposes to exploit uranium from these properties via open pit and underground mining methods, and process the Potentially Economic Material (PEM) via an acid heap leach process with ion exchange (IX) recovery.

The nominated base case calls for the PEM resource of 109 Mt with an average grade of 289 ppm U_3O_8 to be mined over 10 years at 10.9 Mt/a with average annual production of 6.08 M lb U_3O_8. Preliminary results indicate that USD 50 / lb U_3O_8 and 8 % discount rate will yield a post-tax NPV of USD 603.1 and IRR of 41 %. This project is estimated to require an initial capital investment of USD 299.8 M and sustaining capital of USD 43.9 M.

Consideration of the project’s feasibility is outside the scope of this report as Inferred Mineral Resources are included within the PEM estimate which has been used as the basis of this PEA.

1.2 GEOLOGY, MINERALISATION AND EXPLORATION

The Andes represents a large anticlinorium complicated by a series of faults and intrusions and the Late Tertiary and Quaternary rejuvenation by block faulting of an eroded early Tertiary folded mountain range. Topographically the mountains consist of a central dissected plateau (the Altiplano), enclosed by narrow ranges. The Macusani Plateau is located in the Altiplano area.

Uranium mineralisation is found in acidic volcanic pyroclastic rocks of rhyolite composition that cover large areas of the Macusani Plateau which are preserved in a NW-SE trending graben within the Andes. These pyroclastic rocks are dated between 10.0 and 6.7 Ma. Uranium mineralisation is found concentrated along steeply dipping fractures but constrained within horizontal or sub-horizontal zones and is disseminated into the surrounding host rock.

Exploration work completed by the Instituto Peruano de Energia Nuclear as well as the United Nation Development Programme / International Atomic Energy Agency in the mid to late 1970s identified this part of Peru to be uraniferous.

1.3 MINERAL RESOURCES

Table 1-1 summarises the Measured and Indicated Mineral Resources for the project, while Table 1-2 summarises the Inferred Mineral Resources.

Table 1-1: Summary Resource Statement - Measured and Indicated Materials

Complex	Deposit	Mineral Resource Category	Metric Units			Imperial Units		
			Tonne (Mt)	Grade (U ppm)	U_3O_8 Content (000s kg)	Ton (Mt)	U_3O_8 Content (Mlb)	U_3O_8 Grade (lb/ton)
Corachapi	Corachapi*	Measured	1.031	120	146	1.136	0.32	0.28
Kihitian	Chilcuno Chico	Indicated	34.840	218	8 972	38.405	19.78	0.52
Kihitian	Quebrada Blanca	Indicated	5.509	279	1 814	6.073	4.00	0.66
Kihitian	Tantamaco	Indicated	7.393	191	1 661	8.150	3.66	0.45
Isivilla	Isivilla	Indicated	4.568	296	1 597	5.035	3.52	0.70
Corani	Nueva Corani	Indicated	3.397	141	565	3.744	1.25	0.33
Corachapi	Corachapi*	Indicated	10.562	171	2 130	11.643	4.70	0.40
Colibri	Colibri II & III**	Indicated	27.885	203	6 675	30.738	14.72	0.48
Total Measured and Indicated			**95.185**	**210**	**23 559**	**104.924**	**51.94**	**0.50**

** Figures based on October 2010 Mineral Resource Estimates by The Mineral Corporation*
*** Figures based on September 2013 Mineral Resource Estimates by The Mineral Corporation*
Minor discrepancies due to rounding may occur
There are currently no known risks that could materially affect potential development
Density 1.98t/m^3
Cut-off 75 U ppm
Ton is short ton

Table 1-2: Summary Resource Statement - Inferred Materials

Complex	Deposit	Mineral Resource Category	Metric Units			Imperial Units		
			Tonne (Mt)	Grade (U ppm)	U_3O_8 Content (000s kg)	Ton (Mt)	U_3O_8 Content (Mlb)	U_3O_8 Grade (lb/ton)
Kihitian	Chilcuno Chico	Inferred	30.995	294	10 751	34.166	23.70	0.69
Kihitian	Quebrada Blanca	Inferred	13.436	269	4 264	14.811	9.40	0.63
Kihitian	Tuturumani	Inferred	3.300	146	569	3.638	1.25	0.34
Kihitian	Tantamaco	Inferred	35.849	172	7 251	39.517	15.98	0.40
Isivilla	Isivilla	Inferred	7.396	295	2 573	8.153	5.67	0.70
Isivilla	Puncopata	Inferred	5.923	216	1 506	6.529	3.32	0.51
Isivilla	Calvario I	Inferred	1.679	268	531	1.851	1.17	0.63
Isivilla	Calvario Real	Inferred	1.146	90	122	1.264	0.27	0.21
Corani	Nueva Corani	Inferred	6.112	111	799	6.737	1.76	0.26
Corachapi	Corachapi*	Inferred	3.753	195	863	4.137	1.90	0.46
Colibri	Colibri II & III**	Inferred	9.453	167	1 862	10.420	4.10	0.39
Colibri	Tupuramani**	Inferred	10.976	125	1 618	12.099	3.57	0.29
Total Inferred			**130.020**	**213**	**32 708**	**143.322**	**72.11**	**0.50**

** Figures based on October 2010 Mineral Resource Estimates by The Mineral Corporation*
*** Figures based on September 2013 Mineral Resource Estimates by The Mineral Corporation*
Minor discrepancies due to rounding may occur
There are currently no known risks that could materially affect potential development
Density 1.98t/m^3
Cut-off 75 U ppm
Ton is short ton

The current areas explored via drilling and sampling that have allowed delineation of Mineral Resources are relatively small compared to the total Plateau Uranium mining concession footprint that is underlain by the Quenimari Formation rocks. Further radiometric surveys and geological interpretations of the un-explored mining concession areas are required to delineate potential additional targets for drilling and sampling.

1.4 MINING

WAI has conducted a mining study and estimated associated costs. The PEA is based on Mineral Resources, not Mineral Reserves and includes Inferred Mineral Resources that are considered too speculative geologically to have economic considerations applied to be categorised as Mineral Reserves. They do not have demonstrated economic viability and there is no certainty that the preliminary economic assessment will be realised.

This PEA considers a base case where Mineral Resources are recovered from open pits and an underground operation. The open pits are located in three separate geological complexes and consist of ten projected pits. The underground design is on a continuing mineralised horizon from an open pit.

The PEA has considered three of five identified resource areas, the three areas selected are based on uranium metal content and proximity to the proposed central processing area to reduce project CAPEX.

Open pit Mineral Resources contribute 92 % of the life of mine (LOM) projected 30 000 tonnes per day (tpd) mill feed.

A summary of the base case PEM mining inventories is detailed below:

Table 1-3: Base Case In-Situ Complex Mineral Resources to be Extracted

Complex		Tonnage	Grade U_3O_8 (ppm)	Waste (Mt)	Strip Ratio (t_W:t_{PEM})	U_3O_8 Content (t)	U_3O_8 Content (M lbs)
2	Colibri	40.1	232	9.90	0.25	7 760	17.1
3	Kihitian	45.7	309	170	3.72	11 800	26.0
3	Kihitian (UG)	8.23	475	0.00	-	3 440	7.58
4	Isivilla	15.0	373	48.8	3.25	4 670	10.3

Note: Figures have been rounded to 3SF
Mining factors (Loss/Dilution not applied)
Complex 1 and 5 excluded

1.4.1 MINE DESIGN

A base case contains of 101 Mt of Mineral Resources that are projected to be extracted from open pits and 8.23 Mt from underground mining. The base case open pits are planned to extract 30 000 tpd of PEM through conventional drill/blast and truck/shovel methods. The base case includes a LOM average of 10.9 Mt/a at 289 ppm U_3O_8 for a total of 31 400 t of contained U_3O_8 metal and 61.0 M lbs of recovered U_3O_8.

The open pit fleet will move PEM to either the processing area or a mobile crusher, depending on proximity. An allowance has been included in the open pit fleet to haul underground PEM to a central crushing plant.

The underground mine design will produce 2 700 tpd through room and pillar mining methods, extracted by a continuous miner and conveyors. Underground access is made directly via the open pit of Complex 3, allowing direct PEM mining to occur and minimising access capital requirements. The underground designs have been constrained by geotechnical and mechanical limitations to 10 m maximum height. Underground infrastructure will include ventilation shafts and fans and electrical substations.

The proposed operation size is within standard practises and proven methods. All equipment to be selected is readily available.

1.5 CONCLUSIONS

1.5.1 MINING

The project has shown that mining operations are amenable to an open pit operation with supplementary underground extraction. The PEM mining inventory currently contains 56 % Inferred Mineral Resources. Mining designs have been based on preliminary geotechnical designs.

The Corachapi and Corani Complexes have been excluded from this PEA, but they both have identified Mineral Resources which could be considered in an updated optimisation.

1.5.2 RECOVERY METHODS

The proposed method of recovery is based on the developed production schedule of the PEM. The proposed method includes crushing and stacking PEM onto a heap leach pad which is irrigated with an acidic solution to dissolve the uranium. The leach solution would then pass through ion exchange (IX) columns to recover the dissolved uranium and solvent extraction (SX) to recover uranium from the IX eluate to reduce acid consumption. The SX product solution then passes through precipitation to yield a yellowcake precipitate, which is thickened, filtered, dried and packaged for dispatch. Interpretation of metallurgical testwork results determined a processing recovery rate of 88 %, resulting in an average annual production of 6.08 M lb U_3O_8. The nominated process method utilises industry standard equipment and extraction technologies that are used in practice around the world.

1.5.3 FINANCIAL ANALYSIS

A capital cost and operating cost estimate based on the proposed mining and recovery methods, was prepared to the level of a PEA, with an accuracy of -30 % to +30 %. A base economic model was prepared using the capital and operating cost estimates as presented in Section 21, the model parameters outlined in Section 22, and the production schedule outlined in Section 16. A summary of the major financial parameters and results of the base case scenario are as shown in Table 1-4. It should be noted that these results do not demonstrate economic feasibility as the mine production schedule includes Inferred Mineral Resources. The results should be considered as potential results that could be achieved if the Inferred Mineral Resources were able to be converted to Indicated or Measured Mineral Resources in the future.

Table 1-4: Financial Model Parameters and Results

Item	Unit	Pre-Tax Scenario	Post-Tax Scenario
Parameters			
Uranium Price	USD/lb U_3O_8	50	50
Average cost of Production	USD/lb U_3O_8	17.28	17.28
Start-up CAPEX	M USD	299.8	299.8
Sustaining CAPEX	M USD	43.9	43.9
Results			
NPV (8 % discount rate)	M USD	852.7	603.1
IRR	%	47.6	40.6
Payback Period	annum	1.69	1.76

The project currently produces an encouraging IRR and NPV utilising a conservative sale price of USD 50 /lb U_3O_8.

1.6 RECOMMENDATIONS

Based on the results of the PEA it is recommended that a Pre-Feasibility Study (PFS) be completed to give an indication of the project's feasibility. Further definition of the project is required to allow a PFS to be completed and thus, as a minimum, the following is recommended to further develop the project and reduce the technical uncertainty and risk, associated with these variables:

- Infill drilling to upgrade the category of the Mineral Resources;
- Metallurgical testwork to improve confidence in and better define the proposed process; for the uranium and any secondary products
- A cut-off grade trade-off assessment should be undertaken to ascertain potential benefits to the project; and

- Geotechnical testwork to improve confidence of mining design.

It is estimated that these works will cost in the order of USD 4.0 million and take approximately 14 months to complete.

1.7 HIGH GRADE OPTION

As part of 10.9 Mt/a heap leach design GBM were requested to evaluate four additional projects. The estimates were to be conducted at a desktop class 5 level and based on a high-grade resource which used a 200 ppm U_3O_8 cut-off grade, and aimed to produce 5 M lb per annum.

The following four cases were investigated;

- Case 1: Heap leach, open pit mining only
- Case 2: Heap leach, open pit and underground mining
- Case 3: Tank leach, open pit mining only
- Case 4: Tank leach, open pit and underground mining

The tank leach option has been investigated due to the decreased plant footprint and the possible increased recoveries. There is also more confidence in scaling up from testwork to actual conditions for the tank leach option.

The same optimisation parameters from the base case were used for the high grade cases of complexes 2, 3 and 4. By varying the minimum processing cut-off grades the impact upon the NPV was observed. All tonnes and grades reported in Table 1-5 are in-situ results and have not been adjusted for mining loss or dilution.

Table 1-5 High Grade Case In-Situ Complex Mineral Resources to be Extracted

Complex		Tonnage	Grade U_3O_8 (ppm)	Waste (Mt)	Strip Ratio (t_W:t_{PEM})	U_3O_8 Content (t)	U_3O_8 Content (M lbs)
2	Colibri	18.0	341	11.0	0.61	6 120	11.9
3	Kihitian	22.1	493	165	7.46	10 900	21.2
3	Kihitian (UG)	8.23	475	0.00	-	3 440	7.58
4	Isivilla	10.6	470	49.4	4.69	4 960	9.62

Note: Figures have been rounded to 3SF
Mining factors (Loss/Dilution not applied)
In cases 1 and 2 U_3O_8 is extracted by the method detailed in section 17 (Heap leach). Case 3 and 4 are based on a tank leach operation.
Complexes 1 and 5 excluded

1.7.1 COSTING

The capital and operating estimates produced as part of the high grade option have utilised key inputs established as part of the base case project. However where information was not available historical and industry factors had been used.

Table 1-6 Capital Estimates

Case	Initial Capital (000's)	LOM Capital (000's)
Base Case	299.9	358.5
Case 1	247.5	279.4
Case 2	247.5	291.4
Case 3	267.4	299.3
Case 4	267.4	311.3

Table 1-7 Operating Estimates

Case	$/t ROM	$/lb U_3O_8
Base Case	9.6	17.28
Case 1	14.6	17.39
Case 2	13.6	15.95
Case 3	17.6	19.73
Case 4	17.0	18.81

Table 1-8 Financial Results

	Recovered U_3O_8	Pre-Tax		Post-Tax	
Case	M lb / a	NPV	IRR	NPV	IRR
Base Case	6.08	852.8	47.6	603.1	40.6
Case 1	4.26	544.4	41.2	417.4	37.3
Case 2	5.01	733.5	49.4	550.9	43.7
Case 3	4.50	510.2	36.8	397.2	33.9
Case 4	5.30	679.9	43.2	516.1	38.9

SECTION 2 INTRODUCTION

2.1 GENERAL

This report has been prepared on behalf of Plateau Uranium Inc. (Plateau Uranium), a Canadian company listed on the TSX Venture Exchange. Plateau Uranium has serveral uranium prospects on the Macusani Plateau with concessions being held by subsidiary companies, Global Gold S.A.C (Global Gold) and Minergia S.A.C (Minergia). GBM Minerals Engineering Consultants Limited (GBM) is an independent firm of engineering consultants specialising in the development, design and construction of mining projects. GBM was commissioned by Plateau Uranium to carry out a PEA to determine the yellowcake recovery process and to design and cost a process facility to treat the uranium PEM. This included the development of the process flowsheet the processing and infrastructure engineering design the capital and operating cost estimates to an accuracy of +30 % to -30 %, and the compilation of this PEA report.

This report includes all six mineral “complexes” that fall under the Plateau Uranium umbrella. The Mineral Resource estimates for two of these complexes (Corachapi and Colibri) have been previously reported by The Mineral Corporation (Young 2010[1], 2010[2], 2010[3], 2013) and are unchanged. The Mineral Resource estimates for three of the complexes (Kihitian, Isivilla, Corani) have been re-estimated. There are no Mineral Resources for the Sayaña Complex.

The Mineral Corporation (TMC) is a leading advisory firm focused on the provision of geological and mining engineering services and has prepared the Mineral Resource estimates and completed data verification for the project.

Wardell Armstrong International (WAI) is a leading multidisciplinary consultancy who has developed a LOM plan for proposed mining operations and associated cost estimates.

Project feasibility is outside the scope and consideration of this report as Inferred Mineral Resources form part of the PEM upon which this PEA is based. Inferred Mineral Resources cannot be directly converted into Mineral Reserves and due to their uncertainty of existence it cannot be assumed that any part of an Inferred Mineral Resource will ever be upgraded to a higher Mineral Resource category.

Additionally GBM have included for comparison a high grade resource, which has been evaluated with processing by either heap leach or tank leach. The high grade PEM has accounted for adjustment of acid consumption, recovery and costs from the base case. The results demonstrate that the project deposit has numerous options to extract and provide sound returns.

2.2 PRIMARY INFORMATION SOURCES

This report makes use of the following primary information sources:

- Macusani Plateau Uranium Deposits, Peru, (WAI, 2015) (3).
- Consolidated Mineral Resource Estimates for the Kihitian, Isivilla and Corani Uranium complexes, controlled by Plateau Uranium in the Puno District of Peru Report, (TMC, 2015) (4).

GBM has also used various other information sources which are referenced where applicable throughout this report.

2.3 QUALIFIED PERSONS

The GBM Qualified Persons are Michael Short BE (Civil), CEng FIMMM, FAusIMM(CP), FIEAust CPEng and Thomas Apelt BEng, PhD (Chem Eng), CEng MIChemE, MAusIMM(CP).

WAI and TMC have authored sections of this report as detailed in Table 2-1. The WAI Qualified Person is Mark Mounde, B.Eng., CEng MIMMM and the TMC Qualified Person is David Young, BSc (Hons), FGSSA, FSAIMM, FAusIMM, Pr Sci Nat.

2.4 QUALIFIED PERSON SITE VISIT

A number of visits by various Qualified Persons and other parties have been carried out to inspect the project site and verify its characteristics. The following site visits directly applicable to this PEA have been carried out by GBM, WAI and TMC representatives:

- Mark Mounde (WAI) and Stephen Holley (WAI) visited the site in November 2013;
- David Young (TMC) and Stewart Nupen (TMC) visited the site in March 2013;
- David Young (TMC), Michael Short (GBM) and Joe Russell (GBM) visited the site in January 2011;
- Michael Short (GBM), Joe Russell (GBM) and Bruce Pilcher (WAI) visited the site in October 2009; and
- David Young (TMC) and Stewart Nupen (TMC) visited the site in August 2008 and July 2009.

2.5 FINANCIAL INTEREST DISCLAIMER

Neither GBM, TMC, WAI, nor any of their consultants employed in the preparation of this report, have any beneficial interest in the assets of Plateau Uranium.

GBM, TMC and WAI have been paid fees and will continue to be paid fees for this work in accordance with normal professional consulting practices.

2.6 QUALIFIED PERSON SECTIONAL RESPONSIBILITY

This PEA was prepared by or under the supervision of the Qualified Person(s) identified in Table 2-1 against the respective sections of this report.

Table 2-1: Responsible Qualified Persons

Section	Section Title	Qualified Person(s)
1	Summary	GBM (Michael Short)
2	Introduction	GBM (Michael Short)
3	Reliance On Other Experts	GBM (Michael Short)
4	Property Description And Location	GBM (Michael Short)
5	Accessibility, Climate, Local Resources, Infrastructure And Physiography	GBM (Michael Short)
6	History	GBM (Michael Short) / TMC (David Young)
7	Geological Setting And Mineralisation	TMC (David Young)
8	Deposit Types	TMC (David Young)
9	Exploration	TMC (David Young)
10	Drilling	TMC (David Young)
11	Sample Preparation, Analyses, And Security	TMC (David Young)
12	Data Verification	TMC (David Young)
13	Mineral Processing And Metallurgical Testing	GBM (Thomas Apelt)
14	Mineral Resource Estimates	TMC (David Young)
15	Mineral Reserve Estimates	WAI (Mark Mounde)
16	Mining Methods	WAI (Mark Mounde)
17	Recovery Methods	GBM (Thomas Apelt)
18	Project Infrastructure	GBM (Michael Short)
19	Market Studies And Contracts	GBM (Michael Short)
20	Environmental Studies, Permitting, And Social Or Community Impact	GBM (Michael Short)
21	Capital And Operating Costs	GBM (Michael Short) / WAI (Mark Mounde)
22	Economic Analysis	GBM (Michael Short)

Section	Section Title	Qualified Person(s)
23	Adjacent Properties	TMC (David Young)
24	Other Relevant Data And Information	GBM (Michael Short)
25	Interpretation And Conclusions	GBM (Michael Short)
26	Recommendations	GBM (Michael Short)
27	References	GBM (Michael Short)

SECTION 3 RELIANCE ON OTHER EXPERTS

The Qualified Persons have relied on expert opinions and information provided by Plateau Uranium pertaining to environmental considerations, taxation matters and legal matters including mineral tenure, surface rights and material contracts.

For the purposes of Section 4 (Property Description and Location) and Section 23 (Adjacent Properties) of this report the Qualified Person has relied on property ownership data provided by Plateau Uranium and other sources as referenced within the sections. This information is believed to be essentially complete and correct to the best of the Qualified Person's knowledge and no information has been intentionally withheld that would affect the conclusions made herein. The Qualified Person has not researched the property title or mineral rights for the project and expresses no legal opinion as to the ownership status of the property.

For the purposes of Section 19 (Market Studies and Contracts) of this report the Qualified Person has relied on information pertaining to market studies and material contracts provided by Plateau Uranium and other sources as referenced within the section. The Qualified Person has reviewed the information provided by Plateau Uranium and believes this information to be correct and adequate for use in this report.

For the purposes of Section 20 (Environmental Studies, Permitting, and Social or Community Impact) of this report the Qualified Person has relied on information provided by Plateau Uranium and other sources as referenced within the section. The Qualified Person has reviewed the information provided by Plateau Uranium and believes this information to be correct and adequate for use in this report.

For the purposes of Section 22 (Economic Analysis) of this report the Qualified Person has relied on information provided by Plateau Uranium and other sources as referenced within the section, pertaining to taxation. The Qualified Person has reviewed the taxation information provided and believes it to be correct and adequate for use in this report.

SECTION 4 PROPERTY DESCRIPTION AND LOCATION

4.1 LOCATION

From an internal government administration arrangement, Peru is divided into 24 “Departments”, each of which is subdivided into provinces and districts or regions. The Plateau Uranium concessions are located in the Carabaya Province which is a province of the Department of Puno in the south-eastern part of Peru. The Carabaya Province is divided into ten districts or regions. It is bounded to the north by the Madre de Dios Region, on the east by the Sandia Province, on the south by the provinces of Azángaro, Melgar and Putina and on the west by the Cusco Region. The capital of the province is Macusani. The people in the province are mainly indigenous citizens of Quechua descent. Quechua is the language which the majority of the population (84.12 %) learn to speak from childhood, while 15.14 % of the residents use the Spanish language and 0.62 % communicate in Aymara.

The locality of the project area is shown in Figure 4-1. The portfolio comprises the amalgamation of those rights held by Plateau Uranium. This report is focused only a selection of mining concessions within the Macusani project area.

The project is located approximately 650 km east southeast of Lima and about 220 km by road from Juliaca in the south. The town of Macusani is some 25 km to the southeast of the project.

The survey reference system utilised for this report is Universal Transverse Mercator, Zone 19S, using the Provisional South American 1956 datum, hereafter referred to as PSAD UTM Zone 19S. The project concessions lie between the co-ordinates 320 000 and 340 000 East and 844 4 000 and 846 7 500 North.

Figure 4-1: Locality Plan

4.2 MINERAL TENURE

The project includes the mining concessions listed in Table 4-1 and illustrated in Figure 4-2. All of which are valid for all solid minerals.

Table 4-1: Mining Concessions

Mining Concession Code	Mining Concession Name	Date Conferred	Area (ha)	Owner	Complex Name
01-00530-05	Lincoln XXVII	07-Sep-07	900	Minergia SAC	Complex 3 - Kihitian
01-0052-905	Lincoln XXVI	2005	1 000	Minergia SAC	Complex 4 - Isivilla
01-0053-205	Lincoln XXIX	2005	1 000	Minergia SAC	Complex 5 - Corani
01-0117-805	Lincoln XXX	2005	200	Minergia SAC	Complex 5 - Corani
01-03428-97	Corachapi	27-Jul-05	500	Global Gold SAC	Complex 1 - Corachapi
01-02152-04	Taititira	27-Jul-05	100	Global Gold SAC	Complex 1 - Corachapi
01-00070-05	Taypicorani	27-Jul-05	200	Global Gold SAC	Complex 1 - Corachapi
01-00889-05	Colibri II	11-Apr-05	600	Global Gold SAC	Complex 2 - Colibri
01-01218-05	Colibri III	16-May-06	100	Global Gold SAC	Complex 2 - Colibri
01-00695-05	Tupuramani	18-May-05	400	Global Gold SAC	Complex 2 - Colibri
01-00367-05	Kihitian	27-Jul-05	800	Global Gold SAC	Complex 3 - Kihitian
01-00165-05	Triunfador I	10-Jan-05	400	Global Gold SAC	Complex 4 - Isivilla
01-00711-05	Calvario II	18-May-05	400	Global Gold SAC	Complex 5 - Corani
01-00697-05	Calvario III	18-May-05	400	Global Gold SAC	Complex 5 - Corani
01-01113-05	Samilio I	15-Jul-15	1 000	Global Gold SAC	Complex 6 - Sayaña
01-00692-05	Triunfador 4	27-May-05	900	Global Gold SAC	Complex 6 - Sayaña
01-00866-05	Triunfador 5	18-Aug-05	600	Global Gold SAC	Complex 6 - Sayaña
01-00772-05	Tantamaco 3	31-Aug-05	300	Minergia SAC	Complex 3 - Kihitian

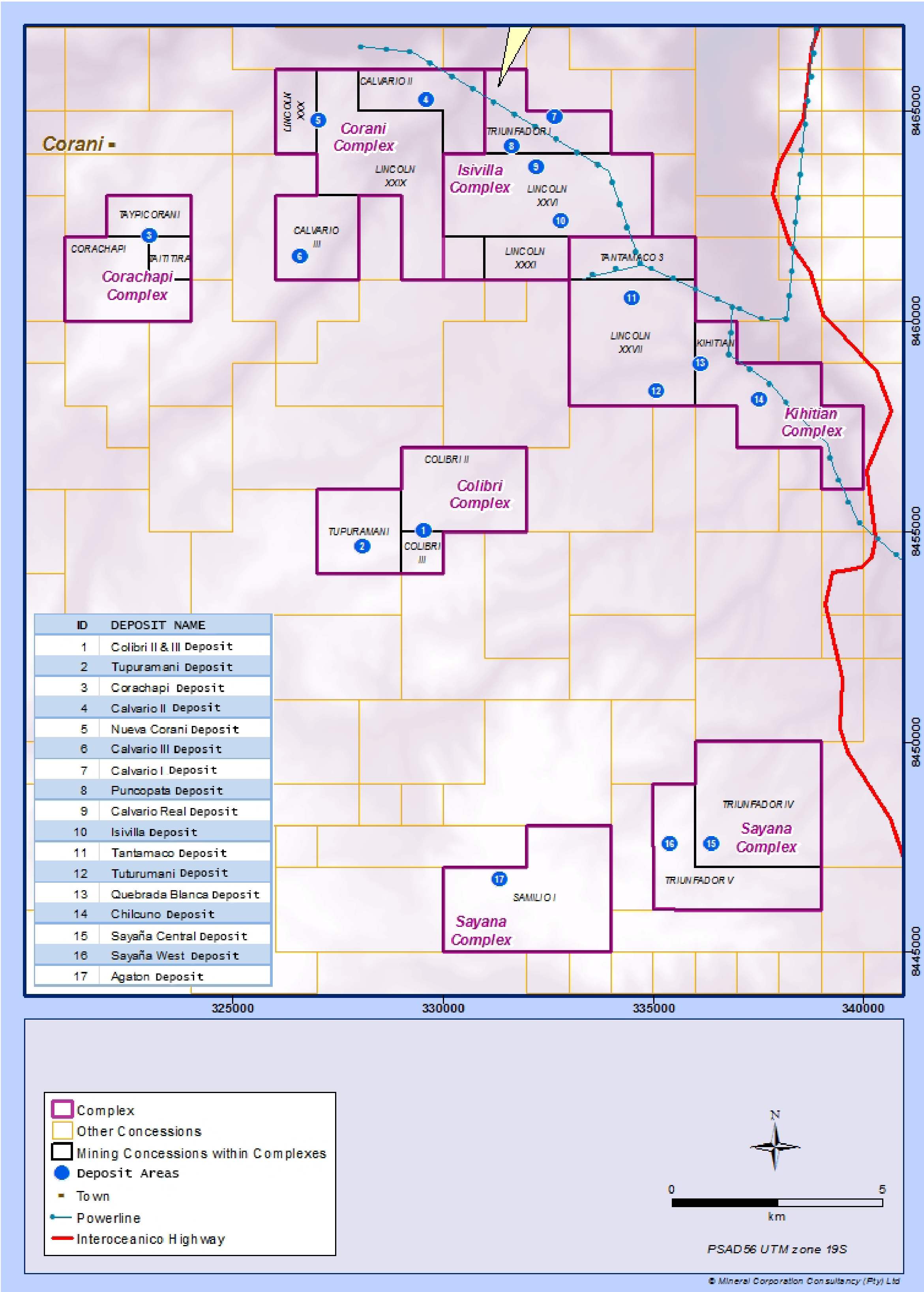


Figure 4-2: Mineral Rights Plan

The Qualified Person has restricted their review of the mineral rights held by Global Gold and Minergia respectively to checking the individual concession boundaries on plans against those depicted on the mining concessions. No legal review of the validity of the process Global Gold and Minergia went through to obtain the mining concessions has been undertaken, nor has an attempt been made to understand the various company structures and ownerships prior to transfer to either Global Gold or Minergia.

4.3 PROPERTY, TITLE AND SURFACE RIGHTS

Plateau Uranium Inc. is a company listed on the TSX Venture Exchange that owns 99.5 % of a Peruvian company, Macusani Yellowcake SAC. The remaining 0.5 % is held by a Peruvian individual as recommended by the Ministry of Energy and Mines (MEM). Plateau Uranium Inc. also owns 100 % of another Peruvian company, Minergia. Global Gold and Minergia hold various mineral rights, however, the focus of this report is entirely on the mining concessions listed in Table 4-1.

The main legislation governing the mining sector is the General Mining Law of Peru gazetted in June 1992 with subsequent amendments. Concessions are granted by the MEM for exploration, exploitation, beneficiation, auxiliary services and transportation. No concession is required for reconnaissance, prospecting or trading.

A mining concession grants its holder the right to explore and exploit minerals within its area and the key characteristics include:

- Concessions are exclusive, freely transferable and mortgageable;
- Location is based on a UTM grid system of minimum 100 ha to 1 000 ha;
- Granted on a first-come, first served basis;
- Indefinite term but with restrictions and an objective based criteria;
- Single annual fee payable. Fee structure based on scale of operations and duration of ownership; and
- Access to the property must be negotiated with surface land owners.

All concessions held by Global Gold S.A.C. and Minergia SAC have been correctly filed at the Ministry of Energy and Mines for all past and present exploration, and all good standing fees made as of July 2017. No set expiration dates has been set on all concessions held by the subsidiaries of PLU, and access to the surface rights will be maintained by continuation of agreements with the local communities located on the Macusani plateau.

The project currently assumes additional land acquisition and surface rights will be obtained in the future to accommodate proposed infrastructure such as access roads and conveyors as well as the area identified for the processing facilities.

4.4 ROYALTIES

The properties are currently not subject to any royalties and it is not yet defined in Peruvian legislation whether a government royalty would be payable, nor its value for uranium extraction. Peruvian royalties are an administrative charge that would be payable to the Puno Department for extracting uranium that is sold. The law with respect to other metallic and non-metallic mineral resources allows the deduction of certain costs and expenses such as indirect taxes, insurance, freight, among others and royalties may be considered for tax purposes as a cost of the mineral that is sold.

Part of the transferral of the Corachapi property to Global Gold was the assumption of a pre-existing net smelter royalty obligation of 1 % of gross proceeds less milling costs to Mining Investments Limited by Contract Uranium (Contact Uranium Limited, 2007) (5). Additionally this concession is currently outside of the PEA.

4.5 ENVIRONMENTAL LIABILITIES AND PERMITS

Due to the draft nature of uranium mining legislation the extent of permitting and environmental liabilities is unknown. The following list is likely to be applicable to future operations and is discussed further in Section 20:

- Right to mine;
- Uranium Licence; and
- Approved Environmental Impact Assessment (EIA).

SECTION 5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 ACCESS AND INFRASTRUCTURE

The project is approximately 650 km south east of Lima and the Interoceanica Highway passes directly to the east. The Interoceanica Highway is a system of sealed roads that link the ports of San Juan and Ilo, on the western coast of Peru, to the western region of Brazil, via the Andes. The nearest towns to the project are Macusani, 25 km to the southeast of the project area, and Corani, 14 km to the northwest of the project. Two unpaved roads connect the project to the Interoceanica Highway, one via Tantamaco, and the other via Macusani. This highway links up to regional logistics infrastructure, including the national road network, Juliaca and other airports, and Ilo (Callao) shipping port.

The closest airport to the project area is Juliaca, situated approximately 220 km to the south. This airport is in good condition and receives numerous daily flights from Lima and Cusco.

There are currently no electricity supplies or telephone communication networks in any part of the concession areas besides those supplying the villages of Tantamaco and Isivilla, and there are high voltage mains adjacent to the project.

5.2 CLIMATE

The Macusani region is classified according to the Köppen classification system as a temperate / mesothermal climate (Group C) and may be more specifically described as a temperate climate with dry winters (Cwb) (Institute for Veterinary Public Health., 2006) (6). This type of climate is characteristic of the highlands inside the tropics of Peru and Bolivia. The region is characterised by dry and cold temperatures from May to October, while from November to April the climate is warmer with more frequent rain and/or snow falls. The rainy season is provoked by tropical air-masses and the dry winters by subtropical high pressure. Mild temperatures are the consequence of altitude which causes it to become cooler year-round. A weather station has been situated on the Macusani Plateau since 2011 (Global Gold, 2013) (7).

The average annual precipitation for the Carabay Province is approximately 600 to 1 000 mm (SENAMHI, 2009) (8) and the average annual precipitation recorded at site is between 470 and 786 (Global Gold, 2013) (7). Most of this precipitation occurs in the wet season, typically between November and April.

There are no available local records of evaporation on the Macusani plateau. The average pan evaporation rate of 1 631 mm per annum of surrounding areas, (with similar altitudes), are assumed to be representative of the Macusani plateau (Ministerio De Aeronautica, 1963) (9).

Maximum and minimum temperatures at site have been recorded as 14.4 °C and -12.3 °C respectively, with an average temperature of below 0 °C recorded on 22 days in 2012 (Global Gold, 2013) (7).

The current design considers a 24 hour 365 day per year operation.

5.2.1 AIR PRESSURE

The maximum site altitude is 4 730 m above mean sea level (Tupuramani) (Global Gold, 2012) (10) and accordingly the site air pressure will nominally be 54.2 kPa (GBM, 2011) (11).

Depending on local variations in altitude and temperature, the air density at site will be approximately 55 % to 60 % of air at standard conditions at sea level. The lack of oxygen and / or lack of air, and other factors relating to reduced air pressure will affect operations due to the impact on personnel and machinery.

5.3 LOCAL RESOURCES AVAILABILITY

5.3.1 SURFACE RIGHTS

The current surface rights agreements allow for exploration activities. It is anticipated that continued good standing and communication with the community will allow for surface rights agreements to be maintained during operations.

5.3.2 POWER SOURCE

The power supply to the project would be from the national grid.

During a site visit, GBM personnel met with the utility authority MEM-DOE, and their representatives confirmed that there would be sufficient capacity in the regional power transmission and distribution network to supply a project of the proposed size. Supply would be primarily sourced from hydroelectric power. This PEA assumes that power will be supplied from a grid connection and that sufficient power would be available at the plant perimeter fence.

The potential for using wind power to deliver a portion of the power requirements at the site was evaluated. The scenario investigated considered that the processing plant would be connected to the San Gaban powerline and if wind resources were available, wind turbines would be used to reduce the energy purchased from the grid as well as “feed in” to the grid when not utilised by the process facility.

The initial analysis suggested that wind power is not a commercially viable option due to the low wind resources in the Macusani region and high cost of installation. However the uncertainty in the data from the Peruvian Wind Atlas (MEM) (12) in this area is significant due to the complex terrain surrounding the project.

Further analysis of this option has not been completed as part of this PEA due to a lack of new information required to revisit assumptions used in the original analyses and investigate potential alternatives.

5.3.3 WATER SOURCE

Inspection of various maps of the region shows the project site to be within the Amazon River catchment area (Universidad Nacionale Ingeniera, 2001) (13). To the west of Corachapi sits the retreating Quelccaya Ice Cap, the largest glacier located within the tropics. It is anticipated that one of the several significant water bodies within close geographical proximity may be a suitable water source. The preferred water source location has been identified as from the valley adjacent to the project. Should this location prove to be incapable of providing sufficient raw water flow (170 m^3/h), additional sources have been identified that should suffice, but these alternatives will increase capital and operating costs due to their increased distance from the project. Year round flows are maintained in the local water courses and therefore the make-up water supply is considered reliable, and not an area of significant risk to the project.

5.3.4 PERSONNEL

Peru has an active mining industry, and it is therefore reasonable to expect that a project workforce consisting of skilled and semi-skilled people would be sourced locally.

5.4 PHYSIOGRAPHY

The concession area is situated on the relatively flat Altiplano of the Eastern Cordillera of the Andes Mountain Range. The climatic conditions and altitude dictate that the properties are vegetated by coarse scrub and grasses.

The Macusani region has a temperate climate with dry winters and is located typically between 4 100 m and 4 700 m above mean sea level (Global Gold, 2012) (10).

There are a number of localised large areas of relatively shallow slope that have been identified on the Macusani plateau as alternatives for the siting of the heap leach pad processing facility and/or tailings management facility. The nature of the topography of these areas would reduce the potential requirement for cut and fill during construction. Additionally, their catchment areas being at the top of the plateau are beneficial in minimising the potential impact on surrounding runoff regimes.

Identification of multiple potential sites for locating the heap leach pad and tailings management facility is beneficial as it minimises the risk to the project.

The Colibri and Isivilla Complexes are considered to have ample areas of land with low undulations and minor gradients to establish a waste rock dump. The Kihitian concession is located on a hillside with steep gradients located towards a valley, in this case the waste dump would need to be located above the deposit or waste material would need to be dumped from the hillside.

SECTION 6 HISTORY

6.1 PREVIOUS OWNERSHIP

6.1.1 BACKGROUND TO CURRENT OWNERSHIP

6.1.1.1 MACUSANI YELLOWCAKE INC.

Macusani Yellowcake Inc. was a Canadian uranium exploration and development company that had been actively exploring on the Macusani Plateau since 2007. The company was incorporated in November 2006 and owned a 99.5 % interest in the Peruvian concessions of Global Gold S.A.C.

6.1.1.2 MINERGIA S.A.C

In 2007, Cameco Corporation (and its wholly owned subsidiary Cameco Global Exploration Limited) (Cameco) entered into a joint venture with Vena Resources Inc. (Vena) with the objective of jointly exploring for uranium in Peru. Minergia S.A.C was formed as the joint venture vehicle, with Cameco providing the funding and Vena undertaking the exploration management. The ownership was founded on 50 % shareholding in favour of each party. The combined portfolio covered an area of 14 700 ha (Henkle, 2014) (14).

Azincourt Uranium Inc. (Azincourt) entered into a definitive share-purchase agreement with joint-venture partners Cameco and Vena to acquire full ownership of Minergia S.A.C, with the acquisition being completed in January 2014 (IRW Press) (15).

Macusani Yellowcake Inc. and Azincourt announced in September 2014 that they had completed the acquisition by Macusani Yellowcake Inc. of Azincourt's adjacent uranium properties located on the Macusani Plateau. Under the terms of the transaction, Macusani acquired 100 % of Azincourt's Peruvian subsidiary, Minergia S.A.C.

Arising from this transaction, there was a consolidation of mining concessions.

6.1.1.3 PLATEAU URANIUM INC.

In April 2015, Macusani Yellowcake Inc. changed its name to Plateau Uranium Inc.

A summary of the transactions which form the history to Plateau Uranium is provided in Figure 6-1.

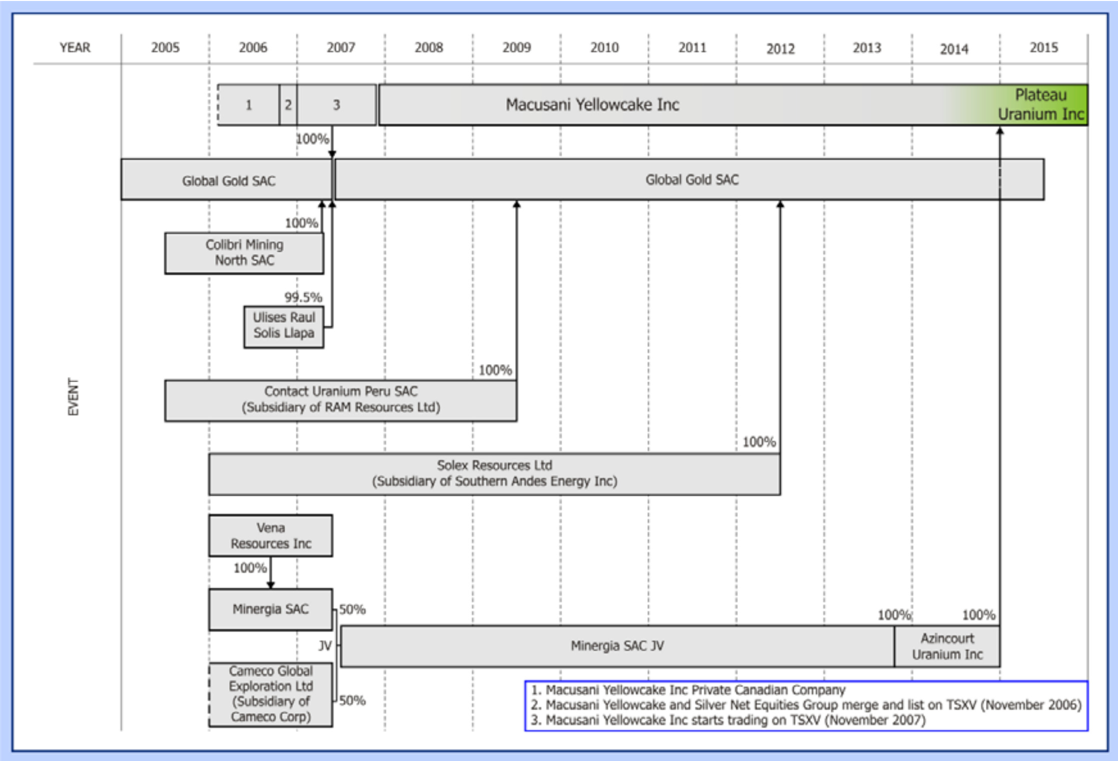


Figure 6-1: History of Plateau Uranium

6.2 COMPLEXES WITH PREVIOUS MINERAL RESOURCE ESTIMATES

6.2.1 CORACHAPI COMPLEX

The Corachapi Complex consists of the Corachapi, Taititira and Taypicorani mining concessions.

These concessions were granted to S.M.R.L. Corachapi, a Contact Uranium Peru S.A.C. (Contact Uranium) entity on 27 July 2005. Contact Uranium was the predecessor of RAM Resources Limited. Macusani, through Global Gold, acquired the concessions from RAM Resources. The concessions were transferred to Global Gold on 18 June 2009.

6.2.2 COLIBRI COMPLEX

The Colibri Complex consists of the Colibri II, Colibri III and Tupuramani mining concessions.

The Colibri II mining concession was granted on 11 April 2005 to Colibri Mining North S.A.C and subsequently transferred to Global Gold on 18 April 2007.

The Colibri III mining concession was granted on 16 May 2006 to Ulises Raul Solis Llapa and subsequently transferred to Global Gold on 18 April 2007.

The Tupuramani mining concession was granted to Minera Macusani S.A.C., a Solex Resources Inc. entity on 18 May 2005. Solex Resources Inc. was the predecessor of Southern Andes Energy Inc. The mining concession was transferred to Global Gold on 26 June 2012.

6.2.3 KIHITIAN COMPLEX

The Kihitian Complex consists of the Kihitian, Lincoln XXVII and Tantamaco 3 mining concessions.

The Kihitian mining concession was granted to S.M.R.L. Corachapi on 27 July 2005. The concession was transferred to Global Gold on 18 June 2009 (Short et al, 2014) (16).

The Lincoln XXVII mining concession was registered to Minergia S.A.C on 7 September 2005 and the Tantamaco 3 mining concession was registered to Minergia on the 7 September 2007.

6.2.4 ISIVILLA COMPLEX

The Isivilla Complex consists of the Triunfador I and Lincoln XXVI mining concessions.

The Triunfador I mining Concession was granted to Minera Macusani S.A.C., a Solex entity, on 10 January 2005. The concession was transferred to Global Gold on 26 June 2012 (Short et al, 2014) (16).

The Lincoln XXVI mining concession was registered to Minergia in 2005.

6.2.5 CORANI COMPLEX

The Corani Complex consists of the Calvario II, Calvario III, Lincoln XXIX and Lincoln XXX mining concessions.

The Calvario II and Calvario III concessions were originally held by Solex until the transfer of the concessions to Global Gold in 2005.

The Lincoln XXIX and Lincoln XXX mining concessions were awarded to Minergia in 2005.

6.2.6 SAYAÑA COMPLEX

The Sayaña Complex consists of the Triunfador 4 and Triunfador 5 mining concessions that were allocated to Solex on 27 May 2005 and 18 August 2005 respectively. These were subsequently transferred to Global Gold. Global Gold also obtained the Samilo 1 mining concession on 15 July 2015 that also forms part of the Complex.

6.3 PREVIOUS EXPLORATION

6.3.1 REGIONAL EXPLORATION

6.3.1.1 INSTITUTO PERUANO DE ENERGIA NUCLEAR – THE PIONEERS

In 1975, the uranium and nuclear activities in Peru were placed under the control of the Instituto Peruano de Energia Nuclear (IPEN). A five year exploration plan (1976-1981) was initiated with the aim of identifying and developing resources in the country. The Macusani East area was the most studied area in southern Peru by IPEN. After IPEN discovered the first 60 uranium showings in 1978, systematic radiometric prospecting and trenching were carried out over an area of approximately 600 km^2, culminating in the discovery of numerous additional uranium showings (Young, 2013) (17).

6.3.1.2 UNDP/IAEA

From mid-1977, a long term United Nation Development Programme/International Atomic Energy Agency (UNDP/IAEA) project was initiated consisting of regional reconnaissance over selected areas. The results of most of the work were negative except for those from a car-borne radiometric survey of the Puno Basin where a significant discovery was made near Macusani in the southern Cordillera Oriental, north of Lake Titicaca. Anomalies were found in the volcanic and interbedded sediments of the Upper Tertiary age Macusani volcanics and the Permian age Mitu Group (Young, 2013) (17).

In the same exploration phase, additional anomalies were located to the south southwest near Santa Rosa in Tertiary age porphyritic rhyolites and andesites.

These (and other discoveries in the Lake Titicaca region) concentrated the exploration in the area. A helicopter spectrometric survey of selected areas was completed in 1980 in Muñani, Lagunaillas and Rio Blanca as an IAEA/IPEN Project and a fixed wing survey was completed in an adjacent area by IPEN. Numerous uranium anomalies were discovered.

In 1984, the Organisation for Economic Co-operation and Development's Nuclear Energy Agency and the IAEA sponsored an International Uranium Resources Evaluation Project Mission (IUREP, 1984) (18) to Peru. The mission estimated that the Speculative Resources of the country fell within the range of 6 000 to 11 000 t of uranium.

6.3.2 LOCAL EXPLORATION

6.3.2.1 CORACHAPI COMPLEX

The Corachapi Complex covers the largest radiometric anomaly that was outlined by IPEN during its work in the 1980s. A ground-based radiometric survey was undertaken to guide the drilling over these anomalies.

Initial exploration was by means of samples collected from 77 trenches cut across the strike of mineralisation at 50 m to 100 m intervals. Contact Uranium undertook a drilling campaign beginning in 2007 which comprised some 193 diamond drillholes. Global Gold subsequently drilled a further 26 drillholes. All holes were drilled to a depth of approximately 50m, bringing the current status of drilling to 11 818m from 219 drillholes. The Mineral Corporation prepared a Mineral Resource estimate based on those drillholes for which completed analytical data was available, totalling 210 holes (Young, 2010[3]).

6.3.2.2 COLIBRI COMPLEX

Apart from the regional exploration described in Section 6.3.1, the Qualified Person is unaware of any previous exploration undertaken prior to the acquisition by Global Gold on the concessions which make up the Colibri Complex. Global Gold delineated potential uranium anomalies through a combination of regional geological understanding and surface radiometric data from the work of IPEN in the 1970's and 1980's (Young, 2013).

A ground radiometric survey was completed over the Complex in 2007. The reading of anomalous values over an area of 4 800m by 600m peaked at 10 000 counts per second (cps). Global Gold then undertook structural mapping which assisted the structural understanding of the area.

A diamond drilling programme was initiated in April 2007 on the target area identified by the ground radiometric anomalies. The initial drilling focus was on Colibri II & III, and subsequently extended onto Tupurumani, and to date 127 diamond drillholes have been drilled. The drilling was undertaken by Minera Colibri, a drilling contractor.

6.3.3 KIHITIAN COMPLEX

During the IUREP project, trenching and limited underground adits were developed at the Chilcuno Chico deposit within the Kihitian Complex. Contact Uranium did not undertake any exploration work.

After acquiring the concession from Contact Uranium, Global Gold undertook a ground-based radiometric survey to guide the exploration drilling. The adits at Pinochio were resampled and drilling was undertaken at the Chilcuno Chico and Quebrada Blanca Deposits and by August 2012, 8104m had been drilled. Minergia undertook drilling on the Tantamaco and Tuturumani Deposits on the basis of the results of ground radiometric surveys.

6.3.4 ISIVILLA COMPLEX

The Calvario I and Puncopata deposits fall within the Triunfador I mining concession. This concession was explored extensively by Solex, between 2000 and 2012. Solex undertook drilling campaigns at the Calvario I and Puncopata deposits which comprised 92 diamond drillholes. During Solex's exploration campaign, they undertook an airborne radiometric survey. This exploration data has been incorporated into the recent drilling work and in combination forms a basis for the Mineral Resource estimates for the Calvario I, Puncopata, Calvario Real and Isivilla deposits described within this report.

A total of 36 core boreholes were drilled at Isivilla and Calvario Real within the Lincoln XXXI mining concession. This drilling funded by Vena and all post-2006 drilling was run as a joint venture between Minergia and Vena funded by Minergia. The drilling work was supervised by geologists employed by Vena.

6.3.5 CORANI COMPLEX

Initial exploration work for the Calvario II deposit was undertaken by Solex. The drilling was undertaken by Frontier Mining, a joint venture between Frontier Pacific and Solex. A total of 32 drillholes from 8 platforms were drilled at the Calvario II deposit, totalling 2 433 m.

Exploration at the Calvario III deposit was undertaken by Solex, and encompassed the drilling of 85 diamond drillholes on 23 platforms, totalling 5 425 m.

In 2006, Vena drilled 8 diamond drillholes at the Nueva Corani deposit, totalling 679 m. From 2007 to 2010, Minergia drilled 48 diamond drillholes at Nueva Corani totalling 6 282 m.

The exploration database for the Corani Complex comprises lithological, sampling, topographical and drillhole survey data. All drillhole and topographic surveys were assumed to be based on PSAD56. A high resolution topographic survey of the area was available. To date, a total of 174 diamond drillholes have been drilled over the Complex totalling 14 628 m.

6.3.6 SAYAÑA COMPLEX

Previous exploration on the Agaton, Sayaña West and Sayaña Central deposits was conducted by Solex through drillhole drilling. On Agaton, a total of 52 drillholes from 12 platforms were drilled, totalling 2 300 m of drilling.

The Sayaña West drilling programme comprised 34 drillholes drilled from 9 platforms. A total of 2 245 m of drilling was completed. Drilling on Sayaña Central concluded with a total of 94 drillholes totalling 8 465 m being drilled. These were drilled from 26 platforms.

Agaton, Sayaña West and Sayaña Central database includes collar, survey, lithological and assay data.

6.4 PREVIOUS MINERAL RESOURCE ESTIMATES

The Qualified Person is unaware of any previous ownership in relation to the concessions which could impact on the Mineral Resource Estimates other than those discussed in the following commentary.

6.4.1 CORACHAPI COMPLEX

In January 2007, an estimate for the Mineral Resources for the Corachapi Project was carried out by Contact Uranium. The estimated Mineral Resources were some 3.793 Mt at an average of 975 ppm uranium (U). In April 2009, a second Mineral Resource estimate was compiled by SRK Consulting. The estimated Mineral Resources (at a 50 ppm U cut-off) were 7.0 Mt with an average grade of 208 ppm U. This estimation was based on gamma down-the-hole estimates (Short et al, 2014) (16). The Qualified Person is not aware of any historical Mineral Reserve estimates being published for the Corachapi Complex.

TMC prepared a statement of the Mineral Resources NI 43-101 for the Corachapi complex based on chemical analytical data. The Mineral Resources were presented in Young (20103) and as they have not been modified in the interim, they remain unchanged. A summary of these Mineral Resources is provided in Table 6-1.

6.4.2 COLIBRI COMPLEX

The Qualified Person is not aware of any historical Mineral Resource or Mineral Reserve estimates being published for the Colibri Complex.

Previous Mineral Resource estimates for the Colibri II & III concessions are those compiled by TMC and reported in Young (2008, 2010[1], 2010[2] and 2013). A summary of these Mineral Resources is provided in Table 6-1.

TMC reported in 2010 a 4 290Mt of Indicated Mineral Resources at a uranium grade of 188ppm above a cut-off grade of 75ppm and 39.207Mt of Inferred Mineral Resources at a uranium grade of 142ppm above the same cut-off. It should be noted that these Mineral Resources were improved in the 2013 estimates compiled by TMC and included the results of the Tupuramani drilling. A summary of these Mineral Resources is provided in Table 6-1.

6.4.3 KIHITIAN COMPLEX

The only previous Mineral Resource estimates which The Mineral Corporation is aware of, for the Chilcuno Chico and Quebrada Blanca deposits, are the NI 43-101-compliant Mineral Resource estimates compiled by TMC in 2012 (Young, 2012) and updated in 2013 (Young, 2013) and provided in Table 6-1.

At the date of acquisition of Minergia by Plateau Uranium, Measured, Indicated and Inferred Mineral Resources were reported for the Tantamaco deposit. In addition, Inferred Resources were reported for the Tuturumani deposit. A summary of these Mineral Resources by Henkle (2014) (14) is provided in Table 6-1.

The Qualified Person is not aware of any other historical Mineral Resource or any historical Mineral Reserve estimates being published for the Kihitian Complex.

6.4.4 ISIVILLA COMPLEX

TMC is not aware of any previous Mineral Resource or Mineral Reserve estimates which have been carried out within the Triunfador Mining Concession (Calvario I and Puncopata deposits) prior to the estimates provided by The Mineral Corporation in 2013 (Young, 2013). A summary of these Mineral Resources is provided in Table 6-1.

At the date of acquisition of Minergia by Macusani, Measured, Indicated and Inferred Mineral Resources were reported for the Isivilla deposit. In addition, Inferred Mineral Resources were reported for the Calvario Real deposit. A summary of these Mineral Resources by Henkle (2014) (14) is provided in Table 6-1.

The Qualified Person is not aware of any other historical Mineral Resource or any historical Mineral Reserve estimates being published for the Isivilla Complex.

6.4.5 CORANI COMPLEX

At the date of acquisition of Minergia by Macusani, Indicated and Inferred Mineral Resources were reported for the Nueva Corani deposit. A summary of these Mineral Resources by Henkle (2014) (14) is provided in Table 6-1.

The Qualified Person is not aware of any other historical Mineral Resource or any historical Mineral Reserve estimates being published for the Corani Complex.

6.4.6 SAYAÑA COMPLEX

The Qualified Person is not aware of any historical Mineral Resource or Mineral Reserve estimates being published for any of the deposits within the Sayaña Complex.

6.5 COMBINED PREVIOUS MINERAL RESOURCES

At the date of the transaction whereby Macusani acquired Minergia the combined Mineral Resources of the two entities are contained in Table 6-1.

Table 6-1: Combined Mineral Resources of Macusani and Minergia prior to the Transaction

Minergia	Tantamaco	Nueva Corani	Isivilla	Tuturumani	Calvario Real
Measured & Indicated	16.02Mlb @0.44lb/t (36.09Mt @ 224ppm U_3O_8)	0.71Mlb @0.20lb/t (3.53Mt @ 101ppm U_3O_8)	1.49Mlb @0.30lb/t (4.92Mt @ 153ppm U_3O_8)	N/A	N/A
Inferred	5.62Mlb @0.35lb/t (15.90Mt @ 178ppm U_3O_8)	3.44Mlb @0.43lb/t (8.09Mt @ 214ppm U_3O_8)	6.44Mlb @0.84lb/t (7.63Mt @ 425ppm U_3O_8)	1.16Mlb @0.19lb/t (6.03Mt @ 97ppm U_3O_8)	0.77Mlb @0.55lb/t (1.41Mt @ 275ppm U_3O_8)
Macusani	**Quebrada Blanca & Chilcuno Chico**	**Colibri 2 & 3 / Tupurumani**	**Corachapi**	**Calvario 1 & Puncopata**	
Measured & Indicated	11.76Mlb @1.27lb/t (8.4Mt @ 635ppm U_3O_8)	14.69Mlb @0.48lb/t (27.9Mt @ 240ppm U_3O_8)	5.02Mlb @0.39lb/t (11.6Mt @ 195ppm U_3O_8)	N/A	
Inferred	17.38Mlb @1.23lb/t (12.80Mt @ 615ppm U_3O_8)	7.67Mlb @0.34lb/t (20.4Mt @ 170ppm U_3O_8)	1.91Mlb @0.46lb/t (3.8Mt @ 230ppm U_3O_8)	3.13Mlb @0.82lb/t (3.5Mt @ 409ppm U_3O_8)	

SECTION 7 GEOLOGICAL SETTING AND MINERALISATION

As already stated, the Macusani concessions are located in the Carabaya Province, Puno Department of south-eastern Peru in the Andes. The Andes are a geographical feature formed by mountain building processes that are active at present which is driven by plate tectonics.

7.1 REGIONAL GEOLOGY

A common geological feature of most mountainous belts in the world is that they are usually structurally and stratigraphically complex. In the Puno region of Peru mainly Paleozoic sediments (520-250 Ma old) that were formed on the western Brazilian Craton (Figure 7-1) have been highly deformed by thrusting and folding due to the westwards movement of the South American tectonic plate (Brazilian Craton) over-riding the Pacific tectonic plate (Nazca Plate) along the western margin of the Americas over the last ±150 Ma. This occurred with the breakup of the Americas from the African and European continents, and the development of the Atlantic Ocean. The main geological units are shown in Figure 7-1 with the Oceanic Trench forming the western margin of the South American plate.

The tectonic history has led to the older sediments being bounded by westward dipping thrusts, intense folding and intrusions of dykes, batholiths and being affected by volcanic activity at various times (Henkle, 2014) (14). The Andes represents a large anticlinorium complicated by a series of faults and intrusions, with the flanks of this superstructure are made up of the coastal Mesozoic and eastern Palaeozoic belts. The Andes represent the Late Tertiary and Quaternary rejuvenation by block faulting of an eroded early Tertiary folded mountain range which occupied the axis of Palaeozoic and Mesozoic geosynclines. Topographically the mountains consist of a central dissected plateau, the Intermontane Depressions and Altiplano enclosed by narrow ranges, the Western Cordillera and the Eastern Cordillera as depicted in Figure 7-1.

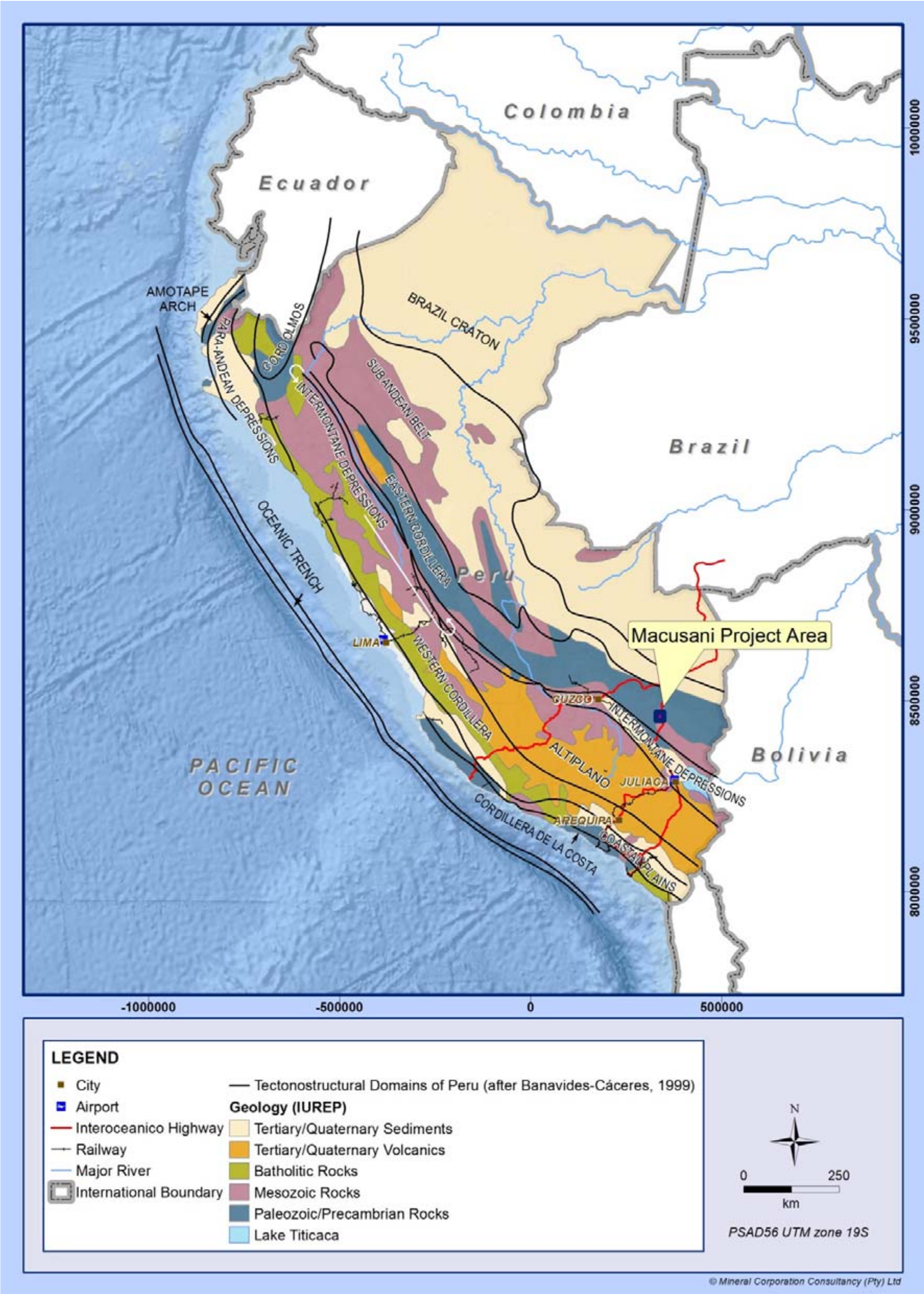


Figure 7-1: Regional Geology Plan

7.2 LOCAL GEOLOGY

In the area of interest, late Tertiary tuffs, ignimbrites and associated sediments are preserved in a northwest-southeast trending graben. Much of the Early Tertiary and Mesozoic cover was eroded prior to deposition of the pyroclastics so they were deposited in part directly on the Palaeozoic rocks including Late Palaeozoic intrusives (Hercynian granites) and extrusives (Mitu volcanics).

The known uranium occurrences in the Macusani region identified by IUREP are associated with these Pliocene and Miocene epoch Quenamari Formation tuffs, ignimbrites and interbedded sediments. Other uranium mineralisation was indicated by IUREP (1984) (18) to be hosted in acidic volcanic rocks of rhyolite composition that cover large areas of the Macusani Plateau in horizontally bedded formations from surface to a depth of about 100 m but these appeared to be lenticular or confined to fracture zones (Young, 2013) (17).

The geological map of the area (Figure 7-2) indicates that the area of interest is underlain by rocks of the Neogene Period, Quenamari Formation (dated between 22.5 Ma to 1.8 Ma). The youngest rocks (Pliocene Epoch) are known as the Yapamayo Member and these outcrop over most of the project area. The older Sapanuta and Chacacuniza Members (Miocene Epoch) underlie the Yapamayo Member. Subdivision of the members into units reflecting separate eruptions as defined by Azincourt in Henkle (2014) (14) was not completed by Global Gold's geologists.

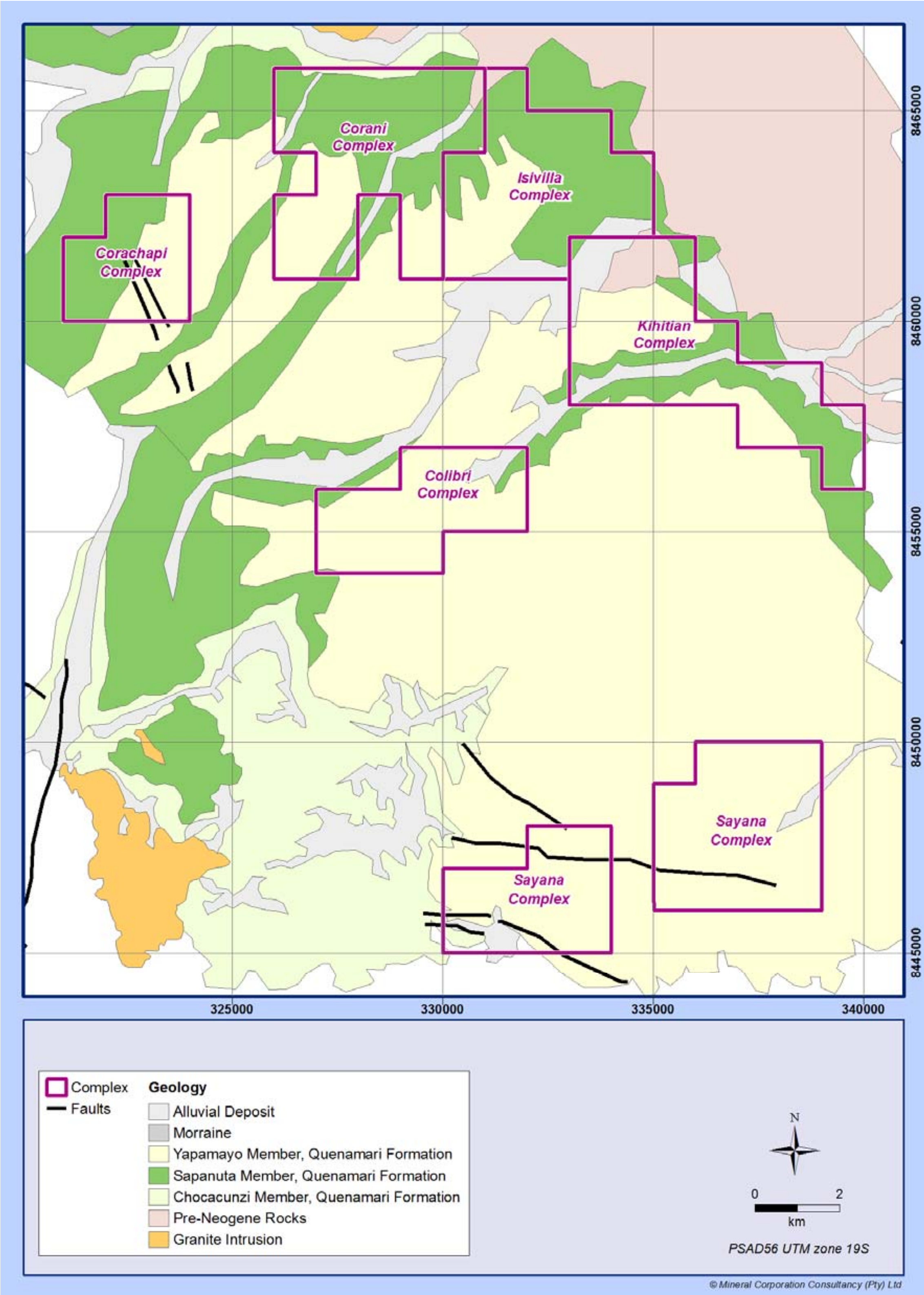


Figure 7-2: Local Geology Plan

7.3 STRUCTURAL GEOLOGY

From the published geological maps of the area structural elements can be measured from the Quenamari Formation rocks. These are mainly observed fault and bedding strike measurements. Based on a length weighting of the faults the rose diagram depicted in Figure 7-3 has been processed to indicate the main fault orientations.

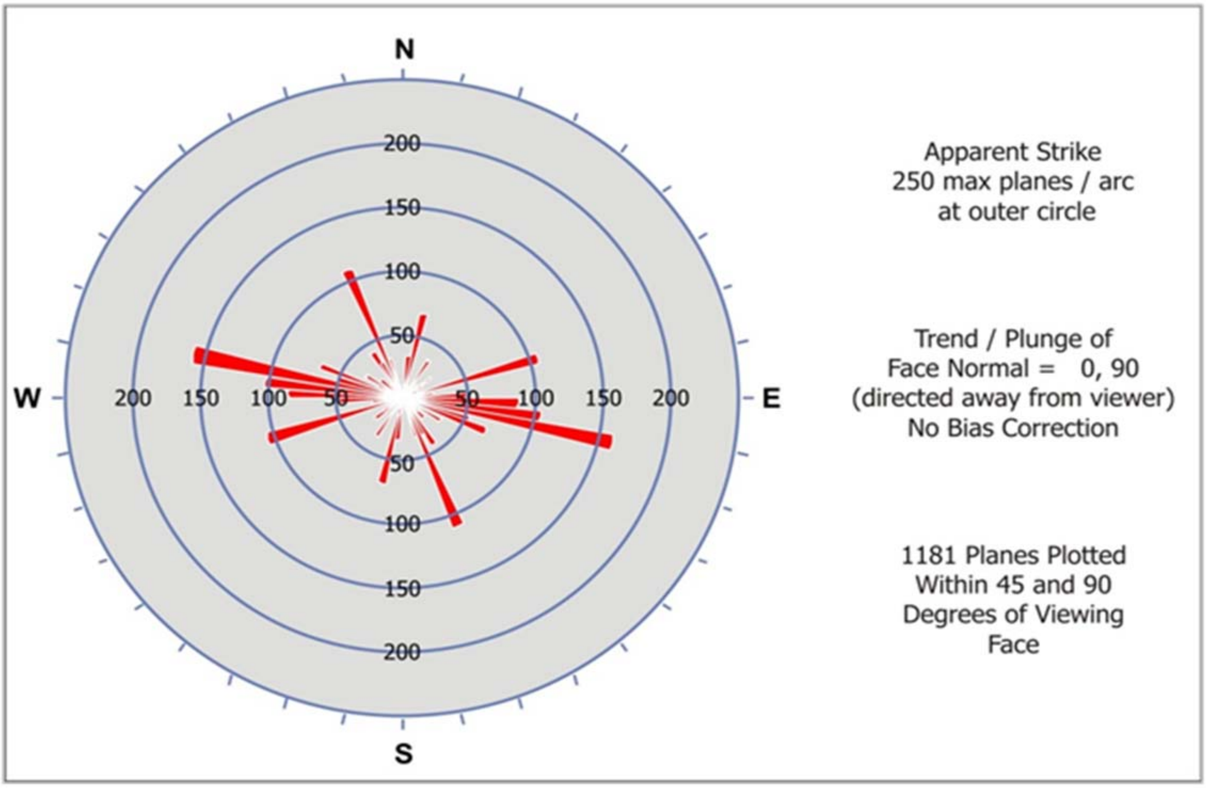


Figure 7-3: Rose Diagram Depicting the Main Regional Fault Orientations

The main fault strike orientations in order of dominance are 102.5˚, 157.5˚, 072.5˚ and 012.5˚. The rocks are noted from the regional plans to dip to the northeast (048˚) by an average of 6˚. Mineralisation is observed along fracture planes and associated dissemination into the wall rocks is observed at the Kihitian and Colibri Complexes. From a number of mineralised fracture orientations measured over these areas it can be noted that the mineralisation has preferential orientations. An analysis of the fracture orientations measured in these complexes is presented in Figure 7-4.

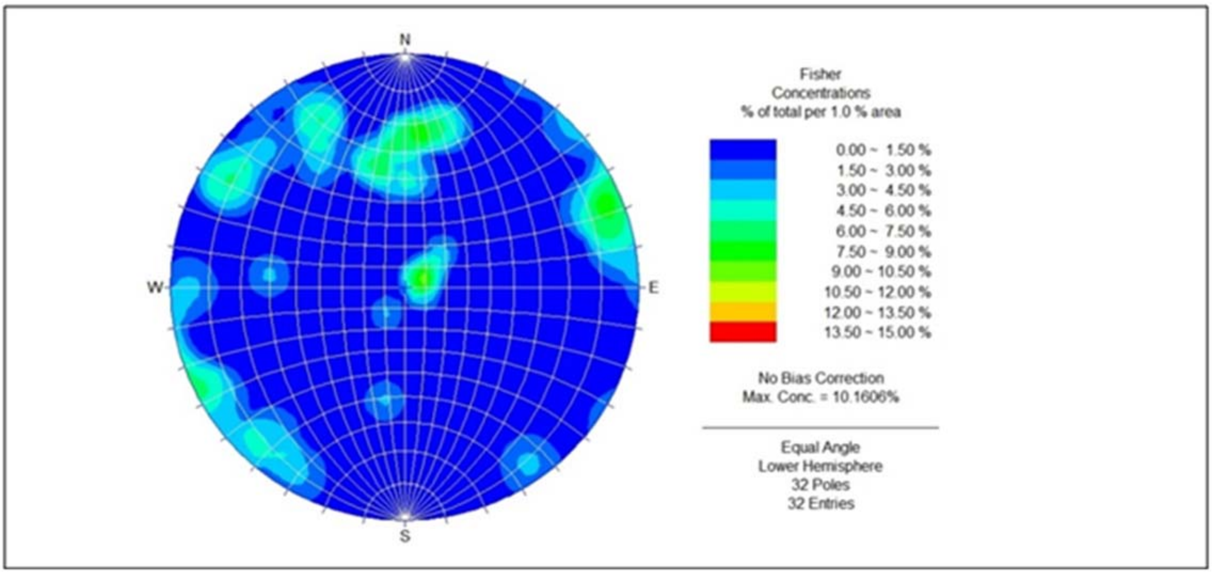


Figure 7-4: Stereogram of Mineralised Fractures

The main structural features emerging from Figure 7-4 indicate that three dominant mineralised fracture orientations emerge:

- A near vertical orientation (85° towards the southwest) striking 158°;
- A steeply dipping orientation (67° towards the south) striking 096°; and
- A flat dipping orientation (10° towards the southwest) striking 150°.

A further three subsidiary mineralised fracture orientations are noted as follows:

- A near vertical orientation (82° towards the southeast) striking 034°;
- A steeply dipping orientation (77° towards the southeast) striking 063°; and
- A near vertical orientation (85° towards the northeast) striking 133°.

Several un-mineralised fractures were measured over the course of the site visits and an analysis of these measurements is contained in Figure 7-5.

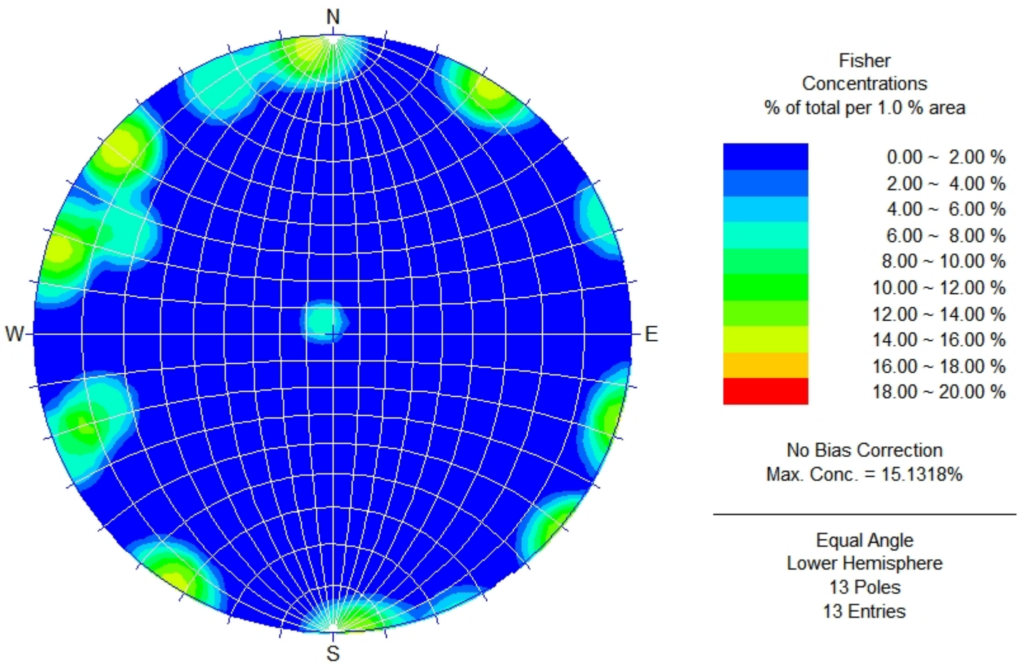


Figure 7-5: Stereogram of the Un-Mineralised Fractures

From Figure 7-5 it can be noted that four dominant and two minor un-mineralised fracture orientations are noted as follows:

- A vertical orientation striking 123°;
- A near vertical orientation (87° towards the south) striking 085°;
- A near vertical orientation (82° towards the east) striking 160°;
- A near vertical orientation (87° towards the east) striking 017°;
- A minor near vertical orientation (85° towards the southeast) striking 040°; and
- A minor flat orientation (5° towards the southeast) striking 050°.

It is possible to correlate the orientations of the faults, mineralised fractures and un-mineralised fractures into common groups as depicted in Table 7-1.

Table 7-1: Fracture Orientation Correlation

Regional Faults	Mineralised Fractures		Un-mineralised Fractures	
Strike	Strike	Dip & Direction	Strike	Dip & Direction
12.5			17	87 E
	34	82 S/E	40	85 S/E
72.5	63	77 S/E	85	87 S
102.5	**96**	**67 S**		
	133	85 N/E	123	90
157.5	**158**	**85 S/W**	160	82 E
	150	**10 S/W**	50	5 S/E

Major mineralised fractures highlighted in bold text

Based on the data in Table 7-1 it would appear that with the limited information collected there is commonality between the mineralised and un-mineralised fracture orientations and that these orientations can be correlated with the regional fault orientations.

No structural measurements at the other Complexes have been taken, but it is considered likely, given the broad similarities between the deposits, that these observations would hold true for the deposits located in the other complexes.

7.4 PROJECT GEOLOGY

7.4.1 INTRODUCTION

The host rocks and PEM are composed of an acidic tuff (Thatcher, 2011) (19) with pyroclasts of size 60 mm to sub-macroscopic. The main minerals constituting the tuff are quartz, orthoclase and plagioclase in a groundmass of amorphous glass. Crude bedding is evident in some outcrops, and is based on “strata” containing larger and smaller pyroclasts. Overall the uranium mineralisation is interpreted to be hosted in a flat dipping acidic tuff.

Uranium mineralisation is observed in these pyroclastic host rocks, and at a local scale is found concentrated along fractures and disseminated into the surrounding host rock. Zones in which the uranium mineralisation (either fracture and / or dissemination) is more concentrated, are identifiable by analysing uranium distribution profiles in drillhole core, and are occasionally observable in outcrop. These mineralised zones, referred to locally as “Manto’s”, typically have a horizontal or sub-horizontal orientation, and can vary from several metres to tens of metres in thickness.

Various samples of rhyolite outcrop with differing amounts and styles of mineralisation were taken on the 2008 and 2011 site visits, from the Colibri Complex and Kihitian Complex respectively, and analyses via optical petrography and under a scanning electron microscope were undertaken

(Thatcher, 2008 (20) and 2011 (19)). The visible uranium mineralisation present was found to be in the form of meta-autunite [$Ca(UO_2)_2(PO_4)_2.6\text{-}8H_2O$], a bright yellow mineral. No other forms of uranium mineralisation were identified and a black amorphous mineral, sometimes associated with the meta-autunite and previously considered to be pitchblende, was found to be a manganese oxide of the romanechite-cryptomelane-hollandite group. The biotite present was found to be a primary constituent of the acid volcanic, and not a product of subsequent alteration.

The petrography of the samples analysed by Thatcher (2011) (19) indicates that the acid volcanics (crystal lapilli tuffs) can have varying composition from rhyolite to dacite to latite which supports the likely presence of stratigraphic layering of the volcanic pile as noted in Section 7.2 and by Cheilletz et al (1992) (21).

7.4.2 CORACHAPI COMPLEX

TMC undertook Mineral Resource estimates for the Corachapi deposit in 2010, and at that time, mineralisation was interpreted to be hosted in a single, sub-horizontal mineralised zone, which was oriented in a northeasterly direction.

7.4.3 COLIBRI COMPLEX

Mineralisation at the Colibri II & III and Tupurumani deposits is interpreted to be hosted in a sub-horizontal, near surface zone, with a dip of approximately 2° - 3° to the northeast. The base of the near surface high-grade zone of mineralisation has a depth of approximately 35 m below surface at Tupuramani and 50 m below surface at Colibri II & III. Lower grade mineralisation is found below the base of the high-grade zone.

7.4.4 KIHITIAN COMPLEX

Within the Kihitian Complex, mineralisation is broadly contained within two distinct zones, referred to as Level A, an upper horizon and Level B, the lower horizon. A dip of 3° towards the southeast is interpreted. Most of the mineralisation for the Complex lies in Level B, while Level A has more sporadic and less continuous mineralisation. The Chilcuno Chico and Tantamaco deposits host both zones while at Quebrada Blanca the mineralisation occurs stratigraphically in Level B and the sparse nature of drilling on the Tuturumani deposits enabled the definition of only the Level A. The Tuturumani deposit has been minimally explored, thus Level B has not been fully tested by drilling.

The contact between Level A and Level B zones has a characteristic grade spike which has been consistent but not continuous across the four deposits. The depth of mineralisation is related to the topography with maximum depths being 200 m and 260 m for Level A and Level B respectively.

7.4.5 ISIVILLA COMPLEX

Uranium mineralisation at the Calvario I, Puncopata, Calvario Real and Isivilla deposits is evident in two sub-parallel, near-horizontal zones, locally referred to as levels, separated vertically by 30 m of barren rhyolite.

At the Isivilla and Calvario Real deposits, two distinct zones referred to as Level A and Level B have been identified. The two levels have a highly diffuse contact with an irregular rhyolite parting. Both levels have been noted to outcrop. Although there are localised variations, generally the contact lies flat dipping up to a maximum of 4° to the northeast. The maximum depth of Level A is approximately 60 m and that of Level B approximately 130 m. During correlation, mineralisation in the Puncopata deposit was noted to be localised exclusively to Level B while in contrast, at the Calvario I deposit, mineralisation is localised exclusively to Level A horizon.

At the Calvario I and Puncopata deposits, the flat-lying interpretation of the mineralised levels is supported by radial radiometric anomalies which encircle the hilltop at the same elevation as the mineralised levels have been interpreted to outcrop (Figure 10-5). Although similar radiometric survey data is absent for the Calvario Real and Isivilla deposits, the extrapolation of the elevation of mineralisation coincides with the elevations as identified from the radiometric data available at the Calvario I and Puncopata deposits.

The uranium mineralisation occurrence is highly isolated although correlatable with the mineralisation in the Puncopata deposit to the north and Isivilla deposit to the south. Mineralisation occurs at variable depths from 30 m to 80 m.

7.4.6 CORANI COMPLEX

The Corani Complex is also interpreted to have two levels, Level A and Level B. These have an approximate dip / dip direction of 3°/157°. At the Calvario III and Nueva Corani deposits, mineralisation occurs within Level A, with less continuous mineralisation in Level B. At the Calvario II deposit, only Level B has been preserved, and Level A had been eroded.

7.4.7 SAYAÑA COMPLEX

The local dip / dip direction within the Sayaña Complex is 5°/117°. Two layers are identified in the geological logging information provided, comprising a polymictic rhyolite upper horizon and a monomictic rhyolite lower horizon (Henkle, 2014) (14). Analysis of the intersections revealed sporadic mineralisation in the Agaton and Sayaña West deposits while Sayaña Central has more continuous mineralisation with thicknesses of about 5 m to 10 m.

7.5 MINERALISATION MODEL

The following observations by TMC need to be taken into consideration in developing a working model for the uranium mineralisation genesis:

- The main type of uranium mineralisation present is meta-autunite where the uranium is in a U^{6+} state;
- Uranium mineralisation occurs along fracture planes that are related to the regional fault orientations;
- Uranium mineralisation also occurs in disseminated form away from fractures;
- Not all fracture planes related to the regional fault orientations are mineralised;
- Mineralisation is concentrated in sub-horizontal zones;
- The uranium mineralisation event was associated with a manganese oxide; and
- The uranium mineralisation is not associated with the normal hydrothermal co-metal mineralisation.

The uranium mineralisation model envisages explosive volcanism of material of differing acidic composition producing a stratified volcanic pile and that mineralisation is concentrated in sub-horizontal zones within these strata. Mineralisation resulted from dissolution of uranium and re-concentration through precipitation controlled by these stratigraphic compositional variations, open fractures and interaction with meteoric waters.

Recent work by Li et al. (2012) (22) would suggest that the uranium mineralisation was formed by leaching of volcanic glass, apatite and monazite, transported as uranyl phosphate complexes and precipitated as autunite ($Ca(UO_2)_2(PO_4)_2.6\text{-}8H_2O$) and subordinate weeksite ($K_2(UO_2)(Si_2O_5)_3\ 4H_2O$) in fractures forming in response to tectonic uplift. U-Pb ages of the autunite indicate initiation of this mineralisation at circa 1 Ma, long after the cooling of the last volcanic eruptions and promote a genetic model that relies on an inter-play of the geomorphology, groundwater movement and its evaporation.

The model by Li *et al*, 2012 (22) represents an evolution in the mineralisation model presented by (Young, 2011) (23) to the current model by Li (2012) (22). The implication for the Mineral Resources and the exploration programme due to this revised model is that confidence in the lateral continuity of the mineralised zones is enhanced.

SECTION 8 DEPOSIT TYPES

The style of mineralisation within fractured acidic pyroclastics is not a common form of uranium mineralisation. The main uranium mineral present is meta-autunite concentrated as disseminations and sometimes massively along fractures. Hence the exploration is based on ground radiometrics followed by evaluation drilling over the potential host rocks of the mineralisation.

SECTION 9 EXPLORATION

Uranium exploration activities in Peru were initiated on the back of the work of IPEN in the 1970s and 1980s. Uranium anomalies were found near Macusani in the Upper Tertiary volcanics and the Permian Mitu Group by the UNDP/IAEA project.

The typical exploration rationale for the Macusani region involves the delineation of potential uranium anomalies through a combination of regional geological interpretation and surface radiometric techniques in order to delineate targets for further investigation through drilling.

Global Gold has conducted ground-based radiometric surveys from a hand-held scintilometer over large portions of its properties (including Colibri II & III and Kihitian; and supported by the previously drilled data from Triunfador I by Frontier Pacific) as a guide for its drilling programmes.

During 2006 and under the management of Vena, radiometric readings from a hand-held scintillometer at 1 785 individual stations as well as 564 Alpha cup readings of radon gas were collected from their concessions. In addition, 44 surface samples were collected from various IPEN showings for target identification and 169 petrographic samples were prepared and examined. (Henkle, 2014) (14). In 2007, Minergia collected 10 301 additional radiometric readings at various concessions in the Macusani area for target identification. In addition, 14 petrographic samples were prepared and examined. The 2009 - 2010 drilling campaign resulted in the completion of 65 additional drillholes and a total of 12 316.8 m of drilling. In addition 155 petrographic samples were prepared and examined. The 2011 drilling campaign resulted in the completion of 62 additional core holes for a total of 11 107 m. The total amount of drilling undertaken on the Minergia properties since 2006 is 232 core holes for a total of 37 958 m (Henkle, 2014) (14).

The current areas explored via drilling and sampling that have allowed delineation of Mineral Resources are relatively small compared to the total Plateau Uranium mining concession footprint (Figure 23-1) that is underlain by the Quenimari Formation rocks (Figure 7-2). Further radiometric surveys and geological interpretations of the un-explored mining concession areas are required to delineate potential additional targets for drilling and sampling.

SECTION 10 DRILLING

The drilling in the Complexes over the deposits was mainly done from a series of platforms, with anything from five to nine drillholes being drilled radially from each platform due to drill site access limitations as illustrated in Figure 10-1. Some deposits, for example Tantamaco and Corachapi, were drilled from a series of regularly spaced drill lines of single holes.

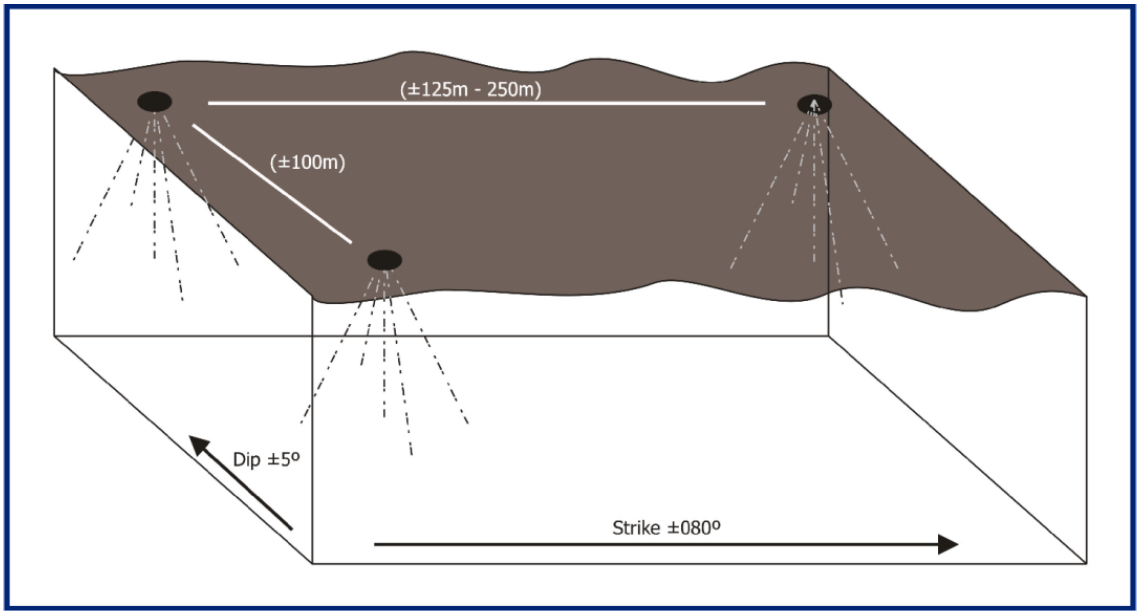


Figure 10-1: Schematic of Drilling Methodology

10.1 CORACHAPI COMPLEX

10.1.1 DRILLING PROGRAMME

Diamond drilling was initiated in 2007 by Contact Uranium over the Corachapi Complex, on target areas identified by ground radiometric surveys as illustrated in Figure 10-2. Contact Uranium drilled 193 drillholes amounting to a metreage of more than 10 000 m. Global Gold drilled an additional 26 drillholes bringing the total drilling to 11 818 m from 219 drillholes as detailed in Table 10-1.

Table 10-1: Drilling Summary – Corachapi Complex

Deposit	Drillholes	Total Drilled (m)	Total Sampled (m)
Corachapi	219	11 818	10 920

The results of 210 of these 219 holes were employed in the estimation of the Corachapi Mineral Resources.

To date, Corachapi has been explored via diamond core that has been cut longitudinally to provide a sample for analysis and one for storage by Contact Uranium. The half core samples have been crushed, pulverized and analysed for uranium abundance. Contact Uranium employed the ALS Chemex laboratory in Lima for their samples and the subsequent samples generated by Global Gold were analysed at the CIMM laboratory (CIMM) in Lima. CIMM erected a sample crushing and pulverizing facility in Juliaca, thus reducing the amount of material that had to be transported to Lima. Global Gold's practise on their newly drilled core is to take whole core samples.

The sampling and analytical QA/QC results for both laboratories were scrutinised and the analytical results to date, were considered acceptable to be employed for the estimation of Mineral Resources.

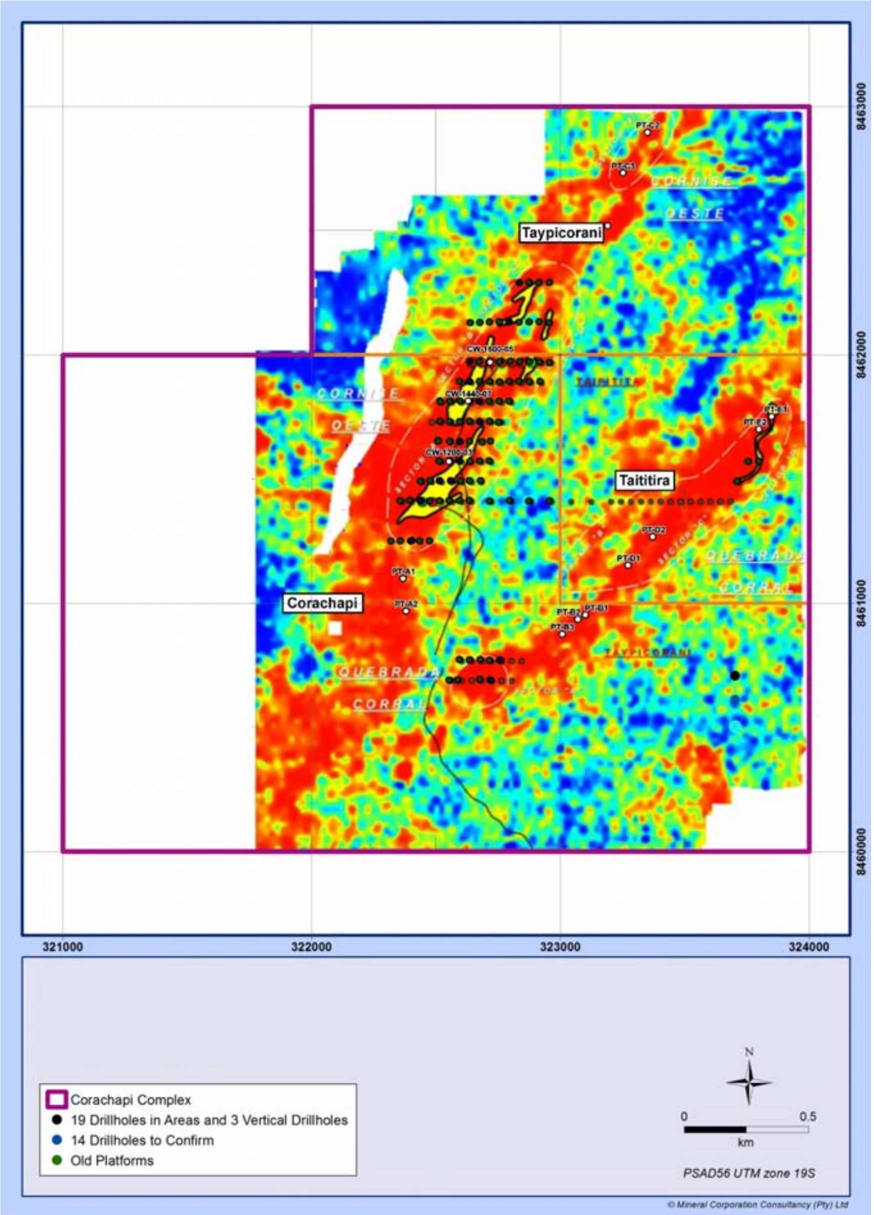


Figure 10-2: Radiometric Survey with Drillholes (source: Global Gold)

10.1.2 DRILLING METHODOLOGY

Contact Uranium drilled the drillholes on 80 m or 160 m spaced east-west lines. The spacing of the holes on the lines was 40 m with generally, two holes drilled from each location, one dipping 50° east and the other 50° west. The Global Gold drilling brought the total drilling to 11 818 m from 219 drillholes. Of these holes, 210 have been chemically analysed, 107 holes were analysed by Contact Uranium and 103 holes were analysed by Global Gold. One hole was analysed by both companies.

10.1.3 SAMPLE RECOVERY AND CORE

Core recovery was reported to be generally over 85 - 90 %.

For additional information, refer to Young (2010[3]) (24).

10.2 COLIBRI COMPLEX

10.2.1 DRILLING PROGRAMME

The diamond drilling was executed by Minera Colibri, a drilling contractor. The Mineral Resources for the Colibri II & III and Tupuramani deposits are based on 149 diamond drillholes, which represent some 12 673.2 m of drilling from 32 platforms. The platform locations are shown in Figure 10-3 while the drilling statistics are shown in Table 10-2.

Table 10-2: Drilling Summary – Colibri Complex

Deposit	Drillholes	Total Drilled (m)	Total Sampled (m)
Colibri II & III	127	8 417.2	11 341
Tupuramani	22	4 256	4 195
Total	**149**	**12 673.2**	**15 536**

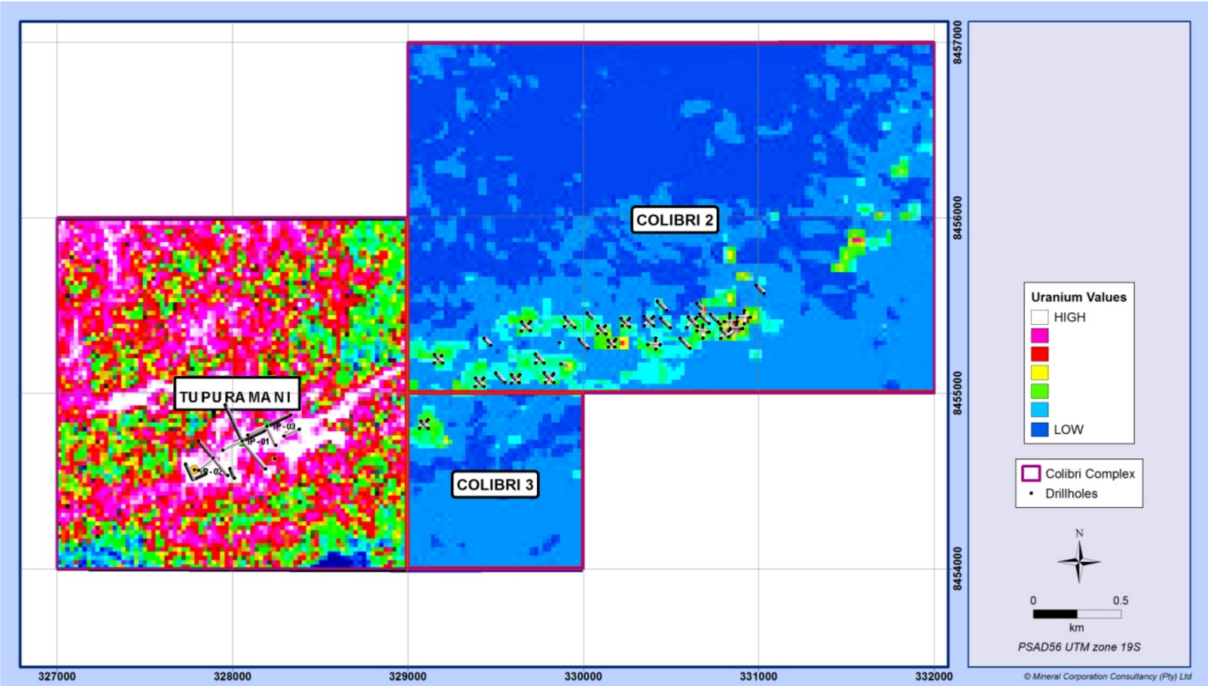


Figure 10-3: Colibri II & III and Tupuramani Radiometrics and Drilling Locations

10.2.2 DRILLING METHODOLOGY

Drillholes were typically drilled from platforms which range from 125 m to 250 m apart. From each platform, a series of drillholes were drilled; one vertical hole and up to four inclined holes. The inclined holes were usually drilled at an angle of 55° from the horizontal and at right angles to each other.

The mineralisation is interpreted to be constrained by flat lying zones which dip gently to the northeast, and as a result the orientation of drilling relative to the mineralisation is different in each direction. This was accounted for in the Mineral Resource estimates by calculating composites which are of equal length relative to the plane of the mineralised zone.

10.2.3 SAMPLE RECOVERY AND CORE

Drilling conditions in the rhyolites are good, and sample recovery is typically close to 100 %.

10.3 KIHITIAN COMPLEX

10.3.1 DRILLING PROGRAMME

Diamond drilling programmes were initiated in 2007 by Global Gold over the Chilcuno Chico and Quebrada Blanca deposits and by Minergia over the Tantamaco and Tuturumani deposits, on target areas identified by ground radiometric surveys. Global Gold drilled 136 drillholes and Minergia drilled 139 drillholes. The drilling summary is shown in Table 10-3.

Table 10-3: Drilling Summary - Kihitian Complex

Deposit	Number of Drillholes	Length of Drilling (m)	No. of Samples
Chilcuno Chico	94	19 060	12 045
Quebrada Blanca	42	5 285	6 042
Tantamaco	128	23 286	8 283
Tuturumani	11	2 477	525
Total	**275**	**50 108**	**26 895**

10.3.2 DRILLING METHODOLOGY

Global Gold drilled 94 drillholes radially from 32 platforms and 42 drillholes radially from 14 platforms for the Chilcuno Chico and Quebrada Blanca deposits respectively, with up to five holes drilled per platform in order to maximise data gathered, as there were surface access limitations. Minergia drilled 128 drillholes in systematic drill lines spaced 100 m apart with the majority being angled at 45° to the east. Eleven exploratory holes were drilled at Tuturumani from a combination of drill lines and platforms. To date, 50 108.5 m of drilling was completed on the four deposits.

The drillhole spacing for the Chilcuno Chico and Quebrada Blanca deposits which were drilled radially resulted in mineralised zone intersection separation distances of up to 175 m. The Tantamaco deposit, which was drilled more linearly, had a series of drill lines 100 m apart and the collar spacing on each drill line being generally 100 m in the north and 200 m in the south of the deposit. The Tuturumani deposit was drilled sparsely.

The platform and drillhole locations are shown in Figure 10-4.

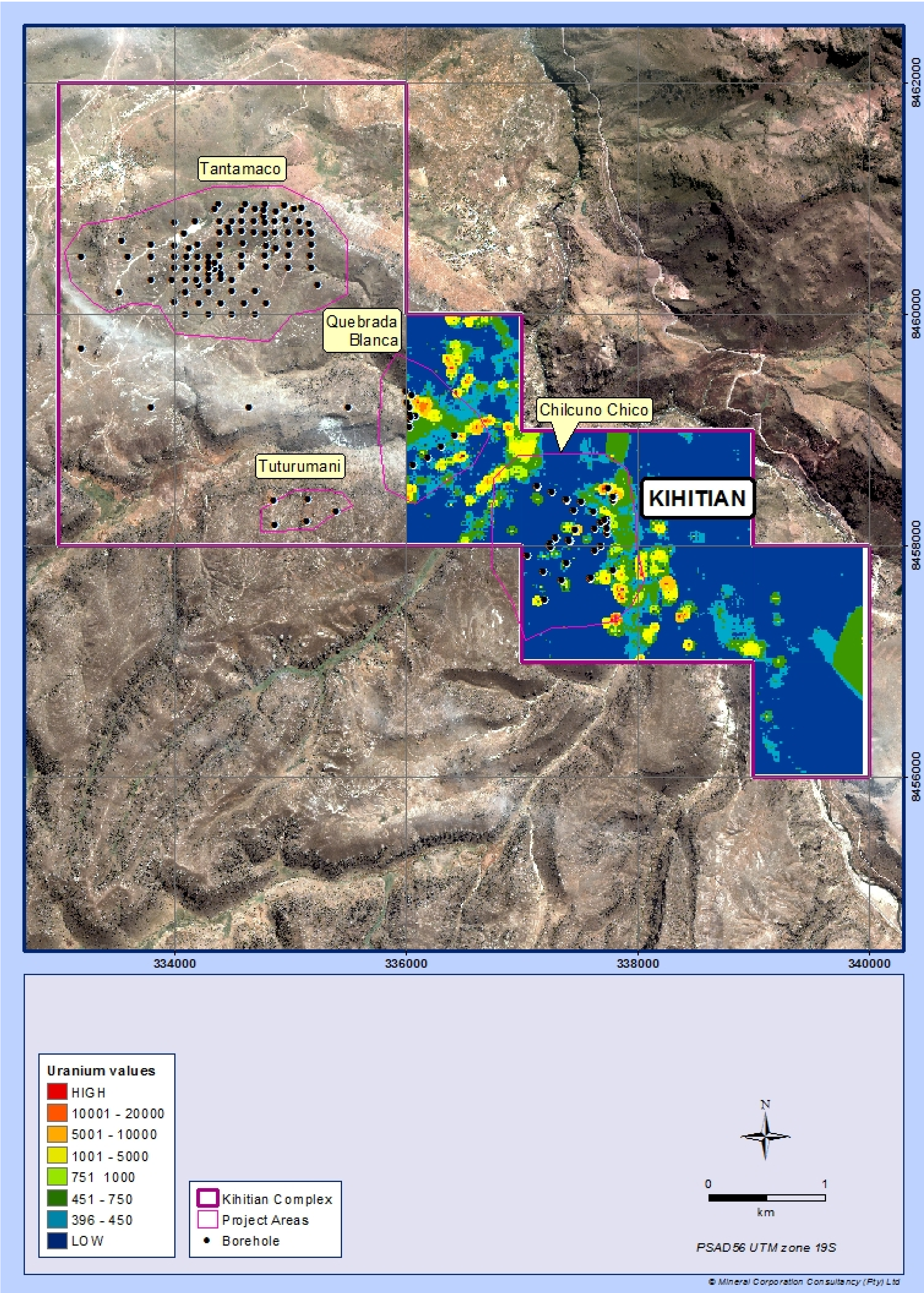


Figure 10-4: Drilling Plan and Available Ground Radiometrics - Kihitian Complex

10.3.3 SAMPLE RECOVERY AND CORE

Drilling conditions at the Chilcuno Chico and Quebrada Blanca deposits were good with core recoveries typically close to 100 % as reported in Young (25). Core recoveries for the Tuturumani and Tantamaco deposits exceeded 95 % (14).

10.4 ISIVILLA COMPLEX

10.4.1 DRILLING PROGRAMME

The original drilling programme for the Puncopata and Calvario I deposits was undertaken by Frontier Pacific. A total of 96 diamond drillholes have been completed from 5 and 13 platforms at the Puncopata and Calvario I deposits respectively.

Within the Calvario I deposit, a total of 58 validated drillholes were accepted by Global Gold. Within the Puncopata deposits, a total of 34 drillholes were accepted.

The Calvario Real prospect was evaluated through 1 628 m of drilling in 9 drillholes. The drilling summary is shown in Table 10-4.

Table 10-4: Drilling Summary – Isivilla Complex

Deposit	Drillholes	Total Drilled (m)	Total Sampled (m)
Calvario I	58	3 857	3 856.72
Puncopata	34	2 261	2 260.00
Calvario Real	9	1 628	1 000.40
Isivilla	27	3 597	3 580.50
Totals	**128**	**11 343**	**10 697.62**

Prior to 2011, only minimal drilling of the Isivilla deposit comprising 7 drillholes totalling 617 m had been carried out. In 2011, a total of 2 980 m was drilled through 20 additional holes. This drilling identified mineralisation that stretches through the southern and central parts of the Isivilla prospects from surface to a depth of 80 m.

10.4.2 DRILLING METHODOLOGY

As with other complexes, the drilling takes place from a series of platforms resulting in separation distance between intersections of the mineralised zones between 100 m and 250 m. The drillhole locations are illustrated in Figure 10-5.

10.4.3 SAMPLE RECOVERY AND CORE

Solex documented core recovery statistics, and as with the other deposits, drilling conditions are likely to have been good, with core recovery very close to 100 % (17). For the Isivilla deposit core recovery exceeded 95 % (Henkle, 2011) (26).

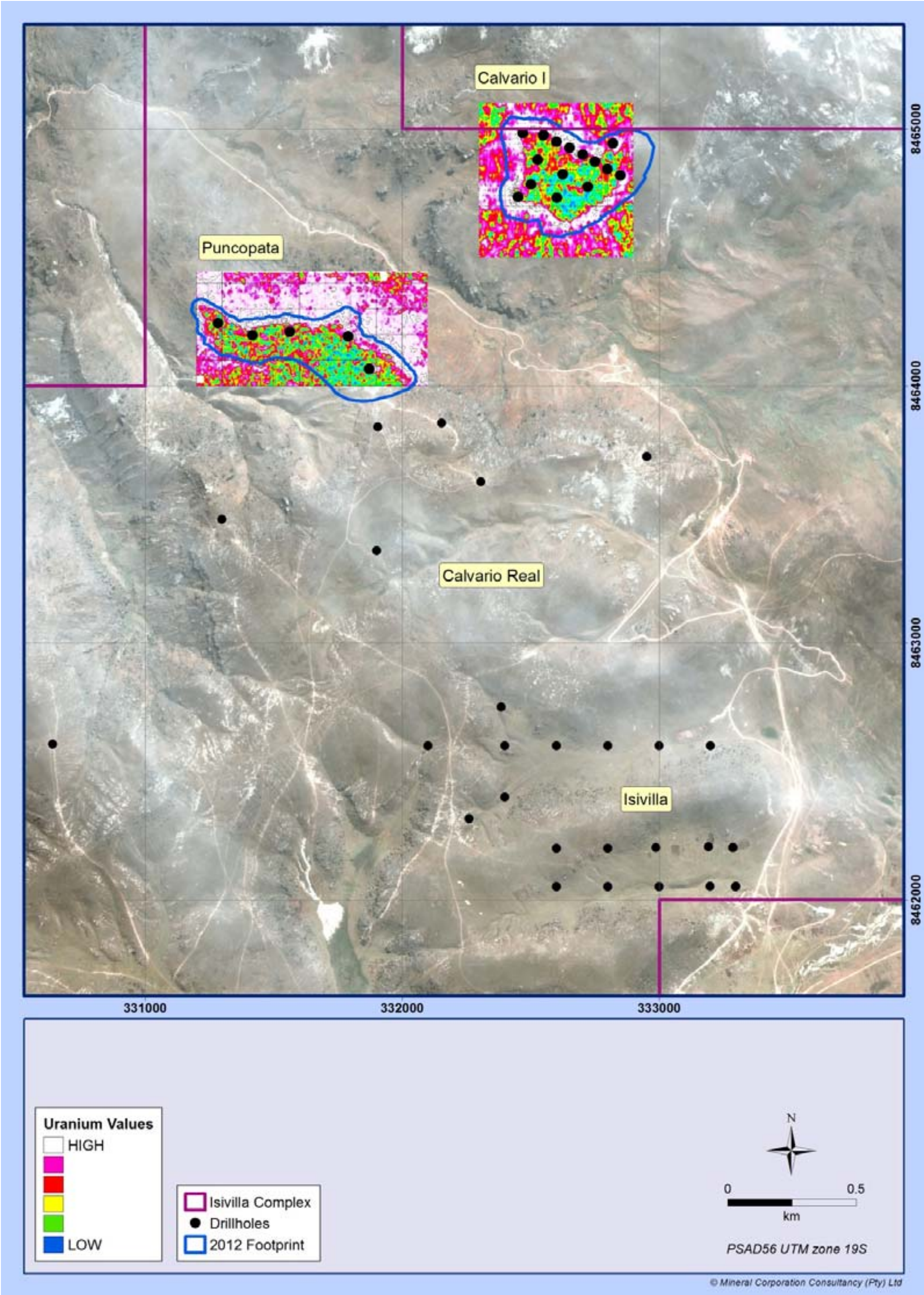


Figure 10-5: Drilling Plan and Available Ground Radiometrics – Isivilla Complex

10.5 CORANI COMPLEX

10.5.1 DRILLING PROGRAMME

Diamond drilling programmes were initiated in 2006 by Vena at the Nueva Corani deposit, totalling 679 m, then from 2007 to 2010 by Minergia, drilling 48 diamond holes totalling 6 282 m. The drilling on the Calvario II and Calvario III deposits was undertaken by Frontier Mining. The drilling summary is provided in Table 10-5.

Table 10-5: Drilling Summary - Corani Complex

Deposit	Drillholes	Total Drilled (m)	Total Sampled (m)
Calvario II	32	2 433	2 425
Calvario III	85	5 425	5 398
Nueva Corani	57	6 770	3 450
Total	**174**	**14 628**	**11 273**

10.5.2 DRILLING METHODOLOGY

At the Calvario II deposit a total of 32 drillholes were drilled from eight platforms, each platform had four drillholes drilled radially in the azimuth directions of 45°, 135°, 225° and 315° as schematically depicted in Figure 10-1. All the drillholes had a dip of 50°. At the Calvario III deposit, a total of 85 diamond holes were drilled totalling 5 425 m. In total, 174 diamond drillholes have been drilled over the three deposits totalling 14 628 m.

10.5.3 SAMPLE RECOVERY AND CORE

Drilling conditions at the Calvario II and Calvario III deposits are not documented. This will have an impact on classification as discussed later in Section 14.6.9. The core recoveries for the Nueva Corani deposit exceeded 95 % (14).

10.6 SAYAÑA COMPLEX

10.6.1 DRILLING PROGRAMME

Diamond drilling programmes were conducted by Solex over the Agaton, Sayaña West and Sayaña Central deposits. A total 180 drillholes were drilled as detailed in Table 10-6.

Table 10-6: Drilling Summary - Sayaña Complex

Deposit	Drillholes	Total Drilled (m)	Total Sampled (m)
Agaton	52	2 301	726.8
Sayaña Central	94	8 464	8 371.5
Sayaña West	34	2 244	1 826.6
Total	**180**	**13 009**	**10 924.9**

10.6.2 DRILLING METHODOLOGY

Solex drilled 52 drillholes radially from 12 platforms, 94 drillholes radially from about 20 platforms and 34 drillholes radially from 9 platforms for the Agaton, Sayaña Central and Sayaña West deposits respectively, as schematically depicted in Figure 10-1. Up to five holes have been drilled per platform in order to maximise data collection, as there were surface access limitations. To date, 13 009 m of drilling has been completed on the three deposits.

10.6.3 SAMPLE RECOVERY AND CORE

No data was available regarding core recoveries, thus the Qualified Person cannot give a statement in this regard. This will, however, have an effect on the classification as discussed in Section 12.6.3.

SECTION 11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 INTRODUCTION

The data which informs the Mineral Resource estimates can be grouped into three populations with respect to the sample preparation, analyses and security protocols. The first population of data is that which has been generated exclusively through the exploration efforts of Global Gold (the Global Gold data). The second population is that which has been generated through the exploration of Minergia (the Minergia data), and acquired by Plateau Uranium in the merger between Macusani Yellowcake Inc. and Azincourt. The third population is that data which was obtained through the merger with Contact Uranium (the Solex data). The description of the sample preparation, analyses and security protocols have thus been grouped in the same way. Table 11-1 is a summary of the data groupings which contribute to the various deposits, and hence the Complexes.

Table 11-1: A Summary of the Data Groupings which Contribute to the Various Deposits

Complex	Deposit	Data Grouping
Corachapi	Corachapi	Solex and Global Gold
Colibri	Colibri II & III	Global Gold
	Tupuramani	Global Gold
Kihitian	Chilcuno Chico	Global Gold
	Quebrada Blanca	Global Gold
	Tantamaco	Minergia
	Tuturumani	Minergia
Isivilla	Calvario I	Solex
	Calvario Real	Minergia
	Puncopata	Solex
	Isivilla	Minergia
Corani	Calvario II & III	Solex
	Nueva Corani	Minergia
Sayaña	Sayaña West	Solex
	Sayaña Central	Solex
	Agaton	Solex

As the Mineral Resources for the Corachapi and Colibri Complexes are unchanged the specific information relating to the sample preparation, analysis and security can be found in the report references Young (2010[3]) (24) and Young (2013) (17) respectively.

11.2 GLOBAL GOLD DATA

The sample preparation, analysis and security protocols for the Global Gold data are pertinent to the Colibri II & III, Tupuramani, Chilcuno Chico and Quebrada Blanca deposits is included in this section.

11.2.1 SAMPLING METHODS

Sampling was from the whole core over the entire length of the drillhole. Individual samples varied from a minimum of 0.25 m to a maximum of 2.0 m with a mean of 0.9 m. Selection of the length to sample was based on visual observation of the mineralisation and assisted by radiometric measurements.

Core from these deposits was scrutinised on the 2010 and 2013 visit, and in both cases the quality of the core recovered was very good.

11.2.2 SAMPLING RECOVERY

The procedure of measuring the core recovery as the core was taken from the core barrel was observed at Colibri II & III, where it would appear that in one case the drillers overestimated the depth of the hole by 4 cm over a run of 1.49 m and in another case underestimated the depth of the hole by 5 cm. The perception gained by scrutiny of the limited core available on site was that although the core could in some cases be somewhat friable, the core recovery was very good and the core pieces fitted together very well in the core boxes prior to sampling.

Another global measure of the sample recovery for the Colibri II & III drilling was provided by the total drilling metres (8 254.87 m) and the total metres sampled and un-sampled (8 103.91 m + 150.96 m = 8 254.87 m) that yielded a global core recovery of 98.17 %.

During the 2013 visit to the Kihitian Complex core from drillhole QB46 was scrutinised and over a drilling run of 36.0 m, 35.19 m of good fitting core was observed. This would indicate a core recovery of 97.75 %.

During the 2013 visit to Tupuramani, TMC logged the core recovery in drillholes TP11 (Vertical) and TP12 (NW) based on the drillers blocks. A recovery of 103 % was noted for a total length of 16.5 m of core that indicates minor errors in the driller’s depth measurements.

It was observed that the nature of the mineralisation, particularly near mineralised fractures, is such that uranium minerals are likely to be washed away while drilling. This occurrence is unlikely to result in a significant drop in core recovery, as the overall integrity of the length of core is maintained. An analysis of the U grade versus core recovery did not reveal any bias related to core recovery.

11.2.3 SAMPLE QUALITY

As the entire core was sampled, the sample taken from the core box is considered representative. The mineralised fractures are friable and it is considered that splitting with a diamond saw would unacceptably jeopardize the integrity of the sample. Thus the method of sampling the whole core is sound, even though no intact library sample was retained. However, a comprehensive photo archive is retained.

11.2.4 SAMPLE PREPARATION, ASSAYING AND ANALYTICAL PROCEDURES

11.2.4.1 SAMPLE PREPARATION

Sample preparation occurred on site at a mobile field station which was located close to the drill rigs and relocated periodically. Once logged and photographed, the entire core identified for sampling was placed into a sampling bag as illustrated in Figure 11-1. The pre-marked aluminium tag was stapled to the sample bag.

Sample depths were recorded together with a basic geological description on a sampling reconciliation log. This log was later captured into an Excel spreadsheet.

Quality control samples in the form of standards were inserted at the permanent field office located in the village of Isivilla. These standards, as well as others, were prepared by Global Gold and certified by ALEPH Group & Asociados S.A.C., Metrologia de las Radiaciones (Radioactivity Measuring Techniques) by having check analyses of the standards completed at the CIMM laboratories in Lima.

Figure 11-1: Whole Core Samples are Bagged on Site

11.2.4.2 SAMPLE DELIVERY PROCEDURES

The complete sample batch, accompanied by a senior representative of the Global Gold exploration team, was sent by road to the town of Juliaca. The samples that were to be analysed at the SGS laboratory in Lima were then dispatched by road to Lima, again accompanied by a senior Global Gold geologist. (It was essentially CIMM's ability to accept core samples in Juliaca that accounted for the decision to use the CIMM laboratory for the analysis of core from subsequent drilling campaigns).

From the preparatory laboratory in Juliaca, the pulverized samples were transported by CIMM to the main CIMM laboratory in Callao, on the outskirts of Lima by either road or as air freight. The CIMM Callao laboratory represents the entry point into the CIMM LIMS system for the samples, and at this point an acid digestion was made from the pulverised sample. A detailed review of the procedures within this laboratory is included in Section 11.2.4.5.

Finally, the sample solutions were analysed at a dedicated CIMM ICP-MS facility in Miraflores, Lima. A detailed review of the procedures within this laboratory is included in Section 11.2.4.6.

TMC examined the sample receiving facilities at all three laboratories and found them to be well organised (Young, 2011) (23). It would appear that the chain of custody of the Global Gold samples from site to final analysis is reasonably secure.

11.2.4.3 SAMPLE PREPARATION AND ANALYSIS (CIMM)

Sample preparation and analysis was carried out through the CIMM Laboratory - Lima. The procedures described below are true for approximately 78 % of the sample database, the remainder of which (22 %) were processed at SGS Laboratory - Lima.

11.2.4.4 CIMM PREPARATION LABORATORY (JULIACA)

The samples were weighed on delivery and given a laboratory code. Drying was completed over a 12 hour period at 100˚C. Crushing was done by two jaw crushers; the first to 6 mm and the second to 2.5 mm. Crushing was completed when the sample was 100 % <2.5 mm. CIMM standards were entered into the stream after the first jaw crusher. The jaw crushers were flushed with quartz material; some of which were sent to the Lima offices for analysis on a regular basis.

One certified reference material, one blank sample and two duplicate samples were incorporated into each batch of 50 samples delivered to CIMM for laboratory analytical quality assurance and control (QA/QC). These results were given to Global Gold on the analysis certificates.

After homogenisation, the crushed sample was riffle split to an approximate 250 g sample that was pulverised by a ring mill. The ring mill was flushed after approximately every five samples or if there was a marked colour change in the crushed material. The preparation facility strives to have the pulverised material at 85 % <200 mesh grain size.

The jaw crushers, riffles and ring mills are all cleaned with compressed air and are located within sub-housings to keep contamination to a minimum. The reject material is kept on site but will eventually be transported to the Global Gold warehouse in Lima.

11.2.4.5 CIMM LABORATORY (LIMA – CALLAO)

The pulverised material was entered into the LIMS system and the sample was manually homogenised. Wet samples were dried before an approximate 0.20 g aliquot (±0.02 g) sample was spooned out and digested with a mixture of $HCl+HNO_3+HF+HClO_4$ acid over a period of 8 hours. The bottles of digested material and acid were dispatched to the CIMM laboratory in Miraflores, Lima.

11.2.4.6 CIMM ICP-MS LABORATORY (LIMA - MIRAFLORES)

The concentration of uranium was read from the acid digested liquid by inductively coupled plasma - mass spectrometry (ICP-MS) for abundances of 0.05 ppm to 10 000 ppm (1 %). Any results greater than 10 000 ppm were re-analysed via inductively coupled plasma - optical emission spectrometry

(ICP-OES). The latter instrument would require a new acid digest to be completed on an aliquot of 0.25 g. The ICP-MS and ICP-OES equipment is calibrated daily with three appropriate standards.

11.2.4.7 SAMPLE PREPARATION AND ANALYSIS (SGS)

In the early stages of exploration, samples were prepared and analysed at SGS Laboratories in Lima.

At SGS, the core samples undergo two-stage crushing (6 mm and 2 mm), homogenisation and a 250 g riffled representative sample is taken. This 250 g sample is pulverized until 95 % passes a 140 mesh grain size. A 20 g aliquot of pulverized sample is taken and digested in three stages; firstly by HNO_3+$HClO_4$ acid, secondly with HF acid and lastly with HCl acid. The concentration of uranium is read from the acid digested liquid by ICP-MS for abundances of 0.1 ppm to 10 000 ppm (1 %).

It is noted from the SGS analytical certificates that the only QA/QC completed by SGS is duplication of approximately 10 % of the samples delivered. Neither standards nor blanks are routinely employed by SGS.

11.2.5 ANALYTICAL QUALITY ASSURANCE AND CONTROL PROCEDURES

11.2.5.1 QA/QC RESULTS

Global Gold inserted standards, blanks and duplicates into the sampling streams, in addition to the QA/QC samples employed by both the SGS and CIMM laboratories themselves.

Table 11-2 contains the overall statistics for the QA/QC samples. A total of 4 550 QA/QC samples were submitted, representing 13 % of the total.

Table 11-2: Summary of Global Gold QA/QC Samples Submitted for Colibri and Kihitian Complexes

	Duplicates		Standards		Blanks		
No Samples	No Inserted by Owner	No Inserted by Laboratory	No Inserted by Owner	No Inserted by Laboratory	No Inserted by Owner	No Inserted by Laboratory	% QA/QC
36 069	713	1 201	565	939	632	500	12.6

11.2.5.2 STANDARD DATA

The standards employed are provided in Table 11-3.

Table 11-3: Standards Used by Global Gold for Colibri and Kihitian Complexes

Standard ID	Expected Value (U ppm)	User
GXR1	34.9	CIMM
GXR1	34.9	CIMM
GXR2	2.9	CIMM
GXR4	6.2	CIMM
ISIVILLA001	15.7	Global Gold
ISIVILLA002	19.1	Global Gold
ISIVILLA004	20.5	Global Gold
ISIVILLA005	13.8	Global Gold
LUGANILLA001	57.2	Global Gold
LUGANILLA004	29.3	Global Gold
SARM86	1 206	TMC
SARM97	101	TMC
SARM98	205	TMC
STD 120C	114.46	CIMM
STD41	3.2	CIMM
STD75	1 767	CIMM
STD74	303	CIMM
STD41	3.3	CIMM
STD 142	73.71	CIMM
STD73	47.98	CIMM

It is the preferred method of TMC to analyse results for standards by error deviation percentage charts, as a sense of proportion is gained from the differences. The definition is as follows:

$$Error\ Deviation = \frac{Xanalysis - Xstandard}{Xstandard}$$

The error deviation results from the expected value is expressed as a percentage using the formula as described above and displayed graphically in Figure 11-2 and Figure 11-3 Positive results indicate an over-estimation by the laboratory, whereas negative results indicate an under-estimation by the laboratory.

Figure 11-2 contains the Global Gold, CIMM and TMC inserted standards for analysis at the CIMM Laboratory. The error deviation dispersion is generally constrained about ±20 %. Only for standards material close to the detection limit are they significantly higher.

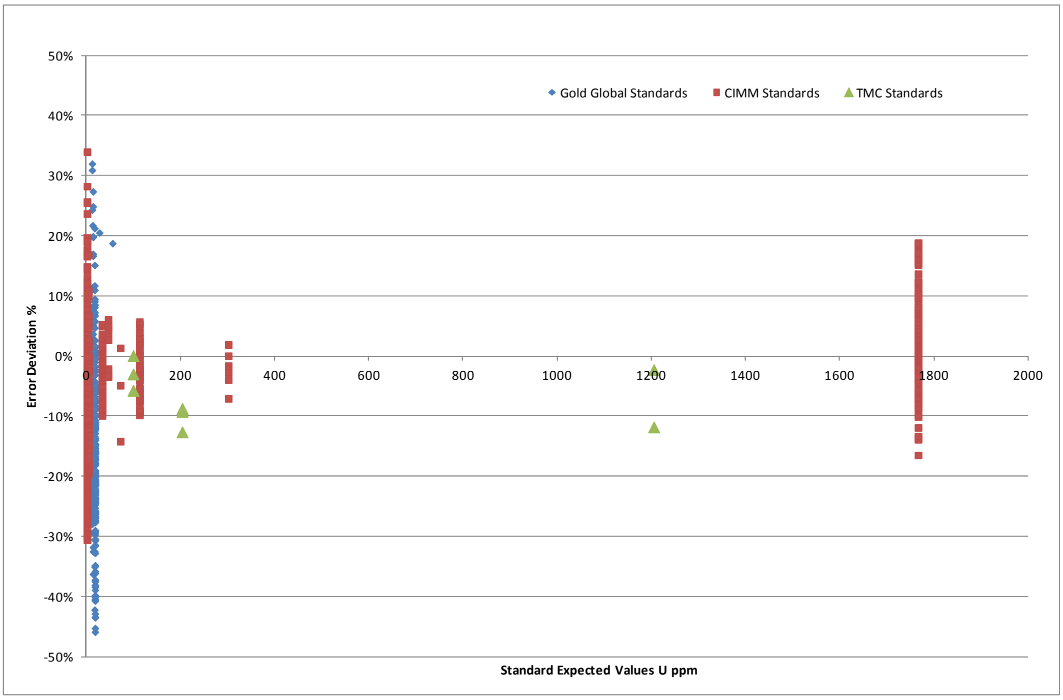


Figure 11-2: Error Deviation (%) for Standards for Global Gold Colibri and Kihitian Complexes Sampling

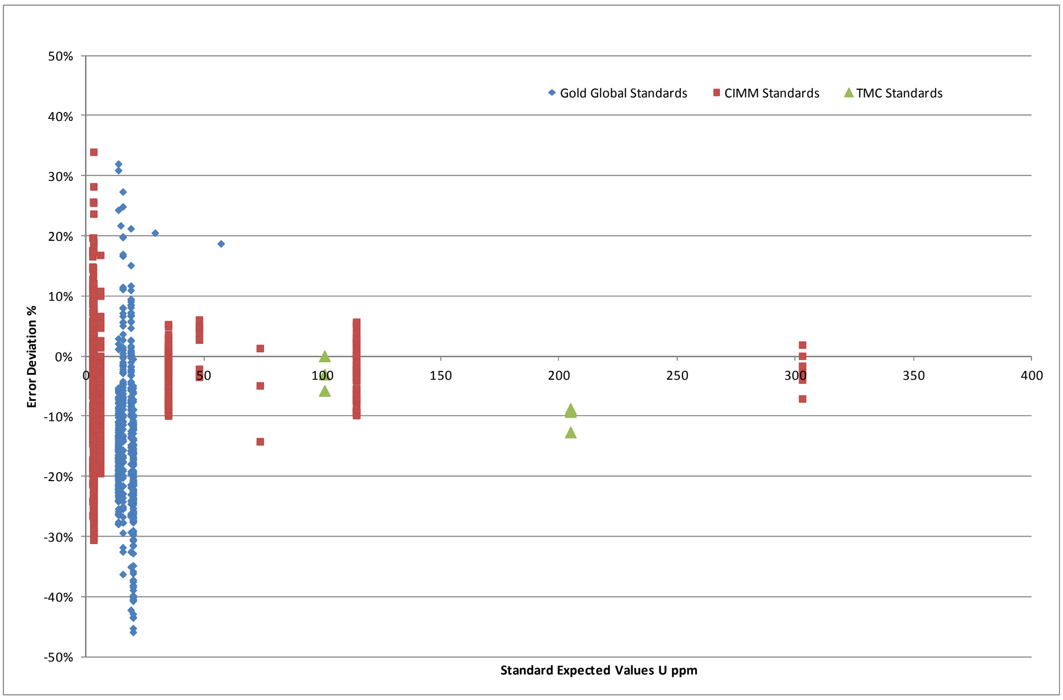


Figure 11-3: Error Deviation for Low Grade Standards for Global Gold Colibri and Kihitian Complexes Sampling

Figure 11-3 shows error deviation for the standards at lower U abundance (<400 ppm U). It can be seen, as depicted in Figure 11-2, that for both Global Gold and CIMM standards, the lower grade standards (<50 ppm) have higher deviations from the expected mean and that these results are typically under-reported.

11.2.5.3 DUPLICATE DATA

Duplicate data are analysed using the mean deviation %. This is calculated as follows:

$$Mean\ Deviation\ \% = \frac{(Xa - Xb) \times 0.5}{Mean\ (Xa, Xb)} \times 100$$

By using this convention, negative deviations are noted as under-reporting and positive deviations are noted as over-reporting of results.

Both laboratory duplicates were based on re-analysing the sample, while the CIMM laboratory also re-analysed a small percentage of pulps. Thus the laboratory duplicates are mainly a measure of the analytical error, with a small proportion being a measure of sampling error. The absolute average mean deviation for CIMM is 1.9 % and for SGS is 2.2 %. The results are shown in Figure 11-4.

Apart from poor precision near the detection limits the laboratories display good analytical precision.

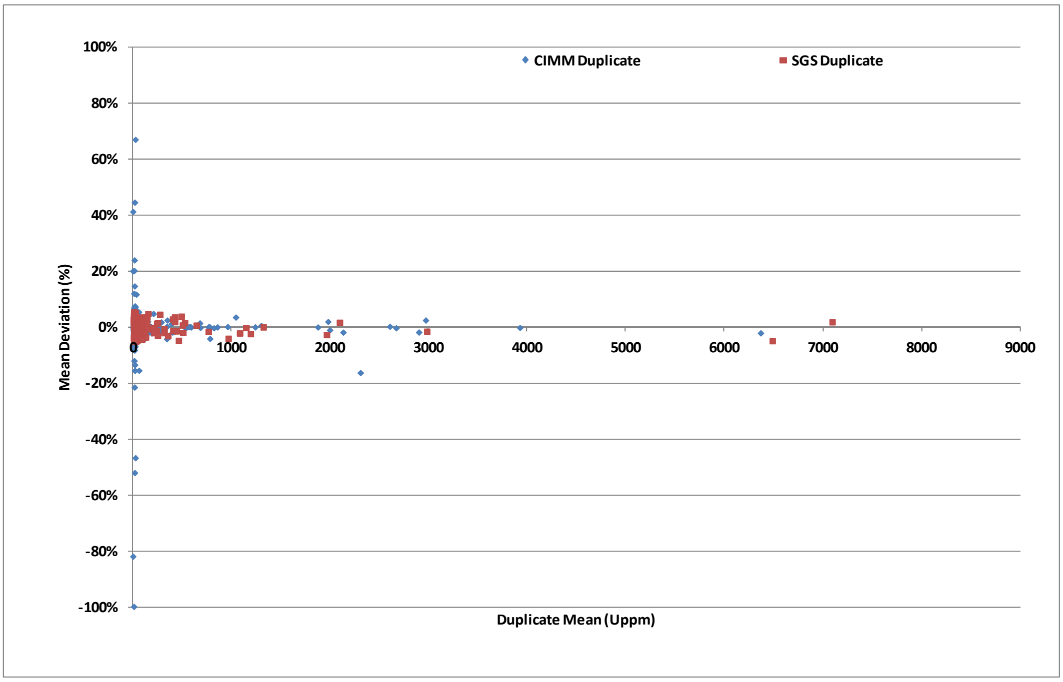


Figure 11-4: Duplicate Sample Results for Global Gold Colibri and Kihitian Complexes

The owner duplicates and those submitted by TMC were based on a re-splitting of crushed core and submission to the laboratory. The owner duplicate is a measure of the core sampling and analytical error and the results are shown in Figure 11-5. Combined with this data are the samples which were taken from crushed core rejects by TMC in 2008 and 2009. Apart from four outliers for both SGS and CIMM the sampling and analytical mean deviation are approximately constrained below 20 %.

The absolute average mean deviation for duplicates submitted by Global Gold is 3.3 % and by TMC is 9.4 %. Apart from poor precision near at very low grades, the analytical precision, including the sampling error, remains acceptable.

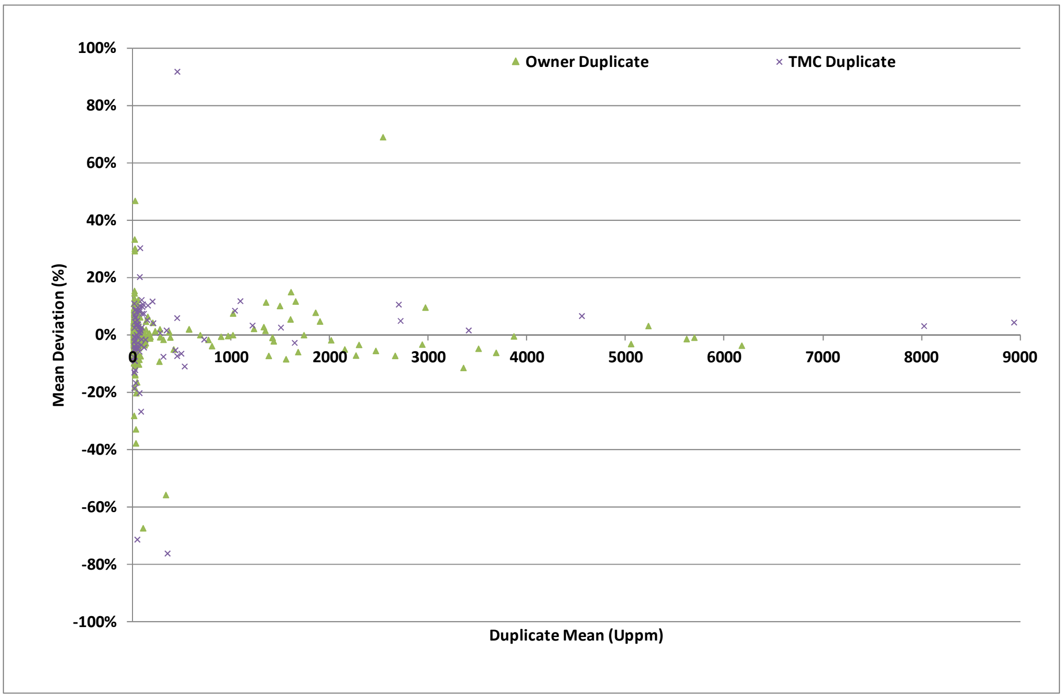


Figure 11-5: Global Gold Group Owner and Third Party Duplicate Analysis for the Colibri and Khitian Complexes

11.2.5.4 BLANK DATA

Blanks were employed to obtain a measure of contamination in the sample preparation and in the laboratory, but uranium is prolific in its presence in rocks in general; the crustal abundance is approximately 2.8 ppm. Thus it is difficult to obtain suitable rocks that contain no uranium. Global Gold employed normal Peruvian cooking flour as a blank material, but the source of the CIMM blank material is not known. The results of all of the blank analytical results are contained in Figure 11-6.

From Figure 11-6 it can be seen that throughout the entire programme only four analyses returned what could be construed as anomalous results above 2.8 ppm. TMC considers the blank results from the Colibri and Khitian Complexes to indicate that no evidence of significant uranium contamination is present in the analytical results.

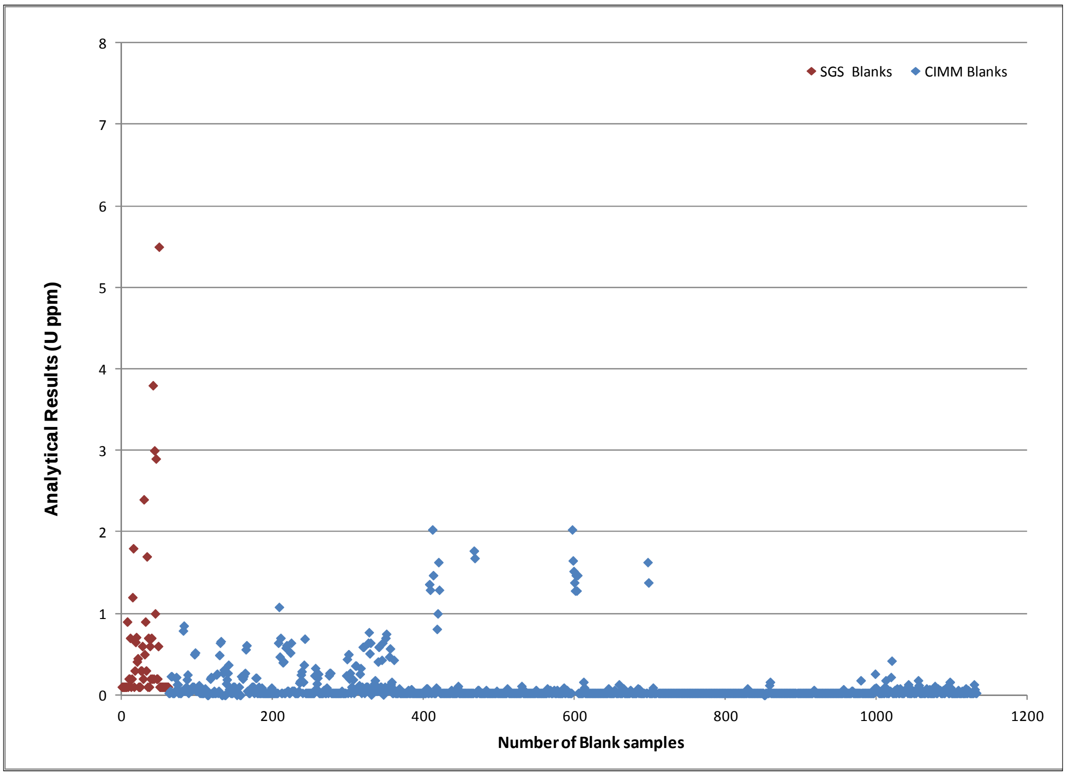


Figure 11-6: Global Gold Group Blank Results for the Colibri and Khitian Complexes

11.2.6 SAMPLE DATABASE

TMC received the drillhole logging results as a series of Microsoft Excel files. The data within these files was re-formatted, and imported in Datamine Studio for further analysis. A check on the accuracy of the transposition of sample results from assay certificate to Excel file was completed by TMC.

11.2.7 OVERALL ADEQUACY STATEMENT

Based on the sample preparation scrutinised for the CIMM Laboratory, the sample security protocols described by Global Gold and the analytical procedures adopted by the SGS and CIMM laboratories, these attributes can be stated as adequate for the estimation of Mineral Resources.

TMC has noted no changes in the tendency to under-report very low grade uranium abundances as previously reported (Young, 2012) (25). As this bias is only evident at low grades, and as the impact is likely to under-estimate rather than over-estimate the low grade material, TMC does not consider that this finding is sufficient to preclude the classification of Inferred or Indicated Mineral Resources. If unresolved, it may however prevent the classification of Measured Mineral Resources in the future.

11.3 SOLEX DATA

The sample preparation, analysis and security protocols for the Solex data are pertinent to the Calvario I, Puncopata, Calvario II, Calvario III, Sayaña West, Sayaña Central and Agaton deposits.

11.3.1 SAMPLING METHODS

TMC has not observed the sampling methodology employed by Solex. However, Solex's Calvario I summary (Solex, 2012) (27) indicates that full core sampling was done, at least for the 72 holes drilled in 2006 and 2007.

11.3.2 SAMPLING RECOVERY

The lithology in which the Solex drilling was undertaken is the same as that which Global Gold had drilled though in generating their exploration data. TMC considers that it is likely that the sample recovery would have been acceptable.

11.3.3 SAMPLE QUALITY

The sample quality is also likely to have been acceptable, as whole core sampling was undertaken.

11.3.4 SAMPLE PREPARATION, ASSAYING AND ANALYTICAL PROCEDURES

TMC has not observed any of the sample preparation, assaying or analytical procedures undertaken for the Solex data.

TMC has had access to the electronic archive, and scans of drillhole logs indicate that for each drillhole, the following work was undertaken / recorded:

- Detailed sample, lithological descriptions;
- Core recovery;
- Core angle;
- Count per second scintilometer results at 5 cm increments; and
- Core photographs.

It is understood that the samples were analysed by ALS Chemex, Vancouver. Five scanned assay certificates have been retained by Global Gold, which confirm this.

11.3.5 ANALYTICAL QUALITY ASSURANCE AND CONTROL PROCEDURES

11.3.5.1 DUPLICATE DATA

It is not clear whether Solex submitted duplicates as part of the exploration protocol. No duplicate results have been retained by Global Gold.

11.3.5.2 STANDARD DATA

It is not clear whether Solex submitted standards as part of the exploration protocol. No standards results have been retained by Global Gold.

11.3.5.3 BLANK DATA

The hard-copy logging sheets retained by Global Gold do illustrate that bank samples were submitted in order to monitor contamination. However, the results of these blank samples were not included in the electronic archive.

11.3.6 SAMPLE DATABASE

The electronic drillhole archive was well ordered and with the exception of the missing duplicate, standard and blank results, was comprehensive. TMC captured the individual electronic drillhole files into a database for the purpose of Mineral Resource estimates.

11.3.7 OVERALL ADEQUACY STATEMENT

Limited information on the sample preparation, security and analytical procedures undertaken by Solex is available. The analytical results from Solex should therefore be treated with caution. TMC is of the view that the analytical data is only suitable for informing Mineral Resource estimates in the Inferred category, provided other evidence for the mineralisation can be identified.

TMC recommends that Global Gold undertake a modest validation drilling programme in order to validate the dataset more comprehensively, and enable their use in the Measured and/or Indicated Mineral Resource categories, as was completed for the Corachapi deposit.

11.4 MINERGIA DATA

The sample preparation, analysis and security protocols for the Minergia data are pertinent to the Tantamaco, Tuturumani, Isivilla, Calvario Real and Nueva Corani deposits. They are based on the descriptions by Henkle (2014) (14) and a QA/QC report prepared by Cuba and Del Carpio (Cuba, 2010) (28).

11.4.1 SAMPLING METHODS

Half core sampling was carried out with individual sample lengths varying from a minimum of 0.25 m to a maximum of 2.0 m. Selection of the length to sample was based on visual observation of the mineralisation and assisted by radiometric measurements.

11.4.2 SAMPLING RECOVERY

Henkle (2014) (14) observed and noted that prior to the logging of the drillholes a technician checked the core boxes to make sure that there were no errors in the depth indicators that were inserted after every drill run. The core recovery in each drill interval was then calculated and the box labels were checked as being correct and legible before the core was handed over to be logged and sampled. No information on core recovery is provided by Henkle (2014) (14).

During sample cutting by a diamond saw and bagging, care was taken to appropriately represent the correct length from which the core was sampled.

11.4.3 SAMPLE QUALITY

During the core logging process, which includes scintillometer measurements, fluorescent screening and geological evaluation, the logging geologist marked the sections of core to be assayed and flagged the higher grade sections of core to be manually split. This was in order to minimize potential mineral grain loss during routine core sawing which uses water as a coolant. Drill samples for assay were taken by cutting the drill core longitudinally with a diamond saw and the mud from the cutting operation was placed in the appropriate sample bags as shown in Figure 11-7.

Figure 11-7: Samples Packed and Ready for Shipment to the Laboratory (Source: Cuba, 2010)

11.4.4 SAMPLE PREPARATION, ASSAYING AND ANALYTICAL PROCEDURES

11.4.4.1 SAMPLE DELIVERY PROCEDURES

Before shipment of samples to the CIMM sample preparation laboratory in Juliaca, a record of each consignment, sequence numbered samples, catalogues and details of each shipment were kept. The samples were secured with security seals and sent after sample transport requisition protocols had been followed. The samples were accompanied by a Minergia geologist, and the CIMM personnel receiving the samples confirmed that the number of samples corresponded to the sample submission form. Sample rejects were collected by a Minergia geologist and these remained stored under strict control in the Minergia core facility in Juliaca (Cuba, 2010) (28).

11.4.4.2 SAMPLE PREPARATION AND ANALYSIS (CIMM)

The sample preparation described by Henkle (2014) (14) is much the same as that described in Section 11.2.4.4. A pulverised aliquot of only 100 g was taken for analysis also by the same procedure described in Sections 11.2.4.5 and 11.2.4.6.

11.4.5 ANALYTICAL QUALITY ASSURANCE AND CONTROL PROCEDURES

11.4.5.1 ANALYTICAL QA/QC RESULTS

Minergia inserted standards, blanks and duplicates into the sampling batches analysed at the CIMM laboratory. A total of 1 044 QA/QC samples were submitted, however, there are a number of QA/QC samples for which there are no analytical results or incomplete data has been provided. These samples have been excluded from the QA/QC dataset. Of the 1 044 samples, only 548 representing 4 % of total samples analysed are considered suitable as QA/QC data. Table 11-4 contains the overall statistics for the analytical QA/QC samples.

Table 11-4: Summary of QA/QC Samples Submitted for Minergia Data

Category	Unidentified	Duplicates	Standards	Blanks	% QA/QC
Samples with complete data	2	225	234	87	4

11.4.5.2 STANDARD DATA

Four types of standards prepared by SGS Laboratory have been used and information relating to the standards is shown in Table 11-5.

Table 11-5: Information in Standards Used at Minergia

Standard ID	Source	Certified Value (U ppm)
ST800019	SGS Laboratory	3 419
ST800019	SGS Laboratory	3 415
ST800020	SGS Laboratory	469
ST800030	SGS Laboratory	0.2

The standards analytical data has been analysed in terms of the % Error Deviation charts, with the % Error Deviation calculated as follows:

$$Error\ Deviation = \frac{Xanalysis - Xstandard\ Certified\ Value}{Xstandard\ Certified\ Value}$$

The error deviation results are portrayed graphically in Figure 11-8 and Figure 11-9. Positive results indicate an over-estimation, whereas negative results indicate an under-estimation of grade. With this technique, Error Deviation with the ±10 % range is considered to signify acceptable levels of accuracy by the laboratory on that standard.

The data presented in Figure 11-8 indicates that at near detection limits the error deviations are very high, which is not surprising, however, when compared to the data in Figure 11-3, they are an order of magnitude higher. This is probably due to the standard being an order of magnitude lower at 0.2 ppm U as opposed to ±3 ppm U in the latter case. For the PEM U abundancies it would seem that CIMM generally under report the grades with the error deviation between -20 % to +10 % (Figure 11-9).

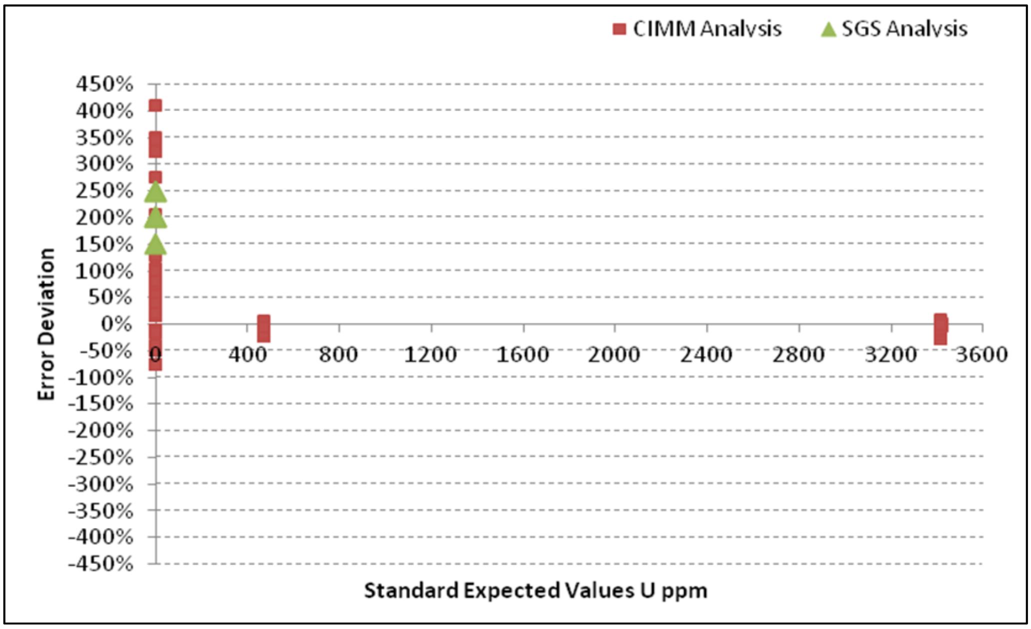


Figure 11-8: Minergia Group % Error Deviation Plot for Standards Analytical Data

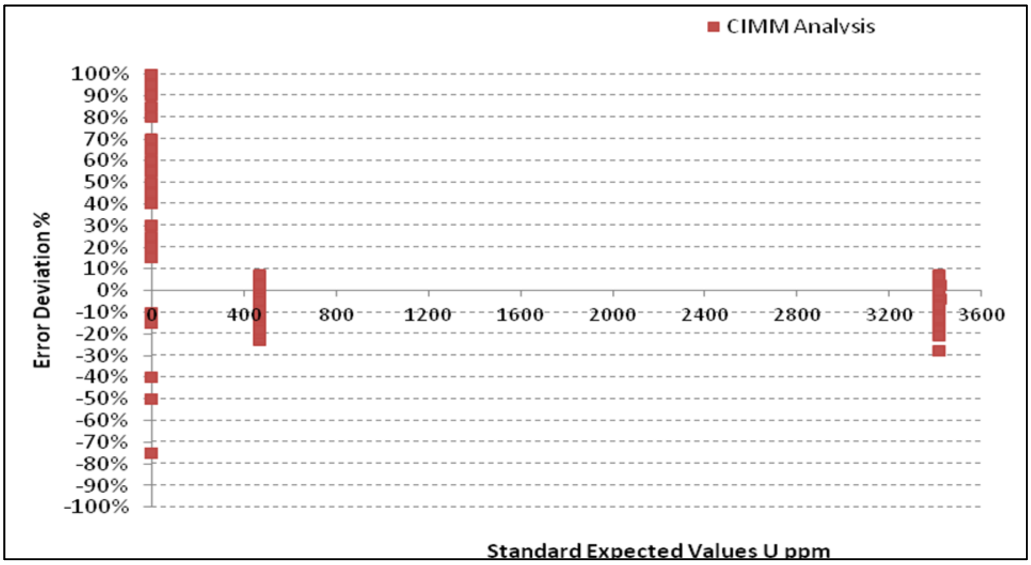


Figure 11-9: Minergia Group % Error Deviation Plot for Standards Analytical Data

11.4.5.3 DUPLICATE DATA

Duplicate analytical data has been analysed for quartered core (Henkle, 2014) (14) using the mean deviation % method, where the Mean Deviation % is calculated as follows:

$$Mean\ Deviation\ \% = \frac{(Xa - Xb) \times 0.5}{Mean\ (Xa, Xb)} \times 100$$

Where Xa and Xb are duplicate pairs.

By using this convention, negative deviations are noted as under-reporting and positive deviations are noted as over-reporting of results. Results of the analysis are presented in Figure 11-10 and summarised in Table 11-6.

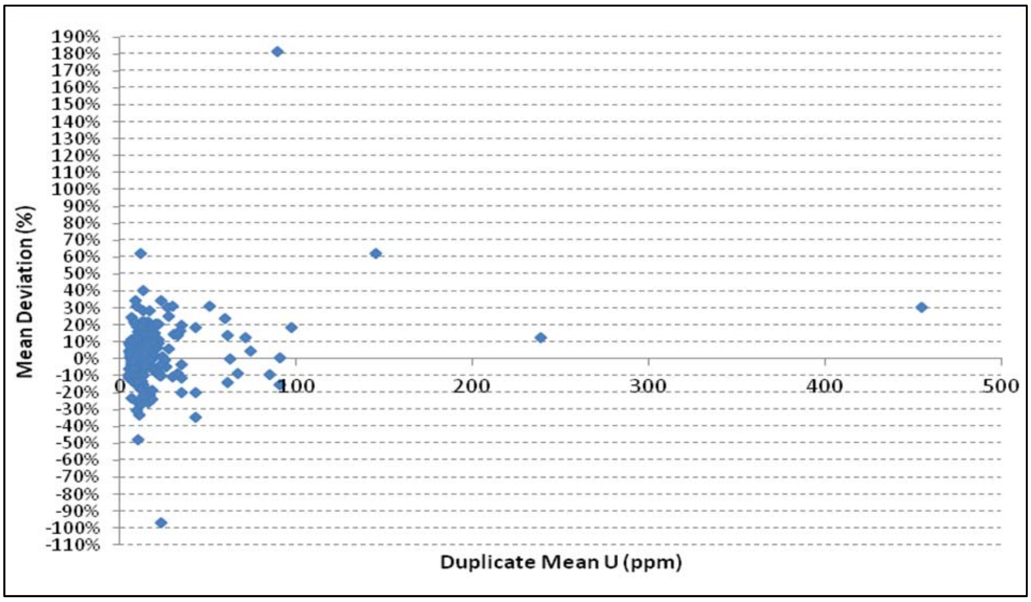


Figure 11-10: Minergia Group Mean Deviation % for Duplicate Data

Table 11-6: Summary of Mean Deviation % Data for the Duplicate Data

Mean Deviation Ranges	±10%	±10%-40%	±40%-100%	±100%-182%
Number of Samples	142	78	4	1
Proportion of Total	63.1 %	34.7 %	1.8 %	0.4 %

The mean deviation data presented in Figure 11-10 and Table 11-6 for quartered core is not anomalous as it contains not only the analytical imprecision but also the sampling imprecision inflated by having the duplicate sample based on a smaller sample than the original.

11.4.5.4 BLANK DATA

The analytical results for the blanks submitted by Minergia are provided in Figure 11-11. TMC has not been able to determine the nature of the blank material used, but it can be interpreted that two different sources of blank were used. The first, which had a value of close to 1 ppm U, was in all likelihood a synthetic blank (such as flour), because it would be unusual to obtain a natural blank with such low uranium abundance. A second blank is evident, with a value of about 7 ppm U. Although three anomalous values are noted, TMC would interpret that the blank results from the Minergia data show no evidence of uranium contamination.

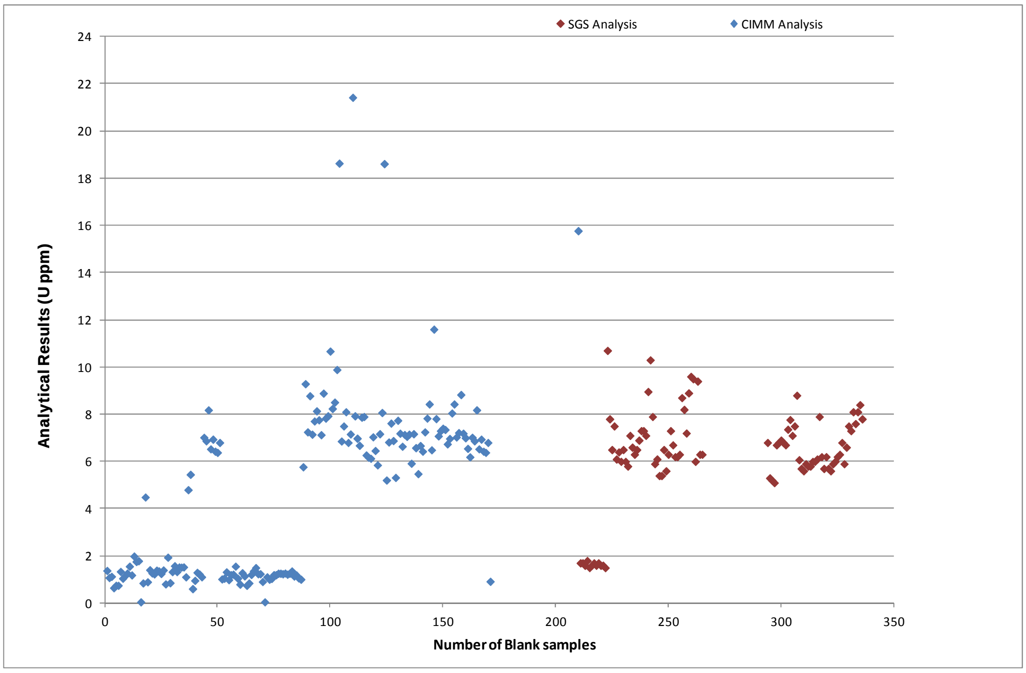


Figure 11-11: Minergia Group Blank Results

11.4.6 SAMPLE DATABASE

TMC received the drillhole logging results as a series of Microsoft Excel files. The data within these files was re-formatted, and imported in Datamine Studio for further analysis. A check on the accuracy of the transposition of sample results from assay certificate to Excel file was completed by TMC.

11.4.7 OVERALL ADEQUACY STATEMENT

Although the analytical methods and protocols for the samples sent to CIMM were not verified by the Qualified Person, the descriptions given by Henkle (2014) (14) match what was observed by Young (2011) (23) and, as such, are accepted. Based on the sample preparation and analytical protocols employed at CIMM, and the sample security protocols described by Minergia, these attributes can be stated as adequate for the estimation of Mineral Resources.

The error deviation indicates the slight under-reporting by CIMM is not material to the reporting of Indicated and Inferred Mineral Resources, but may preclude the reporting of Measured Mineral Resources.

SECTION 12 DATA VERIFICATION

12.1 CORACHAPI COMPLEX

Data verification procedures and results for the Corachapi Complex are documented in Young (2010[3]) (24). As these Mineral Resources are unchanged the specific information relating to the data verification can be located in this reference.

12.2 COLIBRI COMPLEX

Data verification procedures and results for the Colibri Complex are documented in Young (2013) (17). As these Mineral Resources are unchanged the specific information relating to the data verification can be located in this reference.

12.3 KIHITIAN COMPLEX

12.3.1 SITE VISIT

TMC has visited the Chilcuno Chico and Quebrada Blanca deposits and Henkle (2014) (14) undertook a visit to Tantamaco. TMC is of the view that on the basis of the descriptions in Henkle (2014) (14), and our review of the exploration data, it would seem reasonable to interpret that these deposits have the same geological setting and mineralisation characteristics as those already modelled by TMC. TMC has therefore relied on the data verification undertaken by Henkle (2014) (14), in addition to requesting that independent samples analyses be carried out.

12.3.2 DRILLHOLE LOCATIONS

Henkle and Associates verified the locations of select diamond drillholes (Henkle, 2014) (14).

12.3.3 GEOLOGICAL OBSERVATIONS

Henkle and Associates reviewed structural and lithological features both in outcrop and in the various core samples, and the lithological descriptions of the core with the geologists who logged it. They were satisfied that the descriptions were sound. They also reviewed the sample selection and security procedures with the field geologists and exploration management and were satisfied as to their adequacy (Henkle, 2014) (14).

12.3.4 INDEPENDENT SAMPLE ANALYSES

Henkle and Associates reviewed the data accumulated by Minergia as well as that collected during the site visits. A check analysis programme was instituted as part of its data validation and 159 samples of pulps and rejects were taken together with standards, duplicates and blanks and sent them to the Saskatchewan Research Council laboratory in Saskatoon, Canada. The samples were analysed by an ICP-OES method for material with U abundances between 1 ppm and 1 000 ppm. Material that contained a U abundance greater than 1 000 ppm, it was analysed also by an ICP-OES that measures the U_3O_8 abundance from 0.001 % upwards.

The comparison of the Saskatchewan Research Council results with the CIMM laboratory indicates that there is an acceptable correlation between the laboratories. The higher grade samples (1 000 to 13 000 ppm U_3O_8) analysed for U_3O_8 returned consistently higher abundances than those by CIMM by 8 % to 10 %. The lower U abundance material (1 to 85 ppm U) on the other hand returned consistently lower abundance than those by CIMM, also by 8 % to 10 %.

TMC requested that duplicate analytical checks on 22 randomly selected duplicate crushed core rejects, from Tantamaco and Tuturumani be completed. Table 12-1 summarises the results of this re-analysis.

Table 12-1: Independent Analysis for Tantamaco and Tuturumani

Deposit	Drillhole Number	Original Sample No.	Re-analysis Sample No.	Original (U ppm)	Re-analysis (U ppm)
Tuturumani	MA-TU-MI-DDH-2010-012	11387	DJG-09	3 362.00	6 082.00
Tuturumani	MA-TU-MI-DDH-2010-004	11475	DJG-10	340.00	143.00
Tuturumani	MA-TU-MI-DDH-2010-012	11398	DJG-11	933.00	406.00
Tuturumani	MA-TU-MI-DDH-2010-006	11417	DJG-12	10.67	54.08
Tuturumani	MA-TU-MI-DDH-2010-011	11336	DJG-13	7.26	8.46
Tuturumani	MA-TU-MI-DDH-2010-007	11294	DJG-14	11.67	9.86
Tuturumani	MA-TU-MI-DDH-2010-008	11317	DJG-15	41.58	21.00
Tantamaco	TADDHS11-180	14767	DJG-23	41.38	45.42
Tantamaco	TADDHS11-149	13852	DJG-24	272.00	107.00
Tantamaco	TADDHS11-117	13003	DJG-25	10.35	9.06
Tantamaco	MA-TA-MI-DDH-2010-076	10665	DJG-26	1 531.00	1 479.00
Tantamaco	MA-TA-MI-DDH-2010-060	9592	DJG-27	134.00	117.00
Tantamaco	MA-TA-MI-DDH-2010-055	9354	DJG-28	2.31	6.45
Tantamaco	MC-TA-VR-DDH-2008-027	4086	DJG-29	603.00	568.00
Tantamaco	MC-TA-VR-DDH-2008-016	2731	DJG-30	10.70	11.58
Tantamaco	MC-TA-VR-DDH-2008-013	2558	DJG-31	9.00	12.42
Tantamaco	MC-TA-VR-DDH-2008-017	2828	DJG-32	10.90	11.82
Tantamaco	MC-TA-VR-DDH-2008-020	3208	DJG-33	256.80	202.00

Deposit	Drillhole Number	Original Sample No.	Re-analysis Sample No.	Original (U ppm)	Re-analysis (U ppm)
Tantamaco	MC-TA-VR-DDH-2007-007	1615	DJG-34	11 738.00	7 882.00
Tantamaco	MC-NC-VR-DDH-2008-006	5698	DJG-35	22.34	50.74
Tantamaco	MC-NC-VR-DDH-2008-006	5601	DJG-36	63.83	79.30
Tantamaco	MC-NC-VR-DDH-2008-005	5510	DJG-37	91.30	122.00

The re-analysis of samples is not intended as a QA/QC measure, but rather as a check that uranium mineralisation is present in similar abundances as is indicated in the sample database, as a means of testing the database for errors as well as ensuring that the tenor of mineralisation exists. Table 12-1 illustrates a reasonable co-incidence between sample result and re-analysis.

12.3.5 LIMITATIONS OR FAILURE TO CONDUCT SUCH VERIFICATION, AND THE REASONS FOR ANY SUCH LIMITATIONS OR FAILURE

None

12.4 ISIVILLA COMPLEX

12.4.1 SITE VISIT

David Young and Stewart Nupen visited only the Calvario I deposit within the Triunfador concession (Solex data) in March 2013. The collar locations of two drillholes were observed and independent samples taken from Global Gold's storage facility in Macusani. No core or outcrop was examined. No site visits have been made to the Calvario Real and Isivilla deposits (Minergia data) and reliance is placed on Henkle (2014) (14) who visited the deposits recently. No site visit has been made by TMC to the Puncopata deposits.

12.4.2 DRILLHOLE LOCATIONS

Table 12-2 considers the drillhole collar locations as verified at the Triunfador concession.

Table 12-2: Drillhole Collar Locations Verified at Triunfador

	Pad 9		Pad 4	
	X co-ordinate	Y co-ordinate	X co-ordinate	Y co-ordinate
Hand held GPS location	332 633.2	8 464 819	332 808.4	8 464 837
Surveyed location database	332 624.7	8 464 823	332 789.9	8 464 844

12.4.3 GEOLOGICAL OBSERVATIONS

Henkle and Associates reviewed structural and lithological features both in outcrop and in the various core samples, and the lithological descriptions of the core with the geologists who logged it. They were satisfied that the descriptions were sound. They also reviewed the sample selection and security procedures with the field geologists and exploration management and were satisfied as to their adequacy (Henkle, 2014) (14).

TMC has not reviewed any geological information for the Solex data in this mining concession. The outcrops noted in the brief visit to Calvario I indicate it is underlain by similar rhyolite lavas to the other deposits.

12.4.4 INDEPENDENT SAMPLE ANALYSES

In 2013, TMC visited Global Gold's storage facility in Macusani, and selected 14 samples bags containing reject crushed core sample material for re-analysis.

In 2015, a total of 15 samples were selected by TMC by random choice directly from the assay results of the new areas of Calvario Real and Isivilla which were not visited and the reject crushed core sample material on site sent for re-analysis.

In both cases, subjects for re-analysis were a mixture of samples with reportedly high, medium and low uranium grades. The results of the re-analysis are shown in Table 12-3.

Table 12-3: Independent Analysis from the Isivilla Complex (2015)

Deposit	BH Number	Original Sample no.	Re-analysis Sample No	Original (U ppm)	Re-analysis (U ppm)
Isivilla	MA-IS-MI-DDH-2010-007	11791	DJG-08	37.72	19.53
Isivilla	ISDDHS11-008	11913	DJG-07	16.64	14.74
Isivilla	ISDDHS11-030	14573	DJG-16	13.98	12.26
Isivilla	ISDDHS11-028	13509	DJG-17	166.00	76.90
Isivilla	ISDDHS11-012	12673	DJG-18	396.00	290.00
Isivilla	ISDDHS11-014	13125	DJG-19	123.00	162.00
Isivilla	ISDDHS11-021	13081	DJG-20	368.00	432.00
Isivilla	ISDDHS11-018	12288	DJG-21	121.00	105.00
Isivilla	ISDDHS11-024	13327	DJG-22	146.00	134.00
Calvario Real	MA-CR-MI-DDH-2010-009	10838	DJG-01	683.00	936.00
Calvario Real	MA-CR-MI-DDH-2010-009	10841	DJG-02	15.58	12.00
Calvario Real	MA-CR-MI-DDH-2010-009	10854	DJG-03	149.00	190.00
Calvario Real	MC-CR-VR-DDH-2008-001	4951	DJG-04	10.70	12.84
Calvario Real	MC-CR-VR-DDH-2008-001	4955	DJG-05	96.90	171.00
Calvario Real	MC-CR-VR-DDH-2008-001	4944	DJG-06	759.50	288.00

Deposit	BH Number	Original Sample no.	Re-analysis Sample No	Original (U ppm)	Re-analysis (U ppm)
Calvario 1	CAL07-39	14483	D2429	53.00	48.63
Calvario 1	CAL07-39	14489	D2430	510.00	469.00
Calvario 1	CAL07-37	14490	D2431	59.00	56.47
Calvario 1	CAL07-39	14498	D2432	248.00	228.00
Calvario 1	CAL07-39	14501	D2433	53.00	46.55
Calvario 1	CAL07-39	14504	D2434	48.00	42.92
Calvario 1	CAL07-39	14505	D2435	59.00	55.39
Calvario 1	CAL07-39	14506	D2436	48.00	40.45
Calvario 1	CAL07-39	14507	D2437	47.00	40.30
Calvario 1	CAL07-39	14508	D2438	50.00	43.58
Calvario 1	CAL07-39	14510	D2439	45.00	36.70
Puncopata	PUN06-09	1239	D2440	85.00	74.85
Puncopata	PUN06-09	1224	D2441	1 900.00	1 975.00
Puncopata	PUN06-09	1240	D2442	201.00	248.00

Table 12-3 illustrates a reasonable correlation between the original sample result and its re-analysis.

Henkle and Associates' re-sampling and analysis programme is further described in Section 12.3.4.

12.4.5 LIMITATIONS OR FAILURE TO CONDUCT SUCH VERIFICATION, AND THE REASONS FOR ANY SUCH LIMITATIONS OR FAILURE

12.4.5.1 CALVARIO I AND PUNCOPATA DEPOSITS

The verification procedures for the Calvario I and Puncopata deposits were limited by the fact that the drilling was undertaken on behalf of Solex by a third party that is no longer active. No drillhole core is available as a result of full-core sampling.

TMC considered the following factors in concluding the data is suitable for Mineral Resource estimates for at least the Inferred category:

- The well-ordered and reasonably comprehensive nature of the dataset;
- Core photographs;
- Correlation of the mineralisation in core with the radiometric anomaly;
- The data verification d described in 12.4.3 and 12.4.4; and
- The results contained in Table 12-3.

12.4.5.2 ISIVILLA AND CALVARIO REAL DEPOSITS

None

12.5 CORANI COMPLEX

12.5.1 SITE VISIT

TMC has visited the Calvario II and Calvario III deposits. Henkle (2014) (14) undertook a site visit to the Nueva Corani deposit. TMC is of the view that on the basis of the descriptions in Henkle (2014) (14), and TMC's review of the exploration data, it would seem reasonable to interpret that these deposits have the same geological setting and mineralisation characteristics as those already modelled by TMC.

12.5.2 INDEPENDENT SAMPLE ANALYSES

Henkle and Associates reviewed the data accumulated by Minergia as well as that collected during its site visits. There was no detail pertaining to the data collected during the site visit.

TMC requested duplicate analytical checks on 17 randomly selected duplicate crushed core rejects, from Calvario II, Calvario III and Nueva Corani, for re-analysis and Table 12-4 summarises the results of this re-analysis.

Table 12-4: Independent Analysis for Corani Complex

Deposit	BH Number	Original Sample no.	Re-analysis Sample No	Original (U ppm)	Re-analysis (U ppm)
Nueva Corani	MC-NC-VR-DDH-2008-003	5337	DJG-38	19.90	24.52
Nueva Corani	MC-NC-VR-DDH-2008-002	5210	DJG-39	91.20	59.29
Nueva Corani	MC-NC-VR-DDH-2008-002	5168	DJG-40	581.30	708.00
Nueva Corani	MC-NC-VR-DDH-2008-034	7669	DJG-41	461.54	401.00
Nueva Corani	MC-NC-VR-DDH-2008-036	7727	DJG-42	45.80	38.23
Nueva Corani	MC-NC-VR-DDH-2008-040	7996	DJG-43	49.30	13.37
Nueva Corani	MC-NC-VR-DDH-2008-041	8157	DJG-44	791.78	846.00
Calvario 2	CII07-02	20804	DJG-45	78.00	63.96
Calvario 3	CIII08-75	26267	DJG-46	45.00	42.96
Calvario 3	CIII08-69	25980	DJG-47	79.00	82.49
Calvario 2	CII07-01	20658	DJG-48	110.00	90.59
Calvario 3	CIII07-14	5264	DJG-50	121.00	12.92
Calvario 2	CII07-14	21627	DJG-52	1 098.00	833.00
Calvario 3	CIII08-81	26508	DJG-53	911.00	589.00
Calvario 3	CIII08-73	26182	DJG-54	27.00	22.20

Deposit	BH Number	Original Sample no.	Re-analysis Sample No	Original (U ppm)	Re-analysis (U ppm)
Calvario 2	CII07-01	20663	DJG-61	95.00	72.40
Calvario 2	CII07-01	20664	DJG-62	296.00	393.00

Table 12-4 illustrates a reasonable correlation between the original sample result and its re-analysis.

Henkle and Associates' re-sampling and analysis programme is further described in Section 12.3.4.

12.6 SAYAÑA COMPLEX

12.6.1 SITE VISIT

TMC has not visited the Sayaña and Agaton sites (Solex data).

12.6.2 INDEPENDENT SAMPLE ANALYSES

TMC requested duplicate analytical checks on 16 samples, from Sayaña West and Sayaña Central, for independent analysis and Table 12-5 summarises the results of this re-analysis. Samples from the Agaton area were not available for re-analysis.

Table 12-5: Independent Analysis for Sayaña Central and Sayaña West

Deposit	BH Number	Original Sample no.	Re-analysis Sample No	Original (U ppm)	Re-analysis (U ppm)
Sayaña Central	SY06-01	14	DJG-49	254.00	348.00
Sayaña Central	SY06-08	304	DJG-51	144.00	247.00
Sayaña Central	SY06-02	51	DJG-55	856.00	793.00
Sayaña Central	SY06-02	52	DJG-56	3 610.00	2 038.00
Sayaña Central	SY06-06	118	DJG-57	643.00	686.00
Sayaña Central	SY06-06	119	DJG-58	47.00	39.95
Sayaña Central	SY06-07	262	DJG-59	364.00	351.00
Sayaña Central	SY06-08	318	DJG-60	44.00	66.50
Sayaña Central	SY07-44	16581	DJG-63	44.00	12.71
Sayaña Central	SY07-44	16582	DJG-64	21.00	8.07
Sayaña Central	SY07-44	16584	DJG-65	59.00	8.09
Sayaña West	SW08-33	25219	DJG-66	9.00	4.89
Sayaña West	SW08-33	25220	DJG-67	10.00	7.28
Sayaña West	SW08-33	25221	DJG-68	10.00	4.66
Sayaña West	SW08-33	25222	DJG-69	15.00	9.04
Sayaña West	SW08-33	25223	DJG-70	13.00	8.54

Table 12-5 illustrates a reasonable correlation between the original sample result and its re-analysis.

Henkle and Associates' re-sampling and analysis programme is further described in Section 12.3.4.

12.6.3 LIMITATIONS OR FAILURE TO CONDUCT SUCH VERIFICATION, AND THE REASONS FOR ANY SUCH LIMITATIONS OR FAILURE

As there have not been any site visits by an independent Qualified Person to the Sayaña Complex and no QA/QC data is available, these results cannot be employed to report Mineral Resources and/or Exploration Results.

SECTION 13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 INTRODUCTION

The process comprises crushing, uranium extraction by heap leaching and recovery by ion exchange, solvent exchange and precipitation. A summary and review of available testwork is contained in this section.

Metallurgical testing to date has been primarily on samples from the Colibri Complex. Bottle roll testwork (BRT) suggests the Kihitian Complex and Corachapi Complex PEM is very similar to the Colibri Complex PEM. Whilst fewer testwork results are available for the Isivilla Complex PEM, it is assumed that it is also similar to the Colibri PEM. Therefore, column leach test results from samples from Colibri are assumed for this level of study to be applicable to the Kihitian, Corachapi and Isivilla PEM. In future, further metallurgical testwork on representative samples of each individual PEM body will be required to validate this assumption to meet the technical and financial accuracy requirements of a Feasibility Study.

13.2 COMPLETED TESTWORK SUMMARY

Testwork that has been completed for the project to date is listed below:

- Bottle roll leach tests;
- Column leach tests;
- Ion exchange;
- Solvent extraction and precipitation;
- Grade confirmation;
- Size by size assay;
- Mineralogy;
- Residue neutralisation;
- Site water quality; and
- Rock compression tests.

The extent and findings of the completed work are discussed and an assessment of the suitability and adequacy of each is provided. The metallurgical processing testwork that has been conducted to date can be grouped into six phases. A summary of the testwork completed and results from all six phases is contained in Table 13-1. The testwork listed in Table 13-1 was completed predominantly on samples from Colibri Complex. The Minergia testwork was conducted on samples from the Isivilla Complex. Any other exceptions are stated.

Table 13-1: Completed Testwork Summary

Item	Date	Company	Experiment	Key findings
Phase 1: Initial bottle roll tests				
1	Dec 2007	TECMMINE	Leach-bottle roll	Acid consumption: Bottle roll: 33 kg/t - 40 kg/t; Extraction: Bottle roll: 26 % - 39 %
2	Apr 2008	TECMMINE	Leach-bottle roll	Acid consumption: Bottle roll: 28 kg/t, Column: 14.5 kg/t - 17 kg/t; Extraction: Bottle roll: 58 % - 99 %, Column: 84 % - 99 %; Pretreatment increases early but not overall extraction.
Phase 2: Initial column leach, SX and IX tests				
3	May 2009	TECMMINE	Leach-column	Acid consumption: 5-9-13 kg/t for sizes 3-2-1"; Extraction: 78 % - 81 % - 80 %; U distribution: 70 % U in +1 mm; Kinetics: 1-2" similar, 3" slower; Optimum size: 2" possibly.
4	Dec 2009	TECMMINE	Solvent extraction	CYANEX 272 loading: Diluent: Kerosene; Modifier: Solvesso 100; Activator: sulfuric acid; pH: 1.0; O/A: 01; Extraction: 99 %; Time: 3 min; CYANEX 272 stripping: O/A: 2.5; Extraction: 56 %; Stages: 1; Time: 3 min; $NaCO_3$ soln: 0.5M;
5	Dec 2009	TECMMINE	Solvent extraction	ALAMINE Loading: Diluent: Kerosene; Modifier: Solvesso 100; Activator: sulfuric acid; pH: 1.0; O/A: 1; Extraction: 98 %; Stages: 3-4; Time: 1.5 min; ALAMINE stripping: O/A: 4-5; Extraction: 97 %; Stages: 1; Time: 1-2 min; $NaCO_3$ soln: 12 %;
6	2009	TECMMINE	Solvent extraction	D2EHPA loading: PLS tenor: 105 ppm; O/A: 1; Extraction: 90 %; pH: 1.7; Stages: 3; Time: 160 min; D2EHPA stripping: O/A: 5; Extraction: 98 %; Stages: 2; Time: 179 min; Acid water: 24 g/L; ALAMINE Loading: O/A: 1; Extraction: 86 %; Stages: 1; Time: 1.5 min; ALAMINE stripping: O/A: 4; Extraction: 98 %; Stages: 1; Time: 1.5 min; NaCl soln: 0.9M; Precipitation: 98 %; Reagent: MgO
7	Mar 2010	TECMMINE	Solvent extraction	ALAMINE Loading: Diluent: Kerosene; Modifier: Solvesso 100; Activator: sulfuric acid; pH: 1.0; O/A: 1; Extraction: 98 %; Stages: 3; Time: 2 min; PLS: 105 ppm; ALAMINE stripping: O/A: 7; Extraction: 98 %; Stages: 2; Time: 2 min; NaCl soln: 1.2M; Precipitation: 97 %; Reagent: MgO
8	May 2010	Pontificia Universidad Catolica del Peru	PEM characterisation	Natural density: 2.0 g/cm³; Simple compression strength: 3.6 to 19.0 MPa; Compressive stress: 2.0 to 32.5 MPa; Friction angle: 35.6 to 41 ^{0}; Cohesion: 0.120 to 0.075 MPa
Phase 3: Ion exchange screening tests				
9	Sep 2010	CWENGA	Ion exchange	IX: Resin: Strong base; Stages: 2; Exhaustion: incomplete; Elution: Eluant: 4M sulfuric acid; Test status: incomplete-insufficient sample; Precipitation: Reagent: Lime and H_2O_2
Phase 4: Leach extraction mapping tests				
10	Oct 2010	TECMMINE	Leach-bottle roll	Group 1: Grade: 10 to 6817 ppm (avg: 271 ppm); Extraction: 21 % to 99 % (avg: 63 %); Acid consumption: 10 to 32 kg/t (avg: 22 kg/t); Group 2: Grade: 9 to 2971 ppm (avg: 126 ppm); Extraction: 25 % to 92 % (avg: 64 %); Acid consumption: 11 kg/t to 23 kg/t (avg: 16 kg/t); Corachapi: Grade: 45 ppm to 6 933 ppm (avg: 618 ppm); Extraction: 31 % to 95 % (avg: 73 %); Acid consumption: 13 kg/t to 44 kg/t (avg: 24 kg/t);
11	2010	TECMMINE	Leach-bottle roll	Grade: 26 ppm to 19 955 ppm (avg: 943 ppm); Recovery: 54 % to 98 % (avg: 86 %)
Phase 5: Large column leach tests				
12	Jan 2011	TECMMINE	Leach-column	Head grade: 295 ppm; Acid consumption: 10 kg/t; Height: 2.6m; Diameter 16.5"; Irrigation: 8 L/h/m²; Acid soln: 15 g/L; Time: 50 days; Extraction: 97 %; Tail: 8 ppm; PLS tenor: 93 ppm; U dist: 60 % in >6.35 mm.

Item	Date	Company	Experiment	Key findings
13	Jan 2011	TECMMINE	Leach-column	Head grade: 98 ppm; Acid consumption: 10 kg/t; Height: 2.5m; Diameter 16.5"; Irrigation: 8 L/h/m²; Acid soln: 15 g/L; Time: 50 days; Extraction: 93 %; Tail: 7 ppm; PLS tenor: 30 ppm.; U dist: 69.5 % in >6.35 mm.
14	Feb 2011	CIMM	Water analysis	Total hardness: 6 mg/L - 16 mg/L
15	Mar 2011	TECMMINE	Leach-column	Head grade: 383 ppm; Acid consumption: 10 kg/t; Height: 2.6 m; Diameter 16.5"; Irrigation: 8 L/h/m²; Acid soln: 15 g/L; Time: 45 days; Extraction: 92 %; Tail: 30 ppm; PLS tenor: 143 ppm.
16	Mar 2011	TECMMINE	Leach-column	Head grade: 22 ppm; Acid consumption: 25 kg/t; Height: 2.3 m; Diameter 7"; Irrigation: 8 L/h/m²; Acid soln: 15 g/L; Time: 32 days; Extraction: 72 %; Tail: 6 ppm; PLS tenor: 9 ppm; U dist: 78.4 % in >6.35 mm.
17	Mar 2011	TECMMINE	Leach-column	Head grade: 22 ppm; Acid consumption: 24 kg/t; Height: 2.4 m; Diameter 7"; Irrigation: 8 L/h/m²; Acid soln: 15 g/L; Time: 29 days; Extraction: 57 %; Tail: 8 ppm; PLS tenor: 6 ppm; U dist: 71.9 % in >6.35 mm.
18	May 2011	TECMMINE	Leach-column	Head grade: 38 ppm; Acid consumption: 13 kg/t; Height: 2.4 m; Diameter 7"; Irrigation: 8 L/h/m²; Acid soln: 15 g/L; Time: 46 days; Extraction: 85 %; Tail: 6 ppm; PLS tenor: 32 ppm; U dist: 58.6 % in >6.35 mm.
19	Jun 2011	TECMMINE	Leach-column	Head grade: 27 ppm; Acid consumption: 16 kg/t; Height: 2.4 m; Diameter 7"; Irrigation: 8 L/h/m²; Acid soln: 15 g/L; Time: 46 days; Extraction: 80 %; Tail: 5 ppm; PLS tenor: 22 ppm; U dist: 49.4 % in >6.35 mm.
20	Jul 2011	TECMMINE	Leach-column	Lima: Head grade: 12 ppm - 36 ppm; Acid consumption: 10 kg/t - 16 kg/t (Bottle rolls: 13 - 21 kg/t); Height: 2.5 m; Diameter 7"; Irrigation: 8 L/h/m²; Acid soln: 15 g/L; Time: 32 - 46 days; Extraction: 57 % - 75 % (Bottle rolls: 58 % - 85 %); Sedimentation: 0.17 % - 0.25 %; PLS tenor: 126 ppm - 223 ppm; Mass: 80 kg; Size: 1 - 2". Bottle roll conditions: Size: 74 um; Time: 480 min; pH: <2; Mass: 2 kg - 5 kg; Notes: Low grade leach profile suggests disseminated/locked U, with a surge in PLS tenor mid-leach. Isivilla: Head grade: 86 ppm - 504 ppm; Acid consumption: 10 kg/t - 11 kg/t (Bottle rolls: 16 kg/t - 22 kg/t); Height: 2.5 m; Diameter 16.5"; Irrigation: 8 L/h/m²; Acid soln: 15 g/L; Time: 45 - 50 days; Extraction: 83 % - 90 % (Bottle rolls: 91 % - 98 %); Sedimentation: 0.28 % - 1.05 %; PLS tenor: 1 402 – 1 430 ppm; Mass: 500 kg; Size: 1 - 2". Bottle roll acid consumption high due to over-concentration of acid, fine-grained sample, more contact with the uranium PEM.
21	Jul 2011	TECMMINE	Leach-column	Head grade: 85 ppm; Acid consumption: 10 kg/t; Height: 2.6 m; Diameter 16.5"; Irrigation: 8 L/h/m²; Acid soln: 15 g/L; Time: 49 days; Extraction: 91 %; Tail: 8 ppm; PLS tenor: 78 ppm.
22	Feb 2012	TECMMINE	Leach-bottleroll	10 bottle roll tests; Head grade: 32 to 413 ppm; Extraction: 79 to 97 %; Acid consumption: 15 to 34 kg/t; Size: 74 um; Time: 24 hrs; pH <2; Acid soln: 15 g/l; Tail: 8 to 53 ppm.
23	Feb 2012	TECMMINE	Leach-bottleroll	Graphical comparison of feed and residue elemental assays (for the 10 bottle roll tests; Head grade: 32 to 413 ppm; Extraction: 79 to 97 %; Acid consumption: 15 to 34 kg/t; Size: 74 um; Time: 24 hrs; pH <2; Acid soln: 15 g/l; Tail: 8 to 53 ppm)
24	Feb 2012	TECMMINE	Leach-bottleroll	Feed and residue elemental assays (for the 10 bottle roll tests; Head grade: 32 to 413 ppm; Extraction: 79 to 97 %; Acid consumption: 15 to 34 kg/t; Size: 74 um; Time: 24 hrs; pH <2; Acid soln: 15 g/l; Tail: 8 to 53 ppm)
25	Feb 2012	Foremost Geological	PEM characterisation	Technical report by geologist detailing mineralogy, drill assays, property maps and radiometric maps of Corachapi and Kihitian. Autenite and meta-autenite predominate U minerals.

Item	Date	Company	Experiment	Key findings
26	Mar 2012	TECMMINE	Leach-bottleroll	Report (for the 10 bottle roll tests; Head grade: 32 to 413 ppm; Extraction: 79 to 97 %; Acid consumption: 15 to 34 kg/t; Size: 74 um; Time: 24 hrs; pH <2; Acid soln: 15 g/l; Tail: 8 to 53 ppm) Leaching kinetics investigation for dynamic leach model. Two leaching kinetics were apparent and require further investigation. Qemscan analysis recommended. High tails for the slow kinetics tests. Aliquots taken every 10 mins in first hour, every 30 mins for second hour, every 60 mins for third and fourth hour, and one at 24 hours. No stirring after four hours. There maybe slight correlation for dissolution versus grade.
27	Jun 2012	PSE	Leach-column	Development of a dyamic cascade, heap leach simulation model that can manipulate particle size distribution, heap height and irrigation flux. More detailed testwork is required to refine the model.
28	Jun 2013	TECMMINE	Leach-bottleroll	Summary of bottle roll tests from Kihitian (22 tests from Pinocho1 & 2 and Chilcuno 1 & 2 (C-P) and 55 tests from Quebrada Blanca (QB)). Extraction: 60 to 99 %. Acid consumption: 91 - 380 kg/t (C-P); 12 - 37 kg/t (QB). Grade ranges: 2.9 - 14.5 % U (C-P); 0.009 - 2.2 % U (QB).
Phase 6: Minergia testwork				
29	Sep 2011	Cameco	Leach-bottleroll	Desktop study heap leach of whole ore. Assumed recovery of 80 %. Costs estimated and assumed. Acid consumption the biggest cost. Benchmarking placed the proposed project well within its contemporaries.
30	Apr 2012	JK Tech	PEM characterisation	SAG Mill Comminution (SMC) tests conducted. A x b = 390.5, very soft; t10 = 85.3 %, very soft.
31	Oct 2013	Cameco	Leach-column	Study considering tank and heap leach of scrubbed and whole ore. Grade: 260 - 570 ppm and 580 - 800 ppm. Soft ignimbrite host rock. Very porous ore. Selective comminution and tank leaching discounted in favour of whole ore heap leaching (cheaper and equivalent recoveries). Column leaching for 340 hrs achieved 98 % extraction. 95 % of maximum extraction could be achieved in 1/3rd of time. Scaling to 7 m, this equates to 70 days leaching. [Graphical inspection by GBM suggests a leach time of 140 days.] Estimated acid consumption: 7 kg/t. Estimated recovery: 80 - 90 %.
32	May 2015	Queen's University	Mineralogy	Geology PhD. Hosted by extremely fractionated, peraluminous, sillimanite-andalusite-muscovite-biotite rhyolites. Main ore mineral, metaautunite, occurs in fractures in association with weeksite and hydrous Mn- and Fe-mineraloids. The absence of any U4+ minerals, as well as absence of breccias or any hydrothermal alteration associated with the ore, suggests that meta-autunite is the primary U mineral. The timing of ore-formation, as well as the mineralogical and geochemical characteristics of the ore and host rocks, suggests that U was leached from the rhyolites, transported by meteoric waters and precipitated as meta-autunite through evaporation and interaction of the uranyl ion with Fe- and Mn-mineraloids. The majority of known deposits are located on the upper walls of active fluvial canyons. The geomorphological environment that is focusing both groundwater flow and its evaporation was the most favourable for meta-autunite precipitation. The Macusani U deposits are volcanic-hosted, but the ore-forming process is of low-temperature and relates to the climatic changes in the area. Some assays and grades but no quantitative mineralogy presented.

13.3 METALLURGICAL TESTWORK REVIEW

13.3.1 BOTTLE ROLL LEACH TESTS

In December 2007, TECMMINE conducted bottle roll leach tests on high grade samples of 3.19 % U[1] to determine sulfuric acid consumption and uranium extraction (TECMMINE, 2007) (29). Acid consumption was 33 kg to 40 kg acid/t and extraction was between 26 % and 39 %. These results represent high acid consumption and low uranium extraction. The reasons for these poor results were not clear and further duplicate tests were recommended, as were independent duplicate assays (for example, in IPEN and CIMM Peru laboratories).

In April 2008, TECMMINE conducted further bottle roll leach tests and small column leach tests on the same high grade material (Zone X) and two other head grades (low grade (Zone 1) and medium grade (Zone 2)) (TECMMINE, 2008) (30). Whilst not detailed in the report, these zones are understood to be in the Colibri and Kihitian Complexes. The bottle roll samples consisted of 400 g of 80 % passing 200 mesh (74 µm) PEM. In the 5-hour leach tests, uranium extraction was 24 % and 94 % for Zone 1 and 2, respectively. Acid consumption for Zone 1 and 2 was 28.5 kg/t, see Table 13-2.

Table 13-2: Bottle Roll Test Results – TECMMINE April 2008

Element	Unit	Zone 1	Zone 2	Zone X
Uranium Grade	ppm	75 (low)	509 (medium)	31 900 (high)
Uranium extraction 96 hour test 5 hour test	 % %	 58.8 24.0	 99.0 94.3	 - 88.1 / 74.9
Acid consumption	kg/t	28.5	28.5	28.7 / 29.4

An extensive campaign of bottle roll tests on drill core samples ensued. The results were summarised in 2010 (31) (32). The grade assays of the Corachapi drill core samples ranged between 45 ppm to 6 933 ppm, see Figure 13-1. Bottle roll extraction ranged from 40 % to 87 %, see Figure 13-2. Acid consumption ranged from 13 kg/t to 44 kg/t. See Figure 13-3.

[1] TECMMINE and CIMM Peru report PEM grade and leach solution tenor in terms of total uranium (elemental) (108) (109). For assays in terms of U_3O_8 multiply the uranium assay by 1.179.

Analysis of these results suggests:

- Extraction is not strongly related to head grade, see Figure 13-4;
- There is no strong relationship between acid consumption and head grade, see Figure 13-5; and
- There is no strong relationship between extraction and acid consumption, see Figure 13-6.

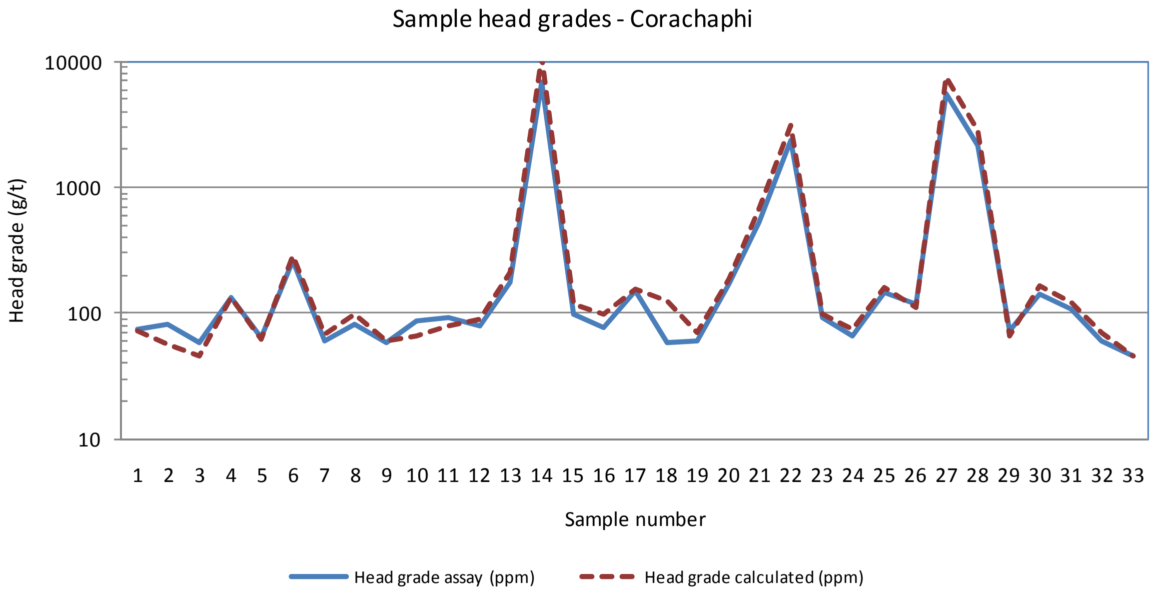


Figure 13-1: Corachapi Drill Core Head Grades – TECMMINE 2010

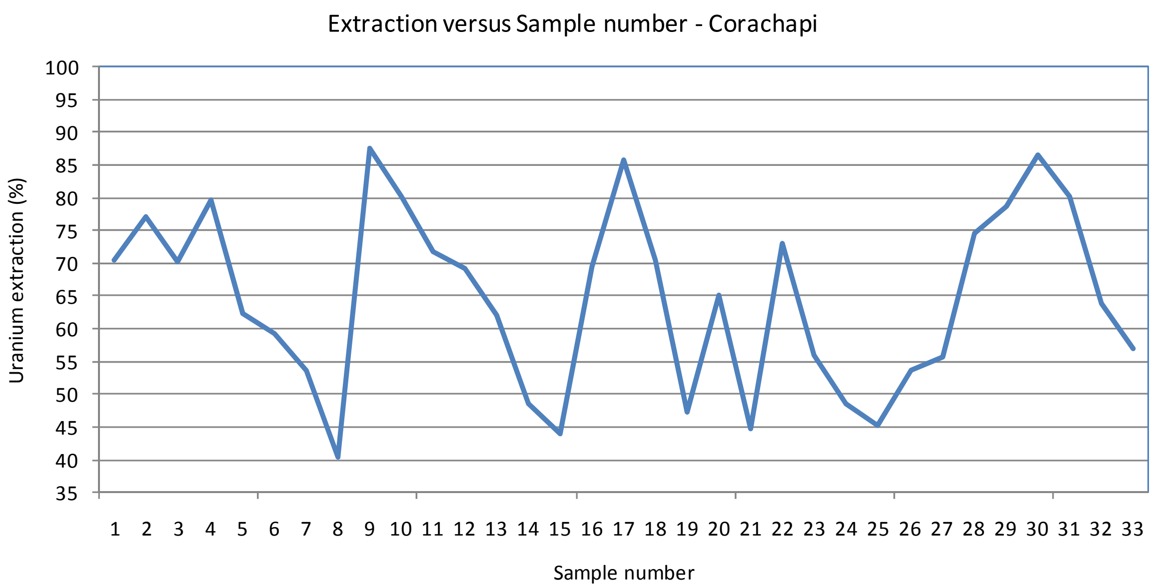


Figure 13-2: Corachapi Drill Core Bottle Roll Extractions – TECMMINE 2010

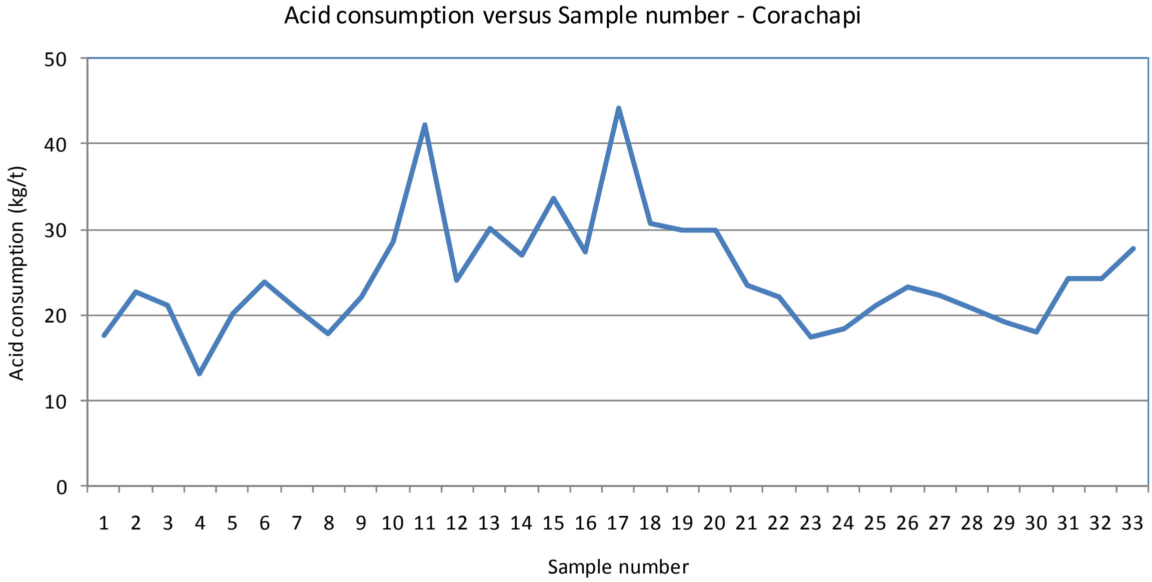


Figure 13-3: Corachapi Drill Core Bottle Roll Acid Consumption – TECMMINE 2010

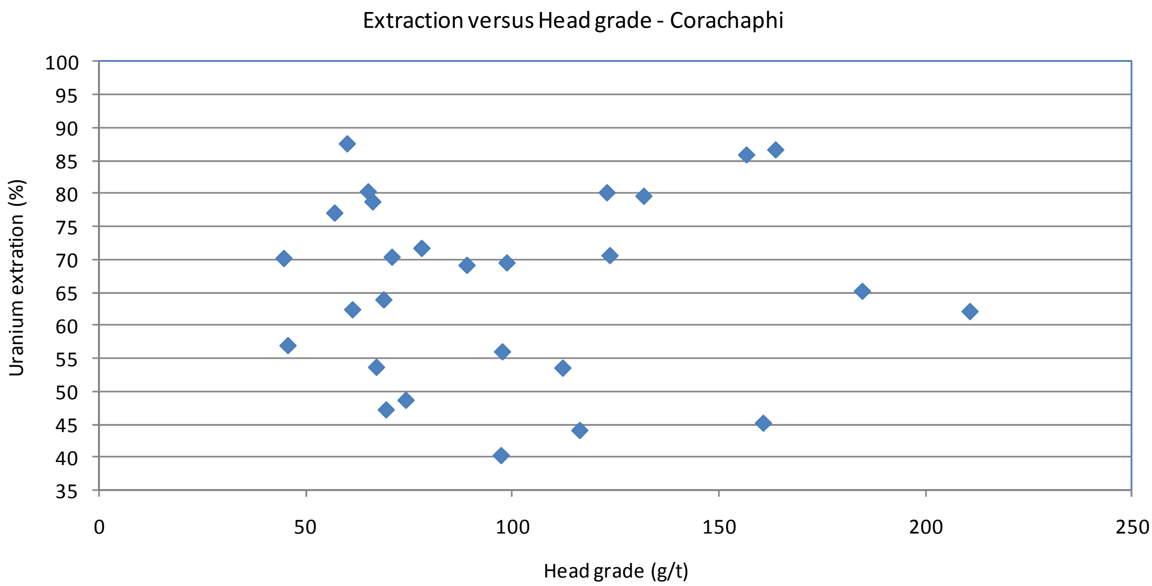


Figure 13-4: Corachapi Drill Core Extraction vs Head Grade – TECMMINE 2010

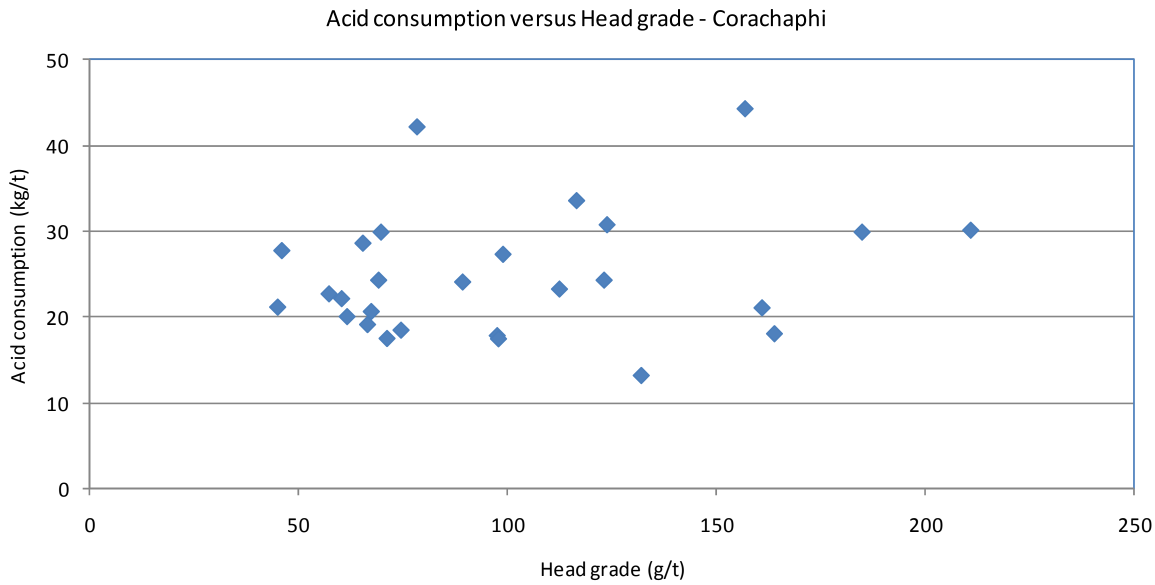


Figure 13-5: Corachapi Drill Core Acid Consumption vs Head Grade – TECMMINE 2010

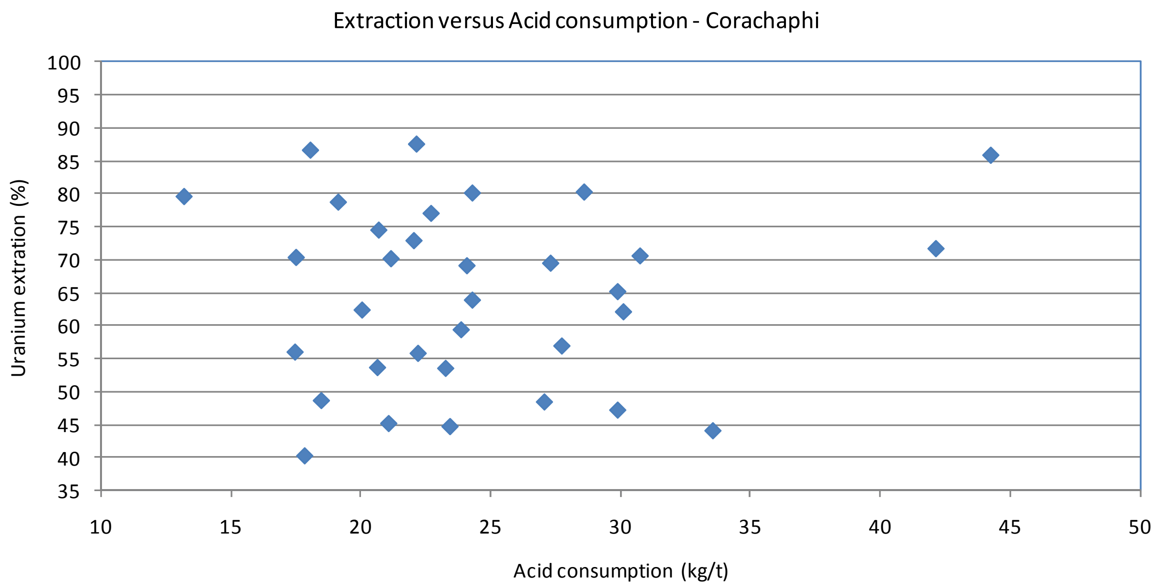


Figure 13-6: Corachapi Drill Core Extraction vs Acid Consumption – TECMMINE 2010

13.3.2 HEAD ASSAY VARIANCE

The possible existence of a consistent discrepancy in the bottle roll sample head assays was noted. The actual laboratory head assay consistently appeared to be lower than the calculated head assay, see Figure 13-1. Assays of the two products from the leach test, the leach solution tenor and the laboratory assay of the residue, are used to back calculate the uranium content of the head (test) sample. Due to the difficulties in obtaining a representative sample for the laboratory head assay, the calculated head assay is often considered as the more accurate of the two. Leach liquor is homogenous and easy to measure and sample and contains the majority of the uranium. Whilst obtaining a representative leach test residue sample is also difficult, there is less uranium in the residue, so analytical error has a smaller effect on the calculated head assay.

A statistical analysis (GBM, 2011) (33) conducted on the 200 bottle roll test assays (TECMMINE, 2010) (31) (32) found that there was a correlation between calculated and laboratory assay head grade figures. For scaling consideration the natural logarithms of the assays were plotted against each other. A regression analysis was performed. Whilst the correlation for all of the data points was good, visual inspection of the normal probability plot indicated that the correlation changed at the natural logarithm of grade value of approximately 4.7, see Figure 13-7. This value equates to a grade of 110 ppm U.

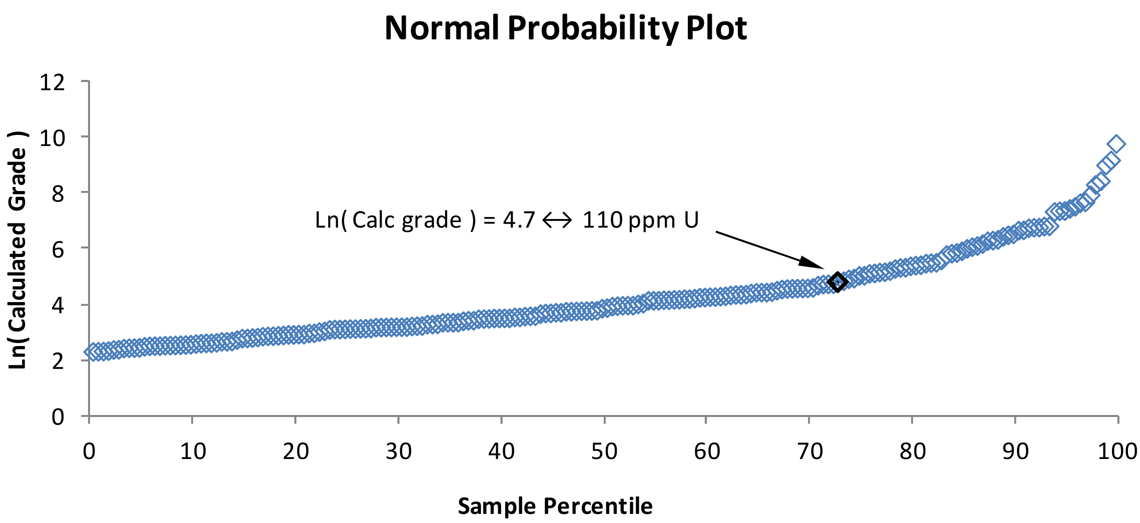


Figure 13-7: Normal Probability Plot of the Logarithm of Calculated Head Grade (All Grades)

The natural logarithm of the calculated uranium grade was plotted against the natural logarithm of the assayed grade for grades less than (or equal to) 110 ppm U, see Figure 13-8, and also for grades greater than 110 ppm U, see Figure 13-9.

The correlations are as follows:

For grade ≤ 110 ppm U: Calculated Grade = Assayed Grade^1.0287

For grade > 110 ppm U: Calculated Grade = Assayed Grade^1.0203

To illustrate the effects of the correlations, for a laboratory uranium assay of 100 ppm U, the analysis suggests the calculated grade is 100^1.0287 = 114 ppm U. As mentioned, the calculated grade is considered a more accurate indication of the grade as it is determined from the amount of uranium proven to exist from a solution assay and a tailings assay of the spent PEM when treated by acid.

For a laboratory assay of 200 ppm, the analysis suggests the calculated grade, the 'true' grade, is 200^1.0203 = 223 ppm U. Both of these examples show a significant difference exists between the laboratory assay and the calculated, true grade, which may have a significant effect on the design of the processing facility.

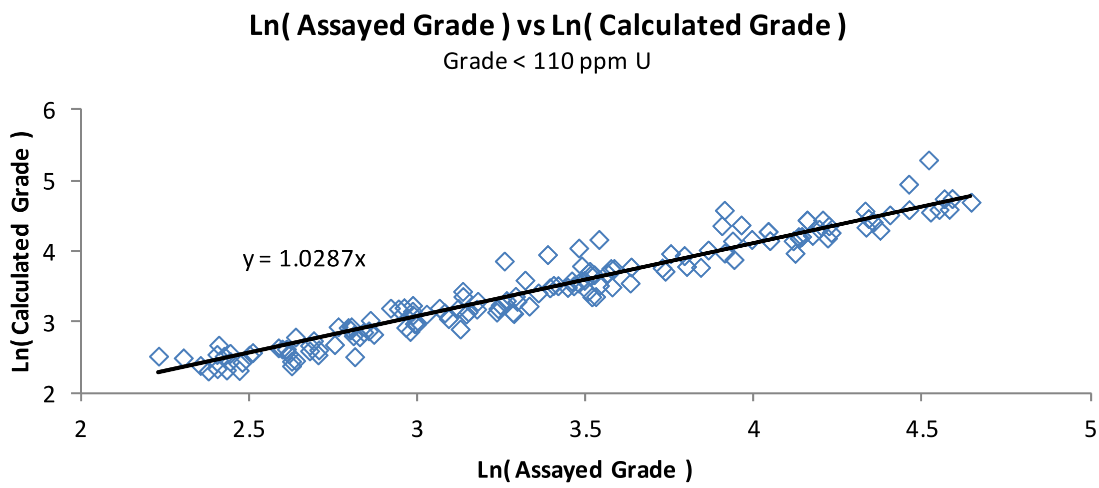


Figure 13-8: Natural Logarithm Plot of Calculated Versus Assayed Grade (Grades < 110 ppm U)

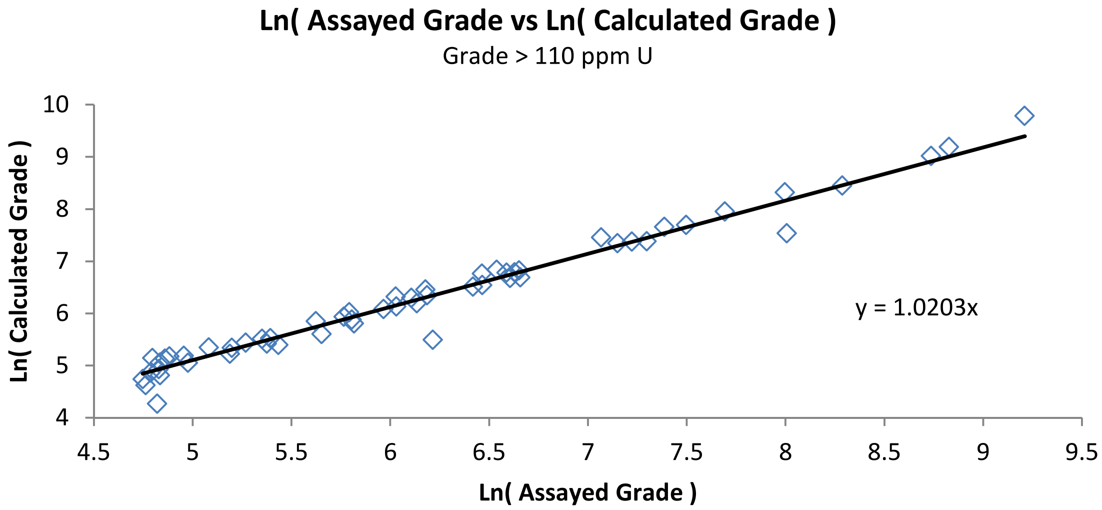


Figure 13-9: Natural Logarithm Plot of Calculated Versus Assayed Grade (Grades > 110 ppm U)

13.3.3 COLUMN LEACH TESTS

Column leach tests have been conducted initially in small column format in Lima using potable and distilled water. The tests have progressed to large column format at Isivilla using site water.

13.3.3.1 SMALL COLUMN TESTS

The TECMMINE testwork in April 2008 included column leach tests (TECMMINE, 2008) (30). Tests were conducted in a 7 inch diameter column in Lima. Zone 1 and Zone 2 samples were tested with and without pre-leach acid treatment. The pre-treatment consisted of adding a strong acid solution (60 g/L) to achieve 10 % moisture and then letting the sample stand for 24 hours before column irrigation commenced. The PEM samples consisted of 85 kg to 95 kg, 100 % passing two inches in size. Open circuit irrigation, at a rate of 8 L/h/m², took place over 31 days. The test conditions are listed in Table 13-3. The uranium extraction and acid consumption results are shown in Table 13-4. The leach extraction curve followed the expected general sigmoidal shape after a delay of six days, as the leach solution percolated down through the column. The initial slow extraction accelerated during days 14 to 18 before slowing again as the test approached completion.

Table 13-3: Column Leach Test Conditions – TECMMINE April 2008

Parameter	Unit	Value
Particle Size (100 % passing)	mm	50.8 (2 inches)
Weight of PEM (No pre-treatment)	kg	92 – 95
Weight of PEM (Pre-treatment)	kg	85 – 87.5
Leach solution flowrate	$L/h/m^2$	8
Acid concentration	g/L	15
Soaking time (Pre-treatment)	hour	24
Leaching time (both methods)	day	31

Table 13-4: Column Leach Test Results – TECMMINE April 2008

Parameter	Unit	Zone 1 Values	Zone 2 Values
PEM uranium grade	g/t	75	509
Uranium extraction (No pre-treatment)	%	84.4	98.6
Uranium extraction (Pre-treatment)	%	77.3	69.5
Acid consumption (No pre-treatment)	kg/t	14.6	14.5
Acid consumption (Pre-treatment)	kg/t	17.0	16.4

Without pre-treatment, extractions of 84 % to 98 % were achieved with an acid consumption of approximately 14.5 kg/t. With pre-treatment, extractions of 69 % to 77 % were achieved with an acid consumption of 16.4 kg/t to 17 kg/t. Pre-treatment was found to increase uranium extraction in the early stages of leaching but the increase was not sustained through to the end of the leach test. The report concluded that pre-treatment did not reduce the leach cycle time. The leach extractions, particularly for the Zone 2 tests, are comparable to the bottle roll leach tests. Hence, it was concluded that the PEM is porous and allows good lixiviant percolation. An additional recommendation was to conduct a size by size assay to delineate the uranium distribution.

In May 2009, TECMMINE conducted column leach tests on PEM samples sized to 100 % passing 1, 2 and 3 inches (34). The results are shown in Table 13-5 and

Table 13-6.

Table 13-5: Column Leach by Size Test Conditions – TECMMINE May 2009

Description	Value
Sample weight, kg	80
Irrigation rate, (L/h)/m^2	8
Acid concentration, g/L	15
Leach solution pH	< 2
Type of continuous irrigation	Drip
Commencement date	6 March 2009

Table 13-6: Column Leach by Size Test Results – TECMMINE May 2009

Crush Size	Extraction, %	Leach time, days	Acid consumption, kg/t	Flowrate, m^3/t
Minus 1 inch	80.63	51	13.29	2.87
Minus 2 inch	81.59	56	9.04	3.13
Minus 3 inch	78.48	65	5.69	3.75

These tests all show similar extraction figures but the time to achieve an extraction of about 80 % increases with particle size; leach time for minus 3 inch material shows about a 30 % increase when compared to that for minus 1 inch material. However, acid consumption is only 43 % of that required for the minus 1 inch material. Comparing the minus 1 inch and minus 2 inch material, the latter requires about 16 % more leach time, but the acid consumption is only 63 % of the quantity consumed by the minus 1 inch material.

In the tests carried out on the minus 2 inch material in April 2008, the acid consumption was 16 kg/t and 17 kg/t (Table 13-4), whereas in this series, it was 5 kg/t to 13 kg/t. It was postulated by TECMMINE that this difference could be due to different content of acinicidas (a term to define acid consuming metal components).

Chemical analysis conducted at CIMM, Peru gave the following uranium head assays for the column material:

- Column 1, minus 1 inch: 32.3 ppm;
- Column 2, minus 2 inch: 40.3 ppm; and
- Column 3, minus 3 inch: 28.7 ppm.

As might be expected, the results observed in both these series of testwork showed a relationship between recovery and head grade. Table 13-7 contains a summary of the small column leach testwork results.

Table 13-7: Small Column Leach Testwork Summary

Sample	Head grade, ppm U	Extraction, %
Zone 1, 2008 testwork	28.3 (calc.)	84.37
Zone 2, 2008 testwork	509	98.58
Column 1, minus 1 inch	32.32	80.63
Column 2, minus 2 inch	40.29	81.59
Column 3, minus 3 inch	28.69	78.38

On the basis of these results, head grades in the region of 30 ppm to 40 ppm would be expected to achieve extractions of approximately 80 %.

13.3.3.1.1 VERY LOW GRADE TESTWORK

A very low grade leach test programme was conducted in Lima, concurrently with the large column tests on low grade PEM in Isivilla. In Lima, very low-grade PEM (20 ppm to 30 ppm) was tested at minus 2 inch and minus 1 inch sizes. The minus 2 inch test conditions and results are contained in Table 13-8 and Table 13-10, respectively.

In July 2011 the results for the minus 1 inch test were reported (TECMMINE, 2009) (35). The test conditions and results are contained in Table 13-9 and Table 13-11 respectively.

Table 13-8: Very Low Grade (minus 2 inch) Column Leach Test Conditions – TECMMINE 2011

Description	Column 1	Column 2
Test location	Lima	Lima
Sample weight, kg	78	77
Bed height, m	2.34	2.37
Column diameter, in	7	7
Irrigation rate, (L/h)/m^2	8	8
Percolation velocity, m/d	1.07	1.05
Acid concentration, g/L	15	15
Leach solution pH	< 2	< 2
Type of continuous irrigation	Drip	Drip
Commencement date	16 Feb 2011	16 Feb 2011
Duration, days	32	29

Table 13-9: Very Low Grade (minus 1 inch) Column Leach Test Conditions – TECMMINE 2011

Description	Column 3	Column 4
Test location	Lima	Lima
Sample weight, kg	~80	~80
Bed height, m	2.37	2.39
Column diameter, in	7	7
Irrigation rate, (L/h)/m^2	8	8
Percolation velocity, m/d	1.40	1.07
Acid concentration, g/L	15	15
Leach solution pH	< 2	< 2
Type of continuous irrigation	Drip	Drip
Commencement date	25 Apr 2011	25 Apr 2011
Duration, days	46	46

Table 13-10: Very Low Grade (minus 2 inch) Column Leach Testwork Summary – TECMMINE 2011

Description	Column 1	Column 2
Head grade (calculated), ppm U	22.3	19.7
Feed 100 % passing size, in	2	2
Extraction, %	72.6	57.9
Acid consumption, kg/t	11.3	9.8
Leach solution tenor, ppm	16	11
Residue grade, ppm	6.1	8.3
Total irrigation, m^3/t	1 808	1 759
Column settlement, %	0.17	0.21

Table 13-11: Very Low Grade (minus 1 inch) Column Leach Testwork Summary – TECMMINE 2011

Description	Column 3	Column 4
Head grade (calculated), ppm U	38.2	27.2
Feed 100 % passing size, in	1	1
Extraction, %	85.4	80.2
Acid consumption, kg/t	12.6	16.0
Leach solution tenor, ppm	33	22
Residue grade, ppm	5.6	5.4
Total irrigation, m^3/t	2 791	2 669
Column settlement, %	0.25	0.71

The extraction results for the very low grade, 2 inch PEM are much lower (57 % to 72 %) than the 1 inch PEM (80 % to 85 %) but they show that leaching is possible.

With the minus 1 inch column leach tests achieving extraction of 80 % to 85 %, the indication is that for low grade material, a finer crush size could yield improved extraction. However, the improved extractions correspond also to higher head grades, some 120 % to 150 % higher (with comparable residue grades). This feature indicates the higher the head grade, the more leachable uranium available for extraction, so the higher the extraction.

Both the 1 inch and 2 inch material appear to load into a column in a stable manner. Column settlement in each test was less than 1 %.

The leach profiles for Column 1 and Column 2 were uniform in shape, similar to the profiles in the large column leach tests described later in Section 13.3.3.2. However, the leach profiles for Column 3 and 4 were non-uniform. The profile for Column 4 is shown in Figure 13-10. The profile appears to proceed as expected initially. However, there is a surge in uranium extraction mid-way through the leaching period. This has been attributed to the very low-grade PEM not displaying any veins of uranium but rather containing only disseminated uranium, typically as rhyolite. It appears that the acid solution requires extra time to penetrate the PEM and reach the ‘locked’ uranium particles. Leaching of this uranium then occurs and reports to the PLS and the PLS tenor increases. The leach kinetics impact upon the heap design.

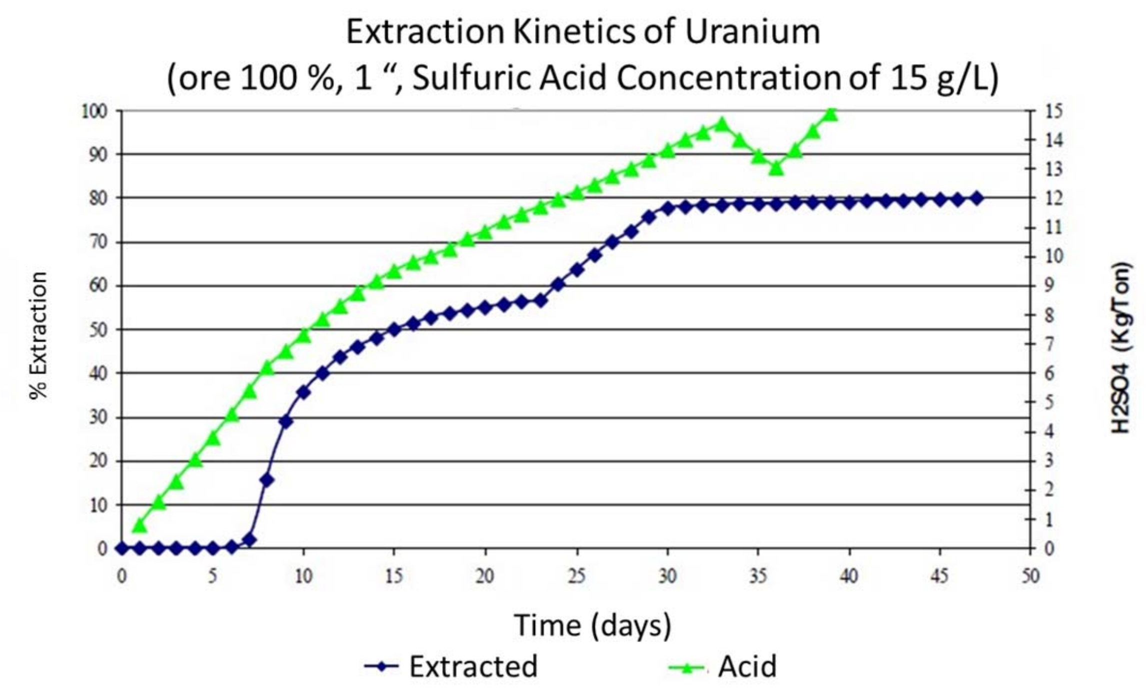


Figure 13-10: Column 4 Very Low Grade Column Leach Test Profile – TECMMINE 2011

13.3.3.2 LARGE COLUMN TESTS

Utilising the testwork knowledge and experience acquired from the Lima leach tests, four column tests were undertaken at Isivilla using site water (with site ambient conditions) from late 2010 to early 2011. The four tests have been completed (TECMMINE (36) (37) (38) (39) and Table 13-12 details the test conditions.

Table 13-12: Large Column Leach Test Conditions – TECMMINE 2011

Description	Column 1	Column 2	Column 3	Column 4
Test location	Isivilla	Isivilla	Isivilla	Isivilla
Sample weight, kg	~490	489	437	427
Bed height, m	~2.58	2.59	2.60	2.54
Column diameter, in	16.5	16.5	16.5	16.5
Irrigation rate, (L/h)/m^2	8	8	8	8
Percolation velocity, m/d	1.03	0.94	1.16	1.08
Acid concentration, g/L	15	15	15	15
Leach solution pH	<2	< 2	< 2	< 2
Type of continuous irrigation	Drip	Drip	Drip	Drip
Commencement date	14 Mar 2011	16 Jan 2011	06 Dec 2010	06 Dec 2010
Duration , days	49	45	50	50

Table 13-13 lists the results of the column leach tests. Uranium extraction achieved is 91 % to 98 %[2]. Acid consumption was 9.6 to 10.5 kg/t. Figure 13-11 shows the leach extraction kinetics for the Column 3 test and the acid consumption curve. The extraction kinetics curves for the Column 2 and 4 tests exhibited similar behaviour. Comparing Column 2 to Column 3 indicates that reducing the feed top size from 1 inch to 2 inch does not significantly impact extraction or acid consumption results. Comparing Column 3 to Column 4 indicates that feed grade does not significantly impact extraction or acid consumption results. In fact, extraction and acid consumption are relatively consistent across the range of grade and feed size tested in the large format column.

[2] Column 2 (Isivilla) test results were initially reported as 92 % extraction, head grade of 383 ppm U and residue grade of 30 ppm U. This was the result of an instrument recalibration (109).

Table 13-13: Large Column Leach Testwork Summary – TECMMINE 2011

Description	Column 1	Column 2	Column 3	Column 4
Head grade, ppm U	85	359	287	98
Feed 100 % passing size, in	1	1	2	2
Extraction, %	91.0	98.3	96.1	93.1
Acid consumption, kg/t	10.2	10.5	9.6	10.3
Leach solution tenor, ppm	78	353	276	91
Residue grade, ppm	8	6	11	7
Total irrigation, m³/t	2.526	2.463	2.948	2.995
Column settlement, %	1.05	0.46	0.65	0.28

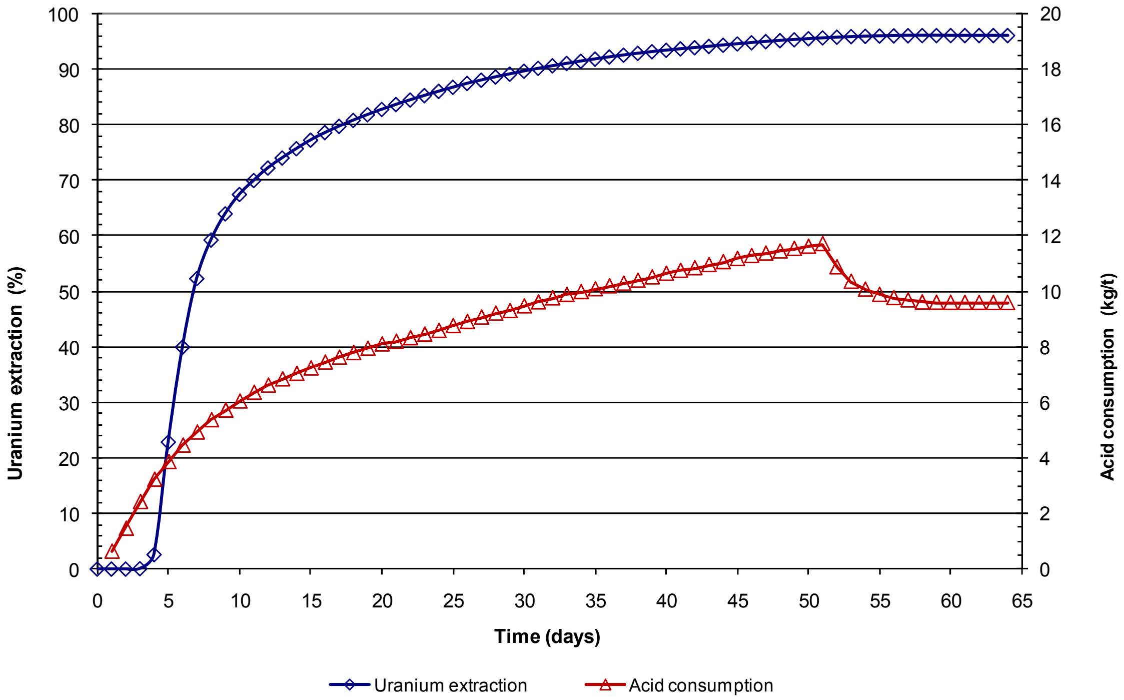


Figure 13-11: Column 3: Kinetics of Extraction of Uranium (100 % passing 2"; 15 g/L of H_2SO_4)

Eight-hour bottle roll tests were conducted on feed samples from each of the eight column leach tests detailed above in Table 13-8 through to Table 13-13. The samples were ground to a 200 mesh (74 µm). The uranium extraction and acid consumption details for the bottle roll tests are contained in Table 13-13 and Table 13-14, which also contains the column leach test results for comparison. Acid consumption in the bottle roll tests are between 33 % and 100 % higher than the column leach tests. The laboratory test report attributes this to the finely ground bottle roll feed sample exposing more uranium and other leachable elements to the sulfuric acid solution (TECMMINE, 2011) (35). Bottle roll extractions are less than or equal to column leach extractions. Whilst not explicitly stated, this is presumably due to the same cause. That is, the acid is also dissolving other accompanying elements in the uranium PEM. If the bottle roll tests proceeded for longer than eight hours, then the uranium extraction may have increased to the column leach extraction results.

Table 13-14: Column Leach and Bottle Roll Test Summary – TECMMINE 2011

Column	Location	Size [inch]	Column leach tests			Bottle roll tests		
			Grade [g/t]	Extraction [%]	Acid consumption [kg/t]	Grade [g/t]	Extraction [%]	Acid consumption [kg/t]
1	Lima	2	22.3	72.6	11.3	44.2	58.7	14.3
2	Lima	2	19.7	57.9	9.8	19.4	57.3	12.6
3	Lima	1	38.2	85.4	12.7	30.0	71.4	16.1
4	Lima	1	27.2	80.2	16.0	30.0	67.3	21.2
1	Isivilla	1	82	91.0	10.2	128	74.8	21.5
2	Isivilla	1	383	92.2	10.5	548	90.4	22.8
3	Isivilla	2	287	96.1	9.6	354	86.2	15.1
4	Isivilla	2	98	93.1	10.3	174	86.0	17.0

These site column leach test results show that the PEM is amenable to leaching. However, significant differences between the Isivilla and Lima extraction and acid consumption exist. Further testwork and analysis that duplicates the latest results should be undertaken to increase confidence in these results and possibly explain the source of the differences.

13.3.3.3 MINERGIA LEACH TESTS

Cameco held an interest in Minergia before it was acquired by Azincourt and conducted early stage testwork in 2011 and 2013 (Cameco Corporation, 2013) (40). For the purpose of this report the testwork will be referred to as the Minergia testwork, being undertaken by Cameco. The uranium grade of the samples tested ranged from approximately 260 to 800 ppm U.

Four column leach tests were conducted in 76 mm diameter, 2 litre columns. The average PEM height was 40 cm. The columns were irrigated with a 1.4 pH sulphuric acid solution at 3 mL/min, which equals approximately 40 L/h/m².

Table 13-15: Column Leach Test Summary – Cameco 2013

Column	Preparation	Size [inch]	Column leach tests			
			Grade [g/t U]	Time [h]	Extraction [%]	Acid consumption [kg/t]
1	Acid agglomeration	1/2	626.3	~ 336	~ 97	~ 5
2	Acid agglomeration + H_2O_2 oxidation to +550 mV Eh	1/2	344.5	~ 336	~ 97	~ 5
3	Acid agglomeration	1/2	344.5	~ 336	~ 97	~ 5
4	Acid agglomeration	3/8	344.5	~ 336	~ 97	~ 5

The highlights of these results are the high extraction and low acid consumption. Test observations of the tenor of the pregnant leach solution (column underflow) indicated that most of the uranium dissolution occurred in the pugging[3] and agglomeration stage and the column leach stage was effectively an PEM washing stage.

Whilst the final extraction level achieved was consistent across the Minergia testwork, the initial rate of extraction for the minus 3/8 inch material was slower, suggesting that “the mass transfer in this material was likely more diffusion controlled due to smaller average pore spaces.” (Cameco Corporation, 2013) (40).

Notable differences in these tests, compared to the earlier column leach tests are:

- Scale: The columns are much smaller, with diameter of 76 mm, compared to 170 mm and 400 mm;
- Particle size: The samples were crushed to minus 12.5 mm and minus 9.5 mm, compared to minus 25.4 mm and minus 50.8 mm; and
- Agglomeration: The crushed samples were agglomerated after pugging with 5 kg/t of sulphuric acid in a 45 % aqueous solution (final moisture content of 10.4 %), compared to no agglomeration/pugging.

The report on the Minergia testwork projected the column leach test results to a full-scale, 7 m heap. Assuming an extraction 5 % less than full possible extraction, a 70 day leach stage was determined. The total leach cycle was not specified.

These laboratory column leach test results show that the PEM introduced into the project through the Minergia acquisition is amenable to leaching, with high extraction and low acid consumption. Further

[3] to mix or knead (clay) with water to form a malleable mass or paste, often in a pug mill (Collins English Dictionary)

testwork and analysis that duplicates these results at a larger scale should be undertaken to increase confidence in these results. Mineralogical analysis of the PEM is required to fully explain the wide range of acid consumption (low for the Minergia testwork, medium for other columns and high for other bottle roll tests).

13.3.3.4 PHENOMENOLOGICAL HEAP LEACH MODELLING AND POROSITY

The length of time required to conduct and report on laboratory and column leach testing highlighted the time it would take to investigate the effect of test conditions, either by way of ad hoc or factored experimentation. In 2012, Process Systems Enterprise (PSE) was commissioned to develop a phenomenological model from first principles to allow desktop investigation of variations in leaching conditions (Process Systems Enterprise, 2012) (41). Such investigation would assist and expedite column leach test condition selection for the determination of optimum design criteria.

Specific goals of the modelling were to study the effect of particle size distribution, heap height and acid solution irrigation rates on the performance of the leaching process. The model would enable the quantification of the importance of each variable and allow the investigation of other operating variables such as leaching duration, cascade leaching configurations and material grade.

The heap leach model was comprised of the following submodels:

1. Hydraulic model. The Van Genuchten equation, which relates water content and pressure in a porous material, was employed to capture the hydraulic behaviour;
2. Kinetic model. Uranium leaching was assumed to be according the dissolution of meta-autunite. A fourth order reaction was assumed, according to the stoichiometry, with the rate estimated from the literature; and,
3. Pellet (with mass transfer) model. Particle size distributions, of sizes of 1, 2 and 3 inches, were assumed and a mass transfer or diffusion coefficient for the system was estimated using a Maxwell-Stefan formulation model for the movement of solutes within the pellets, assuming a single diffusion coefficient.

Three particle sizes, three irrigation fluxes and four heap heights were studied and resulted in a set of thirteen results. This initial version of the phenomenological model indicated a benefit in a smaller crush size and illustrated the effects of heap height and irrigation flux on PLS tenor over time. Predicted acid consumption was significantly lower than test consumption data.

The test report included recommendations for experimental analysis of additional acid consuming mineral species and particle porosity.

The Minergia testwork (Cameco Corporation, 2013) (40) included porosity measurement, by back scattered electron imagery of polished sections of several –12 mm particles, to estimate the solution accessibility and size distribution of the uranium mineral grains in two PEM blends. Accessibilty was found to be 88 – 92 % and average porosity, 15.2 – 18.5 %. The results suggest that some moisture retention after leaching would be expected.

Clemex image analysis software was used to identify pore area and size distribution. The studied particles had “well-textured uniformly distributed pore structures that would likely give overall easy pathways for leach solutions through the rock”. Diffusion controlled mass transport through the PEM during leaching would be expected since >50 % of the pores being <100 µm in size.

13.3.4 RECOVERY

A good deal of column leach test work has been completed on the project PEM. The large column leach test on 2 inch material was judged to provide the best reference case for design criteria specification. Therefore, the results of the Isivilla column leach test on 2 inch material (Column 3) was analysed further to determine the estimated optimum heap height and recovery (GBM, 2012) (42). The analysis considered pad construction and maintenance costs and yellowcake revenue at a series of heap height and leach cycle times. The optimum height was determined to be 7 m, with a corresponding recovery of 88 %.

13.3.5 ISIVILLA CONFORMANCE

Detailed bottle roll test results of samples from within Isivilla are not currently available. However, the 2013 report by Cameco (40) reported extraction rates of approximately 97 % in small column leach tests and projected 80 to 90 % extraction at heap leach scale. These results are consistent with the results achieved for Colibri and Corachapi Complexes. Considering these points, it was concluded that further column leach tests of the Isivilla PEM were not required to complete this PEA.

13.3.6 KIHITIAN CONFORMANCE

Kihitian bottle roll test results were compared to the Colibri and Corachapi bottle roll test results (GBM, 2013) (43) and the leaching behaviour was found to be consistent with that of the Colibri and Corachapi PEM. Therefore, it was concluded that column leach tests of the Kihitian PEM were not required to complete this PEA.

13.3.6.1 BOTTLE ROLL TEST RECOVERY

Kihitian bottle roll test results (44) were collated with results from Colibri and Corachapi (TECMMINE, 2007, 2010, 2011, 2012, 2013) (45) (46) (47) (48) (49) in a database file (43).

The recovery results versus grade results are as shown in Figure 13-12 and demonstrate the Kihitian results are consistent with those from the Colibri and Corachapi Complexes. This leach performance conformance eliminates the requirement for further column leach tests of Kihitian PEM for this report.

The column leach test results (TECMMINE, 2008, 2011, 2013) (30) (35) (36) (37) (38) (39) (40) were also collated with the bottle roll test results. The recovery results versus grade results are as also seen in Figure 13-12 and demonstrate the column leach test results are consistent with the bottle roll tests.

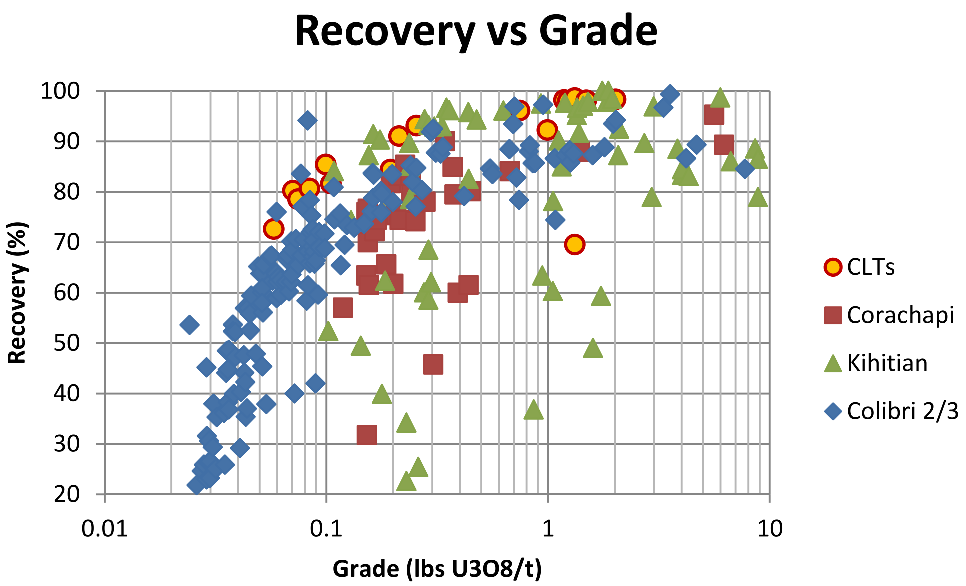


Figure 13-12: Bottle Roll and Column Leach Test Recovery by Grade

13.3.7 ACID CONSUMPTION

Figure 13-13 shows how acid consumption varies with grade. For grades less than approximately 10 lb U_3O_8/t (approximately 3 850 ppm U) acid consumption is relatively consistent and averages approximately 25 kg/t.

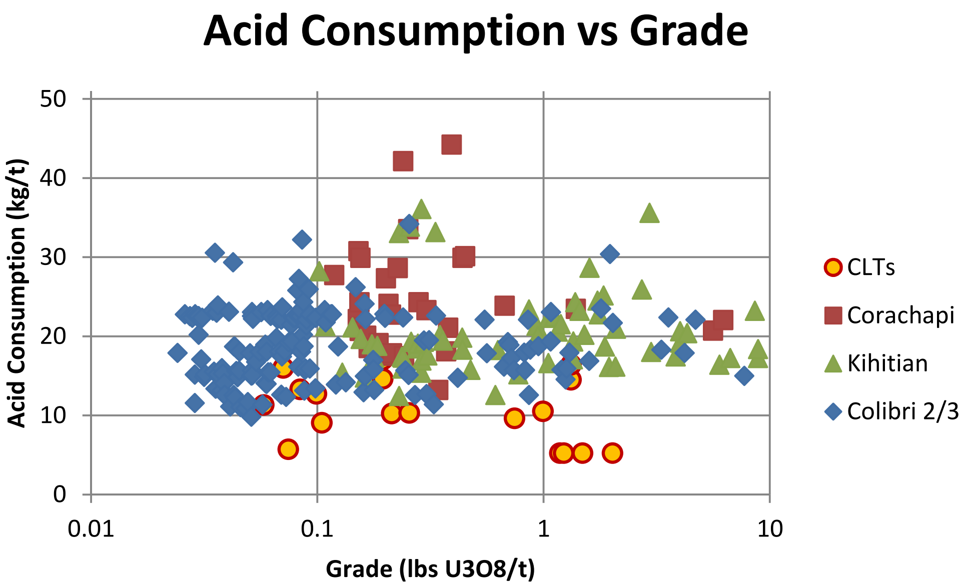


Figure 13-13: Acid Consumption by Grade

The acid consumption shown in Figure 13-13 is total acid consumption, which is made up of theoretical consumption (acid required to extract the contained uranium) and host rock consumption (acid consumption by other minerals in the host rock).

The plot of the theoretical consumption by grade, see Figure 13-14, shows regular, expected behaviour. Acid consumption increases linearly with uranium content. At a grade of approximately 1 lb U_3O_8/t, theoretical acid consumption is approximately 0.3 kg/t.

Subtracting the theoretical acid consumption from the total acid consumption yields the host rock acid consumption, which is plotted against grade in Figure 13-15. Again, for grades less than approximately 10 lb U_3O_8/t host rock acid consumption is relatively consistent and averages approximately 25 kg/t.

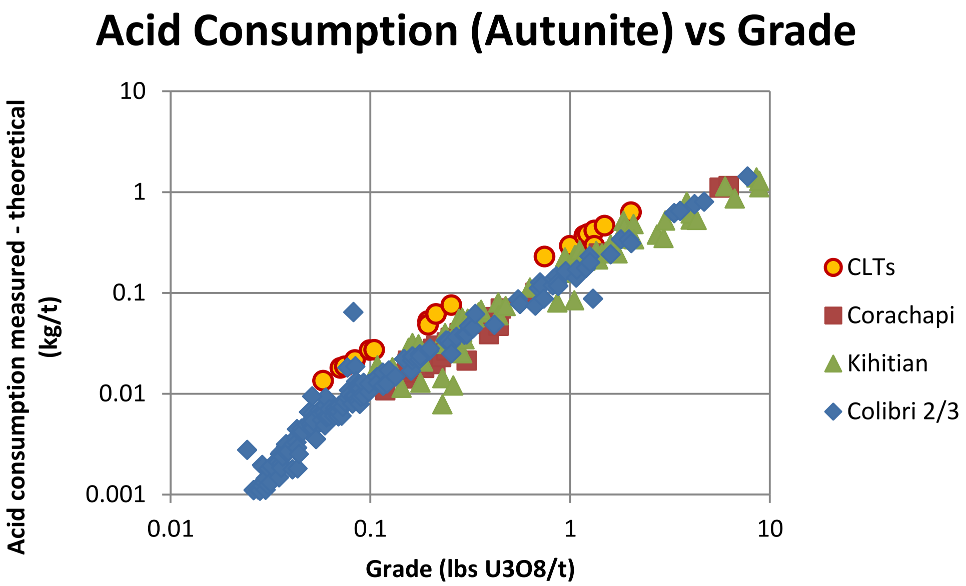


Figure 13-14: Theoretical Acid Consumption by Grade

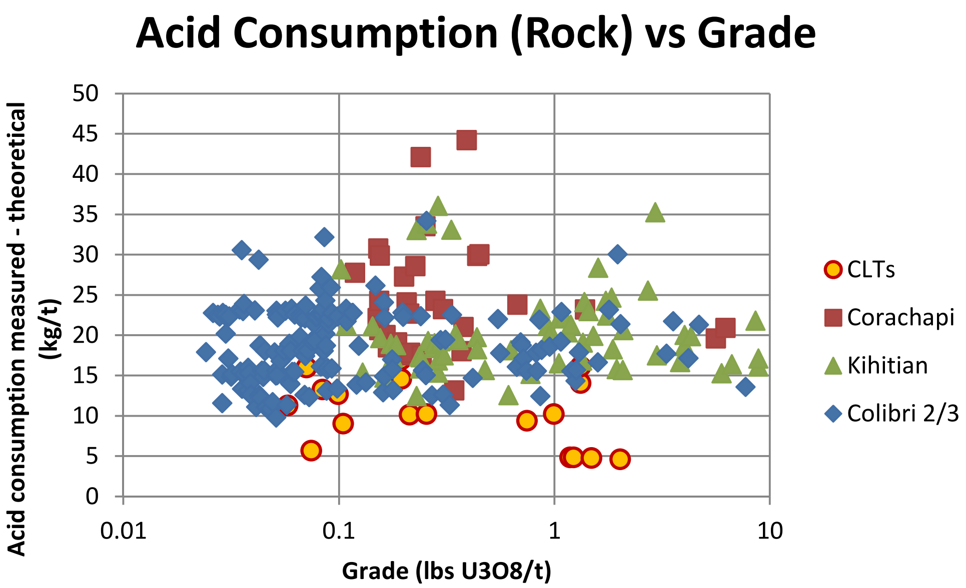


Figure 13-15: Host Rock Acid Consumption by Grade

Note that as grade increases, there is a decrease in acid consumption per pound of yellowcake, see Figure 13-16.

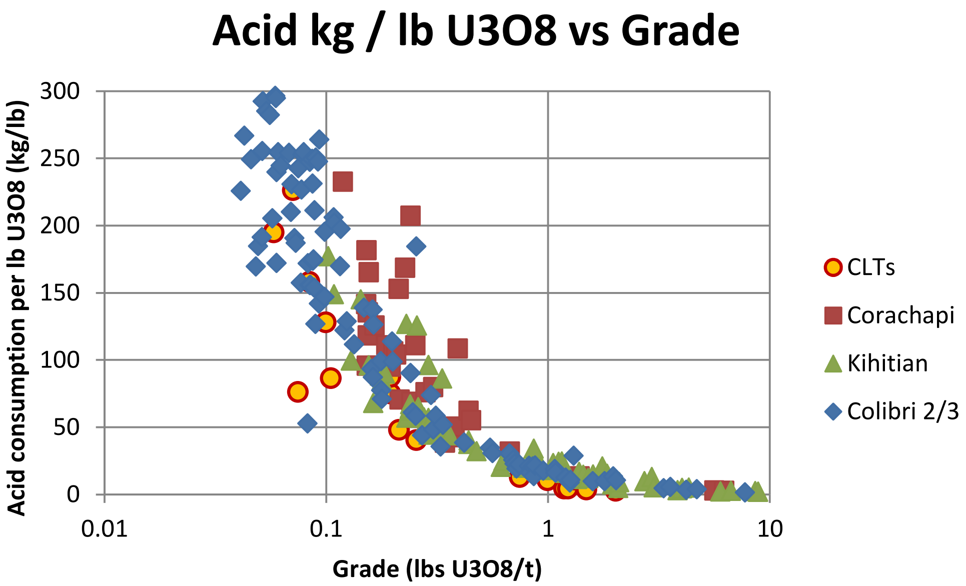


Figure 13-16: Acid Consumption per Pound of Yellowcake by Grade

Comparing Figure 13-14 and Figure 13-15 shows that the underlying high acid consumption is due to host rock acid consumption. Acid consumption is 10 to 100 times higher than the theoretical consumption required to dissolve the contained uranium. Identification and quantification of the acid consuming minerals in the project PEM is required to explain the acid consumption. It is assumed that the bottle roll tests have relatively higher acid consumption due to the bottle roll test length and particle size allowing more non-uranium component consumption. One possible theory to reduce acid consumption is to reduce the acid concentration in the leach solution (increase the pH) and thus reduce its activity and the availability of free acid to react with non-uranium minerals (Bannernan Resources Ltd., 2012) (50).

The grade expected to apply in the current PEA is between approximately 0.5 lb U_3O_8/t and 1 lb U_3O_8/t. From the bottle roll test results, an acid consumption of 20 to 25 kg/t would be expected. However, the column leach test results, also shown in Figure 13-13, lie in the 5 to 17 kg/t range. Therefore, acid consumption for heap leaching is expected to be lower. To assist in the estimation of the projected acid consumption, the consumption per pound U_3O_8 in the column leach tests was analysed against grade (43), see Figure 13-17.

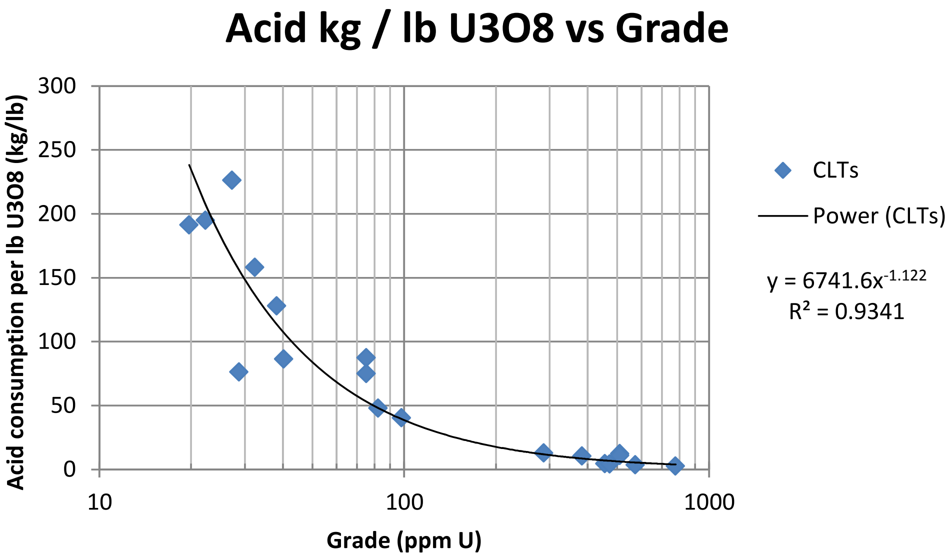


Figure 13-17: Acid Consumption per Pound of Yellowcake by Grade

The power correlation in Equation (1) provides good explanation of the variation of acid consumption with grade.

$$Acid = 6742\ grade^{-1.122}(1)$$

For a feed grade of 288 ppm U_3O_8 (244 ppm U) the estimated acid consumption is 14.1 kg/lb U_3O_8, which translates to 9.0 kg/t.

13.3.8 ION EXCHANGE

There are several competing technologies for uranium recovery from pregnant leach solutions. After investigation and a number of assumptions, ion exchange (IX) has been identified as the most appropriate recovery technology for the currently proposed Macusani heap leach operations (GBM, 2010) (51). It must be understood, however, that this preliminary selection is based upon some very significant assumptions that require further work to confirm.

Another recovery technology due serious consideration is solvent extraction (SX). One of the main differences between the two processes from a physical/equipment perspective is that IX operates by adsorption of uranium from aqueous pregnant leach solution (PLS) onto a solid resin, whereas SX operates by mass transfer of uranium from the aqueous PLS into an organic solvent. The SX process preferentially upgrades the uranium from the aqueous phase and is stripped from the loaded organic phase with concentrated acid.

The characteristics of the PLS, and specifically the uranium tenor of the PLS, are significant when determining the best recovery method. The following statement details uranium recovery process selection based on prevailing process conditions.

"The leach liquor composition is, intrinsically, one of the main factors affecting process selection, with the uranium tenor playing a key role. IX might be favoured over SX when large volumes of leach liquors with low uranium concentrations are to be treated, because solvent losses are primarily related to the volume of solution handled (2). Brown and Hayden concluded that, for concentrations greater than 0.9 g U_3O_8/L, SX is favoured and below 0.35 g U_3O_8/L, IX would be more economical.

For some operations, SX might be the preferred technology, as it can treat a greater range of acidic feed solutions (lower pH values) without a major change in behaviour, while an increase in the acid concentration has an adverse effect on the loading capacity of the IX resin, hence the higher cost of an IX circuit. If a variable acid content in the leach liquor could be handled, it significantly increases the flexibility of the leaching circuit operation.

The presence of impurities or by-products will play a further key role in influencing process selection (e.g. calcium sulfate fouling, the presence of molybdenum and vanadium species that poison or co-load onto the IX resins, or chlorides suppressing extraction)." (52)

This assessment is echoed by Lunt *et al* (53) who state that for high tenor PLS, solvent extraction will, *"in any event, become progressively more attractive and would undoubtedly give superior economic returns at tenors greater than 0.9 g/l U_3O_8."*

For medium to low grade PEM, the PLS tenor, from a cascade heap leach, is expected to be less than 900 ppm U_3O_8 which is at or below the IX-SX transition point. Therefore, the IX process will be the most cost effective, benefiting from comparatively low capital expenditure and relatively smaller operating expenses. The case for IX is further strengthened by the higher solvent losses to evaporation expected at project elevation.

IX testwork conducted by CWENGA has established that IX is a technically feasible technology for the Macusani leach solutions (54). Samples of leach liquors from Macusani were sent to CWENGA for IX tests. The results indicated that IX is an appropriate technology. A rough process design was presented and a demonstration plant was costed.

Four feed solutions, Type A, B, C and D, of tenor 7 to 531 ppm uranium were first assayed for elements (aluminium, iron, calcium) that may be of commercial interest. The results indicated that only uranium was of commercial interest, see Table 13-16.

Table 13-16: Pregnant Leach Solution Assay Results – CWENGA 2010

Contained Elements	TYPE A ppm	TYPE B ppm	TYPE C ppm	TYPE D ppm
Aluminium	1 035	73	548	731
Iron	239	91	232	234
Calcium	587	255	468	594
Uranium	531	77	19	6.7

Each solution was then contacted with four resins to check for uranium selectivity. Feed solution was contacted with each resin for two hours, decanted and analysed for uranium. Table 13-17 contains the before and after tenors for sixteen shaking tests. The uranium removal efficiencies for these single stage tests are shown in Figure 13-18.

Table 13-17: Ion Exchange Resin Shake Test Results – CWENGA 2010

Description	TYPE A ppm U	TYPE B ppm U	TYPE C ppm U	TYPE D ppm U
Feed solution tenor	537	71	18	6.0
Resin I – discharge tenor	80	2.5	1.6	1.4
Resin II – discharge tenor	186	5.0	2.5	1.7
Resin III – discharge tenor	331	15	9.4	4.4
Resin IV – discharge tenor	507	22	15	5.8

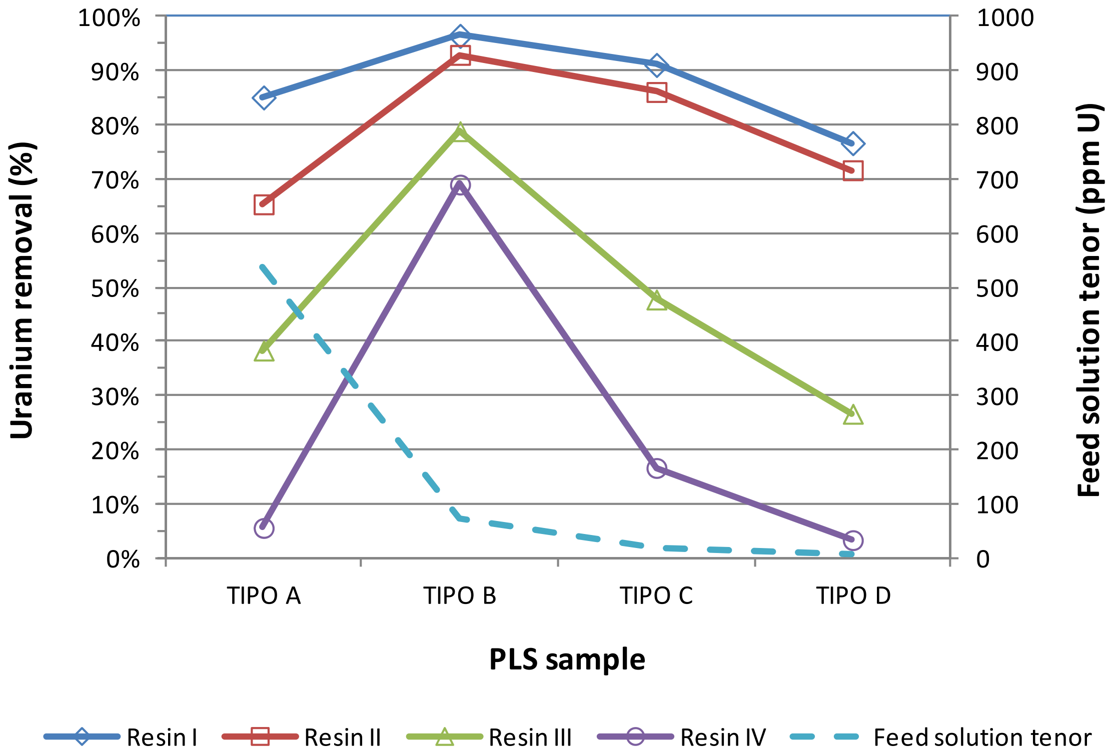


Figure 13-18: Ion Exchange Uranium Removal Results – CWENGA 2010

Two resins exhibited high uranium removal characteristics; Lewatit M500 and Lewatit M62. These two resins were subsequently used for IX testing in standard glass columns, in the two stage arrangement shown in Figure 13-19. Uranium leakage was high for Lewatit M62, isolating Lewatit M500 as the best resin from the four pre-selected resins.

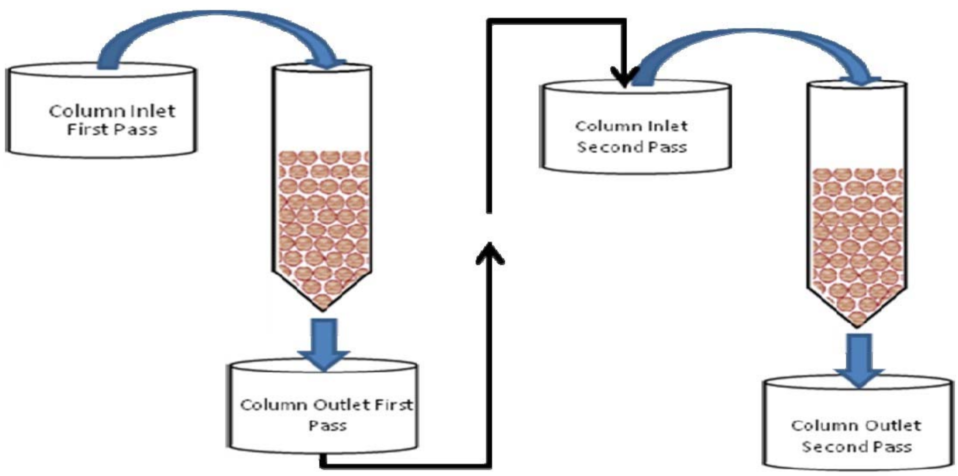


Figure 13-19: IX Column Test Arrangement – CWENGA 2010

Solution sample quantities restricted the number of tests. There was sufficient volume to test Type A (530 ppm U) and a blend of Types B and C (56 ppm U). Figure 13-20 illustrates the two stage IX column tests for Type A. The volume of solution processed is expressed in terms of bed volumes. The bed volume is the volume of the bed of IX resin in each column, 50 mm in this case. The overall uranium removal efficiency throughout the two stage test was greater or equal to 99.6 %. The blended sample test provided comparable results over the volume of solution processed for Type A.

In order to achieve and maintain these high uranium recoveries in a continuous process the resin should be removed from service, before it is completely loaded, and replaced by fresh resin. This extra processing step may have economic implications and environmental concerns. If the resin is changed over when the first stage removal has dropped to 96 % and 4 % of the uranium (21 ppm U for a feed tenor of 530 ppm U - Type A) is still left in, the solution can be recycled back to the leaching process, which eliminates these economic and environmental implications. Such a target change over point is also shown in Figure 13-20.

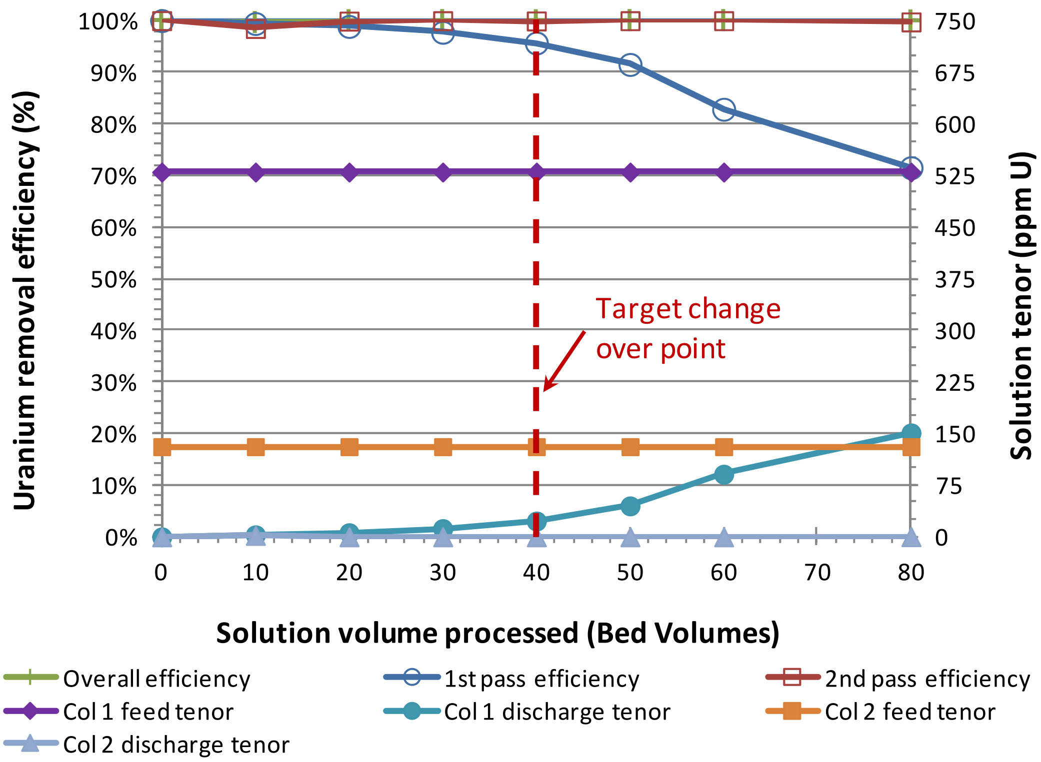


Figure 13-20: Ion Exchange Column Test Results – CWENGA 2010

These IX tests have found a suitable IX resin capable of achieving a minimum of over 96 % removal of uranium from PLS in a single stage and over 99 % for a two stage system. In an industrial process, any remaining uranium would be returned to the process, and no uranium would be released, increasing the final uranium removal from PLS to above 99 %.

The loaded resin from the Type A test was eluted with 4M sulfuric acid solution. However, the quantity of sulfuric acid solution available was insufficient to complete the test. Despite this, the loaded eluate that was collected was oxidized with hydrogen peroxide and then neutralised with milk of lime. The gypsum obtained was filtered off. The filtrate was then further neutralised to produce a yellow precipitate.

CWENGA will conduct a full IX test programme on the solutions produced after the leach testwork has established PLS flow and tenor with a high degree of confidence.

It is also noted that chlorides may suppress uranium extraction in the IX process. This is a variable that needs confirmation by testing the water supply for the Macusani site, as an SX plant could be required (instead of IX) if the PLS were to contain high levels of chloride ions.

In the Minergia testwork (Cameco Corporation, 2013) (40), the column leach apparatus was in closed-circuit with IX columns such that column discharge PLS was fed directly to IX circuit. Regenerated lixiviant (BLS) was returned from the IX circuit to the columns for further leaching. The IX resin (Dowex 21K) was found to load to 15 to 35 g/L uranium, depending on the PLS tenor. There was visual indication that no co-loading was occurring. A strong sulfuric acid solution (200 g/L) was used as the eluent for IX column elution at a rate of 15 mL/min (1 L batch). The eluate (pregnant solution) contained 85 to 137 mg/L of uranium, an order of magnitude higher than the PLS tenor, which had a flow rate of 3 mL/min.

13.3.9 SOLVENT EXTRACTION AND PRECIPITATION

SX testwork on pregnant solutions produced in previous leaching tests has been conducted in Lima (55) (56) (57) (58). Three extractants were tested for their technical feasibility, namely Alamine 336, Cyanex 272 and Di-2-Ethyl-Hexyl-Phosphoric Acid (D2EHPA). The PLS tenors tested had a range of 34 ppm to 906 ppm. The tests show that solvent extraction using these extractants is technically feasible. The construction of extraction isotherms would be required to optimise the number of extraction stages.

The extraction stages yielded various efficiencies. The conditions that achieved the maximum (90 % to 95 %) extraction efficiencies are given in Table 13-18. Stripping solutions of sodium chloride, sodium carbonate and acid water (dilute sulfuric acid) were tested successfully. The stripping stage achieved stripping efficiencies of 82 % to 99 % for the Alamine 336 and D2EHPA reagents. The cause for the poor stripping efficiencies obtained in the tests associated with the Cyanex extractants is not clear.

Tests 4a and 4b in Table 13-18 are details of SX tests conducted to present a complete downstream processing option for the processing of PLS. The processing flowsheet for extraction with D2EHPA is shown in Figure 13-21. This SX test campaign, on a small pilot plant, utilises SX as the primary uranium extraction method for pregnant leach solution. SX is also used for the subsequent processing stages. The PLS is contacted (in three stages) with a D2EHPA-containing organic phase that extracts the uranium from the aqueous PLS phase. The uranium is then re-extracted with acid water (in two stages). The uranium ions are oxidised by air injection. The uranium is then extracted by contact with an alamine-containing organic phase (in one stage) and re-extracted by a sodium chloride aqueous stripping phase (one stage). Magnesium oxide causes magnesium diuranate to precipitate out of solution, which may then be filtered out and dried. Impurities in the barren acid water phases may be removed by a reduction process.

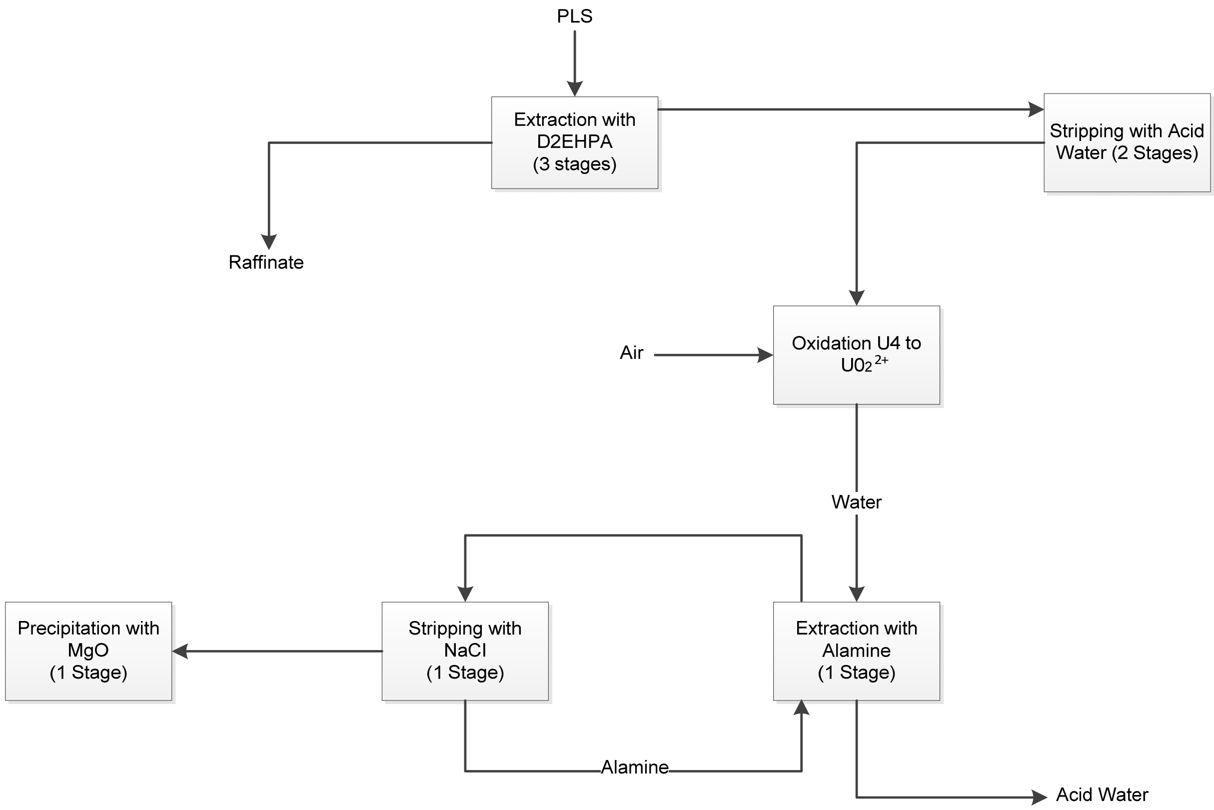


Figure 13-21: Uranium Extraction with Di-2-Ethyl-Hexyl-Phosphoric Acid (D2EHPA)

Table 13-18: Solvent Extraction Testwork Summary – TECMMINE 2009 and 2010

Description		1	2	3	4a	4b
Date		December 2009	January 2010	March 2010	June 2010	June 2010
Leach solution tenor tested, mg U/l	High	870	–	905.6	–	–
	Medium	276	287.2	–	–	–
	Low	34.1	95.1	105.1	105.1	105.1
Organic extractant		Alamine 336 7 % v/v, 0.1M	Cyanex 272 12 % v/v, 0.3M (Cyanex 923 and 921 also tested)	Alamine 336 5 % v/v, 0.1M	D2EHPA 5 % v/v, 0.15M (Primary)	Alamine 336 5 % v/v, 0.12M (Secondary)
Organic diluant		Kerosene	Kerosene	Kerosene	Kerosene	Kerosene
Extraction Stage						
O/A volume ratio		1.0	0.75 – 1.0	1.0	1.0	1.0
Mixing time, mins		2	3	120 – 135	160	1.5
Agitator speed, rpm		380	380	500	500	Not reported
pH		1.1	1.0	1.0	1.7	Not reported
Counter-current decantation stages		3 - 4	3	3	3	1
Extraction efficiency, %		60 - 95.3	95.1	91.7 – 94.3	90.5	95.6
Re-extraction Stage						
Re-extraction O/A volume ratio		4.0 – 5.0	1.0	2.6 – 7.2	5.4	4
Direct stages		1	2	2	2	1
Agitator speed, rpm		380	380	500	500	370
Mixing time, mins		1 -2	3	213	167	1.5
Stripping reagent		Sodium carbonate	Various mixtures of sodium carbonate, sodium chloride, water, ammonium, ethanol and sulfuric acid	Sodium carbonate Sodium chloride	Acid water (pH = 1.0)	Sodium chloride (pH = 7.58 1.0 M)
Re-extraction efficiency, %		84.2 – 97.6	0.05 – 56.47	82.3 – 98.2	99.5	98.5
Precipitation Stage						
Precipitation reagent		Magnesium oxide	Not specified	Magnesium oxide	N/A	Magnesium oxide
Precipitation efficiency, %		Not reported	Not reported	26 – 98	N/A	98.8

Solvent extraction has been shown to be technically feasible for leach solutions expected to arise in this project. Precipitation of yellowcake has been shown to be possible utilising magnesium oxide and hydrogen peroxide. The precipitation process is dependent on the nature of the process solution produced upstream and the upstream uranium recovery technology (IX or SX). To date, precipitation testwork has been limited. Further precipitation testwork is required and should be selected based on the upstream process selection and testwork.

13.3.10 MINERALOGY AND URANIUM DISTRIBUTION

13.3.10.1 MINERALOGY

The Minergia testwork (40) included some mineralogy testing, which described the PEM as consisting of "very soft dacitic ignimbrite tuff that contained low amounts of base metals, zirconium and thorium, virtually no sulphides and increased concentrations of uranium, phosphorous, lithium, rubidium, strontium and barium, which indicated its origin from a well differentiated magma."

"The uranium mineralization consisted of more than 90 % of strontium-bearing and arsenic-bearing autunite that filled ores, cracks and fissures and was likely formed by epithermal or telethermal solutions in contact with phosphorous sources such as apatite and phosphorous-bearing micas and feldspars. Due to this mineralization, and as uraninite made up less than 6 % of the uranium mineralization, no oxidant was needed for successful uranium leaching."

The chemical analysis of the core sections found "very low concentrations of element of concern such as arsenic, tellurium, selenium, molybdenum, vanadium, thorium, and other heavy metals, which did not exceed 10 ppm." It was found that the uranium concentration had a high variability.

Stereoscopy found "scarcely disseminated autunite throughout the drill core that was more abundant in the coarse grained textural ore variety. The mineral formed yellowish-green interstitial grains on the border of quartz fragments and fine muscovite aggregate."

Geological study ((Li, 2012) (22) (Li, 2015) (59)) suggests that meta-autunite ($Ca[(UO_2)(PO_4)]_2(H_2O)_{6-8}$) is the primary uranium mineral. "The timing of ore-formation, as well as the mineralogical and geochemical characteristics of the ore and host rocks, suggests that U was leached from the rhyolites, transported by meteoric waters and precipitated as meta-autunite through evaporation and interaction of the uranyl ion with Fe- and Mn-mineraloids. The majority of known deposits are located on the upper walls of active fluvial canyons. The geomorphological environment that is focusing both groundwater flow and its evaporation was the most favourable for meta-autunite precipitation. The Macusani U deposits are volcanic-hosted, but the ore-forming process is of low-temperature and relates to the climatic changes in the area."

Li (59) also found variability in the chemical composition of the meta-autunite from different localities, with ranges evident in UO_3, CaO and P_2O_5 as shown in Figure 13-22.

Fifty-element analysis on ten bottle roll test feed samples (TECMMINE 2012) (60) found aluminium, potassium, sodium and iron at levels in the 1 to 10 % range, see Figure 13-23. Calcium and phosphorous were the next most abundant elements followed by barium, chromium, lithium, manganese, rubidium, titanium and uranium.

Elemental analysis of the bottle roll test tails allowed the determination of the dissolution of the non-uranium elements. The average elemental dissolution is compared to the average quantity of uranium dissolved in Figure 13-24, which shows greater than 20 times more aluminium and 10 times more potassium are dissolved. Other elements that showed greater dissolution than uranium were calcium, iron, sodium and phosphorous.

These elemental dissolution results tie in with the discussion of host rock acid consumption in Section 13.3.7 and highlights the requirement for the identification and quantification of the acid consuming minerals in the Macusani PEM.

Table 3.1. Chemical composition of meta-autunites (Li, in review); na- not analysed, DL – below detection limit; analysed by EMPI, (*) – analysed by ICP-MS.

Occurrence	Chilcuno	Colibri II	Nuevo Corani	Pinocho	Tantamaco
Sample	Chi-6-1*	Clb-2-1	Mac-207*	Pi-1	Mac-206
SiO_2	0.18	2.72	0.06	0.60	0.52
Al_2O_3	2.64	0.85	2.41	0.15	0.49
K_2O	4.98	0.28	0.19	na	0.05
BaO	0.10	0.00*	0.29	0.28	0.19*
ThO_2	0.49	0.01	0.40	na	0.00
UO_3	76.08	62.49	63.20	66.43	64.52
CaO	1.87	5.44	5.36	5.38	4.17
Na_2O	0.18	0.02	0.01	0.02	0.00
MgO	0.06	0.02	0.08	0.03	0.01
P_2O_5	3.41	15.36	15.65	16.08	15.01
MnO	0.02	0.02	0.01	0.01	0.01
Fe_2O_3	0.09	0.10	0.76	0.59	0.48
TiO_2	0.00	0.02	0.02	0.02	0.03
Cr_2O_3	DL	0.01	0.00	na	0.01
V_2O_5	0.00	0.02	0.00	na	0.01
PbO	0.06	0.03	0.03	na	0.02
SO_2	na	0.01	na	na	0.01
Y_2O_3	0.02	0.07	0.04	na	0.23
Tb_2O_3	0.00	0.03	0.00	na	0.02
CuO	0.00	na	0.01	na	0.02
SrO	0.16	0.53*	1.43	0.89	1.31*
Total	90.16	84.77	89.88	89.87	85.11

Figure 13-22: Macusani Meta-Autunite Chemical Composition [from Li (59)]

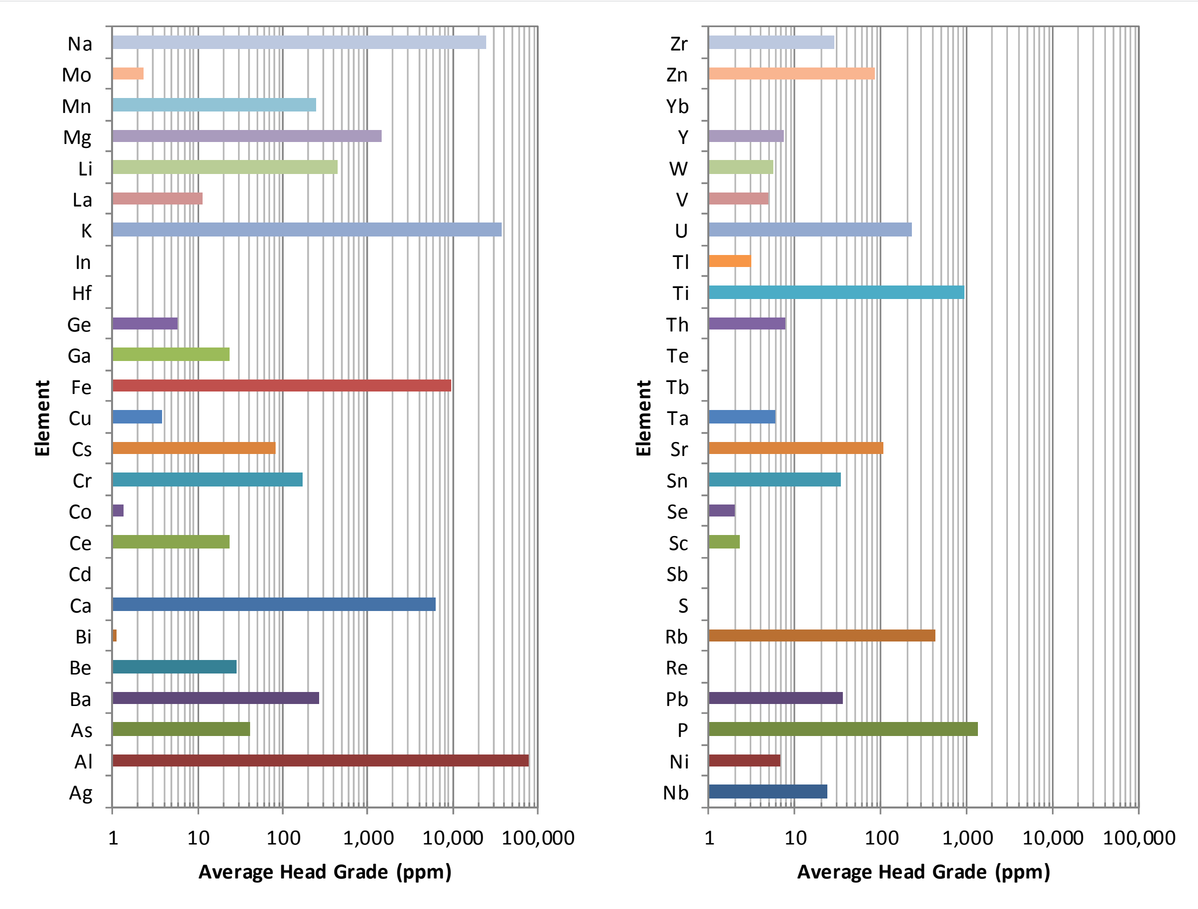


Figure 13-23: Fifty-Element Analysis on Ten Bottle Roll Test Feed Samples – TECMMINE 2012

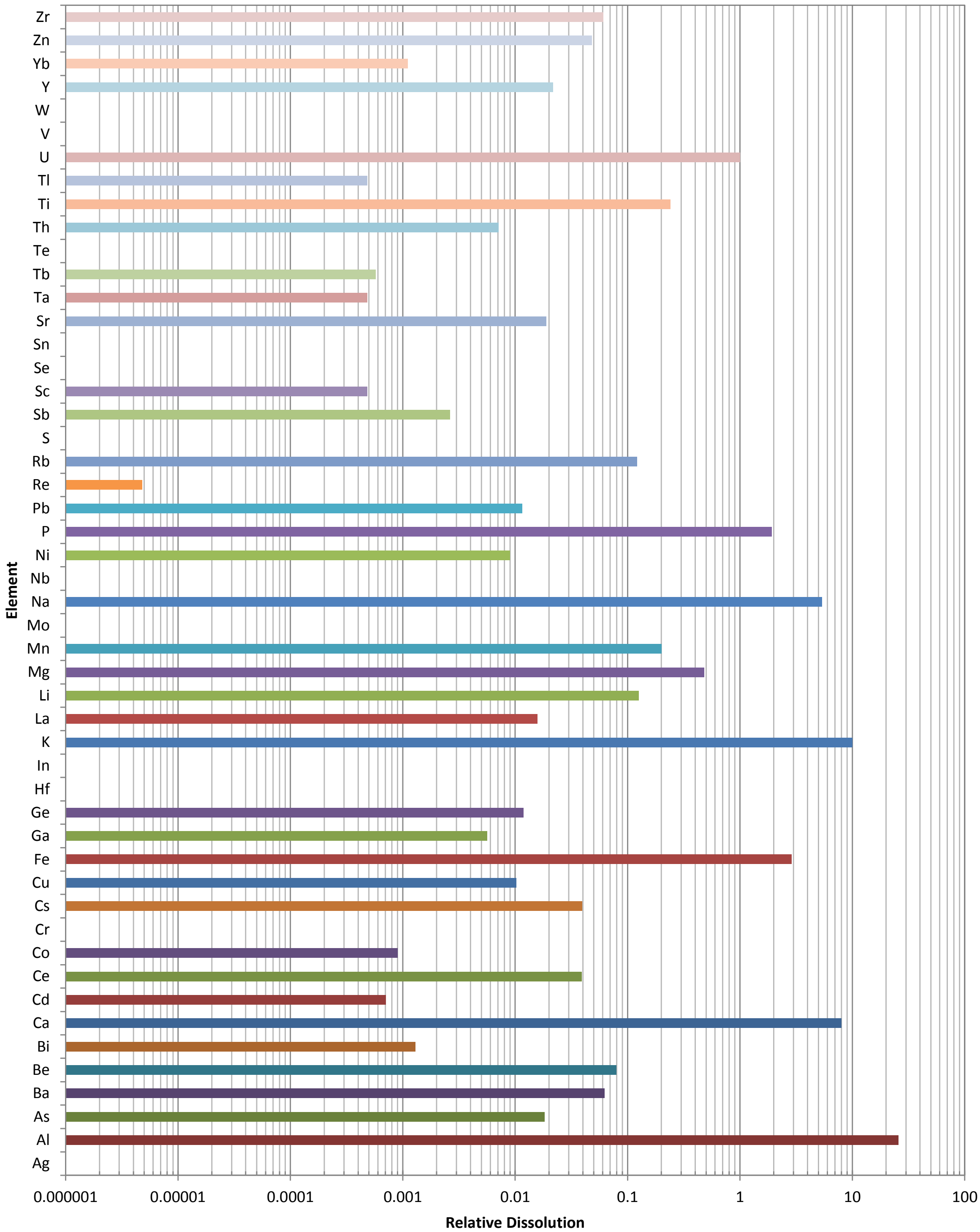


Figure 13-24: Relative Bottle Roll Leach Extraction for 50 Elements – TECMMINE 2012

13.3.10.2 SIZE BY SIZE ASSAY

Feed samples for the column leach tests were assayed by size fraction (TECMMINE, 2011) (37) (38) (61) (62) (2 inch topsize). The grade by size analysis of the low grade sample (approximate uranium content of 30 ppm) (61) is shown in Figure 13-25. The graph indicates that, whilst some 14 % of the uranium is in the fines, >70 % of the uranium is in the plus 2 mm material and >50 % is in the plus 10 mm.

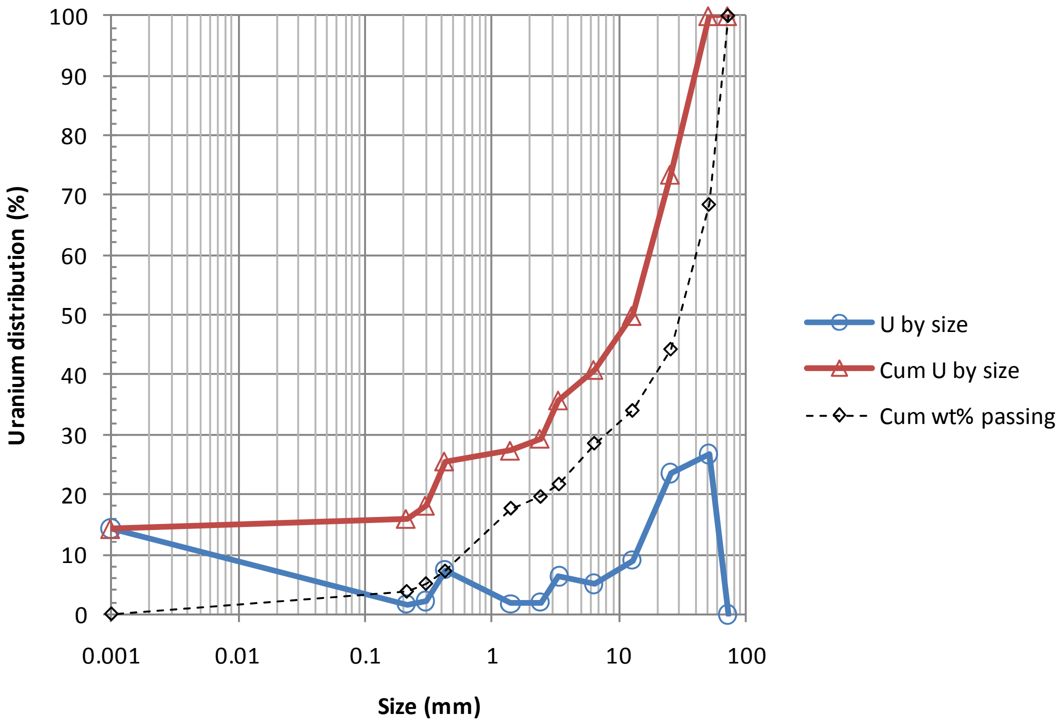


Figure 13-25: Sample Grade by Size Analysis

A grade by size analysis was conducted on Column 3 and 4 feed samples (TECMMINE, 2011) (37) (38) (2 inch topsize). Whilst the sizing did not progress below 6.35 mm, the material exhibited grade by size characteristics consistent with the low grade sample analysis in Lima. Figure 13-26 shows the grade by size analysis for a high grade sample, with >55 % of the uranium is in the plus 10 mm material. Figure 13-27 shows the grade by size analysis for a low grade sample, with approximately 65 % of the uranium in the plus 10 mm material. Between 58 % and 78 % of the uranium was found to be in the plus 6.35 mm size ranges for all samples assessed (TECMMINE, 2011) (37) (38) (61) (62) (63). One exception was for a low grade sample where 49 % of the uranium was in the plus 6.35 mm size ranges (64).

These grade-by-size analyses suggest that there is not a disproportionate amount of uranium in the fines and validates the selection of a 2 inch crush size.

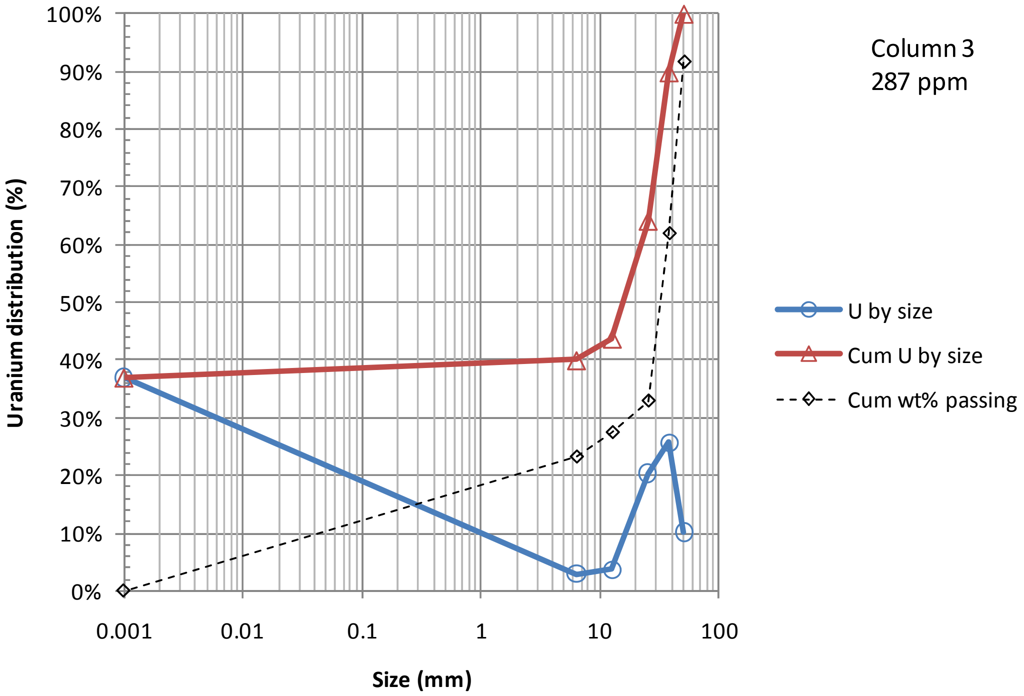


Figure 13-26: Sample Grade by Size Analysis – Column 3

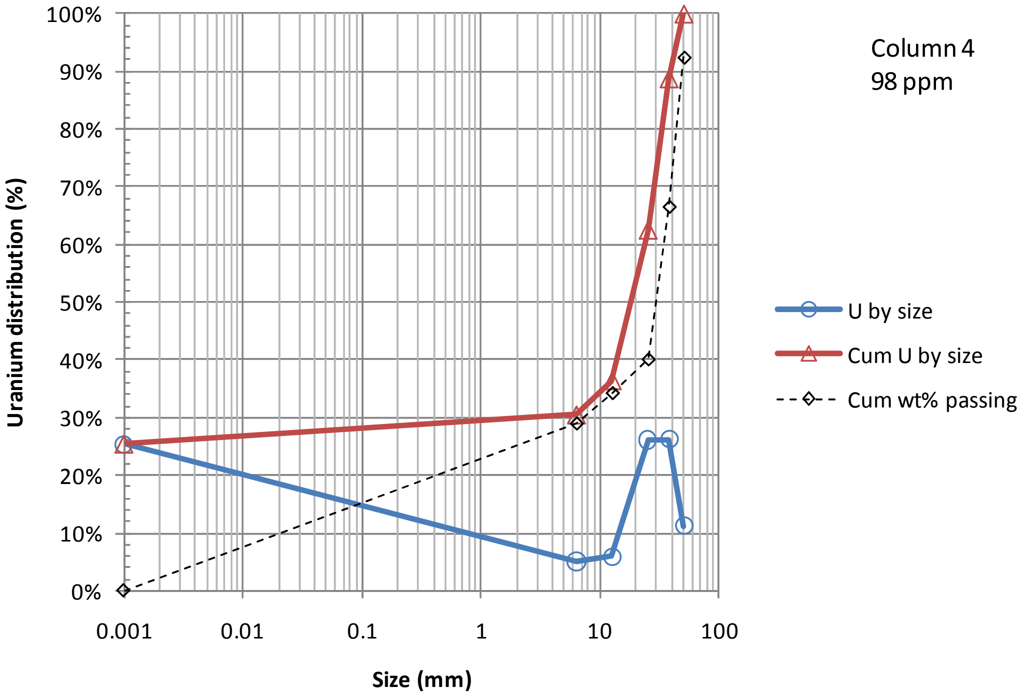


Figure 13-27: Sample Grade by Size Analysis – Column 4

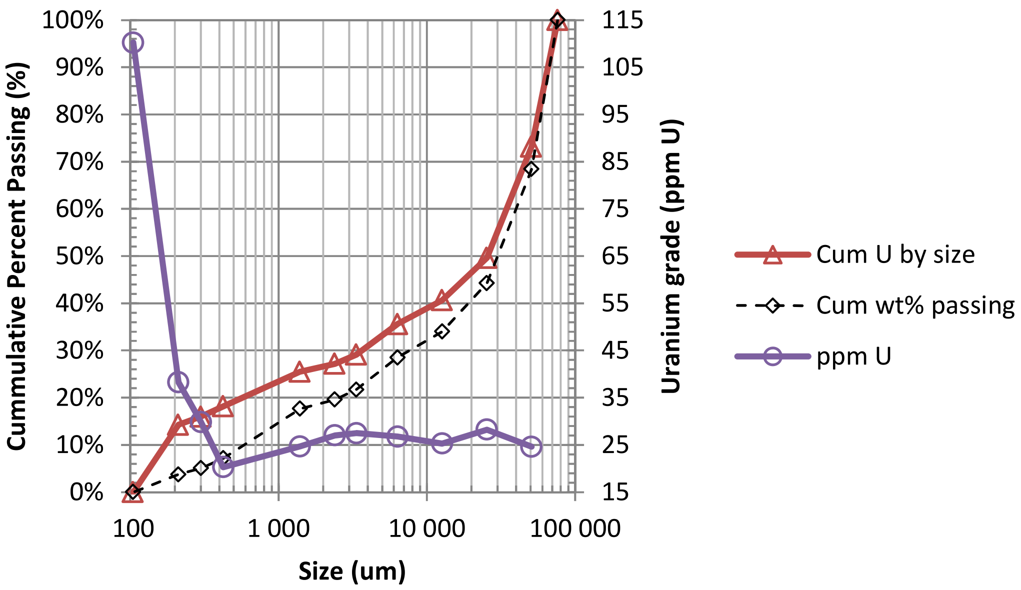


Figure 13-28: Cumulative Sample Mass and Uranium and Grade by Size – TECMMINE 2009

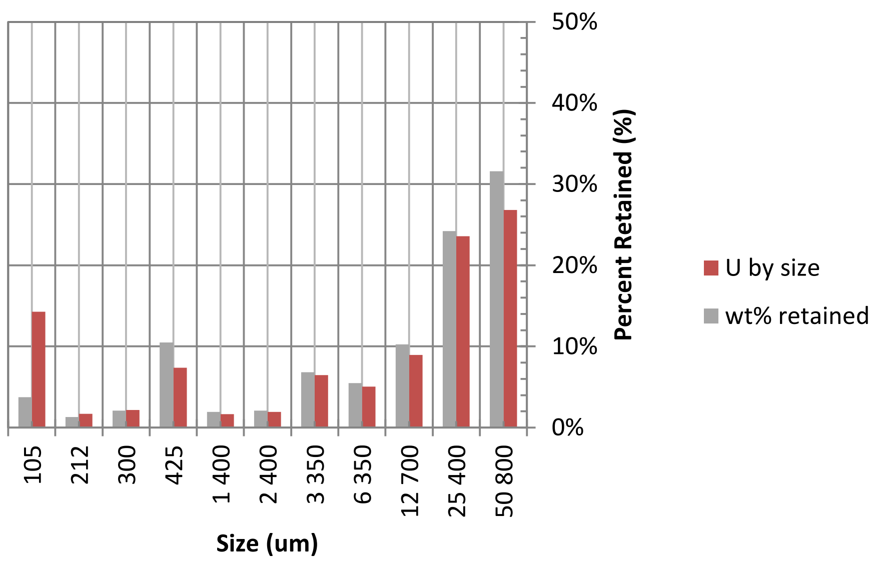


Figure 13-29: Sample Mass and Uranium by Size – TECMMINE 2009

Figure 13-28 and Figure 13-29 show the mass and uranium distribution of the 100 % passing 2 inches column test feed (TECMMINE, 2009) (34). These grade-by-size graphs support the suggestion that there is not a disproportionate amount of uranium in the fines. However, the peak in the grade of the fines highlights the requirement to account for fines in the materials handling and processing of the PEM.

As mentioned in Section 13.3.3.4, the Minergia testwork (Cameco Corporation, 2013)) (40) included porosity measurement, by back scattered electron imagery of polished sections of several –12 mm particles, to estimate the solution accessibility and size distribution of the uranium mineral grains in two PEM blends. High variability in grain size was observed (maximum grain sizes were between 40 µm and 660 µm), which indicated that localised variable leaching rates would be expected. However, an average pore size of between 45 µm and 85 µm would likely result in acceptable uranium leaching kinetics overall.

13.3.10.3 SELECTIVE COMMINUTION AND SCRUBBING

The Minergia testwork (40) also investigated the prospect of selective comminution to reduce the feed rate, scale and CAPEX of a tank leach process at an acceptable recovery loss. The tests involved crushing to12.7 mm or 9.5 mm, tumbling in a mill shell (pulp density of 70 % solids) for 10 to 75 minutes, and then screening.

Screening at 200 µm, after scrubbing for 75 minutes, rejected 40 to 60 % of the mass with a uranium recovery to fines of over 80 % and a leach plant feed grade 2 – 3 higher than the original head grade.

The selective comminution results were interesting, so overall process performance was estimated and compared to estimates of heap leach performance. Uranium recovery for the selective comminution process was 4 to 14 % lower than for the heap leach option.

It was concluded that “selective comminution offers neither higher uranium recoveries nor reagent savings”. Therefore, “heap leaching would be preferred as the simpler, more cost effective process option.”

13.3.11 SITE WATER QUALITY

In January and February 2011 several water analyses were conducted on site (Isivilla) water (CIMM, 2011) (65) (66) (67). The results of these tests are shown in Table 13-19. The water is of good quality with low hardness (dureza), less than 16 mg/L, and low alkalinity (alcalinidad), less than 18 mg/L.

Table 13-19: Site Water Analyses – CIMM 2011

Samples			Elements						
#	Service Code	MON0000	MA0004	MA0016	MA0028	MA0065	MA0066	MA0067
	Element	Sample Type	Alkalinity	$HC0_3^-$ [4]	CO_3^{-2} [5]	Hardness Total	Hardness (Ca)	Hardness (Mg)
	Unit		mg/L	mg/L	mg/L	mg/L	mg/L	mg/L
	Detection Limit		1	1	1	1	1	1
1	IsivillaS2-M0	Water	15	15	<1	14	10	4
2	IsivillaS2-M1	Water	<1	<1	<1	8	5	3
3	IsivillaS2-M2	Water	<1	<1	<1	14	10	4
4	IsivillaS2-M3	Water	<1	<1	<1	9	6	3
1	A050	Water	12	12	<1	7	6	1
2	A100	Water	12	12	<1	7	6	1
3	A150	Water	12	12	<1	7	6	1
4	A200	Water	11	11	<1	6	6	<1
5	A250	Water	11	11	<1	7	5	2
6	A300	Water	11	11	<1	6	6	<1
7	A350	Water	10	10	<1	7	6	1
8	A400	Water	10	10	<1	6	5	1
9	A450	Water	9	9	<1	6	6	<1
10	A500	Water	9	9	<1	6	6	<1
11	A550	Water	8	8	<1	7	6	1
12	A600	Water	7	7	<1	7	5	2
13	A650	Water	6	6	<1	7	6	1
14	A700	Water	5	5	<1	7	6	1
15	A750	Water	5	5	<1	7	5	2
16	A800	Water	4	4	<1	6	6	<1
17	A850	Water	4	4	<1	7	6	1
18	A900	Water	3	3	<1	7	6	<1
19	A950	Water	3	3	<1	7	5	2
20	A1000	Water	2	2	<1	6	6	<1
1	L-OSF	Water	18	18	<1	7	5	2

[4] Equivalent Bicarbonate

[5] Equivalent Carbonate

Samples		Elements						
#	Service Code	MON0000	MA0004	MA0016	MA0028	MA0065	MA0066	MA0067
	Element	Sample Type	Alkalinity	$HC0_3^{-}$ (4)	CO_3^{-2} (5)	Hardness Total	Hardness (Ca)	Hardness (Mg)
	Unit		mg/L	mg/L	mg/L	mg/L	mg/L	mg/L
	Detection Limit		1	1	1	1	1	1
2	L-OF	Water	14	14	<1	8	6	2
3	L-390	Water	<1	<1	<1	13	9	4
4	L-540	Water	<1	<1	<1	12	9	3
5	L-120	Water	<1	<1	<1	13	9	4
6	L-260	Water	<1	<1	<1	16	12	4
7	L-4120	Water	<1	<1	<1	13	9	4

13.3.12 ROCK COMPRESSION AND COMMINUTION TESTS

In May 2010, samples were sent to Pontificia Universidad Catolica del Peru for compression tests (68). Simple compressive strength ranged from 3.6 MPa to 19.0 MPa. Natural density ranged from 1.936 to 2.071 g/cm³. Compressive stress ranged from 2.0 MPa to 32.5 MPa. Friction angle ranged from 35.6 to 41.0˚, with cohesion ranging from 0.120 MPa to 0.075 MPa, respectively. Triaxial compression tests and indirect tensile tests were also conducted.

Some comminution testwork were conducted as part of the Minergia testwork programme (Cameco Corporation, 2013) (40). SAG Mill Comminution (SMC) tests were conducted at SGS Lakefield on a composite feed sample. Table 13-20 contains the SMC test results and indicate the PEM is not suited to AG milling. Extra crushing or SAG milling would be required.

Table 13-20: SMC Test Results – Cameco 2013

SMC Test Parameter	Units	Value	Comment
Brittleness parameter, A	%	86.2	Higher is more brittle and suited to AG milling. Lower is more suited to SAG.
Breakage resistance parameter, b	-	4.53	Lower is more resistant and suited to SAG milling. Higher is more suited to AG milling.
Strength parameter, A x b	-	390.5	Rock strength metric. Lower values indicate higher strength.
Hardness Percentile	-	1	Higher is harder. Ore is very soft (69).
Abrasion parameter, ta	%	4.53	Higher is more amenable to abrasion and AG milling. Lower is resistant to abrasion and more suited to SAG milling.
Drop wight index, DWi	kWh/m³	0.57	Rock strength metric.Correlated to A x b.
Specific energy: coarse particles, Mia	kWh/t	3.3	

SMC Test Parameter	Units	Value	Comment
Specific energy: HPGR, Mih	kWh/t	1.5	
Specific energy: crushing, Mic	kWh/t	0.8	
Relative Density	t/m³	2.24	

13.3.13 TAILINGS REMEDIATION

The Minergia testwork included remediation tests on the column leach test residues (Cameco Corporation, 2013) (40), The remediation tests comprised of three steps:

1. Addition of 30 mg/L barium chloride ($BaCl_2$) and lime (CaO) to pH 10.5 to the final barren solution with three days of circulation through the residue;
2. Addition of 10 mg/L barium chloride ($BaCl_2$) and lime (CaO) to pH 10.5 to the final barren solution with one day of circulation through the residue; and,
3. Progressive pH adjustment of the barren solution from 9 to 11.7, followed by a column rinse via column flooding and draining over serveral cycles of 30 to 60 minutes.

The results indicated that remediation of the heap tails would require approximately 50 % of the time required for leaching, which equates to approximately 35 days in the scale-up estimate. The Lima and Isivilla column leach tests (TECMMINE, 2011) (35) included washing and draining stages that totalled to 10 to 25 % of the leaching time. The water-only washing brought the pH to between 1.5 and 3. Neutralisation would be required before disposal to a tailings management facility. The Minergia testwork (Cameco Corporation, 2013) (40) estimated 8 – 9 kg CaO/lb U_3O_8 (4 – 9 kg CaO/t) would be required for leach residue neutralisation.

Table 13-21 contains the concencentration of “most of the elements of concern” in the final neutralised effluent from the Minergia remediation tests on the column leach residues (Cameco Corporation, 2013) (40). These concentrations were assessed as being “acceptably low”, adding that concentrations with respect to arsenic, uranium and phosphorous should improve as the solution pH approached the target of 7.

Table 13-21: Final Neutralisation Effluent Analysis – Cameco 2013

Element	Units	Column 1	Column 2	Column 3	Column 4
Phosphorous, P	mg/L	11	7.7	9.5	16
Arsenic, As	mg/L	0.094	0.059	0.056	0.13
Iron, Fe	mg/L	1	2	2.1	6.3
Molybdenum, Mo	mg/L	<0.001	<0.001	<0.001	<0.001
Titanium, Ti	mg/L	<0.002	<0.002	<0.002	<0.002
Uranium, U	mg/L	0.14	0.047	0.051	0.11
Vanadium, V	mg/L	0.004	0.001	0.002	0.006

These results appear to be supported by the TECMMINE bottle roll test results (Cameco Corporation, 2012) (70), that showed that calcium, iron, sodium and phosphorous in the PEM showed greater dissolution than uranium, recall Figure 13-24. Hence, these elements would be more likely to be present in the leach solution effluent.

SECTION 14 MINERAL RESOURCE ESTIMATES

14.1 CORACHAPI COMPLEX

The Mineral Resource estimates for the Corachapi Complex were documented by TMC (Young, 2010 (24)) and as they have remained unchanged, a complete description of the process employed is not included in this report; only a summary follows. The Mineral Resources were based on 11 818 m of drilling on 219 drillholes.

The estimates were completed via a block model and interpolation of the uranium abundance by geostatistical methods in the Datamine environment. A block model of base cell sizes of 25 x 25 x 2 m was employed and trimmed by the surface topography model. Multiple Indicator Kriging (MIK) was used to estimate the expected proportion of material above a series of cut-offs, and average grades within grade groups estimated using classical statistics. Due to the highly skewed nature of the 2.5 m bench composite data and the method employed, the Mineral Resource estimates at different uranium cut-offs were checked for material bias by conducting a classical log normal estimation on 5 m composites and a 3D variogram derived block variance. A good correspondence for tonnage was evident between these two methods.

Classification of the Mineral Resource was based on kriging efficiency and indicator group grade estimation errors. The Mineral Resource estimates are provided in Table 14-1.

14.2 COLIBRI COMPLEX

TMC last updated the Mineral Resource estimates at Colibri II & III in September 2013 (Young, 2013 (17)). As they have remained unchanged, a complete description of the process employed is not included in this report, and only a summary follows.

The Mineral Resources at the Colibri II & III and Tupurumani deposits were based on 149 diamond drillholes, which represented some 12 673.2 m of drilling. Full-core samples were taken, owing to the friable nature of the mineralization and host rocks. These samples were crushed and representative samples analysed for uranium. The necessary quality control and assurance was been completed by insertion of reference material, duplicate samples and blank material (Section 11.2).

MIK was undertaken to take cognisance of the log-normal distribution of uranium abundances. Well-structured variograms were obtained for the Colibri II & III deposit, but poorly structured variograms were obtained for the Tupurumani deposit. MIK was employed to estimate the uranium grades into 25 x 25 x 5 m blocks. The estimation was undertaken in 3D into the near surface and deep zones, separated by the base of the interpreted high-grade wireframe.

The Mineral Resource classification was based upon classical log-normal statistical estimation errors per indicator group and weighted by the MIK probability estimates per indicator group. The majority of the near-surface mineralization at Colibri II & III is classified as Indicated whereas the Tupurumani area and the deep zones have been classified as Inferred. The Mineral Resource estimates are provided in Table 14-1.

Table 14-1: Colibri and Corachapi Complex Mineral Resource Estimates as at 31 March 2015

			Metric Units								Imperial Units		
Deposit	Evaluator	Classification	Cut-off	Tonne (000s)	Density (t/m3)	U grade (ppm)	U_3O_8 grade (ppm)	U Content (000s kg)	U_3O_8 Content (000s kg)	U Content (000s lb)	Ton (000s)	U_3O_8 Content (Mlb)	U_3O_8 Grade (lb/ton)
Corachapi	TMC, 2010	Measured	75	1 031	1.98	120	142	124	146	273	1 136	0.32	0.28
Corachapi	TMC, 2010	Indicated	75	10 562	1.98	171	202	1 806	2 130	3 982	11 643	4.70	0.40
Corachapi	TMC, 2010	Inferred	75	3 753	1.98	195	230	732	863	1 613	4 137	1.90	0.46
Colibri II & III	TMC, 2013	Indicated	75	27 885	1.98	203	239	5 661	6 675	12 480	30 738	14.72	0.48
Colibri II & III	TMC, 2013	Inferred	75	9 453	1.98	167	197	1 579	1 862	3 480	10 420	4.10	0.39
Tupuramani	TMC, 2013	Inferred	75	10 976	1.98	125	147	1 372	1 618	3 025	12 099	3.57	0.29

14.3 GENERAL METHOD STATEMENT FOR THE KIHITIAN, ISIVILLA AND CORANI COMPLEX MINERAL RESOURCE ESTIMATES

The current Mineral Resource estimation methodology used for the Kihitian, Isivilla and Corani Complexes was derived from the methodology employed by TMC to estimate the Mineral Resources for the Colibri Complex (Section 14.2). At the Colibri Complex, the continuity of mineralisation is interpreted to be sub-horizontal, and the mineralisation was divided into an upper high-grade zone (level A), and a lower low-grade zone (level B), which were interpreted to be sub-parallel to the host lava stratigraphy. The uranium distribution within the zones is log-normally distributed.

The estimation methodology derived to address the nature of the mineralisation at the Colibri Complex was to treat the upper and lower zones independently, and to use MIK, with a preferred orientation parallel to the interpreted contact between the two zones.

On inspection of the combined (Minergia and Global Gold) datasets for the Kihitian, Isivilla and Corani Complexes, TMC identified that the mineralisation for all three complexes exhibited similar characteristics to those identified at the Colibri Complex, and hence a similar estimation methodology was applied. The broad process steps are described below, and the details of the estimation for each complex are described in Section 14.4 to Section 14.6.

14.3.1 VALIDATION OF DATABASE

If necessary, the Minergia and Global Gold borehole databases were merged, and a visual assessment was made of each complex to confirm that the estimation methodology was appropriate.

14.3.2 IDENTIFICATION OF MINERALISED ZONES

The upper and lower mineralised zones were interpreted by analysis of a combination of the lithological logging, and the uranium distribution down the borehole. The contact between the two zones is often co-incident with a narrow uranium concentration spike. This contact was modelled as a 3-dimensionional surface. The upper mineralised zone is termed Level A and the lower mineralised zone Level B.

14.3.3 CALCULATION OF “BENCH” COMPOSITES

Due to the uranium mineralisation style, samples were taken at varying sampling intervals in order to preserve the sample integrity of higher grade zones. This resulted in varying sample lengths, as opposed to uniform sample lengths. Figure 14-1, illustrating interval 86 m to 123 m for drillhole TADDHS11-091 of Tantamaco illustrates this well.

Composites which had equal, 1.5 m, vertical lengths (“bench” composites) were employed for analysis and estimation due to the grade variability evident in a vertical sense within the zones, and because of the variable drilling orientations. Density information was not available at a drillhole scale, thus composites were not densometricaly weighted.

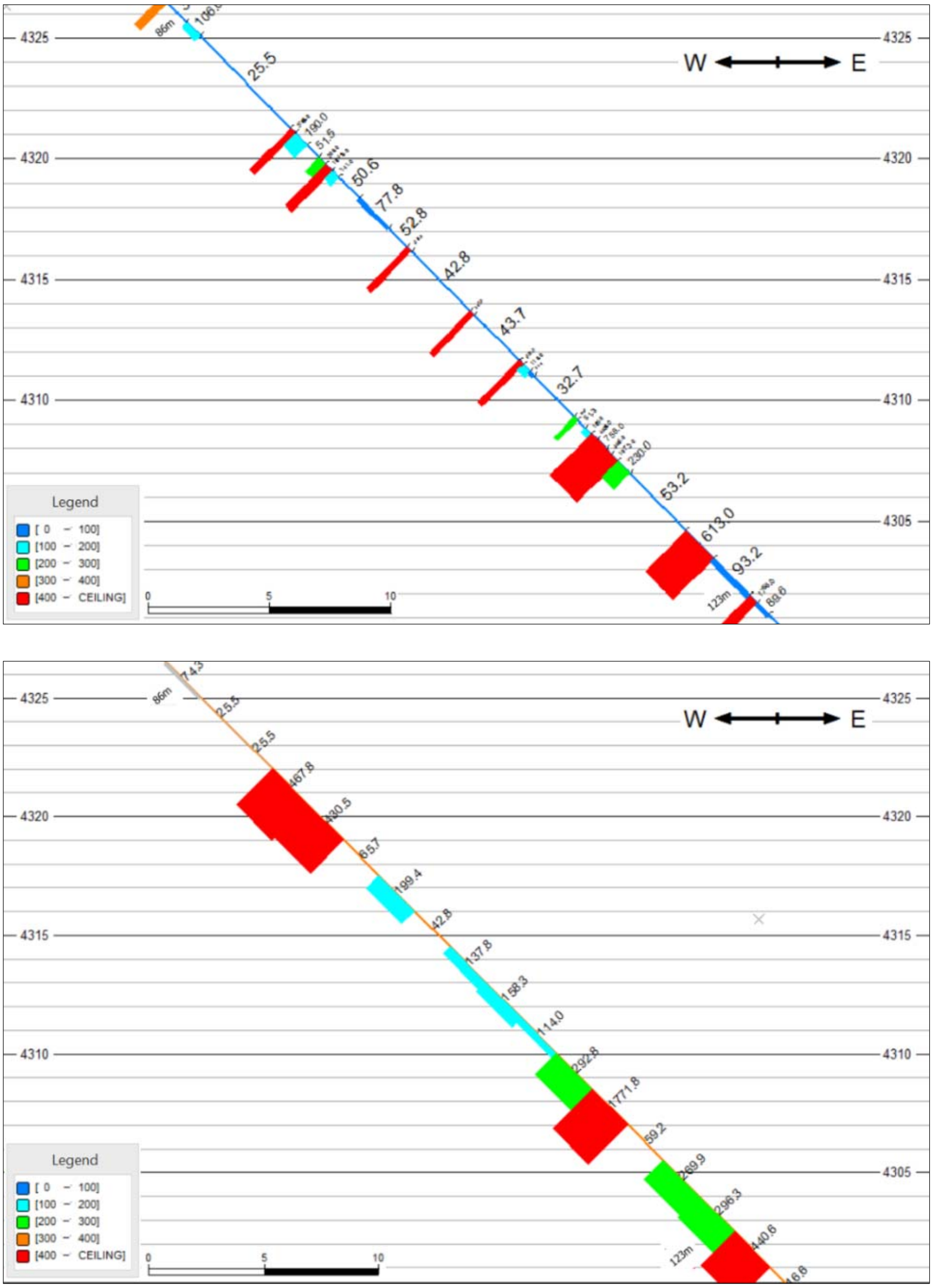


Figure 14-1: Illustration of 1.5 m Bench Composites for TADDHS11-091 rom 86 – 123 m

14.3.4 14.3.4 IDENTIFICATION OF MIK CUT-OFFS

MIK cut-off's for each zone were identified by means of analysis of the inflections within probability plots, as described in Clarke (1993) (71).

14.3.5 DERIVATION OF MIK MEAN GRADES

The mean uranium concentration for the data between the MIK cut-off's (the MIK bins) was estimated using classical statistics, taking into consideration the number of data points and the distribution of the data in the MIK bins. The mean of the data in the lowest cut-off bin, which was typically normally distributed, was used to estimate the bin mean grade. The other MIK bins typically exhibited log normality and thus the log estimates of mean were used to estimate the bin averages. Where the number of samples was few, the “Sichel's-t” lognormal estimate of the mean was used.

14.3.6 MIK VARIOGRAPHY

Variograms were analysed for each MIK bin, within each mineralised zone. Omni-directional variograms were analysed in the plane of the interpreted contact between the zones; these are termed “horizontal” variograms. Grade continuity orthogonal to the horizontal variograms was investigated by means of analysing downhole variograms (or “vertical” variograms).

Anisotropy within the horizontal variograms was investigated in all directions from 0° to 150° in 30° increments. No significant anisotropy was defined from the available data, and thus omni-directional variograms were modelled for all the cut-offs.

Isaaks and Srivastava (1989) (72) advise that in cases where the experimental indicator variograms for several bins can be approximated by a single variogram, the use of a single variogram can produce good results. The variogram for the middle cut-off was used to define the ranges for the estimation search.

14.3.7 MIK ESTIMATION

Block estimates of uranium grade were undertaken, informed by the results of the variography. Appropriate block sizes and search criteria were selected for each zone. The estimation was oriented parallel with the contact between the two mineralised zones, by employing Dynamic Anisotropy, an estimation technique within Datamine Studio™ which allows the search volume and variogram models for each block in the model to be rotated individually, in order to honour the trend of the mineralization.

14.3.7.1 BLOCK MODEL PARAMETERS

A block model with parent cell size of 50 x 50 x 3 m for the well-informed area and 100 x 100 x 3 m for extrapolation areas was utilised. The selection of this block size in the X and Y directions was informed largely by the drillhole spacing. The block height was considered to allow the Mineral Resource estimates to be employed for either exploitation by surface or underground methods. Sub-celling was employed to provide volumetric control of the mineralised zones, but grade estimation took place into the parent cells.

14.3.7.2 SEARCH CRITERIA

The block search criteria comprised three search volumes, with the 1st, 2nd and 3rd searches being equal to, once, twice and thrice the variogram ranges respectively in the XY plane. The Z was kept to ½ variogram range in all three search volumes, to avoid “smearing” of higher or lower grade material vertically. The minimum and maximum numbers of samples for each search volume were five and 24 respectively. Block discretisation to 3 x 3 x 1 was applied.

14.3.8 GEOLOGICAL LOSSES

Due to the perceived recent geologic age of the mineralisation, supported by the observed lack of disturbance of the mineralised zones due to faulting or intrusives (in the drilling), no geological loss factors were applied. The only geological losses that may impact on the Mineral Resource estimates would be un-mineralised dykes, none of which have been identified.

This is considered to be reasonable as the mining of this Mineral Resource is likely to be undertaken by open-pit means, with limited selectivity.

14.3.9 DENSITY DETERMINATIONS

A uniform density of 1.98 t/m^3 has been used for the estimates. This was based on based on the density data reported by Henkle (2014) (14) where a density of 1.98 t/m^3 was obtained from 21 samples from the Tantamaco deposit. In addition, TMC also conducted density tests on two drillholes on the Colibri Complex, and obtained an average density of 2.01 t/m^3 close to surface and 1.98 t/m^3 at depth (Young, 2013) (17).

14.3.10 CUT-OFF GRADE

The Mineral Resources are stated at a cut-off of 75 ppm (U) based on application of basic economic calculations at a cash cost level.

14.3.11 MINERAL RESOURCE CLASSIFICATION

Mineral Resource classification was informed by geostatistical confidence, as expressed in the block Kriging Efficiency (KE) after giving consideration to the confidence in the supporting database (Section 11.2.7, Section 11.3.7 and Section 11.4.7).

KE is defined as follows:

$$KE = \frac{\text{Block Variance} - \text{Kriging Variance}}{\text{Block Variance}}$$

Using the criteria for utilising KE for Mineral Resource classification provided by Mwasinga (2001) (73), blocks with:

- KE > 0.3 can be classified as Inferred;
- KE >0.3 but <0.5 can be classified as Indicated and;
- KE > 0.5 can be classified as Measured.

14.3.12 GRADE ESTIMATION

The final estimated grade of each block was determined by applying the means within each bin to the proportion of the block which was estimated to contain that material.

14.4 KIHITIAN COMPLEX

14.4.1 DRILLHOLE DATABASE

The Excel-based dataset provided was used to build a Datamine Studio™ drillhole database for the purpose of Mineral Resource estimation. A description of the data verification procedures is included in Section 12. The database comprises 275 drillholes from four deposits.

TMC considers the drillhole database to be suitable for the purposes of Mineral Resource estimation.

14.4.2 PEM MODEL

An integration of the drillhole data from the four deposits was done in order to validate the proposed estimation methodology. It is interpreted that the deposits belong to the same mineralisation system, as reasonable correlation across the deposits was noted.

14.4.3 DOMAINS

As described in Section 14.3, Level A and Level B are considered geologically separate units and were treated as such. Furthermore, investigation of the distribution of uranium grades within Level A and Level B revealed the deposits to be statistically distinct populations. As a result, the four deposits were treated as independent populations.

The depth from surface to the top of Level A varies from 20 m to 30 m while the depth from surface to the top of Level B varies from 20 m to 80 m. It should be noted that both leves A and B are not always present in all four deposits.

Both levels have been noted to outcrop and although there are localised variations, the contact is generally flat, dipping up to a maximum of 4° to the northeast.

14.4.4 EXPLORATORY DATA ANALYSIS

14.4.4.1 DISTRIBUTION

Figure 14-2 shows the U abundance distribution for the 1.5 m composites, for Level A and Level B for each of the four deposits. The distributions are highly skewed. Figure 14-2 also contains the descriptive statistics for the same populations. It is noted that the Level A and Level B mean and variances are significantly different, necessitating independent modelling in further statistical and geostatistical analysis.

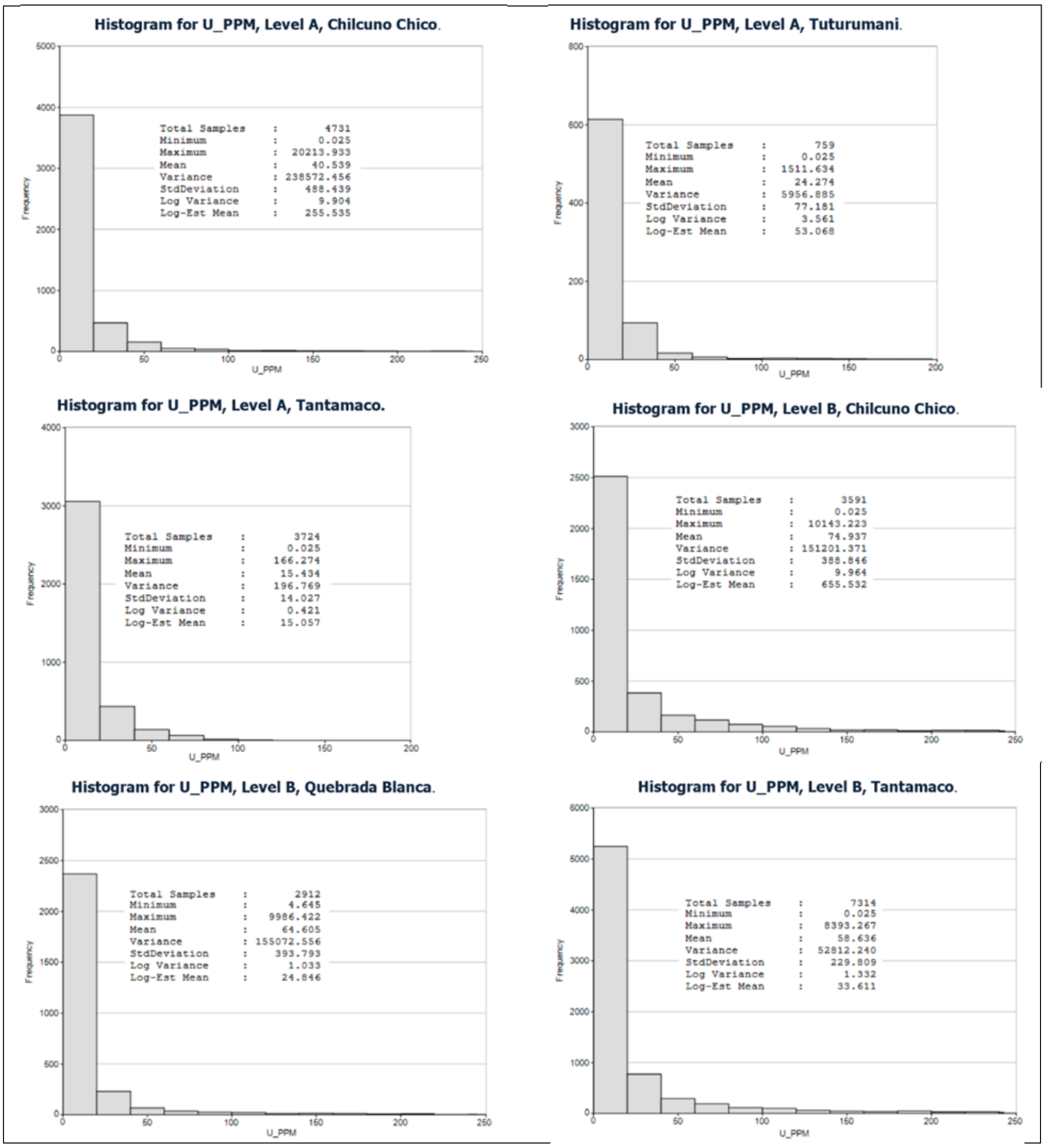


Figure 14-2: 1.5 m Composite U ppm Distribution for the Four Deposits of the Kihitian Complex

14.4.4.2 STATISTICS AND SELECTION OF INDICATOR CUT-OFFS

The probability distribution functions for Level A and Level B were analysed, for each of the four deposits, and the cut-offs selected are contained in Table 14-2.

Table 14-2: Indicator Cut-offs Identified on the 1.5 m Bench Composite data

Deposit	Level A (U ppm)	Level B (U ppm)
Chilcuno Chico	20	11
Chilcuno Chico	80	80
Chilcuno Chico	1 000	1 500
Quebrada Blanca	-	11
Quebrada Blanca	-	80
Quebrada Blanca	-	1 000
Tuturumani	14	-
Tuturumani	58	-
Tuturumani	225	-
Tantamaco	14	11
Tantamaco	65	80
Tantamaco	100	1 000

14.4.5 PREFERRED MINERALISATION ORIENTATION

The contact between Level A and Level B was geologically correlated over the four deposits and a wireframe of this surface was created, in order to control the preferred mineralisation interpolation in a sub-horizontal orientation.

14.4.6 VARIOGRAM ANALYSIS

Indicator variograms using the indicator cut-off values contained in Table 14-2 were analysed, within each deposit. The vertical variograms were used to derive the nugget effect and range in the Z direction.

The interpreted variogram model parameters are contained in Table 14-3. The detailed variograms are contained in Appendix 7 of Young (2015) (74).

Table 14-3: Variogram Model Parameters

Deposit	Level	Cut-off	Nugget	Range(X) (m)	Range(Y) (m)	Range(Z) (m)	Sill
Chilcuno Chico	A	20	0.03	70	70	20	0.149
Chilcuno Chico	A	80	0.013	50	50	6	0.038
Chilcuno Chico	A	1 000	0.0029	100	100	3	0.003
Tuturumani	A	14	0.0015	200	200	42	0.2315
Tuturumani	A	58	0.005	50	50	15	0.038
Tuturumani	A	225	0.001	50	50	6	0.016
Tantamaco	A	14	0.075	200	200	18	0.1475
Tantamaco	A	65	0.002	70	70	4	0.018
Tantamaco	A	100	0.001	50	50	6	0.003
Chilcuno Chico	B	11	0.025	311	311	37	0.2490
Chilcuno Chico	B	80	0.03	120	120	10	0.101
Chilcuno Chico	B	1 500	0.008	50	50	3	0.01
Quebrada Blanca	B	11	0.05	220	220	12	0.241
Quebrada Blanca	B	80	0.015	90	90	10	0.066
Quebrada Blanca	B	1 000	0.004	50	50	6	0.012
Tantamaco	B	11	0.02	310	310	27	0.249
Tantamaco	B	80	0.02	50	50	15	0.097
Tantamaco	B	1 000	0.005	50	50	3	0.0093

Relatively well-structured long range variograms were obtained for the lowest cut-off after which ranges drop to 50 % or less for the upper cut-offs as the relationship between higher-grade samples deteriorates. The ranges from the variogram of the middle cut-offs were used to define search distances.

14.4.7 LOCAL MEANS

Table 14-4 contains the MIK bin grade averages.

Table 14-4: MIK Bin Grades

	Level A		Level B	
Deposit	Grade Bin (U ppm)	Bin Grade (U ppm)	Grade Bin (U ppm)	Bin Grade (U ppm)
Chilcuno Chico	0 - 20	10.82	0 – 11	9.27
Chilcuno Chico	20 - 80	35.64	11 – 80	25.46
Chilcuno Chico	80 – 1 000	232	80 – 1 500	272
Chilcuno Chico	>1 000	4 110	>1 500	3 029
Quebrada Blanca	-	-	0 – 11	9.12
Quebrada Blanca	-	-	11 – 80	20.89
Quebrada Blanca	-	-	80 – 1 000	279
Quebrada Blanca	-	-	>1 000	3 011
Tuturumani	0 - 14	10.10	-	-
Tuturumani	14 - 58	21.6	-	-
Tuturumani	58 - 225	120	-	-
Tuturumani	>225	427	-	-
Tantamaco	0 - 14	9.35	0 – 11	8.03
Tantamaco	14 - 65	26.09	11 – 80	23.44
Tantamaco	65 - 100	74	80 – 1 000	257
Tantamaco	>100	125	>1 000	1 865

14.4.8 MODEL CONSTRAINTS

Although unconstrained estimation was employed, the following surfaces were used to limit the extents of estimation.

14.4.8.1 SURFACE TOPOGRAPHY

The block model was clipped against the topography.

The topographic surface was estimated from Ikonos imagery by Photosat in Vancouver, Canada. It was noted that there is a discrepancy between the topographic elevation as estimated by the Ikonos data, and the local elevation as per the government topographic plans. It was determined to use the Ikonos topography as the reference for the block model.

14.4.8.2 DEPTH CONSTRAINT

A surface was created that limited the depth of estimation to the bottom of Level B.

14.4.8.3 PLANAR EXTENTS

Although the relationship between Level A and Level B was defined where possible across the whole Complex area with the definition of a geologically continuous contact, the extent of the estimation was limited to three times the variogram range, amounting to the extrapolation distances shown in Table 14-5.

Table 14-5: Extrapolation Distances

Deposit	Level A (m)	Level B (m)
Chilcuno Chico	150	360
Quebrada Blanca	-	270
Tuturumani	150	-
Tantamaco	210	150

The block models are depicted in Figure 14-3 and Figure 14-4

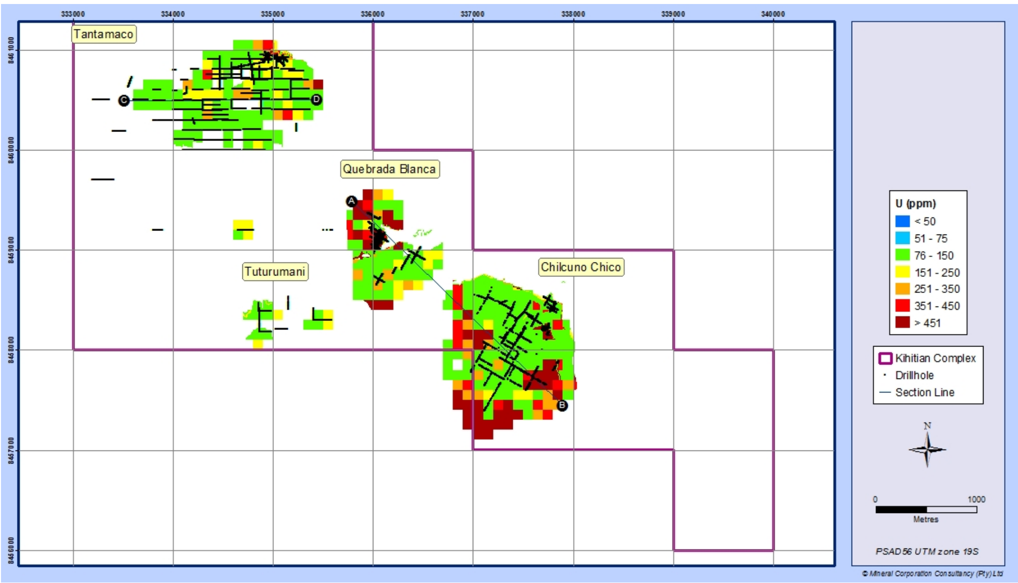


Figure 14-3: Kihitian Complex Mineral Resource Area Showing Uranium Abundance (U ppm) Projected to Surface with Section Lines A-B and C-D Indicated

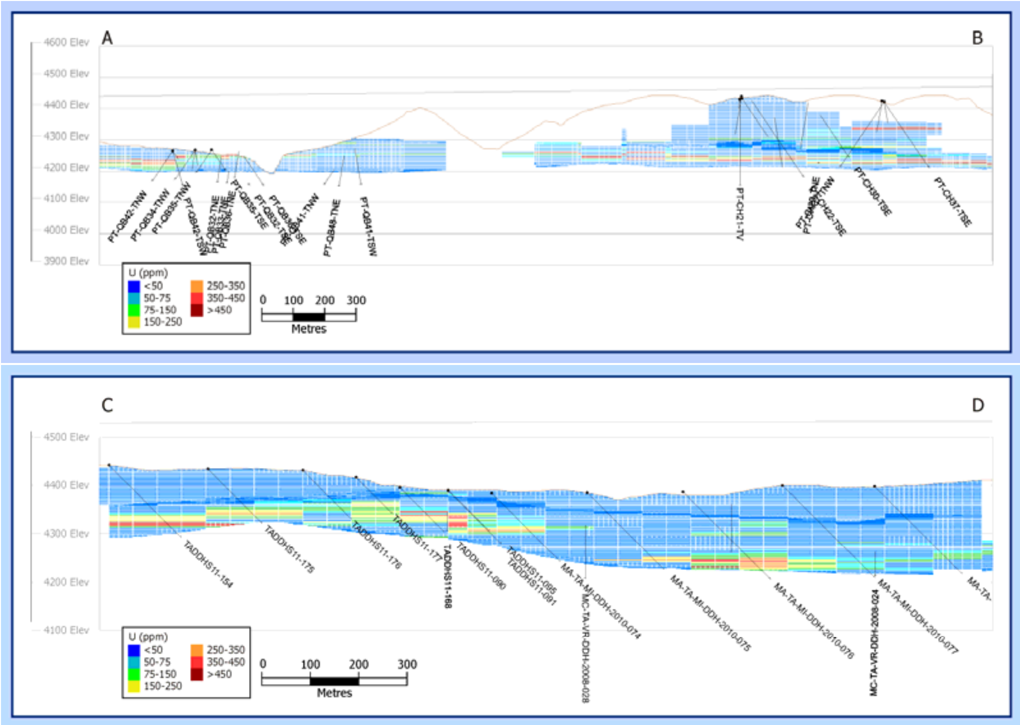


Figure 14-4: Cross Section A-B across Chilcuno – Quebrada Blanca and C-D across Tantamaco Mineral Resource Area Showing Uranium Grade (U ppm)

14.4.9 MINERAL RESOURCE CLASSIFICATION

The Kriging Efficiency (Section 14.3.11), search volume supported by geological confidence and QA/QC considerations were used as guidelines for classification of the Mineral Resources.

14.4.9.1 GEOLOGICAL CONFIDENCE

The geology of the area is fairly simple and well understood. In modelling the zones, the extents and continuity of mineralisation by inspection, the data density and mineralisation outcrop have resulted in the geological confidence being one of the primary drivers for the classification of the individual deposits.

14.4.9.2 DOWNHOLE SURVEY

No downhole surveys have been carried out.

14.4.9.3 QA/QC

As stated in sections 11.2.7 and 11.3.7, the quality of data is considered acceptable for Mineral Resource estimation.

14.4.9.4 MINERAL RESOURCE CLASSIFICATION

Indicated and Inferred Mineral Resources were defined for the Kihitian Complex, and classification plots are shown in Figure 14-5 and Figure 14-6.

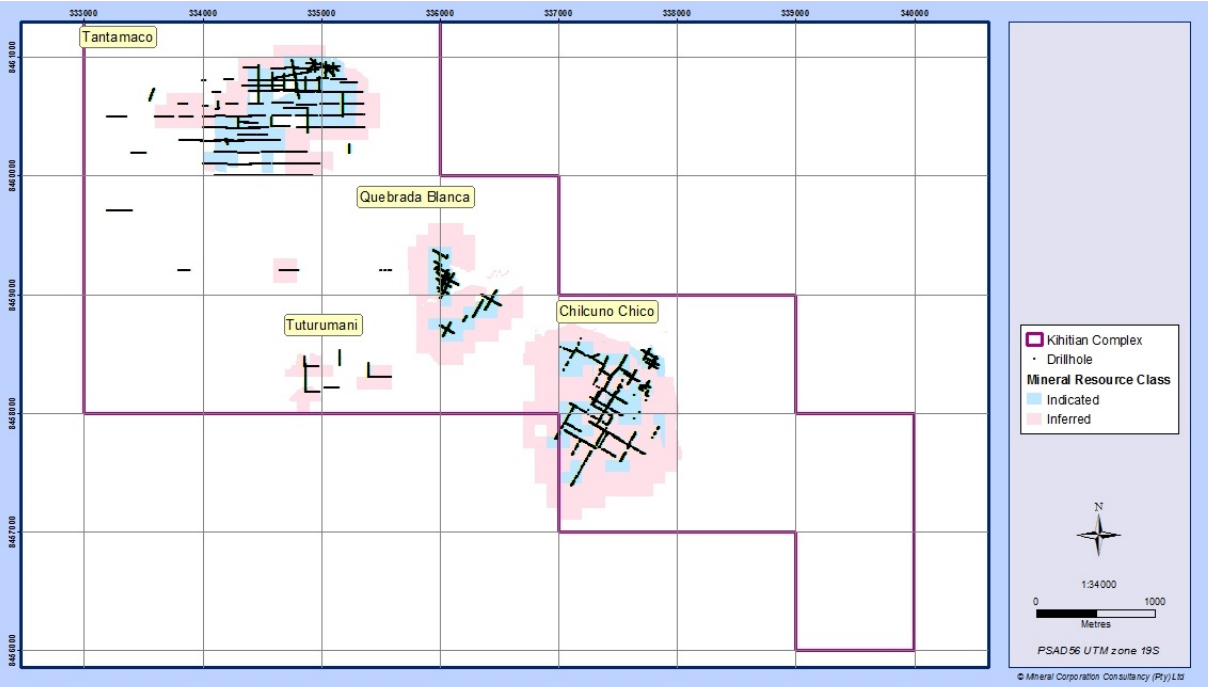


Figure 14-5: Plan View of the Block Model (Projected to Surface) Showing Classification with Section Line A-B Indicated

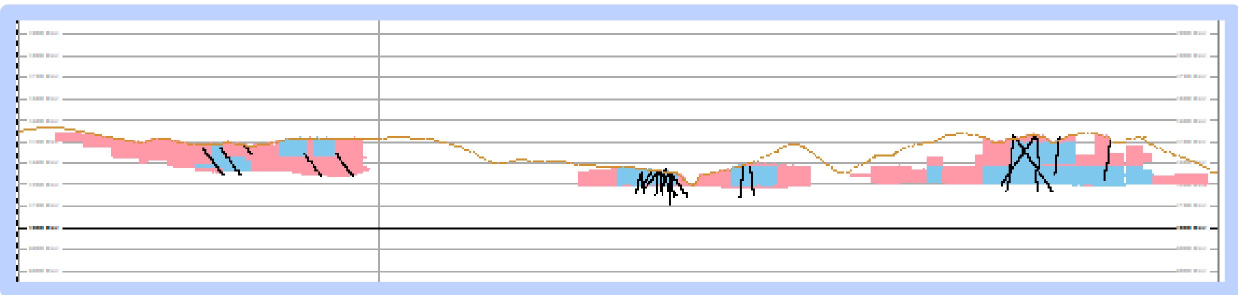


Figure 14-6: Block Model Section (A-B) Showing Classification

14.4.10 MINERAL RESOURCE STATEMENT FOR THE KIHITIAN COMPLEX

The estimated Mineral Resources for the Chilcuno Chico, Quebrada Blanca, Tuturumani and Tantamaco deposits at a 75 ppm U cut-off are summarised in Table 14-6.

Table 14-6: Kihitian Complex Mineral Resource Estimates as at 31 March 2015

			Metric units						Imperial units			
Deposit	**Classification**	**Cut-off (U ppm)**	**Tonne (000s)**	**Density (t/m3)**	**U grade (ppm)**	**U_3O_8 grade (ppm)**	**U Content (000s kg)**	**U_3O_8 Content (000s kg)**	**U Content (000s lb)**	**Ton (000s)**	**U_3O_8 Content (Mlb)**	**U_3O_8 Grade (lb/ton)**
Chilcuno Chico	Indicated	75	34 840	1.98	218	258	7 608	8 972	16 773	38 405	19.78	0.52
Chilcuno Chico	Inferred	75	30 995	1.98	294	347	9 117	10 751	20 099	34 166	23.70	0.69
Quebrada Blanca	Indicated	75	5 509	1.98	279	329	1 538	1 814	3 391	6 073	4.00	0.66
Quebrada Blanca	Inferred	75	13 436	1.98	269	317	3 616	4 264	7 972	14 811	9.40	0.63
Tuturumani	Indicated	75	0	1.98	0	0	0	0	0	0	0.00	0
Tuturumani	Inferred	75	3 300	1.98	146	172	482	569	1 063	3 638	1.25	0.34
Tantamaco	Indicated	75	7 393	1.98	191	225	1 409	1 661	3 106	8 150	3.66	0.45
Tantamaco	Inferred	75	35 849	1.98	172	202	6 148	7 251	13 555	39 517	15.98	0.40
Total Indicated		**75**	**47 743**	**1.98**	**221.08**	**261**	**10 555**	**12 447**	**23 270**	**52 627**	**27.44**	**0.52**
Total Inferred		**75**	**83 581**	**1.98**	**231.68**	**273**	**19 364**	**22 834**	**42 689**	**92 132**	**50.34**	**0.55**
Total Mineral Resources		**75**	**131 323**	**1.98**	**227.82**	**269**	**29 919**	**35 281**	**65 959**	**144 759**	**77.78**	**0.54**

14.4.11 ASSUMPTIONS FOR MINING

The depth consideration for the surface mining was taken at 150 m below surface. Mineral Resources below this depth are thus considered for potential underground mining techniques. From Figure 14-7 two broad areas are identified as possibly being suitable for underground mining in the northwest and southeast of the Kihitian Complex with a central zone of potential surface mining. Table 14-7 contains a breakdown of the Mineral Resources per potential mining method.

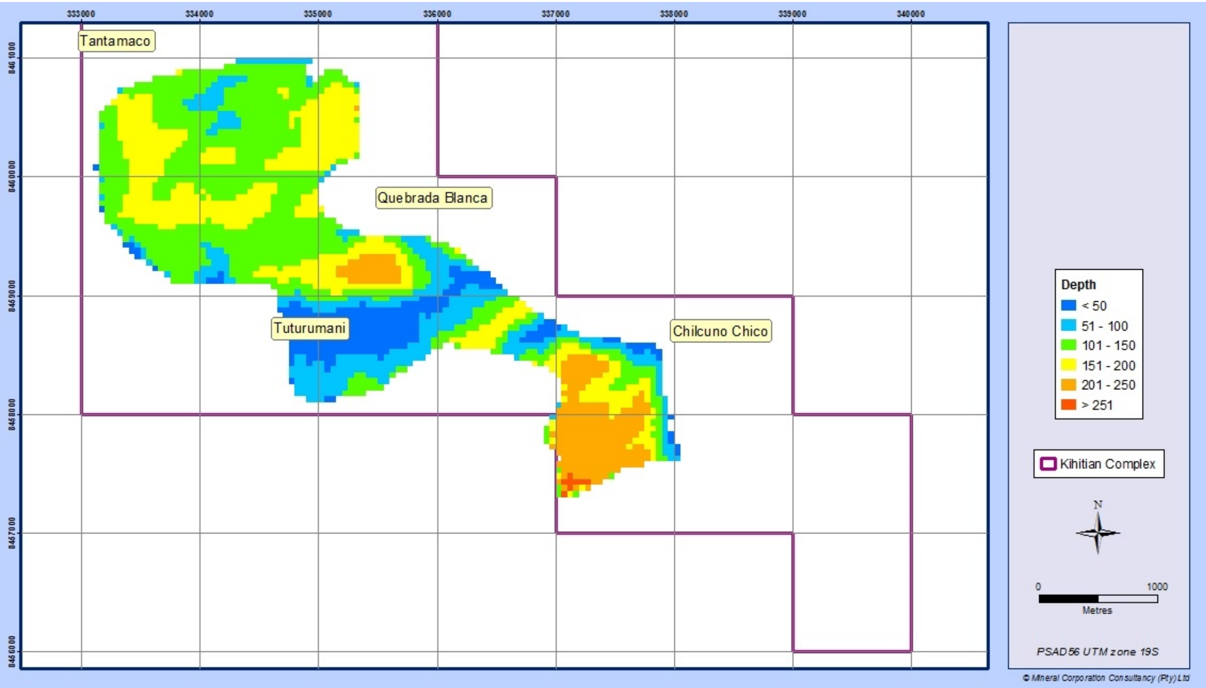


Figure 14-7: Depth to the Base of Level B for the Kihitian Complex

Table 14-7: Kihitian Complex Mineral Resource Tabulation Depicting Mining Method

				Metric units						Imperial units			
Deposit	Mining Method	Classification	Cut-off	Tonne (000s)	Density (t/m3)	U grade (ppm)	U_3O_8 grade (ppm)	U Content (000s kg)	U_3O_8 Content (000s kg)	U Content (000s lb)	Ton (000s)	U_3O_8 Content (Mlb)	U_3O_8 Grade (lb/ton)
Chilcuno Chico	Surface	Indicated	75	6 352	1.98	190	225	1 210	1 426	2 667	7 002	3.14	0.45
Chilcuno Chico	Surface	Inferred	75	11 836	1.98	242	285	2 861	3 374	6 308	13 047	7.44	0.57
Chilcuno Chico	Underground	Indicated	75	28 488	1.98	225	265	6 399	7 546	14 107	31 402	16.64	0.53
Chilcuno Chico	Underground	Inferred	75	19 159	1.98	327	385	6 256	7 377	13 791	21 119	16.26	0.77
Quebrada Blanca	Surface	Indicated	75	5 509	1.98	279	329	1 538	1 814	3 391	6 073	4.00	0.66
Quebrada Blanca	Surface	Inferred	75	12 816	1.98	264	311	3 383	3 990	7 459	14 127	8.80	0.62
Quebrada Blanca	Underground	Indicated	75	0	1.98	0	0	0	0	0	0	0.00	0.00
Quebrada Blanca	Underground	Inferred	75	620	1.98	375	442	233	274	513	684	0.60	0.88
Tuturamani	Surface	Indicated	75	0	1.98	0	0	0	0	0	0	0.00	0.00
Tuturamani	Surface	Inferred	75	3 300	1.98	146	172	482	569	1 063	3 638	1.25	0.34
Tantamaco	Surface	Indicated	75	6 721	1.98	198	233	1 328	1 566	2 928	7 408	3.45	0.47
Tantamaco	Surface	Inferred	75	24 348	1.98	177	208	4 298	5 069	9 476	26 840	11.17	0.42
Tantamaco	Underground	Indicated	75	673	1.98	120	141	81	95	178	742	0.21	0.28
Tantamaco	Underground	Inferred	75	11 501	1.98	161	190	1 850	2 182	4 079	12 678	4.81	0.38

14.4.12 RECONCILIATION

Mineral Resources for Chilcuno Chico and Quebrada Blanca were estimated by TMC in 2013, using a constrained volume defined by two horizons, which were constructed on a section by section basis and within which uranium grades were estimated using Ordinary Kriging. Mineral Resources for the Tantamaco and Tuturamani deposits were previously estimated by Henkle (2014) (14), using the polygonal estimation technique.

Since then, no more drilling has been carried out on the deposits; however, a consolidation of the databases has resulted in a better understanding of the mineralisation, prompting a review and standardisation of the estimation technique applied.

14.4.12.1 LIKE FOR LIKE RECONCILIATION

The previus total Mineral Resource estimates by Henkle (2014) (14) and Young (2013) (17) are presented in Table 14-8.

Table 14-8: Previous Mineral Resource Estimates Reported in Henkle (2014) (14) and Young (2013) (17)

	Cut-off	Tonne	Grade	Grade	Content	Content
Deposit	U ppm	Mt	U ppm	ppm (U_3O_8)	kg (U_3O_8)	Mlb (U_3O_8)
Quebrada Blanca	75	2.14	503	593	1 269 099	2.80
Chilcuno Chico	75	19.04	533	628	11 964 869	26.36
Tantamaco	77	45.90	162	191	8 764 800	19.32
Tuturumani	77	5.50	85	100	550 000	1.21
Totals		**72.60**	**282**	**311**	**22 548 768**	**49.69**

A comparison of the current estimates within the same footprint used by Henkle (2014) (14) within the Kihitian Complex are summarised in Table 14-9. The two estimates utilised the same borehole database.

Table 14-9: Current Mineral Resource Estimates (Amended Footprint)

	Cut-off	Tonne	Grade	Grade	Content	Content
Deposit	U ppm	Mt	U ppm	ppm (U_3O_8)	kg (U_3O_8)	Mlb (U_3O_8)
Quebrada Blanca	75	3.58	259	305	1 093 738	2.41
Chilcuno Chico	75	47.25	224	264	12 476 994	27.51
Tantamaco	77	36.32	174	206	7 463 422	16.45
Tuturumani	77	2.37	137	161	382 122	0.84
Total		**89.52**	**203**	**239**	**21 416 275**	**47.21**

14.4.13 RECONCILIATION OF TONNAGE AND GRADE DIFFERENCES

The differences between the previous and current estimates are attributed to the following:

14.4.13.1 ESTIMATION METHOD

The current estimates were carried out using MIK, while previously, for Chilcuno Chico and Quebrada Blanca, 2-D Ordinary Kriging was applied. For the Tuturumani and Tantamaco deposits, upper and lower evaluation cuts were applied to polygons. These changes in methodology have lead to changes in the tonnage and uranium grades in each deposit.

14.4.13.2 FOOTPRINT

The comparison footprint used for Zones A and B are shown in Figure 14-8 below.

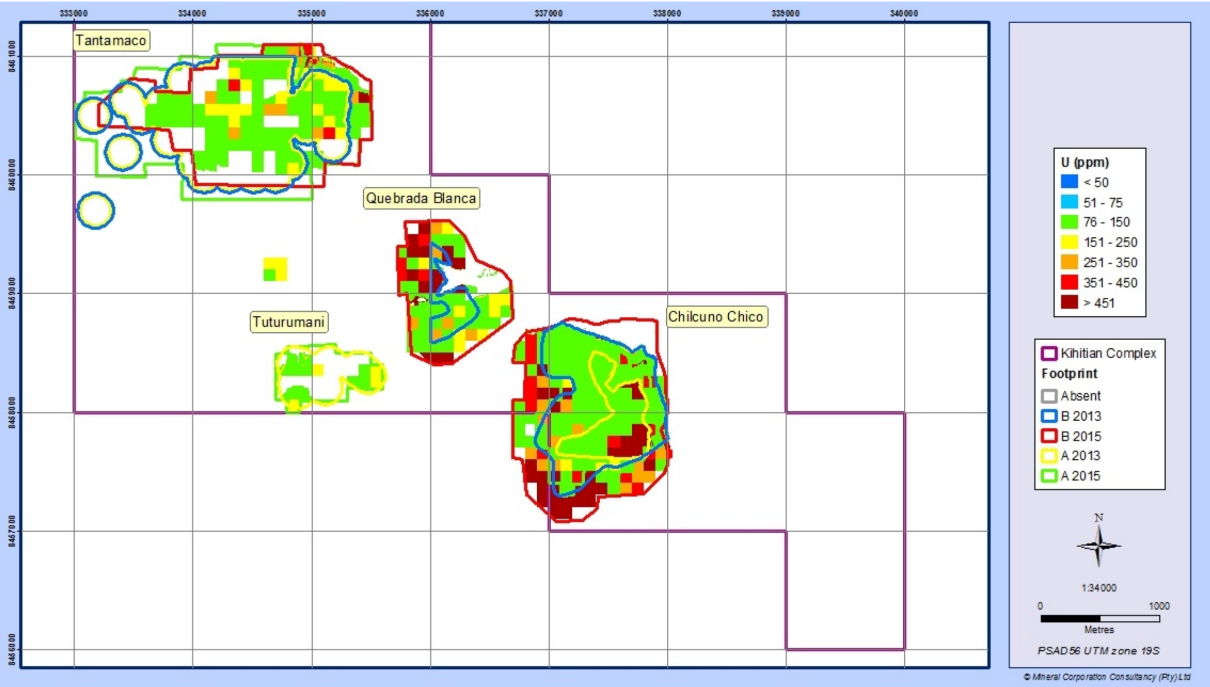


Figure 14-8: Comparison of the Previous vs. Current Upper Zone (Level A) and Lower Zone (Level B) Footprints

The extent of the footprints is discussed in Section 14.4.8.3. In addition, Quebrada Blanca and Chilcuno Chico previously had a tenure boundary constraint which was removed resulting in an increase in the estimates.

14.4.13.3 DENSITY

The following density criterion was used previously, and was honoured in the reconciliation:

Table 14-10: Density

Deposit	Density (t/m^3)
Chilcuno Chico	1.95
Quebrada Blanca	1.95
Tuturumani	1.98

14.4.13.4 U GRADE AND CONTAINED U

The revised MIK methodology has essentially lead to the estimation of lower U grades in the Quebrada Blanca and Chilcuno Chico deposits, that are compensated for by a higher tonnage resulting in similar contained U content (Table 14-9). The reverse is true for the Tantamaco and Tuturamani deposits where slightly lower tonnages are compensated for by higher U grades resulting in similar contained U content (Table 14-9).

14.5 ISIVILLA COMPLEX

14.5.1 DRILLHOLE DATABASE

The database which informs the Calvario I and Puncopata Mineral Resources was entirely sourced from the previous owners, Solex, whereas the database which informs the Calvario Real and Isivilla Mineral Resources was sourced from Minergia. TMC captured the electronic exploration archive, which comprised Microsoft Excel-based drillhole files, including collar, lithological logs, structural logs, radiometric logs and analytical results into a database suitable for use in Datamine Studio™. A summary is provided in Table 14-11.

Table 14-11: A Summary of the Drillhole Database

Deposit	Drillholes	Total Drilled (m)	Total Sampled (m)
Calvario I	71	3 856.72	3 856.72
Puncopata	34	2 720.54	2 260.00
Calvario Real	9	1 625.70	1 000.40
Isivilla	27	3 580.50	3 580.50
Totals	**141**	**11 783.46**	**10 697.62**

14.5.2 OREBODY MODEL

An integration of the drillhole data from the four deposits was done in order to validate the proposed estimation methodology. It is interpreted that the deposits belong to the same mineralisation system, as reasonable correlation across the deposits was noted.

14.5.3 DOMAINS

Mineralisation at Calvario I is interpreted to be exclusively in Level A, at Puncopata exclusively Level B, and in both Level A and Level B at Calvario Real and Isivilla. The depth from surface to the top of Level A varies from 20 m to 30 m while the depth from surface to the top of Level B varies from 20 m to 80 m.

Both levels have been noted to outcrop and although there are localised variations, the contact is generally flat, dipping up to a maximum of 4° to the northeast.

U abundance in all four deposits exhibits high variability across drillholes while grade distribution down a drillhole indicates either only one or two clearly defined peaks.

14.5.4 EXPLORATORY DATA ANALYSIS

14.5.4.1 DISTRIBUTION

The U variable for the 1.5 m composites has a highly skewed-distribution in all deposits, as shown in Figure 14-9.

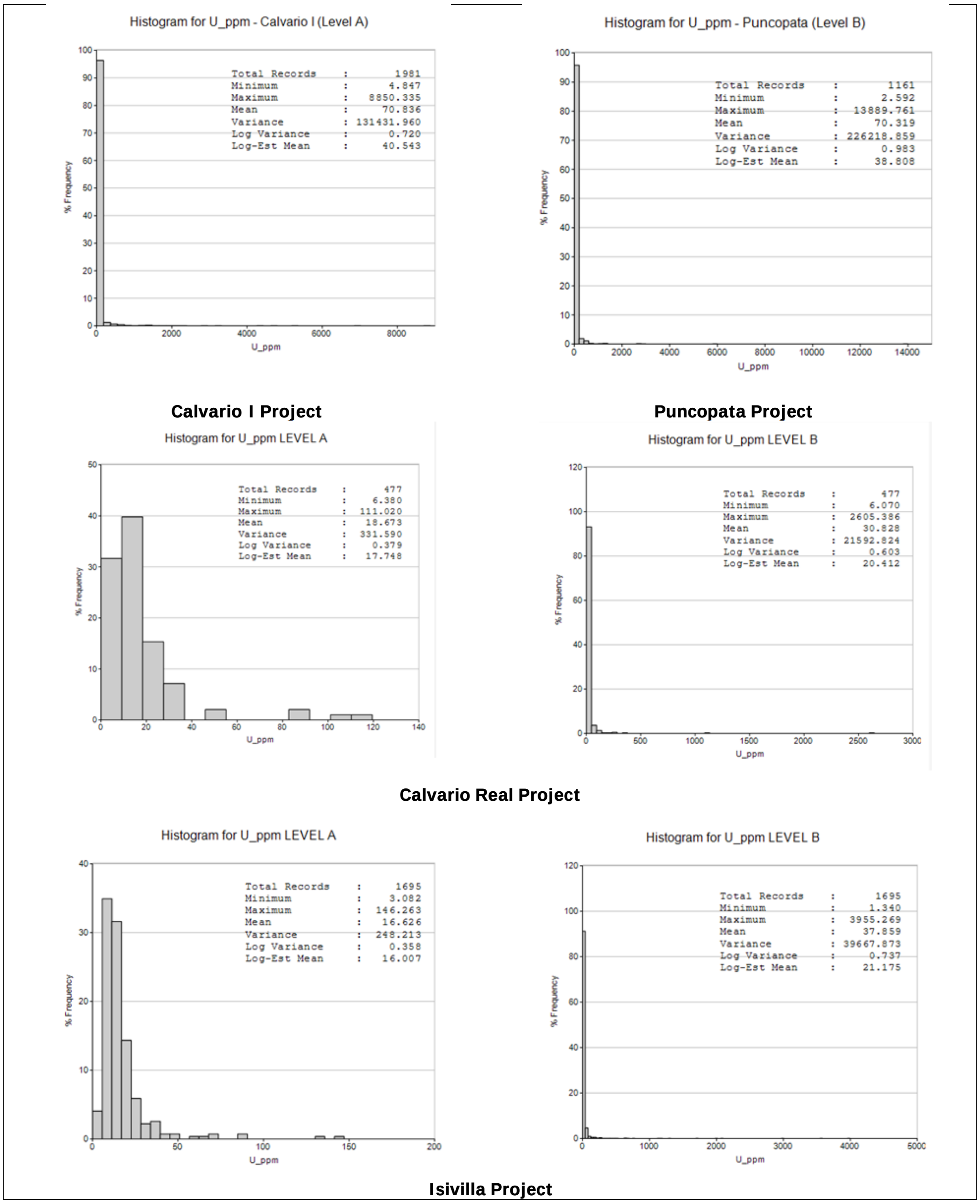


Figure 14-9: 1.5 m Composite U ppm Distribution for the Four Deposits of the Isivilla Complex

The probability distribution functions for the Level A and Level B were analysed for each of the four deposits, and the cut-offs selected are contained in Table 14-12. It was noted that the U distribution only differed between Level A and Level B, but did not differ between the deposits. As a result, a single set of indicator cut-off's was selected for the deposits.

Table 14-12: Summary of Indicator Cut-offs Selection (Tab 39)

Deposits	Cut-off	
	Level A (U ppm)	Level B (U ppm)
All deposits within Isivilla Complex	15	20
All deposits within Isivilla Complex	140	250
All deposits within Isivilla Complex	750	750

14.5.5 PREFERRED MINERALISATION ORIENTATION

The contact between Levels A and B was established and then geologically correlated over the four deposits through a wireframe created for this surface, in order to control a mineralisation, interpolation in a sub-horizontal orientation.

14.5.6 VARIOGRAM ANALYSIS

Indicator variograms using the indicator cut-off values contained in Table 14-12 were analysed, within each deposit. The vertical variograms were used to derive the nugget effect and range in the Z direction.

The interpreted variogram model parameters are contained in Table 14-13. The detailed variograms are contained in Appendix 8 of Young (2015) (17).

Table 14-13: Summary of Variogram Parameters (for all Deposits within Isivilla Complex)

Level A					
Cut-off	Nugget	Range (X)	Range (Y)	Range (Z)	Sill
15	0.002	90	90	90	0.056
140	0.010	90	90	90	0.033
750	0.001	35	35	35	0.011
Level B					
Cut-off	Nugget	Range (X)	Range (Y)	Range (Z)	Sill
20	0.106	175	175	175	0.208
250	0.014	50	50	50	0.024
750	0.003	40	40	40	0.012

14.5.7 LOCAL MEANS

Table 14-14 contains the MIK bin mean grades.

Table 14-14: MIK Bin Grades

	Level A		Level B	
Deposit	**Grade Bin (U ppm)**	**Bin Grade (U ppm)**	**Grade Bin (U ppm)**	**Bin Grade (U ppm)**
All deposits within Isivilla	0 - 15	11	0 – 20	12
All deposits within Isivilla	15 - 140	30	20 - 250	50
All deposits within Isivilla	140 - 750	378	250 - 750	415
All deposits within Isivilla	>750	2 317	> 750	2 359

14.5.8 MODEL CONSTRAINTS

Although unconstrained estimation was employed, the following surfaces were used to limit the extents of estimation.

14.5.8.1 SURFACE TOPOGRAPHY

The model was constrained by the surface topography.

14.5.8.2 DEPTH CONSTRAINT

A surface was created that limited the depth of estimation to the bottom of Level B

14.5.8.3 PLANAR EXTENTS

Although the relationship between Level A and Level B was defined where possible across the whole Complex area, with the definition of a geologically continuous contact, the extent of the estimation was limited to three times the variogram range, amounting to the extrapolation distances shown in Table 14-15.

Table 14-15: Extrapolation Distances

Deposits	Level A (m)	Level B (m)
All deposits within Isivilla Complex	270	150

The block models are depicted in Figure 14-10 and Figure 14-11.

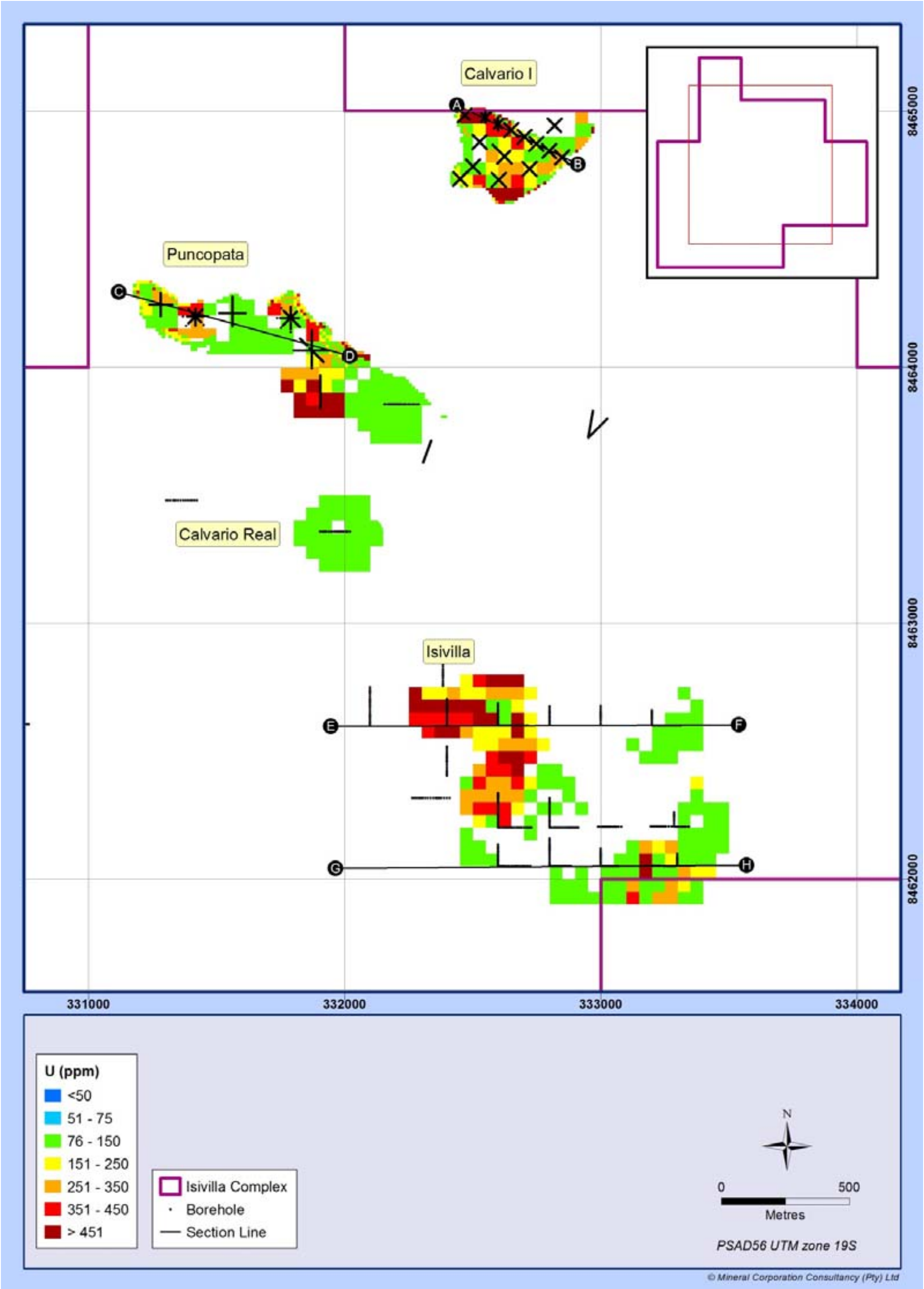


Figure 14-10: Isivilla Complex Mineral Resource Area Showing Uranium Abundance (U ppm), Projected to Surface, with Section Lines A-B, C-D, E-F and G-H Indicated

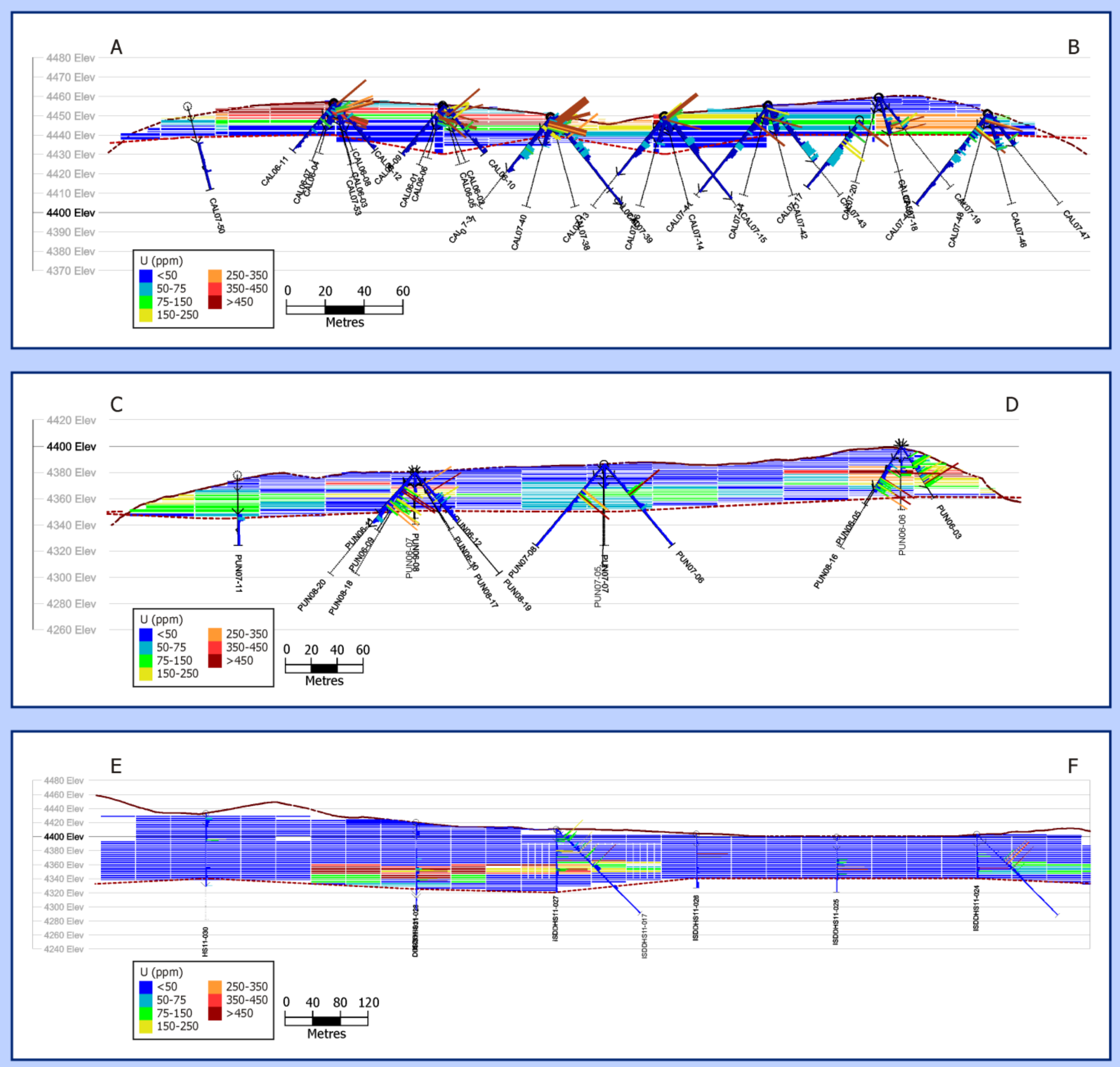


Figure 14-11: Block Model Cross-Sections through Calvario I (A-B), Puncopata (C-D) and Isivilla (E-F)

14.5.9 MINERAL RESOURCE CLASSIFICATION

The Kriging Efficiency (Section 14.3.11), search volume supported by geological confidence and QA/QC considerations were used as guidelines for classification of the Mineral Resources.

14.5.9.1 GEOLOGICAL CONFIDENCE

The deposits within the Isivilla Complex demonstrate mineralisation which conforms generally to the geological and mineralisation models for the other Complexes, and confidence in the geological continuity would be sufficient for the reporting of Indicated and Inferred Mineral Resources.

14.5.9.2 DOWNHOLE SURVEY

No downhole surveys have been carried out.

14.5.9.3 QA/QC

The limited information relating to sample preparation, security and analytical procedures for the data informing the Puncopata, Calvario I and Calvario Real deposits limits the reporting of Mineral Resources to the Inferred classification. The Isivilla deposit data can be employed for the reporting of Mineral Resources in the Indicated and Inferred categories.

14.5.9.4 MINERAL RESOURCE CLASSIFICATION

Indicated and Inferred Mineral Resources were defined for the Isivilla Complex, and classification plots are shown in Figure 14-12 and Figure 14-13.

14.5.10 MINERAL RESOURCE STATEMENT

The estimated Mineral Resources for the Calvario I, Puncopata, Isivilla and Calvario Real deposits at a 75 ppm U are summarised in Table 14-16.

Table 14-16: Isivilla Complex Mineral Resource Statement as at 31 March 2015

			Metric Units						Imperial Units			
Deposit	**Classification**	**Cut Off (U ppm)**	**Tonne t (000s)**	**Density (t/m^3)**	**U Grade (ppm)**	**U_3O_8 Grade (ppm)**	**U Content (000s kg)**	**U_3O_8 Content (000s kg)**	**U Content (000s lb)**	**Ton (000s)**	**U_3O_8 Content (Mlb)**	**U_3O_8 Content (lb/ton)**
Puncopata	Inferred	75	5,923	1.98	216	255	1,277	1,506	3,320	6,529	3.32	0.51
Calvario I	Inferred	75	1,679	1.98	268	316	450	531	1,170	1,851	1.17	0.63
Calvario Real	Inferred	75	1,146	1.98	90	106	104	122	269	1,264	0.27	0.21
Isivilla	Indicated	75	4,568	1.98	296	349	1,354	1,597	3,520	5,035	3.52	0.7
Isivilla	Inferred	75	7,396	1.98	295	348	2,182	2,573	5,673	8,153	5.67	0.7
Total Indicated		**75**	**4,568**	**1.98**	**296**	**349**	**1,354**	**1,597**	**3,520**	**5,035**	**3.52**	**0.70**
Total Inferred		**75**	**16,145**	**1.98**	**249**	**293**	**4,013**	**4,732**	**10,432**	**17,797**	**10.43**	**0.59**
Total Mineral Resources		**75**	**20,713**	**1.98**	**259**	**306**	**5,367**	**6,329**	**13,952**	**22,832**	**13.95**	**0.61**

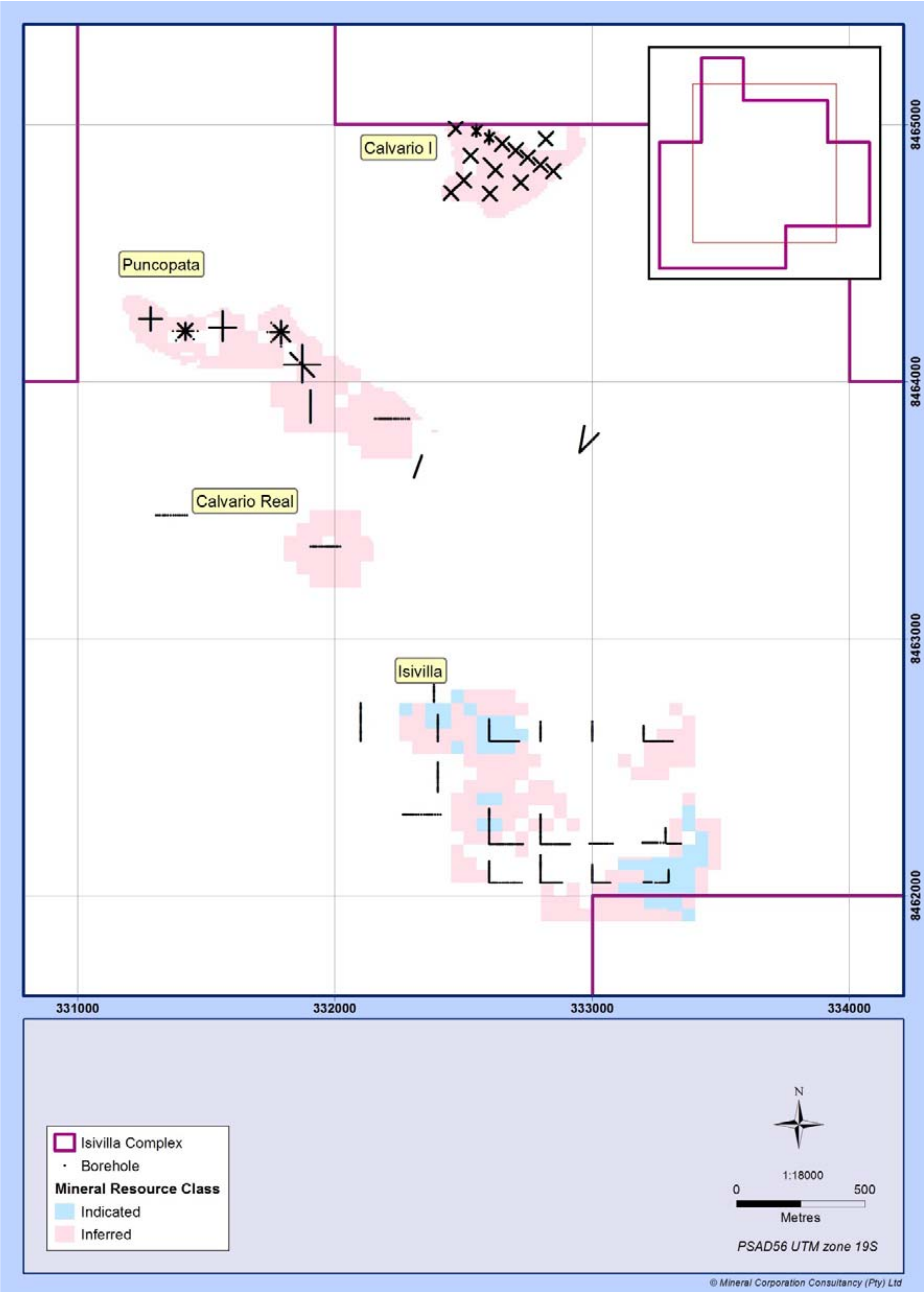


Figure 14-12: Mineral Resources Classification for Calvario I, Puncopata, Calvario Real and Isivilla at 75 ppm Cut-off

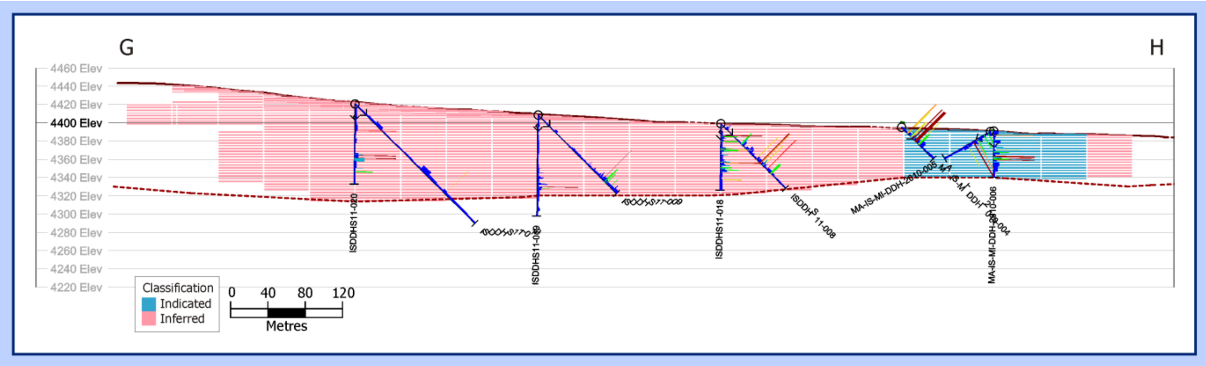


Figure 14-13: Block Model Cross-sections Through Isivilla (G-H)

14.5.11 ASSUMPTIONS FOR MINING

All of the mineralisation within the Isivilla Complex is within 150 m of surface, and hence the assumption is made that the Mineral Resources would be mined by open pit methods.

14.5.12 RECONCILIATION

In 2012, TMC estimated Mineral Resources for part of the Isivilla Complex and reported separate estimates for the Calvario I and Puncopata deposits. It should be noted that the methods adopted for use in 2012 and in the update in 2015 are materially different.

A reconciliation of the estimates reported, confined to the extents used in the 2012 estimates, has been carried out and the reconciled figures summarised are in Table 14-17.

Table 14-17: Reconciliation of Mineral Resources Reported as at September 2012 and April 2015

Project	Category	Tonne (000s t)	Density (t/m^3)	U (ppm)	U_3O_8 Content (000s kg)	U_3O_8 Content (000s lb)	U_3O_8 Content (lb/t)	U_3O_8 Content (lb/ton)
Puncopata 2012	Inferred	2 456	1.98	269	779	1 717	0.70	0.63
Puncopata 2015	Inferred	5 923	1.98	216	1 509	3 326	0.56	0.51
Calvario I 2012	Inferred	1 024	1.98	530	641	1 413	1.38	1.25
Calvario I 2015	Inferred	1 679	1.98	268	531	1 170	0.70	0.63
Isivilla 2012	Indicated	4 500	1.98	127	675	1 262	0.33	0.30
Isivilla 2012	Inferred	6 900	1.98	356	2 898	5 418	0.93	0.84
Isivilla 2015	Indicated	4 568	1.98	296	1 597	2 985	0.77	0.70
Isivilla 2015	Inferred	7 396	1.98	295	2 573	4 811	0.77	0.70

To simplify the reconciliation, the footprint, density and cut-offs were kept constant.

A reasonable reconciliation of reported in-situ grade estimation was achieved in the Puncopata deposit owing to the very robust definition of the PEM. The significant increase in the footprint of the Puncopata deposit has been attributed to the new drilling information south of the area delineated in 2012, and a much wider zone of potentially economic mineralisation. The wider mineralised zone is attributed to the use of MIK within a broader mineralised zone, rather than discreet wireframes.

A reasonable reconciliation of tonnage estimation was achieved in the Calvario I deposit owing to its close proximity to surface hence lending no scope for significant difference in thickness. However, the significant decrease in reported in-situ grade at the Calvario I deposit can be attributed to the impact of the change in methodology, which has resulted in the inclusion of much lower grade material which previously had been excluded using wireframes.

There is an increase in the uranium grade within the Indicated Mineral Resource estimates for the Isivilla deposit, and a decrease in the Inferred Mineral Resource grade. There has been an overall increase in the total contained metal at Isivilla.

14.6 CORANI COMPLEX

14.6.1 DRILLHOLE DATABASE

The Microsoft Excel-based dataset provided has been used to build a Datamine Studio™ drillhole database for the purpose of Mineral Resource estimation. A description of the validation procedures is included in Section 12. The database comprises 174 drillholes from three deposits.

TMC considers the drillhole database to be suitable for the purposes of Mineral Resource estimation.

14.6.2 PEM MODEL

An integration of the drillhole data from the three deposits was done in order to validate the proposed estimation methodology. It is interpreted that the deposits belong to the same mineralisation system, as reasonable correlation across the deposits was noted.

14.6.3 DOMAINS

The Level A and the Level B are considered geologically separate units and were treated as such. Furthermore, investigation of the distribution of uranium grades within Level A and Level B at Nueva Corani revealed the deposit to have a distinct population, and was treated independently.

The Level A unit is not present at Calvario II and the uranium mineralisation in Level B at Nueva Corani is extremely poor and no Mineral Resources have been estimated.

14.6.4 EXPLORATION DATA ANALYSIS

14.6.4.1 DISTRIBUTION

Figure 14-14 shows the U abundance distribution for the 1.5 m composites, for the two levels. The distribution is highly skewed. Figure 14-14 also contains the descriptive statistics for the same population.

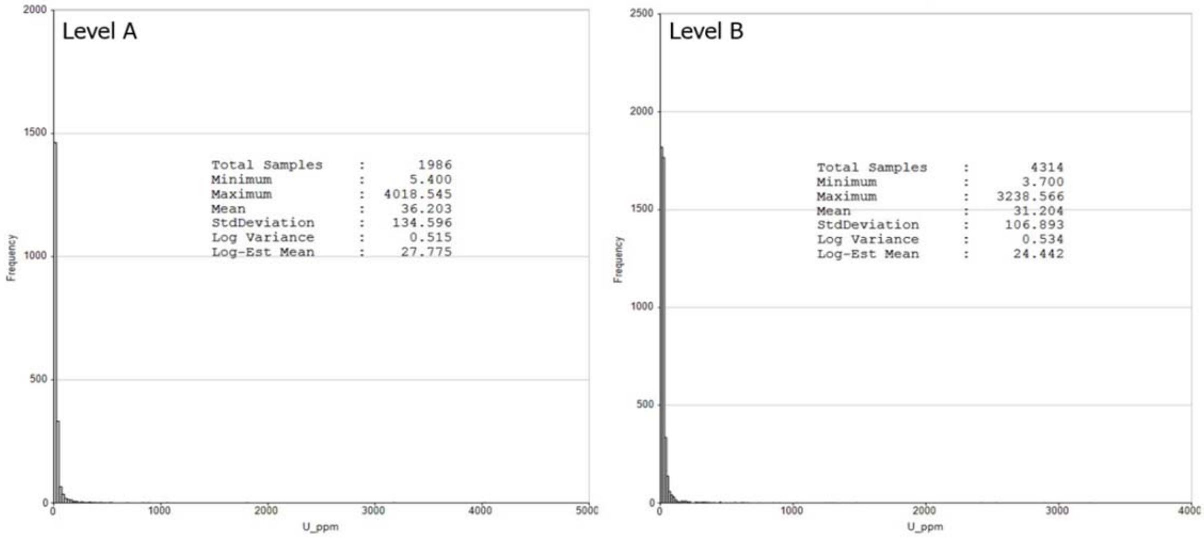


Figure 14-14: 1.5 m Composite U ppm Distributions in Normal Space for Calvario II, Calvario III and Nueva Corani Deposits

The Nueva Corani dataset shows significantly different variances compared to the regional dataset, and it was thus modelled independently. The distribution for the Nueva Corani dataset and descriptive statistics are given in Figure 14-15.

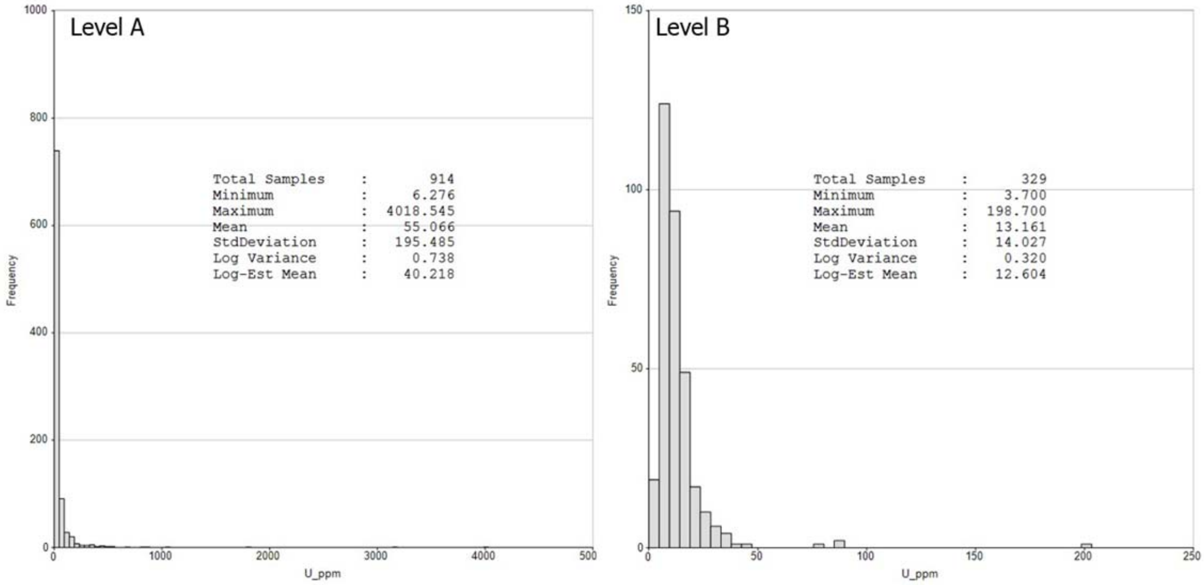


Figure 14-15: 1.5 m Composite U ppm Distributions in Normal space for Nueva Corani

14.6.4.2 STATISTICS AND SELECTION OF INDICATOR CUT-OFFS

The probability distribution functions for Level A and Level B for each of the three deposits were analysed, and the cut-offs selected are in Table 14-18.

Table 14-18: Indicator Cut-offs Identified on the 1.5 m Composite Data

	Cut-Off	
Deposit	**Level A (U ppm)**	**Level B (U ppm)**
Calvario II and III	35	35
	170	120
	550	700
Neuva Corani	50	-
	185	-
	500	-

14.6.5 PREFERRED MINERALISATION ORIENTATION

The contact between Levels A and B was established and then geologically correlated over the three deposits through a wireframe created for this surface, in order to control a mineralisation interpolation in a sub-horizontal orientation.

14.6.6 VARIOGRAM ANALYSIS

Indicator variograms using the indicator cut-off values contained in Table 14-18 were analysed, within each deposit. The vertical variograms were used to derive the nugget effect and ranges in the Z direction.

While variograms were assessed for each cut-off, the variograms within each domain were relatively similar and hence were approximated by a single variogram (Section 14.3.6).

Relatively well-structured long range variograms were obtained for the lowest cut-off, after which ranges drop to 50 % or less for the upper cut-offs as the relationship between higher-grade samples deteriorates. The ranges from the variogram of the middle cut-off were used to define search distances. The parameters for the modelled variograms are summarised in Table 14-19. The detailed variography is contained in Appendix 9 of Young 2015 (74).

Table 14-19: Variogram Model Parameters

Deposit	Level	Cut-off (U ppm)	Nugget	Range (X) m	Range (Y) m	Range (Z) m	Sill
Calvario II	A	35	0.04	421	421	36	0.132
	A	170	0.0136	268	268	10	0.023
	A	550	0.0018	264	264	4	0.0038
	B	35	0.04	8	8	12	0.118
	B	120	0.01	5	5	10	0.024
	B	700	0.0005	15	15	4	0.0025
Calvario III	A	35	0.04	421	421	36	0.132
	A	170	0.0136	268	268	10	0.023
	A	550	0.0018	264	264	4	0.0038
	B	35	0.04	8	8	12	0.118
	B	120	0.01	5	5	10	0.024
	B	700	0.0005	15	15	4	0.0025
Nueva Corani	A	50	0.04	421	421	36	0.132
	A	185	0.0136	268	268	10	0.023
	A	500	0.0018	264	264	4	0.0038
	B	-	-	-	-	-	-
	B	-	-	-	-	-	-
	B	-	-	-	-	-	-

14.6.7 LOCAL MEANS

Table 14-20 contains the MIK bin mean grade.

Table 14-20: Local Means

Deposit	Level A		Level B	
	Grade Bin (U ppm)	Bin Grade (U ppm)	Grade Bin (U ppm)	Bin Grade (U ppm)
Calvario II and III	0 - 35	17.87	0 - 35	16.43
	35 - 170	66.09	35 - 120	56.66
	170 - 550	303	120 - 700	276
	>550	1 748	>700	1 575
Neuva Corani	0 - 50	22.30	-	-
	50 – 185	91.60	-	-
	185 – 500	314	-	-
	>500	1457	-	-

14.6.8 MODEL CONSTRAINTS

Although unconstrained estimation was employed, the following surfaces were used to limit the extents of estimation.

14.6.8.1 SURFACE TOPOGRAPHY

The block model was clipped against the topography.

14.6.8.2 DEPTH CONSTRAINT

A surface was created that limited the depth of estimation to the bottom of Level B.

14.6.8.3 PLANAR EXTENTS

Although the relationship between Level A and Level B was defined where possible across the whole Complex area, with the definition of a geologically continuous contact, the extent of the estimation was limited to three times the variogram range, amounting to the extrapolation distances shown in Table 14-21 per project.

Table 14-21: Extrapolation Distances

Deposit	Level A (m)	Level B (m)
Calvario II	-	200
Cavario III	250	-
Neuva Corani	536	-

The block models are depicted in Figure 14-16 to Figure 14-19.

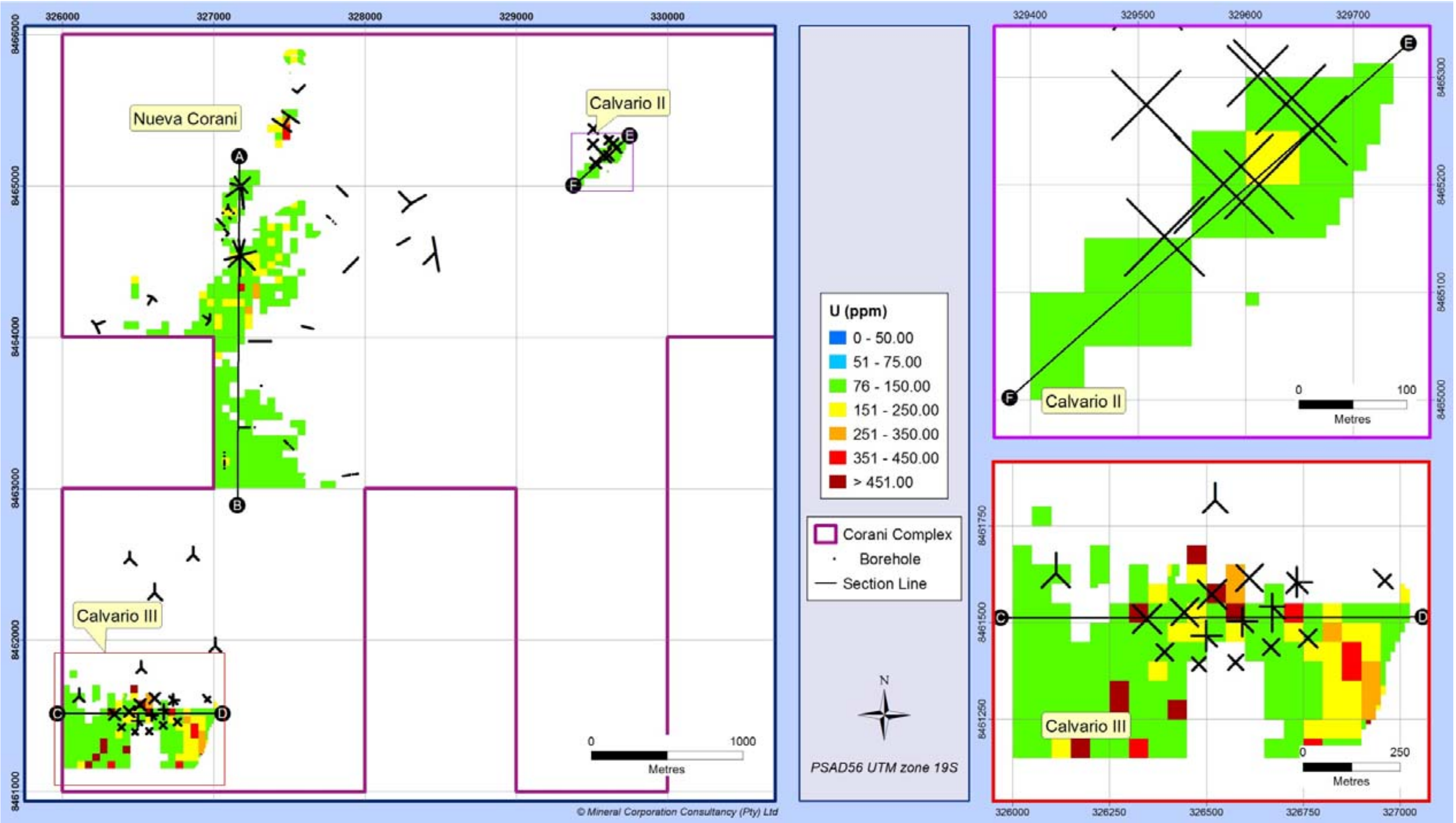


Figure 14-16: Calvario II, Calvario III and Nueva Corani Plan Showing the Block Model at a 75 ppm U Cut-off

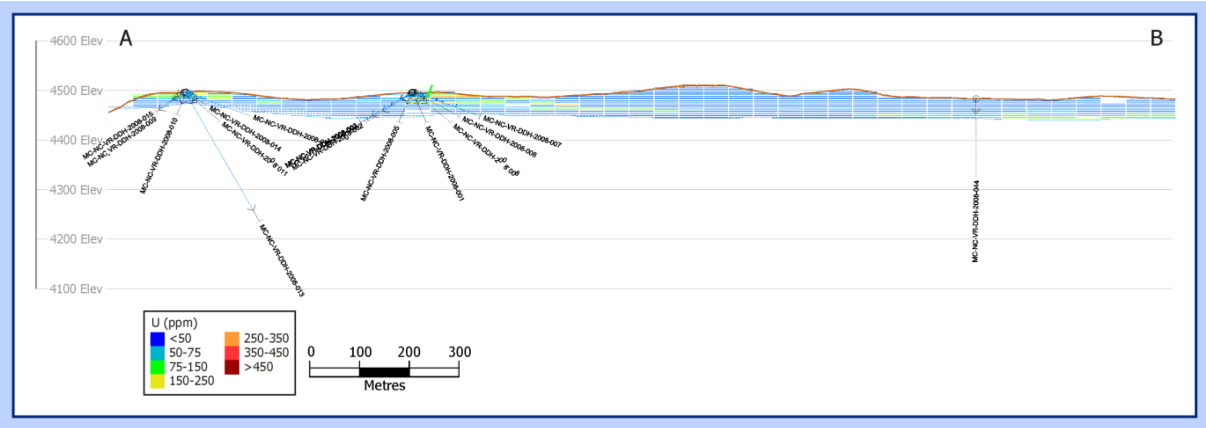


Figure 14-17: North-South Section Through Neuva Corani

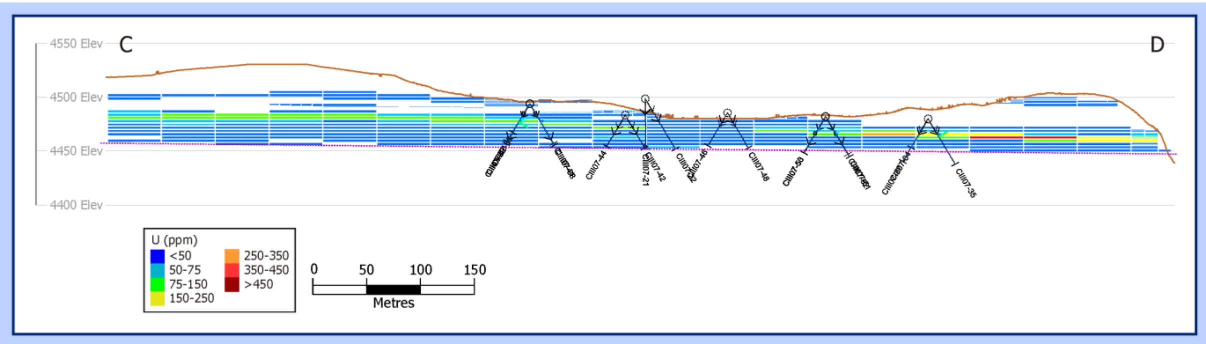


Figure 14-18: West-East Section Through Calvario III

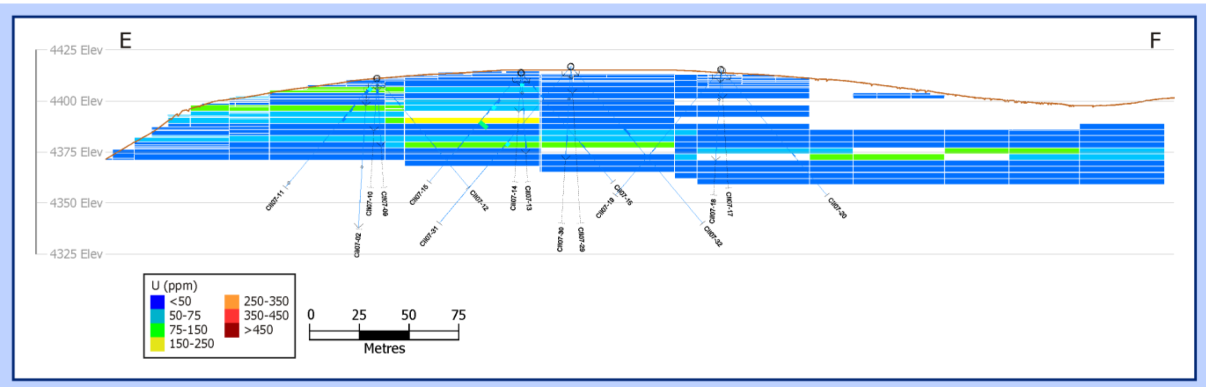


Figure 14-19: North-East South-West Section Through Calvario II

14.6.9 MINERAL RESOURCE CLASSIFICATION

The kriging efficiency (Section 14.3.11), search volume supported by geological confidence and QA/QC considerations were used as guidelines for classification of the Mineral Resources.

14.6.9.1 GEOLOGICAL CONFIDENCE

The deposits within the Corani Complex demonstrate mineralisation which conforms generally to the geological and mineralisation models for the other Complexes, and confidence in the geological continuity would be sufficient for the reporting of Indicated and Inferred Mineral Resources.

14.6.9.2 DOWNHOLE SURVEY

No downhole surveys have been carried out on the Complex.

14.6.9.3 QA/QC

The limited information relating to sample preparation, security and analytical procedures for the data informing the Calvario II and Calvario III deposits limits the reporting of Mineral Resources to the Inferred classification. The Nueva Corani deposit data can be employed for the reporting of Mineral Resources in the Indicated and Inferred categories.

14.6.9.4 MINERAL RESOURCE CLASSIFICATION

Indicated and Inferred Mineral Resources were defined for the Corani Complex, and classification plots are shown in Figure 14-20 and Figure 14-21.

14.6.10 MINERAL RESOURCE STATEMENT

The estimated Mineral Resources for the Calvario II, Calvario III and Nueva Corani deposits at 75 ppm U cut-off are summarised in Table 14-22.

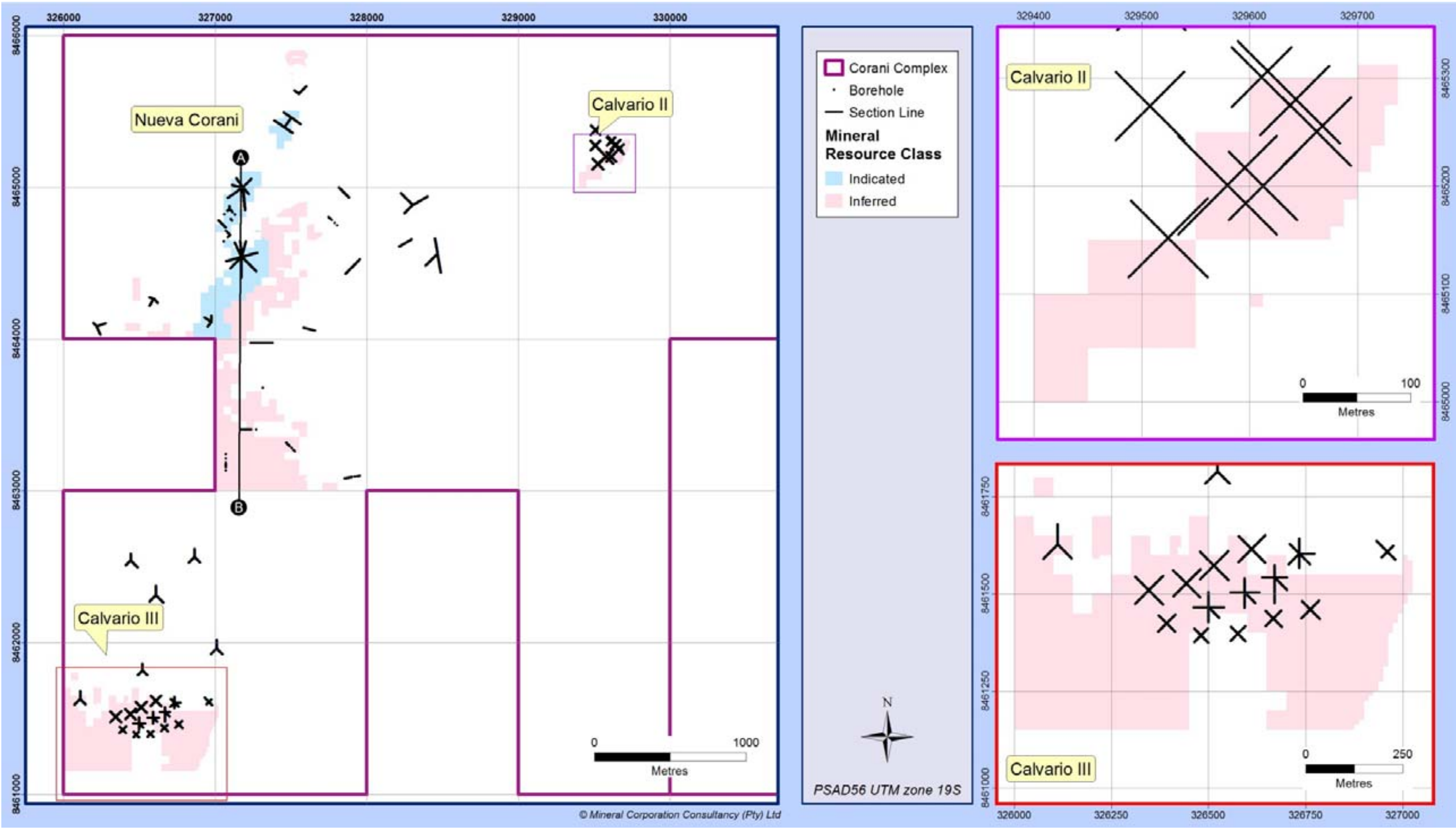


Figure 14-20: Calvario II, Calvario III and Nueva Corani Plan Showing the Block Model Mineral Resource Classification at a 75 ppm U Cut-off

Table 14-22: Corani Complex Mineral Resource Estimates as at 31 March 2015

Deposit	Classification	Cut-off (U ppm)	Metric Units						Imperial Units			
			Tonne (000s)	Density (t/m3)	U grade (ppm)	U_3O_8 grade (ppm)	U Content (000s kg)	U_3O_8 Content (000s kg)	U Content (000s lb)	Ton (000s)	U_3O_8 Content (Mlb)	U_3O_8 Grade (lb/ton)
Calvario II	Inferred	75	337	1.98	94	110	32	37	70	372	0.08	0.22
Calvario III	Inferred	75	3,845	1.98	177	208	679	801	1,497	4,238	1.77	0.42
Nueva Corani	Indicated	75	3,397	1.98	141	166	479	565	1,056	3,744	1.25	0.33
Nueva Corani	Inferred	75	6,112	1.98	111	131	678	799	1,494	6,737	1.76	0.26
Total Indicated		**75**	**3,397**	**1.98**	**141**	**166**	**479**	**565**	**1,056**	**3,744**	**1.25**	**0.33**
Total Inferred		**75**	**10,294**	**1.98**	**135**	**159**	**1,389**	**1,637**	**3,061**	**11,347**	**3.61**	**0.32**
Total Mineral Resources		**75**	**13,691**	**1.98**	**136**	**161**	**1,868**	**2,202**	**4,117**	**15,091**	**4.86**	**0.32**

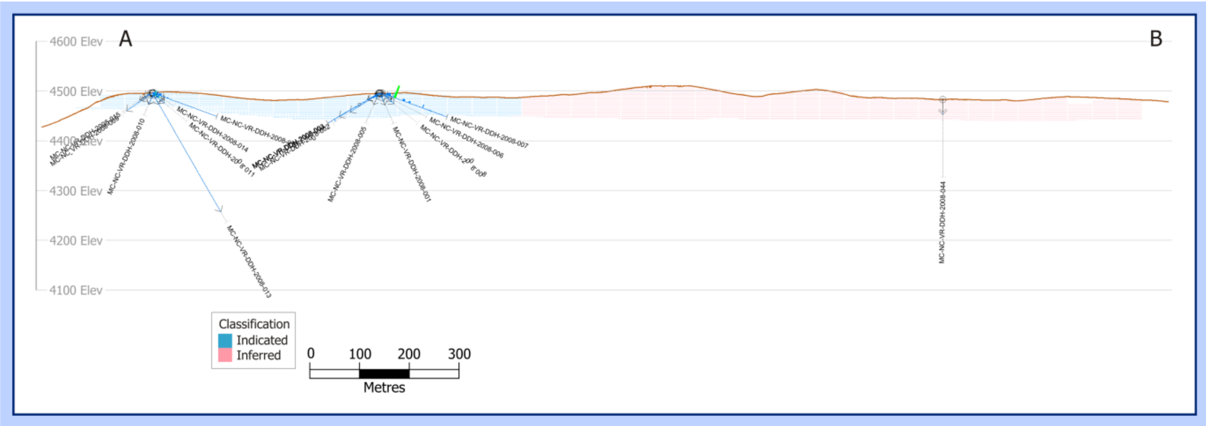


Figure 14-21: North-South Section through Neuva Corani Showing Classification

14.6.11 ASSUMPTIONS FOR MINING

All of the mineralisation within the Corani Complex is within 150 m of surface, and hence the assumption is made that the Mineral Resources would be mined by open pit methods.

14.6.12 RECONCILIATION

Only the Mineral Resources at Nueva Corani were previously estimated by Henkle (2014) (14), using the polygonal estimation technique. Since then no further drilling has been conducted on the Complex. The previous Mineral Resource estimates are in Table 14-23.

Table 14-23: Mineral Resource Estimates Compared with Henkle (2014)

Author	Deposit	Classification	Cut-off (U ppm)	Tonne (000s)	Density (t/m3)	U grade (ppm)	U Content (000s kg)	U_3O_8 Content (000s kg)	U_3O_8 Content (Mlb)	U_3O_8 Grade (lb/ton)
Henkle 2014	Nueva Corani	Indicated	77	3 200	1.98	85	271	320	0.71	0.20
Henkle 2014	Nueva Corani	Inferred	77	7 300	1.98	178	1 300	1 533	3.38	0.42
Young 2015	Nueva Corani	Indicated	75	3 396	1.98	141	479	565	1.25	0.33
Young 2015	Nueva Corani	Inferred	75	6 112	1.98	111	678	799	1.76	0.26

A footprint of the reconciliation of the estimates by Henkle and Associates compared to TMC estimates is shown in Figure 14-22.

The difference between the two estimates is attributed to the following:

- Henkle and Associates estimated into two zones, the upper and lower zone, while TMC only estimated into Level A, the equivalent of the upper zone;
- Henkle and Associates used a polygonal estimation method with a defined thickness, where TMC use MIK to define the continuity of the mineralisation within each zone.
- The footprint covered by the Henkle and Associates estimate is more extensive than the current estimates.

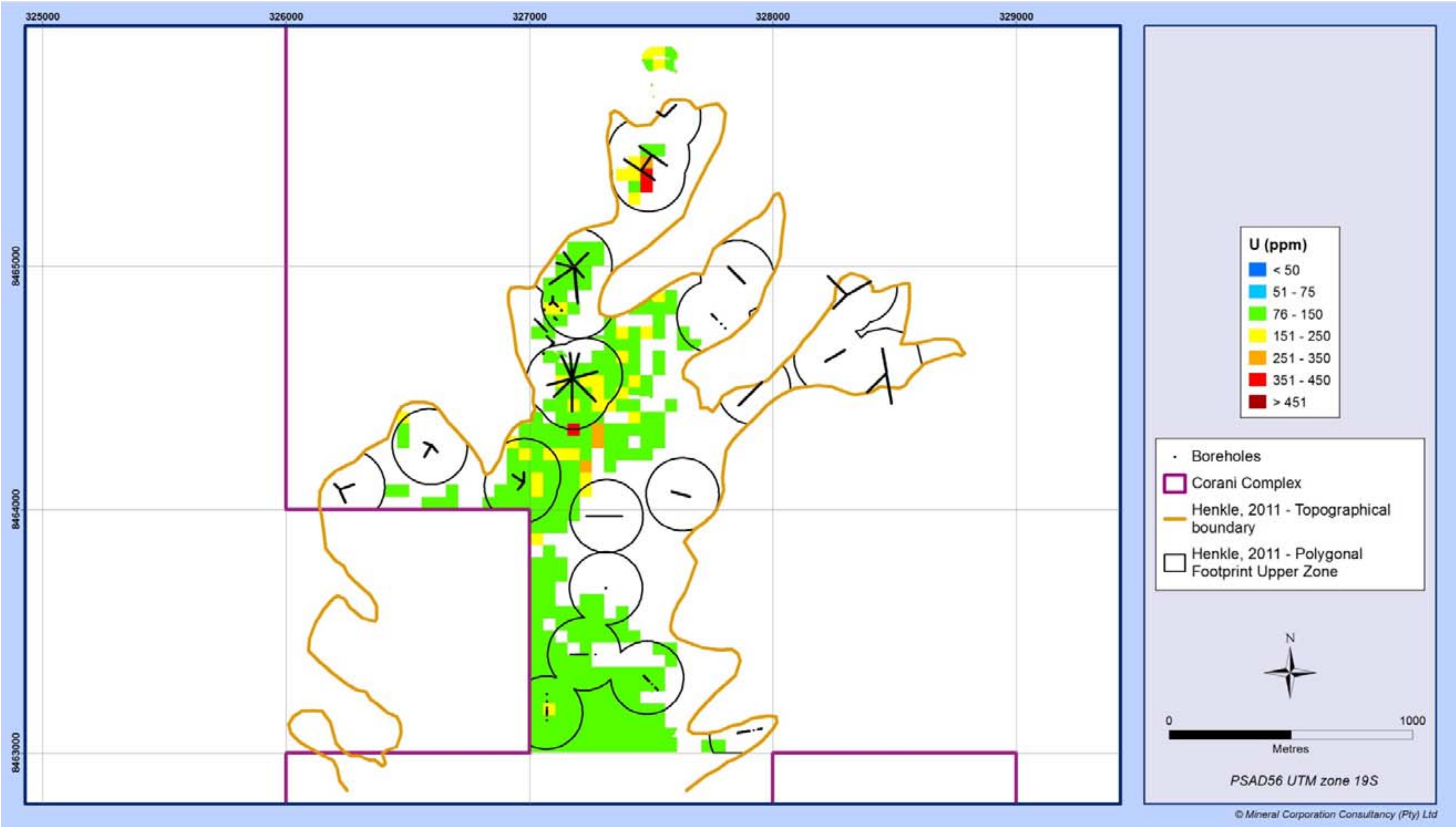


Figure 14-22: Reconciliation of the Estimation Footprint Employed by Henkle (2014) and TMC (this report)

14.7 GRADE TONNAGE RELATIONSHIPS

The sensitivity of the tonnage and grades to varying U ppm cut-offs is illustrated in Figure 14-23 and Figure 14-24. In both graphs, the primary Y-axis is U_3O_8 ppm and the secondary Y-axis is tonne.

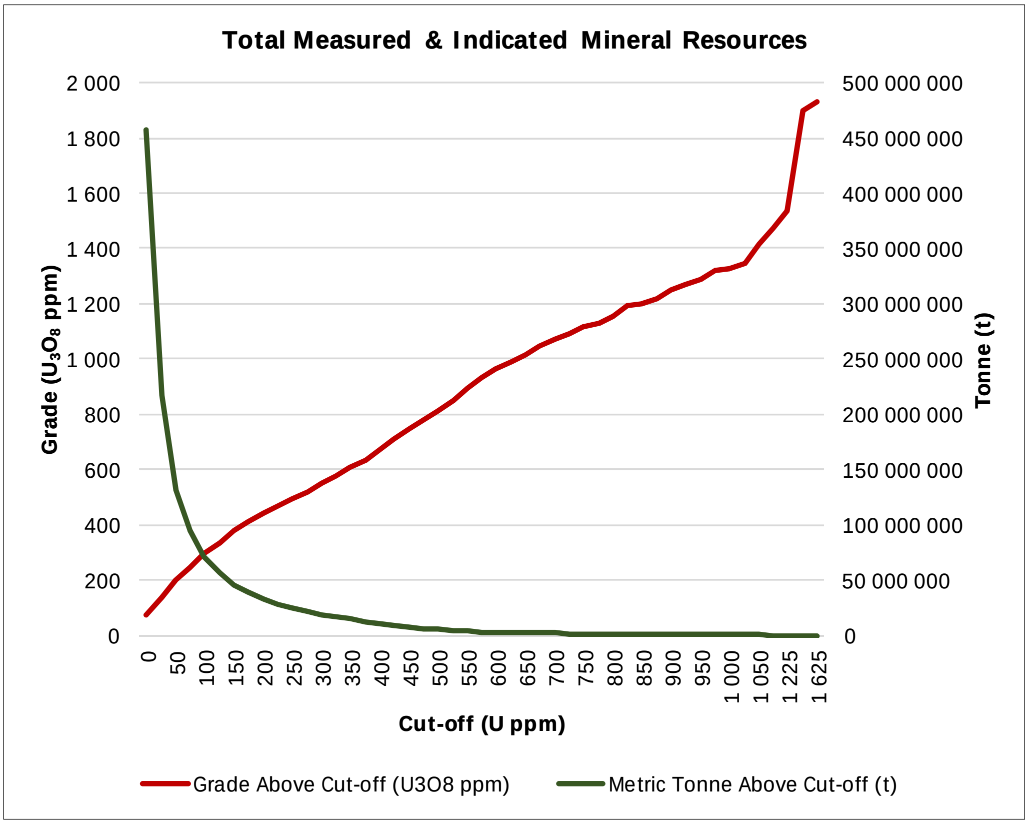


Figure 14-23: Grade-Tonnage Curves for Total Combined Indicated and Measured Mineral Resources

The graph in Figure 14-23 is provided to allow a reader to gauge the sensitivity of the Measured and Indicated Mineral Resources at higher U ppm cut-offs. If for example a 200 U ppm cut-off is chosen the Measured and Indicated Mineral Resources above this cut-off are 33.47 M tonne grading 445 ppm U_3O_8 containing 32.8 M lbs (14.9 M kg) U_3O_8.

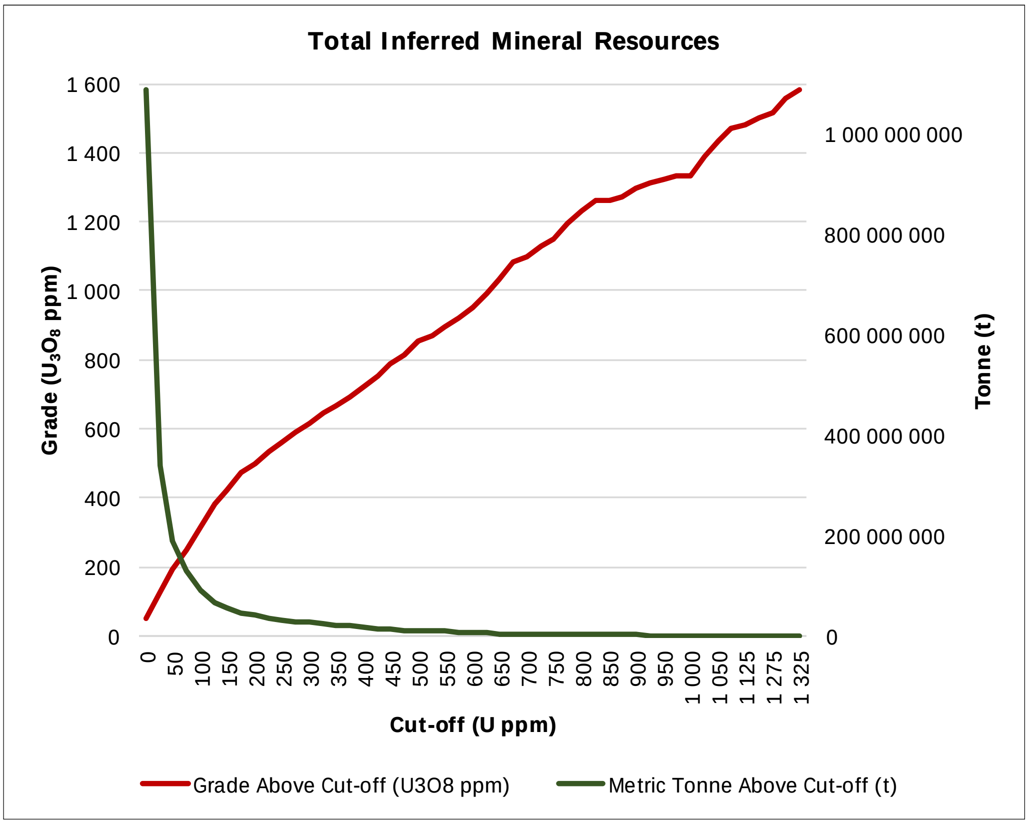


Figure 14-24: Grade-Tonnage Curves for Total Inferred Mineral Resources

The graph in Figure 14-24 is provided to allow a reader to gauge the sensitivity of the Inferred Mineral Resource at higher U ppm cut-offs. If for example a 200 U ppm cut-off is chosen the Inferred Mineral Resources above this cut-off are 41.62 M tonne grading 501 ppm U_3O_8 containing 54.9 M lbs (20.9 M kg) U_3O_8.

The detailed cut-off data for each Complex and classification of Mineral Resources is given in Appendix 10 of Young, 2015 (74).

14.8 SUMMARY RESOURCE STATEMENT

Table 14-24 summarises the Measured and Indicated Mineral Resources for the project, while Table 14-25 summarises the Inferred Mineral Resources, both at a 75 U ppm cut-off.

Table 14-24: Measured and Indicated Resource Statement

Complex	Deposit	Mineral Resource Category	Metric Units			Imperial Units		
			Tonne (Mt)	Grade (U ppm)	U_3O_8 Content (000s kg)	Ton (Mt)	U_3O_8 Content (Mlb)	U_3O_8 Grade (lb/ton)
Corachapi	Corachapi*	Measured	1.031	120	146	1.136	0.32	0.28
Kihitian	Chilcuno Chico	Indicated	34.840	218	8 972	38.405	19.78	0.52
Kihitian	Quebrada Blanca	Indicated	5.509	279	1 814	6.073	4.00	0.66
Kihitian	Tantamaco	Indicated	7.393	191	1 661	8.150	3.66	0.45
Isivilla	Isivilla	Indicated	4.568	296	1 597	5.035	3.52	0.70
Corani	Nueva Corani	Indicated	3.397	141	565	3.744	1.25	0.33
Corachapi	Corachapi*	Indicated	10.562	171	2 130	11.643	4.70	0.40
Colibri	Colibri II & III**	Indicated	27.885	203	6 675	30.738	14.72	0.48
Total Measured and Indicated			**95.185**	**210**	**23 559**	**104.924**	**51.94**	**0.50**

* Figures based on October 2010 Mineral Resource Estimates by The Mineral Corporation
** Figures based on September 2013 Mineral Resource Estimates by The Mineral Corporation
Minor discrepancies due to rounding may occur
There are currently no known risks that could materially affect potential development
Density 1.98t/m^3
Cut-off 75 U ppm
Ton is short ton

Table 14-25: Inferred Materials Resource Statement

Complex	Deposit	Mineral Resource Category	Metric Units			Imperial Units		
			Tonne (Mt)	Grade (U ppm)	U_3O_8 Content (000s kg)	Ton (Mt)	U_3O_8 Content (Mlb)	U_3O_8 Grade (lb/ton)
Kihitian	Chilcuno Chico	Inferred	30.995	294	10 751	34.166	23.70	0.69
Kihitian	Quebrada Blanca	Inferred	13.436	269	4 264	14.811	9.40	0.63
Kihitian	Tuturumani	Inferred	3.300	146	569	3.638	1.25	0.34
Kihitian	Tantamaco	Inferred	35.849	172	7 251	39.517	15.98	0.40
Isivilla	Isivilla	Inferred	7.396	295	2 573	8.153	5.67	0.70
Isivilla	Puncopata	Inferred	5.923	216	1 506	6.529	3.32	0.51
Isivilla	Calvario I	Inferred	1.679	268	531	1.851	1.17	0.63
Isivilla	Calvario Real	Inferred	1.146	90	122	1.264	0.27	0.21
Corani	Nueva Corani	Inferred	6.112	111	799	6.737	1.76	0.26
Corachapi	Corachapi*	Inferred	3.753	195	863	4.137	1.90	0.46
Colibri	Colibri II & III**	Inferred	9.453	167	1 862	10.420	4.10	0.39
Colibri	Tupuramani**	Inferred	10.976	125	1 618	12.099	3.57	0.29
		Total Inferred	**130.020**	**213**	**32 708**	**143.322**	**72.11**	**0.50**

* Figures based on October 2010 Mineral Resource Estimates by The Mineral Corporation
** Figures based on September 2013 Mineral Resource Estimates by The Mineral Corporation
Minor discrepancies due to rounding may occur
There are currently no known risks that could materially affect potential development
Density 1.98t/m^3
Cut-off 75 U ppm
Ton is short ton

SECTION 15 MINERAL RESERVE ESTIMATES

The project's Inferred Mineral Resources were used in the LOM plan together with the project's defined Indicated Mineral Resources. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resources would be converted into Mineral Reserves. Mineral Reserves can only be estimated as a result of an economic evaluation as part of a Pre-Feasibility Study or a Feasibility Study of a mineral project. Accordingly, at the present level of development, there are no Mineral Reserves at the project.

SECTION 16 MINING METHODS

16.1 GENERAL

The Colibri, Kihitian and Isivilla Complexes (Complex 2, 3, and 4 respectively) are planned to be recovered from open pit mining operations. Due to the orientation and topography of the Kihitian Complex, significant Mineral Resources are uneconomic to mine via the open pit method due to the excessive stripping costs and, as such, an underground operation is planned for mineralised material above cut-off.

This PEA has not been carried out to a level of detail equivalent to a pre-feasibility study or feasibility study. Consequently, no Mineral Reserve estimate has been prepared and any resources and conceptual production stated herein should not be considered representative of a Mineral Reserve. Inferred Mineral Resources were used in the LOM plan together with Indicated Mineral Resources. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resources will be converted into Mineral Reserves.

16.2 OPEN PIT OPTIMISATION

16.2.1 ECONOMIC LIMITS

The five main complex areas are shown in Figure 16-1.

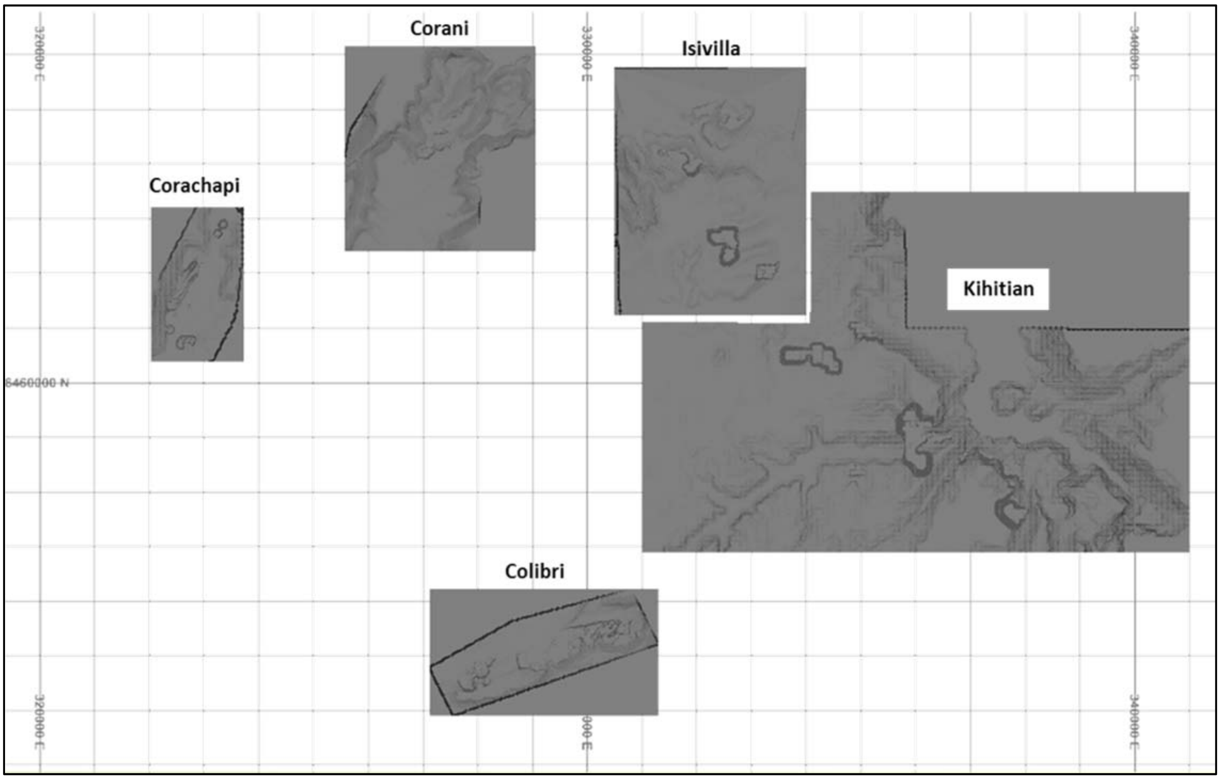


Figure 16-1: Project Area by Complex, 1 km Grid Squares

The complexes are located over a 17 km x 13 km area and the bulk of the deposits (Colibri, Kihitian and Isivilla Complexes) are located to the east over a smaller area. Based on a central processing model, the Colibri, Kihitian and Isivilla Complexes will be used for this report because a central processing area can be located no more than 5 km from the extents of these three Complexes.

16.2.2 MINERAL RESOURCE BLOCK MODEL

The 3D Mineral Resource block model, developed by TMC, was used as the basis for deriving the economic pit shell limits for the Macusani project. The block model dimensions were determined by complex and are stated in Table 16-1.

Table 16-1: Block Model Dimensions

Complex		Block Base Dimensions		
		X	Y	Z
2	Colibri	25	25	5
3	Kihitian*	50	50	3
4	Isivilla*	50	50	3

*Note: * 100m x 100m x 3m was used for extrapolated areas.*

16.2.3 INPUT PARAMETERS

Estimates have been made for metal price, mining dilution, off site costs, and royalties. Mining OPEX has been estimated from a contractor quote provided by Plateau Uranium and processing and general administration OPEX has been calculated by GBM based on processing throughput. Geotechnical parameters were based on a previous study conducted by WAI (74). Input parameters are detailed in Table 16-2.

Table 16-2: Open Pit Optimisation Input Parameters

Parameter	Unit	Value	Source
Product Price			
U_3O_8	USD/lb	50.00	Plateau Uranium
U_3O_8	USD/g	0.110	
Selling Cost*			
U_3O_8	USD/lb	1.50	
U_3O_8	USD/g	0.003	
Royalty	%	3.0	Estimated (1-12 %)
Operating Costs			
Waste Mining Cost	USD/t	2.40	Plateau Uranium
PEM Mining Cost	USD/t	2.40	Plateau Uranium
Processing Cost	USD/t PEM	4.97	GBM
G&A	USD/t PEM	1.18	GBM
Mining Factors			
Processing Recovery	%	88.00	GBM
Overall Pit Slope Angles	°	55	WAI Estimate
Mining Dilution	%	5.00	WAI Estimate
Mining Recovery	%	95.00	WAI Estimate
Discount Rate	%	8.00	

*Note: * Selling cost composed of royalty payments*

The Mineral Resource block model for the Macusani project was then used with NPV Scheduler™ (NPVS) open pit optimisation software to determine optimal mining shells, based on these input parameters. As noted in Section 16.1 the PEM included within the economic shell limits is formed of Indicated and Inferred Mineral Resources.

16.3 CUT-OFF GRADE

NPVS was used to determine cut-off grades. NPVS determines an economic cut-off grade, which is equivalent to a breakeven cut-off grade at the mill, where revenue is equal to processing cost. The cut-off grade should be used for comparative purposes.

Economic cut-off grade = (Processing Cost per unit of PEM x Mining Dilution) / ((Product Price – Selling Cost – Product Processing Cost) x Recovery)

Economic cut-off grade = 68.58 ppm U_3O_8

NPVS uses the basic economic cut-off grade as a test to determine which blocks can be considered as PEM and which can be considered as waste. From the PEM test, factors including mining cost, depth of block, proximity to high-grade areas and mining rate are considered to determine economic shell limits. This takes account of the time taken for mining and block location as to whether a block is economic in the mining sequence.

A basic mining cut-off grade shown for reference, considering mining losses, is shown below.

Open Pit Mining cut-off grade = 100.43 ppm U_3O_8

16.4 OPTIMISATION RESULTS

NPVS uses the Lerch-Grossmann method to produce an open pit shell yielding the maximum profit for a given metal price; the method was used to determine a series of optimised pit shells, known as Lerch-Grossmann shells or phases, up to 120 % of the economic product price. The results were analysed with pit shells chosen as the basis for ultimate pit limits and preliminary phase selection.

The results of the pit optimisation evaluation on the complexes for varying price factor values are summarised in Figure 16-2, Figure 16-3, and Figure 16-4. Note that NPV in the optimisation summary does not take into account CAPEX as it is used only as a guide in shell selection and determination of the mining pit shapes. The actual NPV of the project is summarised in the economics section of this report.

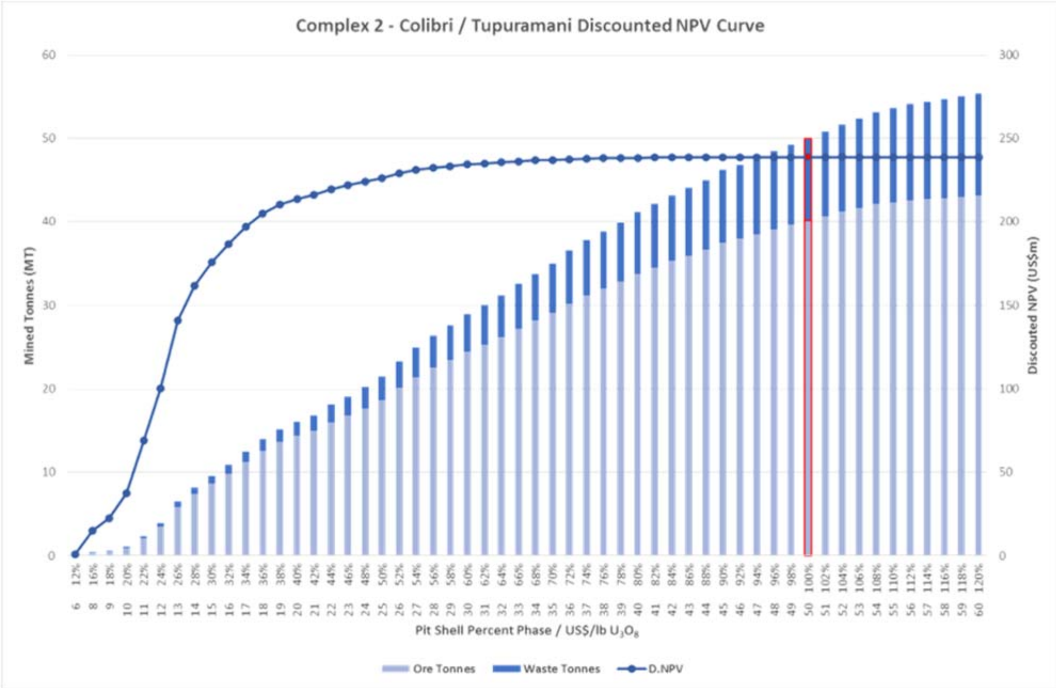


Figure 16-2: Colibri Complex Open Pit Optimisation Cumulative Results

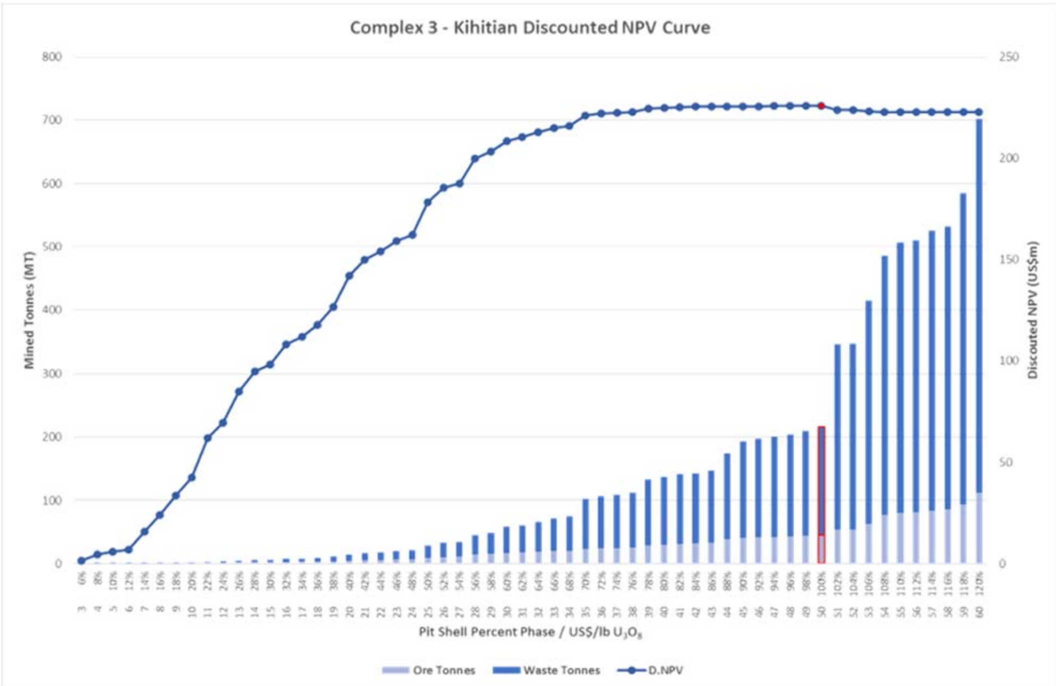


Figure 16-3: Kihitian Complex Open Pit Optimisation Cumulative Results

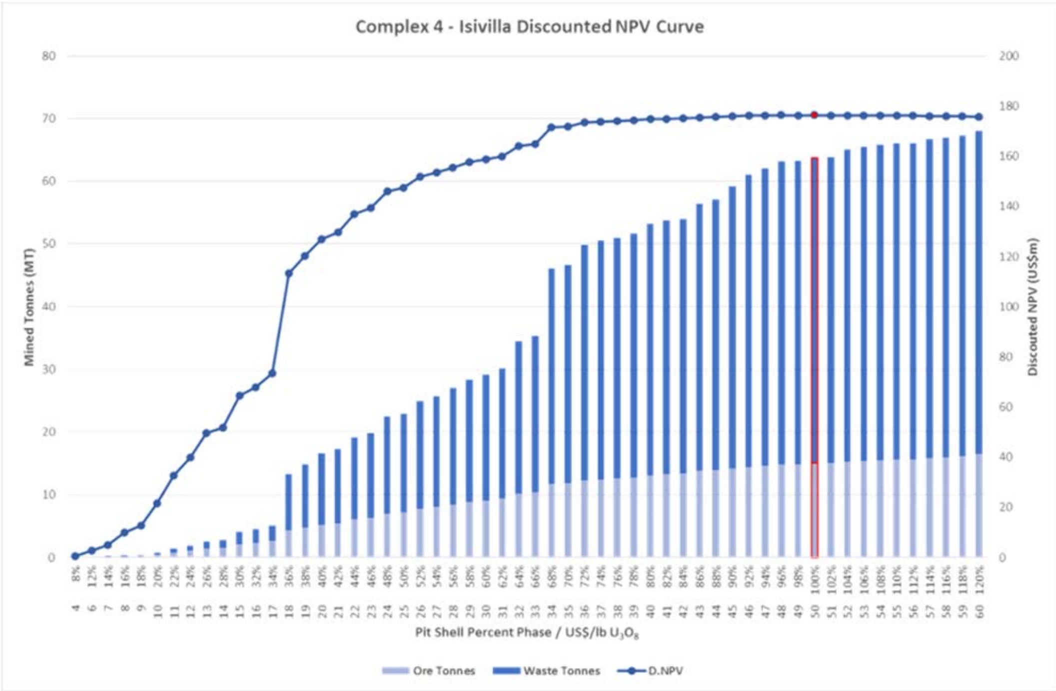


Figure 16-4: Isivilla Complex Open Pit Optimisation Cumulative Results

16.4.1 COLIBRI OPTIMISATION RESULTS

The Colibri Complex shows a typical ‘idealised’ profile, where incremental increases in mineralised rock and waste tonnages occur with increasing pit shell phases. The majority of NPV is reached roughly from the 66 % pit shell phase (33 USD/lb U_3O_8 shell). As the profile of the Lerch-Grossmann phases shows no large step increase in waste tonnage, there is no effect of incremental strip ratio that affects the NPV, therefore the 100 % pit shell was selected to maximise produced metal for optimum NPV.

16.4.2 KIHITIAN OPTIMISATION RESULTS

The Kihitian deposit shows minor step increases in strip ratio, up to the 100 % pit shell phase. The increases are minor which have little negative effect on the NPV curve. Beyond the 100 % pit shell, large increases in the strip ratio are present which have a negative effect on the NPV. The 100 % pit shell phase is the largest phase before negative effects of strip ratio are seen, therefore to maximise produced metal the 100 % pit shell phase is selected, optimising NPV, without compromising for strip ratios.

16.4.3 ISIVILLA OPTIMISATION RESULTS

The Isivilla deposit shows step increases in strip ratio at pit shell phases 36 %, 64 %, and 68 %, but shows marginal increases beyond. Because of the steady increase, the 100 % pit shell phase will be selected, allowing for maximum produced metal.

Based on the analysis of the shells and the preliminary schedule, the 100 % pit shell was chosen for all complexes for further phasing and project scheduling. The LOM plan PEM included both Indicated and Inferred Mineral Resources, of which Indicated Mineral Resources represent 44 % (47.7 Mt) of the material planned to be processed (PEM). The shell contents are shown in Table 16-3 by complex. Table 16-4 summarises the LOM plan, while Table 16-5 shows LOM by Mineral Resource classification.

Table 16-3: In-Situ Complex Mineral Resources to be Extracted

Complex		Tonnage	Grade U_3O_8 (ppm)	Waste (Mt)	Strip Ratio (t_W:t_{PEM})	U_3O_8 Content (t)	U_3O_8 Content (M lbs)
2	Colibri	40.1	232	9.90	0.25	7 760	17.1
3	Kihitian	45.7	309	170	3.72	11 800	26.0
3	Kihitian (UG)	8.23	475	0.00	-	3 440	7.58
4	Isivilla	15.0	373	48.8	3.25	4 670	10.3

Note: Figures have been rounded to 3SF.

Table 16-4: Complex Mineral Resources to be Extracted in LOM Plan

Parameter	Unit	Value
Mine Production Life	a	10
Diluted Process Feed Material	Mt	109
Diluted U_3O_8 grade (mill head grade)	ppm	287
Recovered U_3O_8	t	27 500
Recovered U_3O_8	M lbs	60.6
Waste	Mt	224
Total Material	Mt	332
Strip Ratio	t_W:t_{PEM}	2.05

Note: Figures have been rounded to 3SF.

Table 16-5: Mineral Resources to be Extracted in LOM Plan by Classification

Resource Category	Tonnage (Mt)	U_3O_8 (ppm)	Contained U_3O_8 (t)	Contained U_3O_8 (M lbs)
Indicated	47.7	279	13 300	29.3
Inferred	61.2	293	17 900	39.5

Note: Figures have been rounded to 3SF.

16.4.4 DILUTION & LOSS FACTORS

The dilution and loss factors that have been applied to the Mineral Resources contained within the open pit shells have been based on WAI's experience in open pit mining in similar operations worldwide.

The dilution and loss factors applied are a simplification appropriate with this level of study, leading to an approximate 5 % global loss estimate for the Mineral Resource and approximately 5 % average dilution.

16.5 OPEN PIT MINE DESIGN

Mine planning for the Macusani project was conducted using Datamine NPVS. The base 3D block model was continued for mine design, with pushback selection and production scheduling undertaken using NPVS software.

The selected shells from the optimisation were used for final pit dimensions, and then NPVS was used to determine optimal pushbacks. The 100 % pit shell phase was used for the mining shape limit for each complex as shown in Figure 16-5, Figure 16-6, and Figure 16-7. The sections show a plan and section view of the ultimate pit shape of each complex, with topography in grey and pit outlines shown in red.

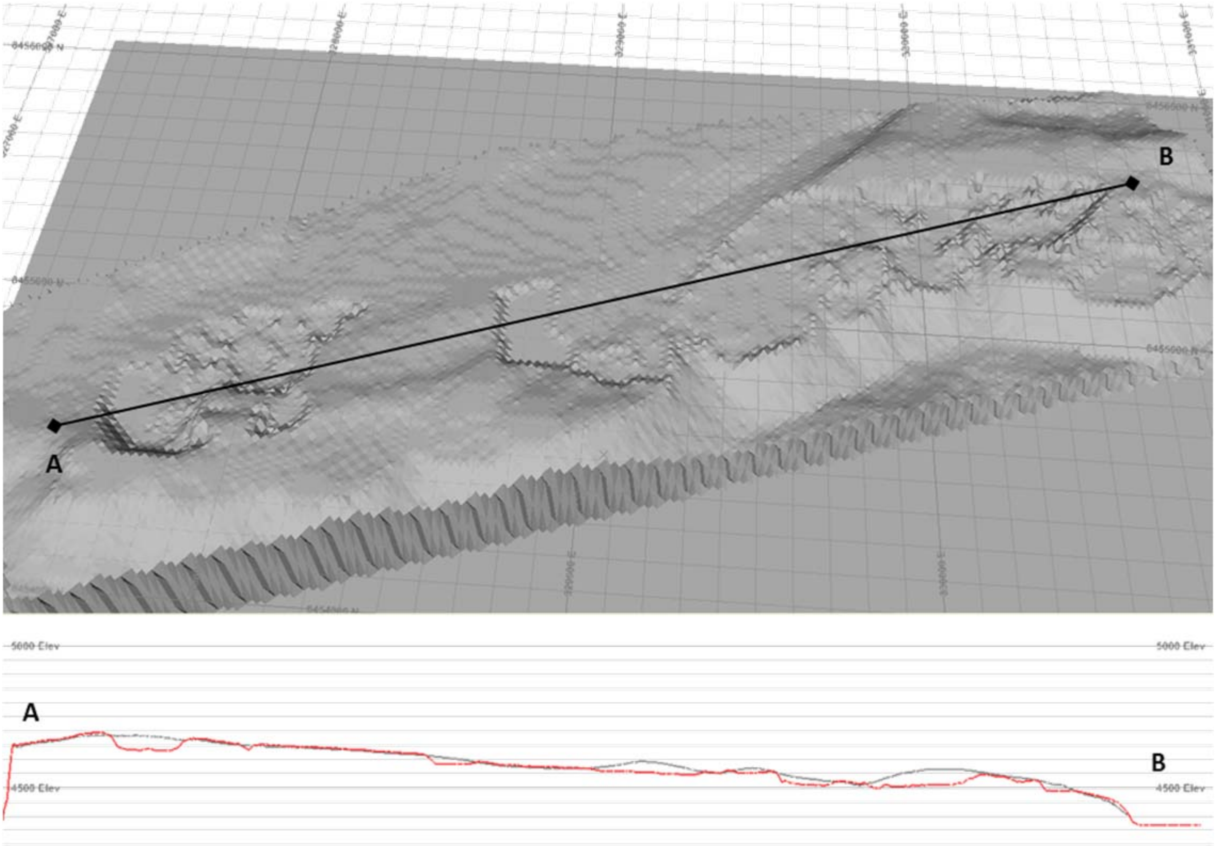


Figure 16-5: Colibri Complex – Isometric View and Cross Section

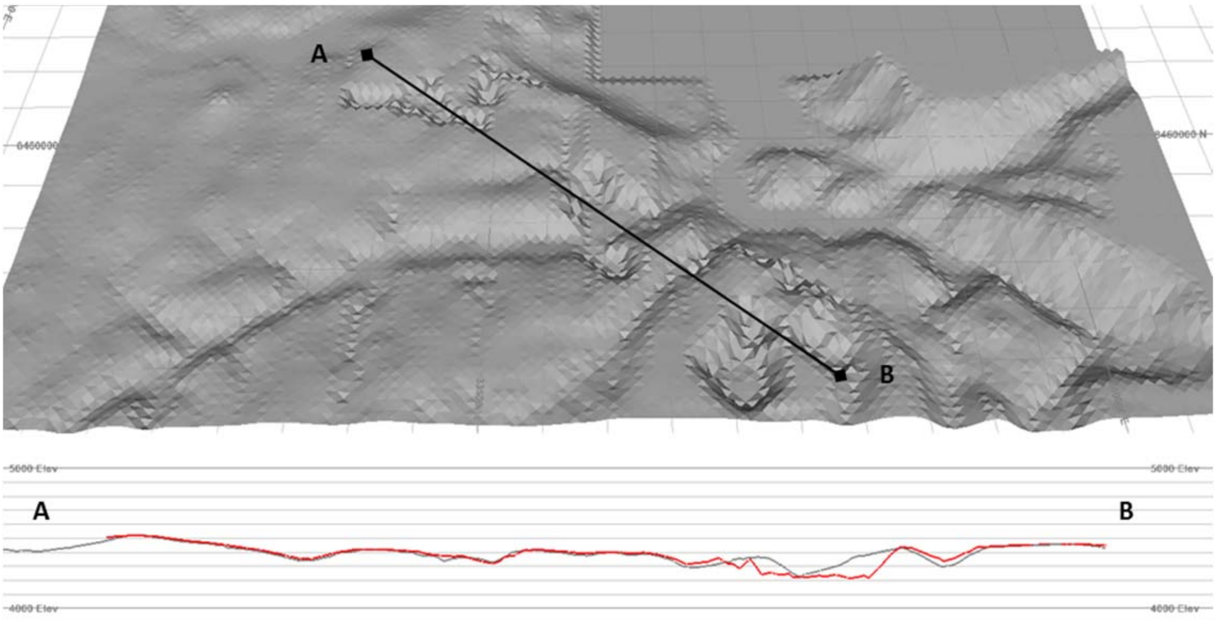


Figure 16-6: Kihitian Complex – Isometric View and Cross Section

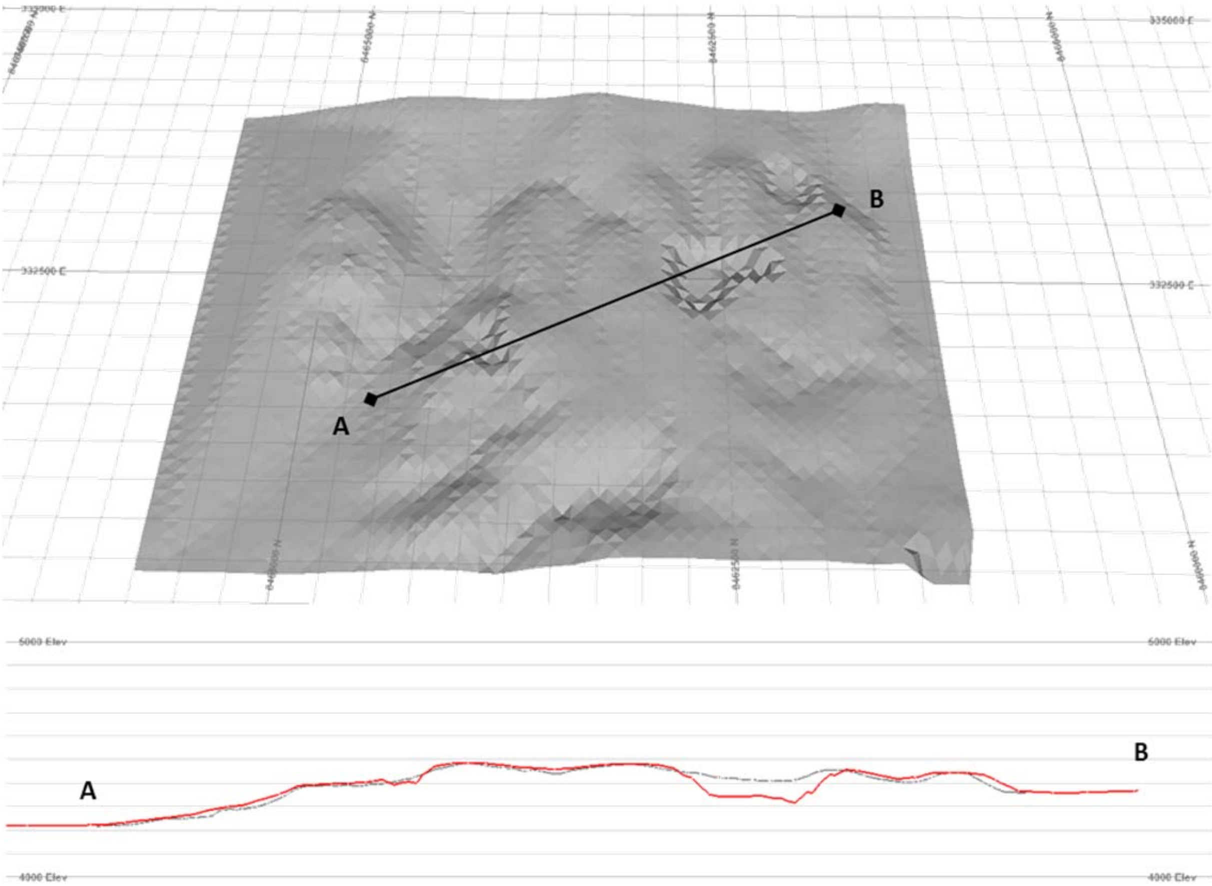


Figure 16-7: Isivilla Complex – Isometric View and Cross Section

16.6 MINE SEQUENCING

NPVS uses the optimised pit extraction sequence and the Lerch-Grossman pit phases to determine practical pushback shapes. The analyses were completed to ensure a balance between PEM grade, stripping ratio, and location. A minimum mining width of at least 30 m was specified in the pushback generation.

The pushback phases were based on higher-grade material being mined first where possible. During active mining and processing of the PEM, the waste would be placed into a waste rock facility adjacent to the final shell limits. All mineralised material would be hauled to a central process facility.

The pushback tonnages, grades, and contained metal for each complex are shown in Table 16-6, Table 16-7 and Table 16-8.

Table 16-6: Colibri Complex Pushback Tonnages and Grade

Pushback	Total Diluted Mill Feed				Waste (Mt)	Strip Ratio (t_W:t_{PEM})	Total (Mt)
	Tonnage (Mt)	Grade U_3O_8 (ppm)	U_3O_8 Content (t)	U_3O_8 Content (M lbs)			
1	10.4	291	3 020	6.66	0.76	0.07	11.2
2	8.39	245	2 060	4.54	1.40	0.17	9.78
3	11.9	184	2 190	4.83	4.11	0.35	16.0
4	7.51	159	1 200	2.64	2.91	0.39	10.4
5	1.82	188	342	0.76	0.74	0.41	2.57
TOTAL	**40.0**	**220**	**8 810**	**19.4**	**9.92**	**0.25**	**50.0**

Note: Figures have been rounded to 3SF.

Table 16-7: Kihitian Complex Pushback Tonnages and Grade

Pushback	Total Diluted Mill Feed				Waste (Mt)	Strip Ratio (t_W:t_{PEM})	Total (Mt)
	Tonnage (Mt)	Grade U_3O_8 (ppm)	U_3O_8 Content (t)	U_3O_8 Content (M lbs)			
1	7.25	427	3 100	6.83	50.8	7.00	58.0
2	4.89	244	1 200	2.64	7.39	1.51	12.3
3	9.58	229	2 200	4.84	20.1	2.10	29.6
4	7.65	358	2 740	6.04	26.6	3.48	34.3
5	7.10	205	1 460	3.21	23.7	3.34	30.8
6	9.18	295	2 710	5.98	41.6	4.53	50.7
TOTAL	**45.7**	**293**	**13 400**	**29.5**	**170**	**3.73**	**216**

Note: Figures have been rounded to 3SF.

Table 16-8: Isivilla Complex Pushback Tonnages and Grade

Pushback	Total Diluted Mill Feed				Waste (Mt)	Strip Ratio (t_W:t_{PEM})	Total (Mt)
	Tonnage (Mt)	Grade U_3O_8 (ppm)	U_3O_8 Content (t)	U_3O_8 Content (M lbs)			
1	3.06	340	1 040	2.29	3.85	1.26	6.91
2	3.31	321	1 060	2.34	6.75	2.04	10.1
3	2.60	502	1 310	2.88	14.4	5.52	17.0
4	3.59	351	1 260	2.78	22.0	6.12	25.6
5	1.59	309	493	1.09	1.62	1.02	3.21
6	0.83	183	151	0.33	0.24	0.29	1.06
TOTAL	**15.0**	**355**	**5 310**	**11.7**	**48.8**	**3.26**	**63.8**

Note: Figures have been rounded to 3SF.

16.7 OPEN PIT MINE OPERATIONS

The open pit mining activities for the Macusani complexes were assumed to be undertaken by a contractor operated fleet as the basis for this PEA. The average unit mining costs used in the project economics was 2.40 USD/t of rock mined. Plateau Uranium has sourced the cost estimate from contractors operating in the region on similar sized operations. The open pit mining cost is provided as all-in-inclusive and covers variations in haulage profiles and equipment selection.

Labour rates have been based on local information from Plateau Uranium and are shown in Table 21-10.

16.7.1 EQUIPMENT

The contractor did not provide equipment lists, so WAI has used the generated mining schedule to produce an indicative fleet list for the operations. The equipment estimate was built from first principles and is based on experience of similar sized open pit operations and local conditions. The proposed mining schedule detailed was used for the equipment estimate, along with deposit and geometry constraints. Table 16-9 summarises the assumed major open pit equipment requirements for the project. All equipment is diesel operated.

Table 16-9: Major Open Pit Equipment Estimate

Equipment Type	No. of Units (max)
150 mm dia. DTH, Crawler Drill	3
15 m^3 Shovel	4
13.8 m^3 Wheel Loader	2
180 t Haul Truck	23
16M Class Grader	1
D10 Class Dozer	3
Water Truck	1

16.7.1.1 UNIT OPERATIONS

Indicatively, the 150 mm diameter blast hole drill rigs are planned to perform the production drilling in the operations, for both mineralised and waste rock. The main loading and haulage fleet is estimated to consist of 180 t haul trucks, loaded primarily with the diesel powered 15 m^3 shovels or the 13.8m^3 wheel loader depending on pit conditions and equipment availability.

As conditions dictate, the D10 class dozers are planned to rip and push material to the excavators, conduct bench clean ups, and maintain the waste dumps and stockpiles. Additional equipment listed in Table 16-9 is planned to be used to maintain and build access roads and to meet various site facility requirements.

Mechanical availability was based by equipment type. For drills, hydraulic excavators, and haul trucks 90 % was used; for wheel loaders, dozers, and graders 88 % was used; and all others used 85 %. Utilisation was set to 87 % which is equivalent to good conditions/operators; while 50 % was used for mobile lights and light vehicles and 12 % for the crane.

16.7.1.2 OPEN PIT OPERATION CONSUMABLES

Based on the anticipated fleet and the preliminary schedule consumables have been estimated as shown in Table 16-10.

Table 16-10: Open Pit Consumable Estimate

Item	Unit	Total (LOM)	Year									
			1	2	3	4	5	6	7	8	9	10
Diesel	kl	217 000	28 900	26 100	31 600	32 000	26 500	24 800	13 000	11 900	11 400	10 800
Lubricants	kl	838	101	94.2	109	110	96.2	91.5	62.4	59.6	58.2	56.6
Explosive	t	18 900	3 100	2 800	2 830	2 960	2 060	2 370	730	720	716	670

16.8 OPEN PIT PERSONNEL TABLE

The maximum mine workforce to be employed is estimated at 344 persons, with 178 on site at any one time. The crew schedule is a rotation schedule, with four shifts working 12 hour shifts, on a dayshift/nightshift rotation, with two crews off at a time. Maintenance will use the same schedule with day and night shifts to support mining operations. Management and technical personnel may be on a varied schedule but will primarily follow the same rotation as operations, working only dayshift, unless required by operations. Table 16-11 summarises the expected open pit personnel requirements.

Table 16-11: Open Pit Summary Personnel Table

Area of Operation	Average (no.)	Max (no.)	Max at any time (no.)
Management	9	9	6
Operations	133	174	87
Technical Services	33	33	19
Maintenance	79	98	51
H&S / Welfare / Warehouse	30	30	15
TOTAL	**284**	**344**	**178**

16.9 UNDERGROUND MINE DESIGN

The Mineral Resources available for underground mining that forms the project LOM plan are contained within a portion of the Mineral Resource block model that is excluded from current economic open pit limits, but which is judged potentially mineable by underground methods. These Mineral Resources contain a significant quantity of Inferred material. For conceptual production, Inferred material has been included within the design and includes estimates of mining dilution and mining recovery.

The proposed underground mine design supports the extraction of 2 700 tpd of PEM through room and pillar mining utilising continuous miners. Paste back fill has not been considered for the design, instead in-situ pillars are planned to be used for support. Conveyor systems have been incorporated for the majority of mining activities.

Material handling from the underground workings to the surface is based on conveyor haulage through the workings, with PEM being hauled directly from the underground to the ROM stockpile through the underground access. Drill and blast equipment has been included (Jumbo / Charge Wagon / Wheel Loader) to allow for the establishment of areas via drill and blast to allow continuous mining.

Mining ventilation will be accomplished by two dedicated exhaust shafts to surface, each equipped with modern fans that will draw fresh air in at the main entrance and through the workings of the mine before being exhausted through the dedicated return airways to minimise exposure to radiation. The two exhaust shafts are located within the mining areas, in the northern and southern extents of the mine layout.

16.9.1 METHOD SELECTION / MINERAL RESOURCE BLOCK MODEL

Mineral Resources which are considered for the underground are shown below. The Mineral Resources are blocks, which are excluded from the open pit design. The bulk of underground Mineral Resources are held within the Kihitian Complex, and specifically at the Chilanco Chico deposit as shown in Figure 16-8.

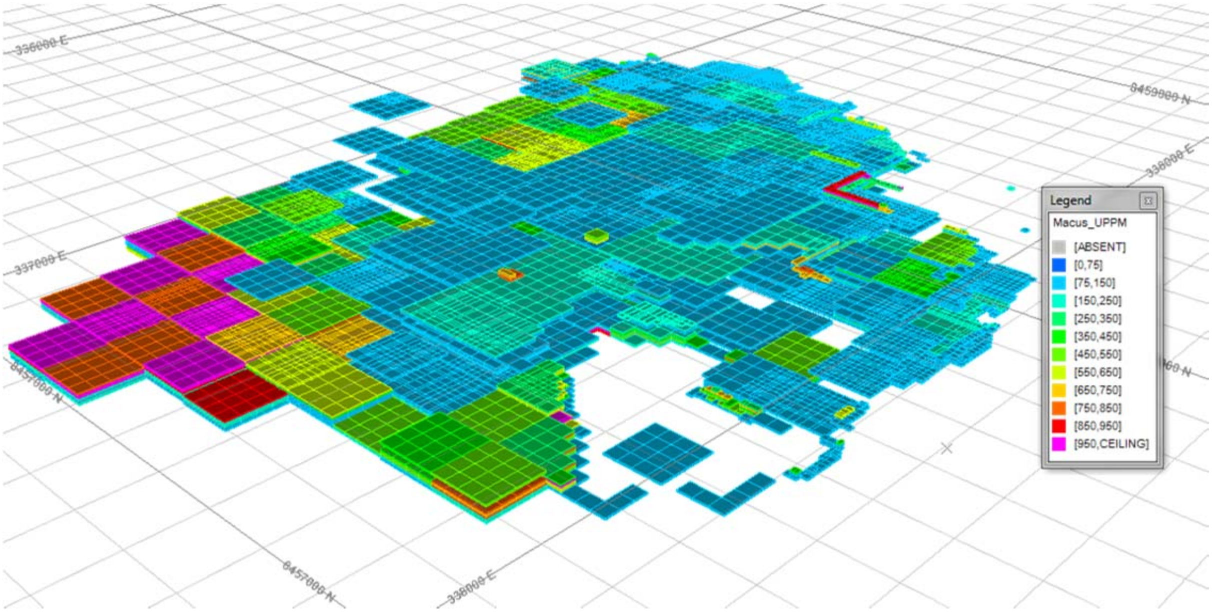


Figure 16-8: Isometric View of Underground Resource Blocks

(Filtered U_3O_8 >100 ppm, Chilanco Chico deposit, PEM blocks cut with Optimised Open Pit Shell)

Mineral Resources at Chilanco Chico are ideally suited to mining via horizontal mining methods. Mineralisation generally varies from 6.0 m and up to 24 m in thickness, averaging approximately 10 m, although low-grade mineralisation extends for some further distance, this material consists of uneconomic grades. The strong continuity and thickness of the mineralisation zones combined with weak rock strength at Chilanco Chico are amenable to horizontal room and pillar mining by continuous miner. This method was selected to benefit from the productivity and low mining costs associated with the method. The method allows for multiple working areas and a fast mining cycle. Conventional drill and blast mining methods were considered, however horizontal room and pillar mining was retained in view of the following considerations:

- Owing to the weak rock strength, bulk-mining methods were ruled out due to high levels of dilution and associated sizing of regional pillars that would be required;
- The mineralisation often occurs with high-grade areas contained within specific horizontal bands, thus requiring some selectivity within the mining method; and
- Highly selective methods such as mechanised overhand cut and fill are less productive than a continuous miner and would incur a higher unit cost of extraction. An increase in direct mining costs in the order of 15 - 20 USD/t mined to support highly selective mining would significantly decrease the operating margin.

16.9.2 DILUTION & LOSS

Estimates for mining dilution and loss were based on WAI's experience and quoted factors based on site experience or published work. Given the mining method is non blasting, blast damage is not going to be a contributing factor, however block fallout from structural features is likely to occur. Mining losses are anticipated to occur from pillars required for stability, and minor material transport losses. For reporting, mining dilution is taken at 5 % and mining loss at 5 %.

16.9.3 STOPE DESIGN RESULTS

WAI used the Mineable Shape Optimiser module from the Datamine 5D Planner mine planning software package to produce conceptual stope shapes. The cut-off grade and design criteria are described below.

16.9.3.1 CUT-OFF GRADE

Table 16-2 was used for product, selling, and processing costs. The previous study conducted by WAI (74) produced an underground mining operating cost of 8.73 USD/t mined, which was used in the following formula:

Economic cut-off grade =	(Mining Cost x Processing Cost x G&A x Transport Cost) x Dilution
	((Product Price x Royalties% – Product Refining Cost) – Loss) x Recovery
Economic cut-off grade =	174.79 ppm U_3O_8
Payability Margin =	10 % > Stope cut-off grade = 192.27 ppm U_3O_8 > Rounded = 200 ppm U_3O_8

Stopes have been designed to match the mining method selected, based on using a continuous miner, the maximum cutting range in a single pass is generally limited to 5 m. Continuous miners are commonly planned to mine multiple passes; however, the impact of the weak rock strength has limited the mine design to two passes or a maximum mining height of 10 m. Stope span sizing is from Section 16.13. Table 16-12 outlines the stope design criteria used.

Table 16-12: Stope Design Criteria

Parameter	Unit	Value
Stope Cut-off Grade	ppm U_3O_8	200
Level Spacing	m	10
Stope Span	m	35
Minimum Mining Width	m	4.5
Maximum Mining Width	m	10
Minimum Waste Pillar Width	m	>100
Minimum Footwall Dip	°	90
Minimum Hangingwall Dip	°	90

The mine design has been set up to develop the highest-grade stope shape up to a maximum design height of 10 m. This is has been achieved by using a large waste pillar in a vertical orientation to limit the stope development to a single shape within the PEM area.

Stope shapes were filtered to exclude peripheral stope shapes that required excessive development or in isolated areas. The final LOM plan is comprised of approximately 792 individual stoping units, illustrated in Figure 16-9.

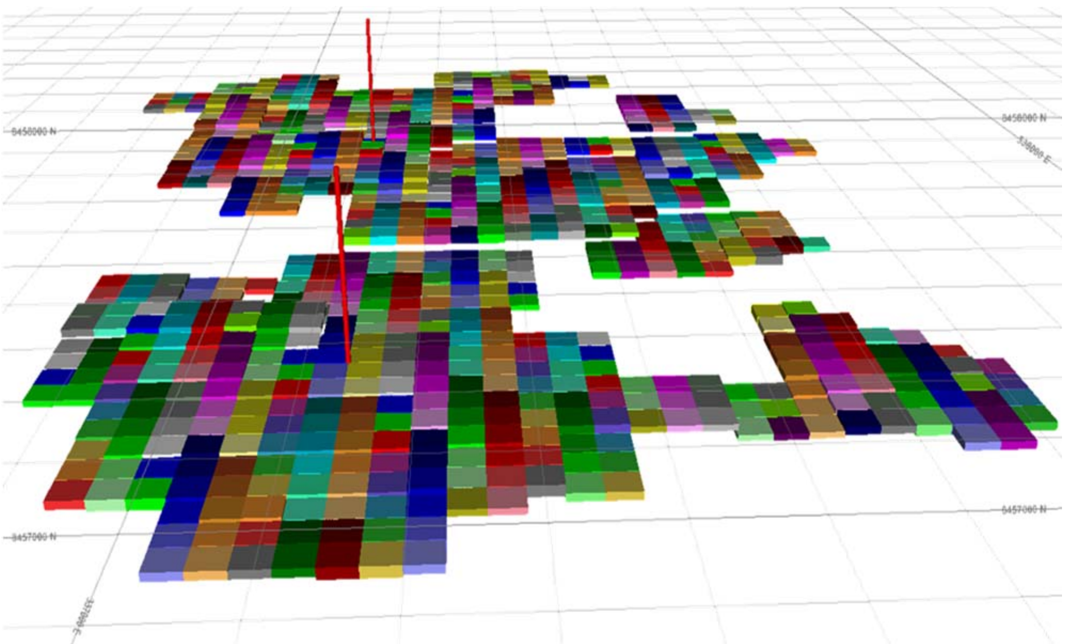


Figure 16-9: Isometric View of Mineable Stope Shapes

16.9.4 SUMMARY OF MINEABLE STOPE SHAPES

This PEA is based on Mineral Resources, not Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. The summary of Mineral Resources available for underground mining shown in Table 16-13 are reported based on the conceptual mine design. The figures are reported inclusive of dilution and recovery estimated as discussed in the report. Figure 16-10 presents the spatial distribution of these Mineral Resources by category as applied to each of the individual stopes in the LOM plan.

Table 16-13: Summary of Mineral Resources Available for Conceptual Underground Mining

Resource Category	PEM (Mt)	Grade (ppm U_3O_8)	Contained Metal	
			Tonnes (U_3O_8)	M lbs (U_3O_8)
Measured	-	-	-	-
Indicated	4.06	438	1 780	3.92
Measured and Indicated	4.06	438	1 780	3.92
Inferred	4.18	511	2 140	4.72

Note: Figures have been rounded to 3SF.

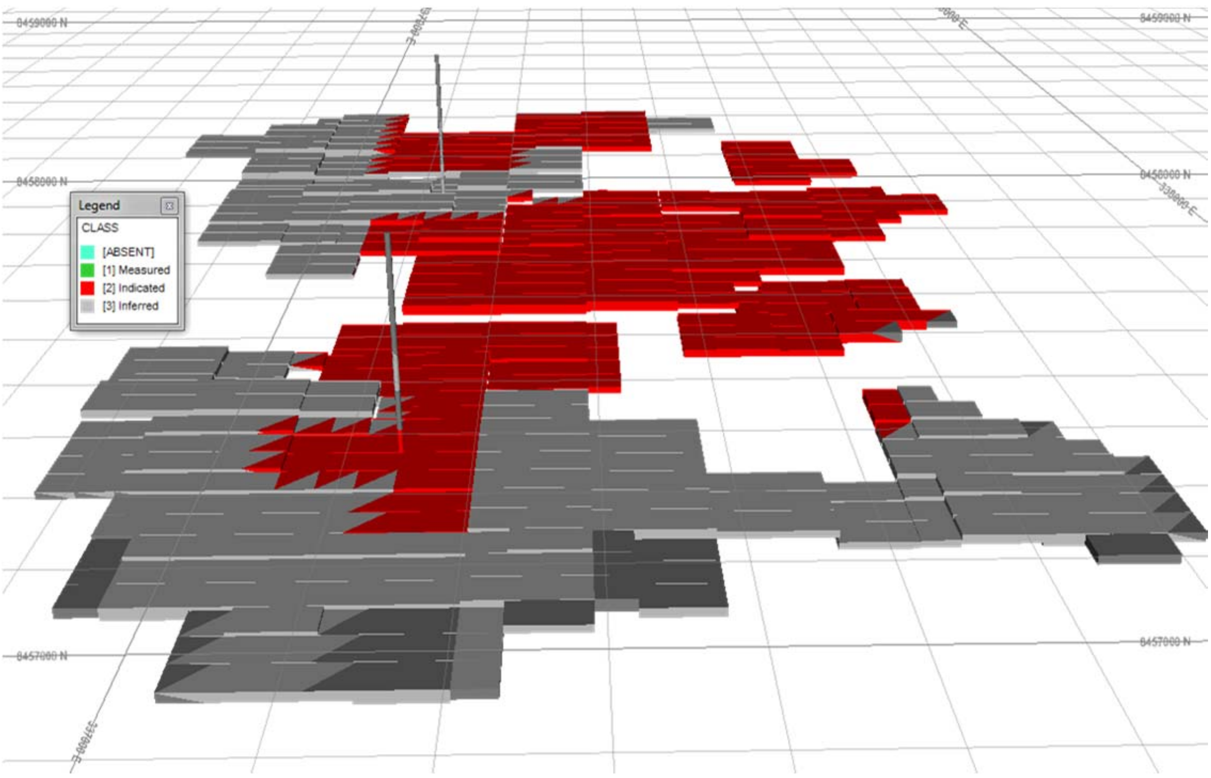


Figure 16-10: Isometric View of Mineable Stope Shapes shown by Mineral Resource Category

16.9.5 MOBILE EQUIPMENT

Current underground mining is planned to use modern continuous miners assisted by conveyors for material handling. In addition to a continuous mining unit, an underground drilling rig has been included for ground support and minor drill and blast duties where required. A wheel loader has also been included for clean-ups and associated works if required. Equipment utilisation and availability was modelled at 90 % respectively.

The breakdown of fleet and equipment to support the LOM underground mining plan is shown in Table 16-14.

Table 16-14: Mobile Equipment

Machine	Units (Max)
Continuous Miner	1
Bridge Conveyor	1
Belt Storage Unit	1
Conveyor	3
Wheel Loader	1
Jumbo Drill Rig	1
Charge Wagon	1
Service Truck	1
Light Vehicles	5

16.9.5.1 UNDERGROUND OPERATION CONSUMABLES

Based on the anticipated fleet and the preliminary schedule, the following unit consumables have been estimated in Table 16-15.

Table 16-15: Underground Operation Consumable Estimate

Item	Unit	Total (LOM)	Year									
			1	2	3	4	5	6	7	8	9	10
Diesel	kl	**51.2**	2.73	5.92	5.53	6.11	6.02	6.13	6.86	6.30	5.66	51.2
Lubricants	kl	**782**	46.7	74.8	87.5	87.7	87.8	101	101	101	95.7	782
Explosive	t	**286**	31.8	31.8	31.8	31.8	31.8	31.8	31.8	31.8	31.8	286
Power	MW	**(av.) 0.81**	0.85	0.65	0.77	0.77	0.77	0.90	0.90	0.90	0.83	0.81

16.10 UNDERGROUND PERSONNEL TABLE

The maximum underground mine workforce to be employed is estimated at 141 persons, with 74 on-site at any one time. The crew schedule is a rotation schedule, with four shifts, working 12 hour shifts, on a dayshift/nightshift rotation, with the two crews off at one time. Maintenance will use the same schedule with day and night shifts to support mining operations. Management, technical, and some service/conveyor personnel will work on a varied schedule but will primarily follow the same rotation as operations, working only dayshift, unless required by operations. Table 16-16 summarises the estimated underground personnel requirements.

Table 16-16: Underground Summary Personnel Table

Area of Operation	Average (no.)	Max (no.)	Max at any time (no.)
Management	9	9	6
Operations	37	40	20
Technical Services	30	30	15
Maintenance	65	65	33
TOTAL	**140**	**141**	**74**

16.11 MINE SCHEDULE

The stope sequence is shown in Figure 16-11 by year of extraction. Mining development starts from the southern end of the deposit, with access made from the planned open pit. Underground mining then develops to the west to intersect the southern ventilation shaft. The mine will then complete the southern area before developing north, to intersect the secondary ventilation shaft, before opening final extents.

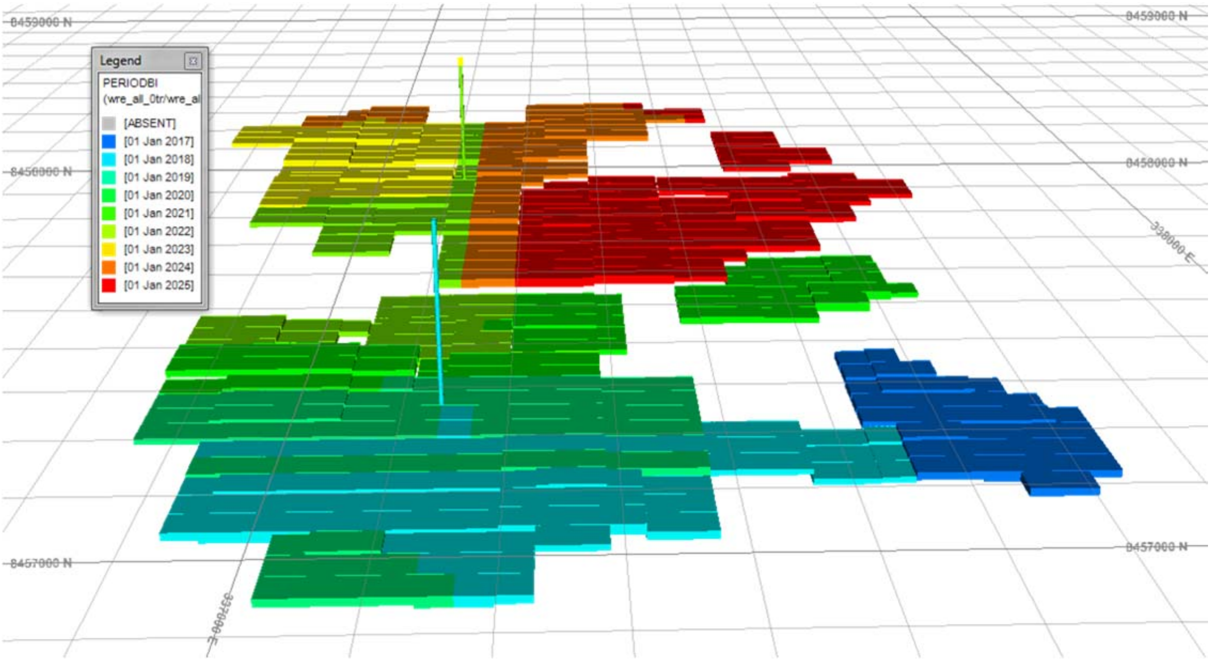


Figure 16-11: Isometric View of Mineable Stope Shapes by Sequence

16.12 UNDERGROUND INFRASTRUCTURE

16.12.1 MAIN LEVELS AND MINING BLOCKS

The stopes have been optimised to take the highest-grade material up to a maximum mineable height of 10 m, which is equivalent to mining in two passes, top down. Scheduling has modelled panel development starting from the open pit and developing into the Mineral Resource, primarily targeting an exhaust raise, before developing to the extents of the design.

16.12.2 VENTILATION

16.12.2.1 OVERVIEW

Ventilation requirements were based on recommended air transit time guidance for uranium mines by Gherghel & Souza (2008) (75), who recommend 10-15 minutes of residence time to limit to 10-20 % of the theoretical yield of radon progeny products.

The air requirement was based on the daily mine production and associated equipment, taking into account the target transit time. The LOM primary ventilation circuits for the underground mine comprises of the following:

- Two intake portals, southern opened initially, northern developed in year 4; and

- Two exhaust raises, south opened in year 2, north developed in year 5. Each would be fitted with a primary fan located at the top of the raise.

Typically, fresh air will flow from the portals through the workings and exhaust via the exhaust shafts, located centrally in the two main mining areas. Suitably sized development fans and ducting will direct air to the workings areas. The exhaust air ultimately reports to an exhaust raise that is directly connected to surface. Figure 16-12 shows the main location of intake and exhaust raises.

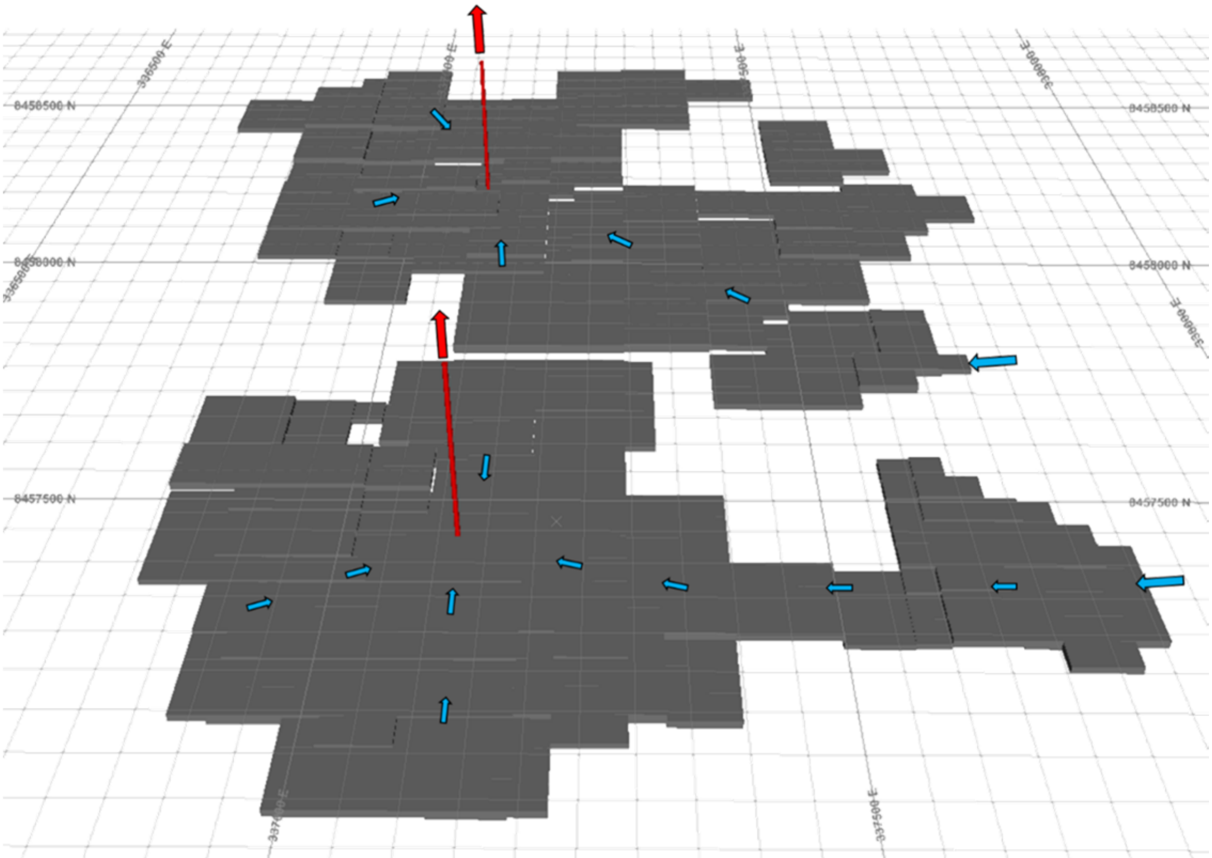


Figure 16-12: Isometric View of LOM Ventilation Network

16.12.2.2 DIESEL ENGINE EXHAUST EMISSION DILUTION AIRFLOW REQUIREMENTS

In determining the diesel engine exhaust emission airflow requirements for the mine, WAI has referred to typically accepted guidelines for ventilation based on number of machines and machine engine power. This is equivalent to 0.05 $m^3/s/kW$ and has been used to determine diesel engine exhaust emission airflow. Table 16-17 shows the airflow requirement based on the anticipated diesel equipment fleet.

Table 16-17: Diesel Engine Exhaust Emission Airflow Requirements

Machine	Number	Engine Power (kW)	Machine Airflow (m^3/s)	Total Airflow (m^3/s)
Continuous Miner*	1	80	2.1	2.1
Wheel Loader	1	600	22.5	22.5
Light Vehicle	5	70	2.6	13.1
Service Truck	1	64	2.4	2.4
Jumbo	1	97	0.9	0.9
Charge Wagon	1	65	0.02	0.02
			Total	**42.99 > 40.0**

*Note: *Continuous Miner cutting head, electrical power, included for dust emissions.*
Fleet usage included in estimate

In the specified airflow, the addition of the cutting head of the continuous miner, although electric has been included to account for heat generated and dust produced. A reduction in airflow has been applied for the anticipated use of the equipment based on the mine schedule; equipment predominantly powered by electricity (e.g. jumbo) has not been taken into account.

Other than active mining areas, no other infrastructure (work bays, fuel bays, or explosive magazines) has been included, however airflow has been increased where required to ensure airflow transit time is minimal and corrected for atmospheric density at the altitude of the site. Total airflow requirements are summarised Table 16-18.

Table 16-18: Total Airflow Requirements

Area	Data
Diesel Engine Exhaust Emission Airflow Requirement	40 m^3/s
Residence Air Time Requirement	100 m^3/s
Total Air Requirement	140 m^3/s
Estimated Fan Pressure	178.5 Pa
Input Power	41.65 kW
Circuit Time (@ 1km)	12.5 min

Note: Based on site elevation.

16.12.3 EMERGENCY EGRESS

The main accesses carry fresh air and are the primary means of egress. Secondary egress via raises has not been included owing to the fan infrastructure and return airflow. Additional means of egress can be made via the secondary portals developed into the open pit.

16.12.4 DEWATERING

No information for estimation of groundwater inflow is currently available. Underground adits developed on site displayed no evidence of groundwater. Owing to the high altitude location of the project ground water inflow has not been considered to be a major influence on the project.

A small mobile face pump has been included for use with the drilling jumbo for drill water.

16.12.5 EXPLOSIVE MAGAZINE

Explosives have been included within the mining method. The explosive use is minor as limited drill and blast has been included to allow for initial opening of headings that may not support a continuous miner. Approximately 32 tonnes of bulk explosive (ANFO) is required. A small skid mounted mobile explosive magazine, with 14.5 t capacity has been included.

16.12.6 ELECTRICAL DISTRIBUTION

Underground electrical requirements and reticulation have been estimated for the project. Equivalent sized continuous miners require a 1 000 V supply. The mine is anticipated to have an 11 kV incoming power supply that will service the underground reticulation, including section circuit breakers. Panel sub-stations will then transform the incoming power supply to a 1 000 V supply for distribution. Substations are planned to be rated to 2 MVA, which will incorporate an 11/1.05 kV transformer, 1 000 V switchgear and protection relays, and PLC network and control equipment.

The operating panel distribution and control board receives the 1 000 V supply from the panel substation via a 150 mm^2 trailing cable and provides separate outlets for the distribution control and protection of the 1 000 V supplies to various development panel machines.

Typically, the continuous miner will require a 120 mm^2 cable, up to 150 m in length, while auxiliary cables (fans/feeder/pumps) are rated at 35 mm^2 at 110 m. A secondary smaller transformer will be required to supply the underground fans. Darling, P. (2011) (76).

16.12.7 MINE SERVICES

Compressed air lines have not been included as the bulk of the mining will be completed by continuous miners, not requiring compressed air. The drill jumbos included have on-board compressors that supply the required air to the machine.

Mine water reticulation underground is made via 110 mm mine poly pipe. A header tank on surface will be used to supply the underground workings. Guidance from National Institute for Occupational Safety and Health for dust control with continuous miners indicates 160 l/min is required for water sprays. The header tank is sized for 24 hours capacity for the continuous miner.

Underground communication is by radios supported on a leaky feeder truck system. Surface communication is by phone and radios.

16.13 GEOTECHNICAL

Geotechnical information is limited for the project. The Catholic University in Peru carried out basic geotechnical rock property testwork in 2010 (77) . Two drill holes were tested, PT16-TV and PT19-TV. The hole locations are not known. The two holes were tested at three different hole lengths for Ultimate Compressive Strength (UCS) and density. The results are shown in Table 16-19.

Table 16-19: Summary of Geotechnical Results

Hole	Depth (m)	Density (t/m^3)	UCS (MPa)	Poisson's Ratio	Young's Modulus
PT16-TV	4.50 – 7.50	2.071	19.04	0.36	9.73
PT16-TV	39.00 – 40.50	2.018	17.75	0.40	12.13
PT16-TV	54.00 – 55.50	1.994	16.09	0.23	12.31
PT19-TV	24.00-27.00	2.010	11.29	0.28	12.27
PT19-TV	39.00 – 42.00	1.936	7.55	0.40	6.26
PT19-TV	61.50 – 64.50	1.955	4.52	0.58	3.80
Average		2.00	12.71	0.38	9.42

The nature or state of weathering of the drill core was not known. The results indicated that hole PT19 TV has significant reduction in strength compared to PT16-TV. For the geotechnical analysis, drill hole PT19-TV has been excluded as during a site visit underground adits were visited which displayed competent rock strength.

16.13.1 ROCK MASS CLASSIFICATION

The basis of the rock mass classification assessment was made during the site visit in 2013. Rock mass classification was determined from data gathered on the site visits using the NGI Q System and Barton's RMR89 system. Supplemental data from the Mineral Resource estimate was also checked (Young, 2015) (78).

Rock mass classification summary results and workings are shown in Table 16-20, while the NGI Q System classification is summarised in Table 16-21 and the GSI converson is summarised in Table 16-22.

Table 16-20: Summary of RMR_{89} Classification

Parameter	Value	Classification Rating	Comment
Strength	19 MPa	2.0	Testwork
RQD	90 %	20	Observed
Joint Spacing	1.5 m	15	Observed
Joint Condition	15	15	3-10 m Length, Separation, Slightly Rough Soft Infilling <5 mm, Moderately Weathered
Groundwater	15	15	Dry
Basic RMR		**67**	Good Rock – Class 2
Joint Adjustment		**-5**	Fair Adjustment
Adjusted RMR		**62**	
Mining Adjustment		**-80 %**	
MRMR		**49.8**	

Table 16-21: Summary of NGI Q System Classification

Parameter	Classification Rating	Comment
RQD	90	Testwork
Jn	9.0	3 Joint Sets
Jr	1.5	Rough, Irregular Planar
Ja	2.0	Clay free, Non Softening
Jw	1.0	Dry
SRF	1.0	Medium Stress
Q Rating	**7.5**	Fair Rock

Table 16-22: Summary of GSI Conversion

Rating	GSI Value
GSI Conversion from RMR_{89}	65.2
GSI (Sonmex & Ulusay)	69.0
GSI (JCond RMR_{89})	67.5
GSI (Jr/Ja)	67.3
GSI Direct	70.0
Q Rating	68

16.13.2 SLOPE STABILITY

The previous slope angles of 55° have been maintained from the previous studies by WAI (74).

16.13.3 STOPE STABILITY

Tributary area analysis was used for the underground opening analysis. The factors described in Table 16-23 were used for the analysis.

Table 16-23: Summary of Tributary Analysis

Parameter	Value	Comment
Depth	200 m	
Density	19.72 kN/m^3	Testwork
Roadway (Wo)	10 m	
Pillar Width	25 m	
Pillar Height	5 m / 10 m	1st Pass Cut / Final Height
Pillar Length	25 m	
Average Pillar Strength	**11.20 MPa**	
σv	3.94 MPa	
TA factor	1.96	
σp	**7.73 MPa**	
FoS	**1.45 / 2.35**	FoS for 5 m Height / 10 m Height
Extraction Ratio e	**49 %**	

16.13.4 GROUND SUPPORT

Empirical conceptual ground support has been estimated using the NGI Q-System. Based on the Q system classification results it can be estimated that the maximum unsupported span that can be achieved is 7.2 m. This is an empirical estimate, but indicates rock bolting is required.

The Q System also estimated rock bolt length required, the system indicates that 3.4 m bolt lengths are required. Typical lengths of supplied bolts are 3.0 m, which have been incorporated into the design.

A conceptual analysis was completed using Rocscience Examine 2D V8.0 with the output of the analysis shown in Figure 16-13.

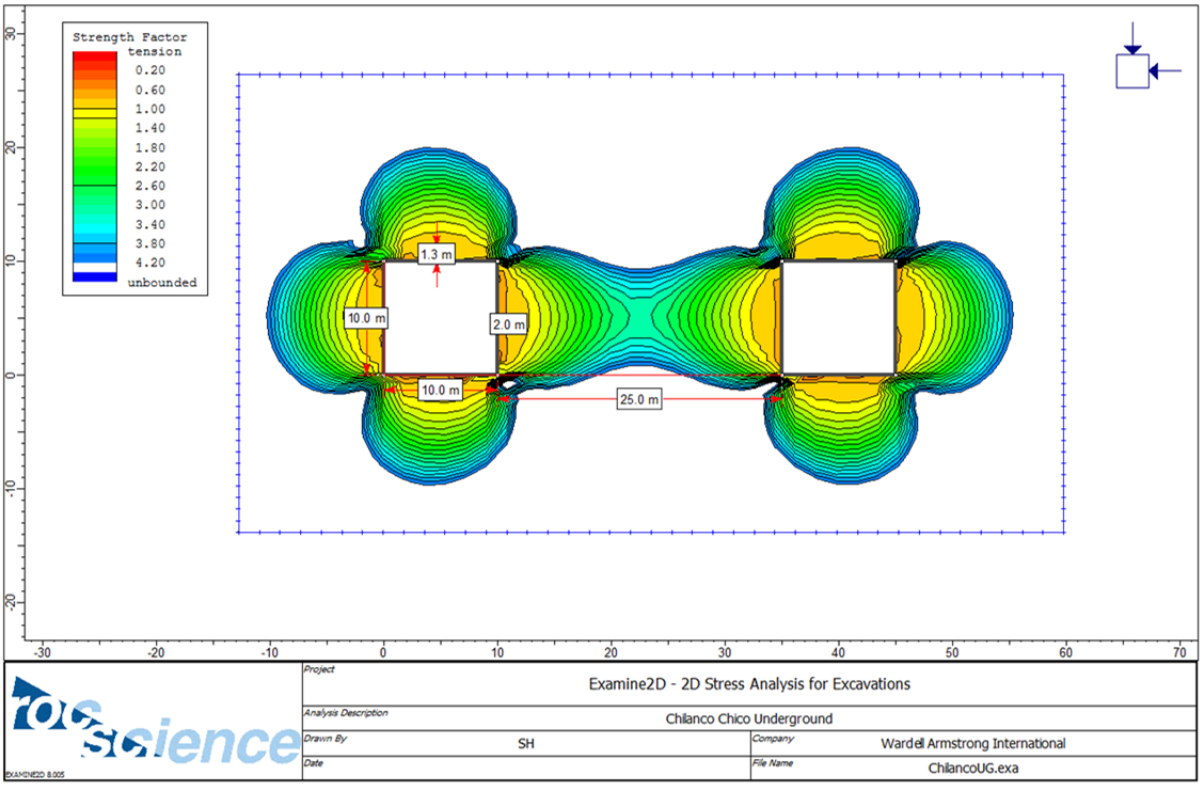


Figure 16-13: Examine 2D Conceptual View of Underground

(Coloured by Strength Factor)

The analysis is conceptual, but has been coloured by strength factor. Strength factor is a measure of induced stress against rock strength, when the strength factor is <1.0 this indicates rock damage, which is likely to be seen as block fall out. The depth of strength factor <1.0 is indicated to be limited to 1.3 m in the backs and 2.0 m in the sidewalls of the excavation. This depth supports the length of 3.0 m bolts for the conceptual plan.

For bolt pattern spacing, a dice five pattern has been selected, along with guidelines for spacing, which results in a bolt spacing of 1.5 m x 1.8 m or 1 bolt/2.7 m^2.

16.14 PROJECTED PRODUCTION SCHEDULE

16.14.1 GENERAL

The approach taken for the project production schedule is to maximise produced U_3O_8 and give consideration to the underground workings. The anticipated project life is ten years and the PEM contained within the design equates to annual ROM production of 10.9 Mt. The underground mine design has been based on production capability of a single continuous miner, which equates to 2 697 t/d or 973 kt annually, for a planned mine life of nine years.

16.14.2 LOM CONCEPTUAL PRODUCTION

Open pit scheduling by deposit was completed using Datamine NPVS, for the underground scheduling Datamine 5D Planner and Enhanced Production Scheduler were used. Both open pit and underground scheduling have targeted higher-grade blocks where possible. The open pit schedule also prioritises the Chilcuno Chico pushback blocks in the first year of production, to enable underground extraction to fit in with the ten year project life. Table 16-24 shows the conceptual production plan summary and Figure 16-14 shows a summary plot of conceptual production.

The PEM mining inventory is detailed in Table 16-24, the breakdown of Mineral Resource classification is not shown, but the LOM conceptual schedule contains 44 % Indicated material and 56 % Inferred material.

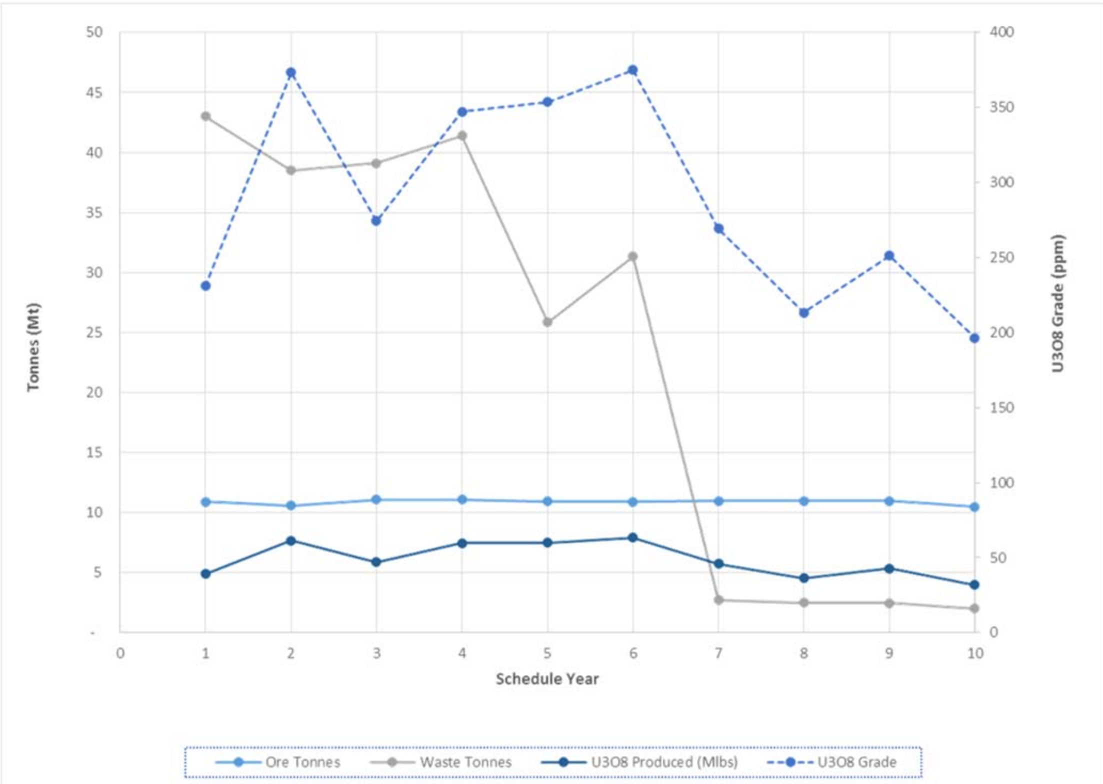


Figure 16-14: Summary Conceptual Schedule Output

Table 16-24: Conceptual Production Plan Summary

Complex	Output	Unit	Year											Total
			0	1	2	3	4	5	6	7	8	9	10	
Colibri Complex	ROM Tonnes	Mt							0.403	10.0	10.0	10.0	9.64	**40.0**
	U_3O_8 Grade	ppm							315	256	198	235	186	**220**
	Waste Tonnes	Mt							0.220	2.70	2.52	2.46	2.02	**9.92**
	Total Tonnes	Mt							0.623	12.7	12.5	12.5	11.7	**50.0**
	Strip Ratio	$t_W:t_{PEM}$							0.55	0.27	0.25	0.25	0.21	**0.25**
	Contained U_3O_8	t							127	2 560	1 980	2 350	1 790	**8 810**
	Recovered U_3O_8	t							112	2 250	1 740	2 070	1 580	**7 750**
	Recovered U_3O_8	M lbs							0.246	4.96	3.85	4.56	3.48	**17.1**
Kihitian Complex	ROM Tonnes	Mt		10.9	10.1	10.1	10.1	4.5	-	-	-	-	-	**45.7**
	U_3O_8 Grade	ppm		231	370	229	305	391	-	-	-	-	-	**293**
	Waste Tonnes	Mt		43.0	38.5	39.1	41.4	8.1	-	-	-	-	-	**170**
	Total Tonnes	Mt		53.9	48.6	49.2	51.5	12.7	-	-	-	-	-	**216**
	Strip Ratio	$t_W:t_{PEM}$		3.94	3.81	3.87	4.10	1.80						**3.72**
	Contained U_3O_8	t		2 520	3 740	2 310	3 080	1 760	-	-	-	-	-	**13 400**
	Recovered U_3O_8	t		2 220	3 290	2 030	2 710	1 550	-	-	-	-	-	**11 800**
	Recovered U_3O_8	M lbs		4.89	7.26	4.48	5.98	3.41	-	-	-	-	-	**26.0**
Isivilla Complex	ROM Tonnes	Mt						5.45	9.53	-	-	-	-	**15.0**
	U_3O_8 Grade	ppm						313	379	-	-	-	-	**355**
	Waste Tonnes	Mt						17.7	31.1	-	-	-	-	**48.8**
	Total Tonnes	Mt						23.2	40.6	-	-	-	-	**63.8**
	Strip Ratio	$t_W:t_{PEM}$						3.25	3.26					**3.26**
	Contained U_3O_8	t						1 710	3 610	-	-	-	-	**5 320**

Complex	Output	Unit	Year											Total
			0	1	2	3	4	5	6	7	8	9	10	
	Recovered U_3O_8	t						1 500	3 180	-	-	-	-	**4 680**
	Recovered U_3O_8	M lbs						3.32	7.00	-	-	-	-	**10.3**
Kihitian Complex UG	ROM Tonnes	Mt			0.507	0.984	0.984	0.984	0.987	0.984	0.984	0.984	0.832	**8.23**
	U_3O_8 Grade	ppm			438	738	779	407	361	403	370	418	317	**475**
	Waste Tonnes	Mt			-	0.01	-	-	0.01	-	-	-	-	**0.02**
	Total Tonnes	Mt			0.507	0.994	0.984	0.984	0.996	0.984	0.984	0.984	0.832	**8.25**
	Strip Ratio	$t_W:t_{PEM}$			-	0.01	-	-	0.01	-	-	-	-	**0.00**
	Contained U_3O_8	t			222	726	767	400	356	397	364	411	264	**3 910**
	Recovered U_3O_8	t			195	639	675	352	313	349	320	362	232	**3 440**
	Recovered U_3O_8	M lbs			0.430	1.410	1.490	0.776	0.691	0.770	0.706	0.797	0.512	**7.58**
Total	**ROM Tonnes**	**Mt**		**10.9**	**10.6**	**11.1**	**11.1**	**10.9**	**10.9**	**11.0**	**11.0**	**11.0**	**10.5**	**109**
	U_3O_8 Grade	**ppm**		**231**	**373**	**274**	**347**	**354**	**375**	**269**	**213**	**251**	**196**	**289**
	Waste Tonnes	**Mt**		**43.0**	**38.5**	**39.1**	**41.4**	**25.8**	**31.3**	**2.70**	**2.52**	**2.46**	**2.02**	**229**
	Total Tonnes	**Mt**		**53.9**	**49.1**	**50.2**	**52.5**	**36.8**	**42.2**	**13.7**	**13.5**	**13.4**	**12.5**	**338**
	Strip Ratio	**$t_W:t_{PEM}$**		**3.94**	**3.63**	**3.53**	**3.74**	**2.36**	**2.87**	**0.25**	**0.23**	**0.22**	**0.19**	**2.10**
	Contained U_3O_8	**t**		**2 520**	**3 962**	**3 036**	**3 847**	**3 870**	**4 093**	**2 957**	**2 344**	**2 761**	**2 054**	**31 400**
	Recovered U_3O_8	**t**		**2 220**	**3 485**	**2 669**	**3 385**	**3 402**	**3 605**	**2 599**	**2 060**	**2 432**	**1 812**	**27 700**
	Recovered U_3O_8	**M lbs**		**4.89**	**7.69**	**5.89**	**7.47**	**7.51**	**7.94**	**5.73**	**4.56**	**5.36**	**3.99**	**61.0**

16.15 HIGH GRADE OPTION

The optimisation for each complex was run using a series of cut-off grades in 100 ppm U_3O_8 increments. The results were then reviewed using an estimated mine life, based on a processing rate of 5Mtpa, and a project NPV (NPV reported from NPV scheduler) estimated. The results of the optimisations are detailed in Figure 16-15.

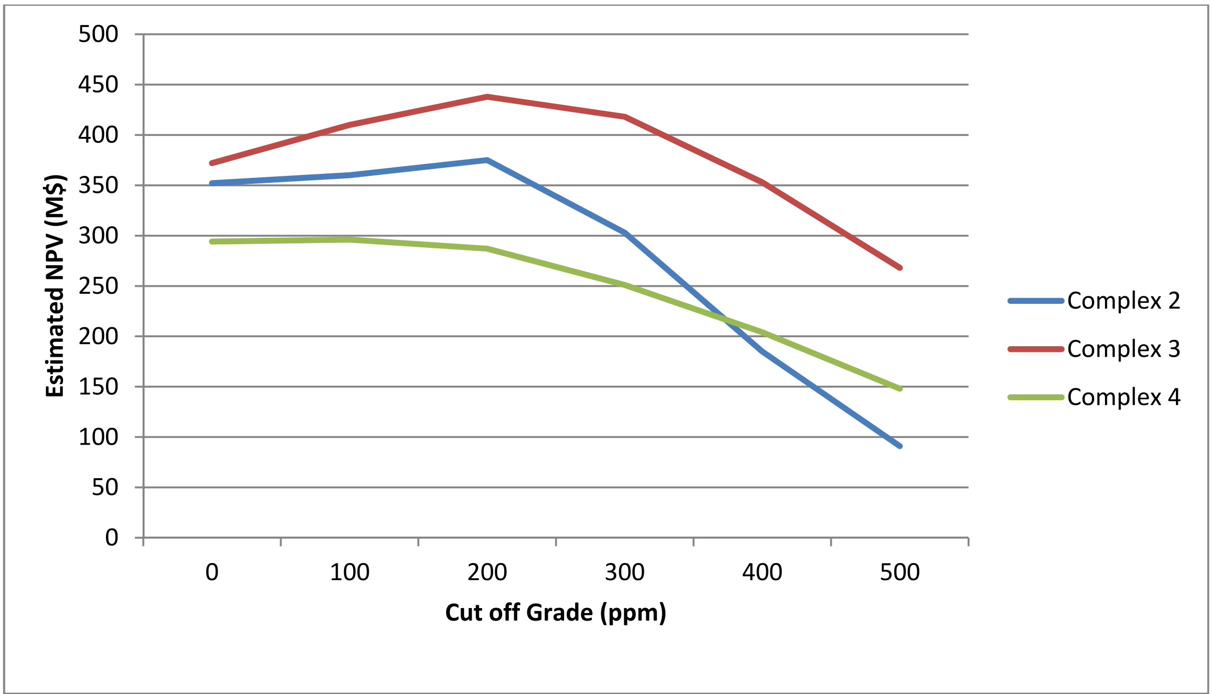


Figure 16-15 NPV v Cut Off Grade

A U_3O_8 cut-off grade of 200 ppm was selected and resulting tonnes and grades of material were used in the evaluation.

16.15.1 RESOURCE CONVERSION

A comparison was drawn against the Potential mineral resource inventory outlined in section 15, and the same percentage conversions were used to interpret Table 16-25 and state the resulting numbers in Table 16-26.

Table 16-25 Selected Mining Complexes 200ppm U_3O_8 Cut-off

Complex	Tonnage (Mt)	Grade U_3O_8 ppm
2	18.1	340
3	54.2	469
4	11.1	464
Total / Average	**83.5**	**440**

Table 16-26 Resource Conversion

Tonnage (Mt)			Grade U_3O_8 ppm			Metal U_3O_8				
Mineral Resource	Mining	%	Mineral Resource	Mining	%	Mineral Resource (t)	Mining (t)	Mineral Resource (Mlbs)	Mining (Mlbs)	%
83.5	50.8	61	440	434	99	36,700	22,000	81.0	48.6	60
Note: Complex 2, 3, & 4 only.										

16.15.2 MINING SCHEDULES

Using the 200 U_3O_8 ppm cut-off grade, conceptual schedules have been generated for analysis. An open pit only schedule, sequenced in the following order: Complex 2, Complex 3, and Complex 4, based on NPV and produced U_3O_8, and a schedule allowing for underground methods starting from Complex 3, Complex 2, and Complex 4.

SECTION 17 RECOVERY METHODS

The information detailed in Section 13 enabled the development of a process block diagram as shown in Figure 17-1.

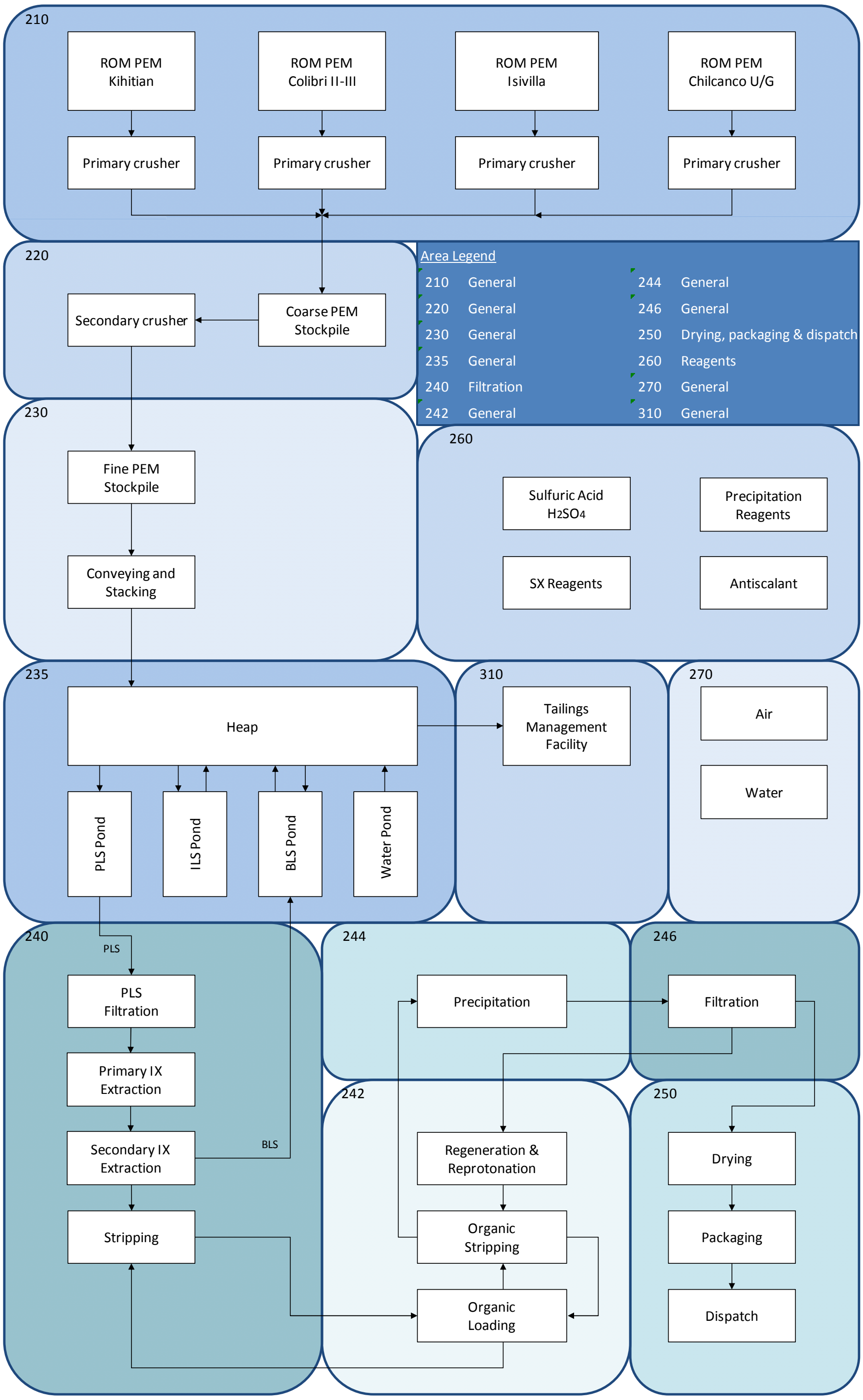


Figure 17-1: Process Block Diagram

17.1 DESIGN CRITERIA

The developed production schedule (Table 16-24) was used as the basis to inform the selection of the proposed plant equipment as well as key design criteria as shown in Table 17-1. Process facility design criteria are derived where possible from vendor information or are otherwise calculated from reference material or sourced in-house.

Table 17-1: Design Criteria

Description	Value	Unit	Reference
Life of mine	10	a	Assumed
Plant operating hours	24	h/day	Assumed
Plant design throughput	1 463	t/h	Calculated
Leach method	Heap-leach		Plateau Uranium
Leach Pad Cells	6	per leach pad	Calculated
Leach Heap Height	7	m	Calculated
Recovery method	Ion-exchange	–	Section 13
Precipitation method	Ammonium hydroxide precipitation	–	Assumed
Overall Leach recovery of U_3O_8	88	%	Calculated (refer to Section 13)

The equipment was selected on a non-optimised preliminary plant design basis and the major cost items have received most attention as is appropriate at this preliminary phase of the project.

A significant consideration during the equipment selection is the effect of the low density air at the plant's high altitude (approximately 4 100 to 4 700 m above mean sea level). During design and equipment selection attempts were made to minimise construction and avoid the inefficiencies associated with un-acclimatised people working at altitude. All combustion engines would be required to be turbo or supercharged and/or tuned for high altitude operation to avoid significant inefficiency with their operations. Pump selection was also made with care as many pumps would not tolerate the low NPSH available.

17.1.1 EFFECTS OF ALTITUDE ON ELECTRICAL EQUIPMENT

Table 17-2 outlines the effects of altitude on electrical equipment for the project.

Table 17-2: Effects of Altitude on Electrical Equipment

Guidelines to Altitude De-rating - Macusani – 4 500 m			
Item	**Calculation method**	**Derating - To be verified by Suppliers**	**Reference**
Motors	Multiply motor shaft kW by 130 % and choose next highest motor	70 % of nameplate	SEED Ltd
VS Drives	Extrapolation from table	Voltage 12 % Current 5 % – 10 % Note: Drives would be rated for the nameplate rating of the motor further derating may not be necessary	Siemens Sinamics catalogue p2/8
Transformers	0.3 % / 100 m above 1 000 m	15 % – 20 % (HV terminations to be de-rated see below)	ANSI C57.92/81
HV Switchgear (up to 33 kV)	Multiply equipment ratings by 0.65	Voltage de-rating 65 % - e.g. 16 kV switchgear operating at 10 kV Current de-rating 7 %	Industrial Power Engineering and Applications Handbook. Siemens Tech Topics No. 30
LV Switchgear	Use LV switchgear rated to operate at 690 V	690 V rated equipment can operate at 500 V Current rating at 95 %	Schneider Masterpact catalogue
Generators	20 % for 3 500 m + 1.4 % per 100 m	65 % of nominal rating	Connell Mining products
Overhead lines		Equipment can only be used at 65 % of its rated voltage.	

It is important that all electrical equipment is rated for the high altitude operation. For example, medium voltage switchgear rated at 16 kV can only be operated at 10 kV at this altitude. Cables and drives for motor feeders would be sized for the nameplate rating of the motor and would not normally require any further de-rating.

The selection of motors with higher ratings to compensate for the reduced cooling effects at high altitude will result in motors operating below their full load rating and therefore with a low power factor.

Detailed site survey information as well as operating experience in the area will be necessary to confirm de-rating requirements as the information sources quoted above indicate that the figures given are to be used as a guide only.

17.2 PROCESS DESCRIPTION

17.2.1 CRUSHING

The PEM sample assessed showed excellent leaching characteristics even at a particle size of 100 % passing 2 inches (50 mm). Therefore, to reduce the assumed ROM feed from 100 % passing 600 mm to 100 % passing 50 mm, the PEM would be crushed in two stages. The first stage is open circuit primary crushing at the ROM pad. Two primary crushers would be in operation; one at Colibri, the other at Kihitian. Crushed ROM PEM is conveyed on a series of overland conveyors from the coarse crushing facility to the coarse PEM stockpile, located adjacent to the fine crushing station near to the leach pad facility. Coarse PEM is reclaimed from the coarse PEM stockpile and crushed at the open-circuit fine crusher. The fine crushed PEM is then stockpiled on the fine PEM stockpile.

ROM PEM is crushed into an acid permeable product size. The crushing plant operates in cooperation with the overland conveying and heap stacking areas. Conveying and stacking is to be undertaken 24 hours per day. Crushing is to be undertaken 20 hours per day. An intermediate stockpile would ensure the supply of crushed material to the heap is maintained. ROM PEM would feed into the primary crushing circuit via an apron feeder. This would enable the control of the PEM feed to the circuit. The ROM feed bin capacity is 190 m^3. The primary sizer would have a reduction ratio of 4 to give a product with an 80 % passing size of 126 mm.

Crushed ROM PEM is conveyed on a series of overland conveyors from the coarse crushing facility to the coarse PEM stockpile, located adjacent to the fine crushing station near to the leach pad facility. Coarse PEM is reclaimed from the stockpile and crushed by the secondary sizer to 80 % passing 42 mm (100 % passing 50 mm), a size reduction ratio of approximately 3:1. The fine crushed PEM is then stockpiled on the fine PEM stockpile.

Fine PEM from the stockpile feeds the heap leach cells via an overland conveyor system.

The process described above is highly dependent upon the PEM characteristics. Further testwork is required to classify the PEM and its amenability to crushing and screening.

17.2.1.1 ROM BIN

The ROM bins, at the front end of the crushing plants, accept ROM product from what is assumed to be a mixture of both underground and conventional open pit mining operations, involving drilling, blasting, and loading by shovel, and truck haulage of the mined PEM from multiple sites.

The ROM bins receive PEM from the mining fleet and would be installed to allow the use of dumping equipment (truck or front end loader). The construction of a retaining wall negates the requirement to use a more costly bridge arrangement, which would have higher structural steel requirements (the bridge itself and additional supports). However, a bridge arrangement reduces the total required civil works and can be favourable if there is a requirement to relocate the crushing plants throughout the life of the mine. In this project, one crushing installation would be fixed and one would be mobile.

In the event of crushing/stacking downtime, ROM pad storage areas would be available as a transition dump so as to not disrupt the mining process. Mobile equipment would then feed PEM to the crusher in times when mining is unavailable and is included in the mining cost section of this report.

17.2.1.2 APRON FEEDER

The apron feeder is a steel track that steadily draws material out from the ROM bin, and feeds it into the primary crusher. The track speed is adjusted to control the material flowrate into the crushing circuit, as measured and controlled by a downstream weightometer. The feeder would tolerate the PEM being dumped directly onto its feeding track surface, which should enhance reliability and operational flexibility. The apron feeder is part of the crusher assembly package and as such is semi mobile.

17.2.1.3 CRUSHERS

The crushing equipment selection is driven by the dimensions and the hardness of the mineral deposit. With little testwork of the PEM available, qualitative methodology must be applied. It is assumed the Macusani PEM may be characterised as relatively soft, with moderate abrasiveness, when compared to other PEM crushing operations and has been modelled on a size distribution sourced for blasted limestone.

The feed top size is assumed as 600 mm, the desired product size is 50 mm. For this application and capacity, gyratory crushers and various arrangements of multiple jaw and/or cone crushers were briefly considered. However, none of these options have been found to be ideal. The gyratory crushers suffer from very high capital and installation costs. Jaw and cone crushers require multiple units to achieve the design throughput, leading again to high capital costs. Their operational costs are also significant. Due to their crushing mechanism, all of these equipment options would also produce more fines in comparison to the proposed solution.

The recommended solution is to use sizers for primary and secondary crushing. Sizers are well suited to producing the 50 mm product size required. Sizers have superior throughput for a given equipment size, significantly lower capital costs and should also provide the benefit of less fines production. MMD Equipment was approached to supply a quote for the equipment as they have significant experience with sizing equipment in these types of applications, and have branches in the Peruvian region.

Dust control sprays have been included at the feeder and crushers primarily for dust control and maintenance of the health and safety of mine operators. Dust suppression is particularly important, as

PEM dust particles (blown on the wind or otherwise) should not be allowed to contaminate the area around the operation.

17.2.1.4 CRUSHING CIRCUIT SCREENS, CONVEYORS AND CHUTES

Sizing screens will not be required as no oversize is produced from the sizers and thus no recirculation circuit and conveyors are required. A basic network of conveyors and chutes exists to transport the PEM between stages of crushing.

The conveyors have the ability to be relocated. They have been designed with minimal footings, and are short and easy to maintain as they will all be the same width and motor sizes.

17.2.1.5 AGGLOMERATION

It has been assumed that agglomeration will not be required, due to factors such as the relatively large product size, and the effective size control offered by sizers. Further testwork will be required to determine if the crushed PEM requires agglomeration.

17.2.1.6 STOCKPILE FOR CRUSHED PEM STORAGE

It has been assumed a stockpile for buffer storage would be required. This assumption removes the requirement for synchronous operation of the crushing and stacking equipment. Storage capacity to buffer between the mining and process facility operations is to be provided by ROM pad stockpiles. It may also be determined that the process requires feed blending for grade control, which stockpiling could facilitate. However, this PEA assumes the mining operation provides the process facility with PEM of the appropriate head grade.

17.2.2 CONVEY, STACK AND RECLAIM

Fine crushed PEM is reclaimed from the stockpile and is conveyed on a series of overland conveyors from the fine crushing area to the heap leach site. Conveyed PEM is then stacked in a continuous fashion onto the heap at a height of 7 m and a cell width of 225 m. After the PEM has been stacked it is dressed and prepared for irrigation with dilute sulfuric acid. Whilst the heap is continuous, the collection and irrigation piping divides the heap into 6 cells. The stacking rate for the entire pad matches the cell leaching/drain cycle times such that the stacker and reclaimer operate continuously through cycles.

The cell leach cycle (based on current testwork) is outlined in Table 17-3.

Table 17-3: Cell Schedule

Description	Value	Unit	Fraction
Reclaim	8	days	0.057
Reclaim Stack Gap	4	days	0.028
Stack	8	days	0.057
Pipe layout	1	days	0.007
ILS leach	45	days	0.319
BLS leach	45	days	0.319
Total leach	90	days	0.638
Wash	21	days	0.149
Drain	9	days	0.064
Pipe collect	0	days	0.000
Total	**141**	**days**	**1.000**

Fine PEM would travel on a corridor conveyor from the fine PEM stockpile and a tripper conveyor would trip the PEM onto the stacker cross conveyor. The stacker stacks material on the leach pad to a height of approximately 7 m. The stacking and reclaiming conveyors function as one unit during operation. The conveying, stacking and reclaiming system is designed to operate 24 hours per day.

After each cell cycle is complete, the spent PEM is reclaimed and transported, over a series of conveyors, to the tailings management facility

17.2.2.1 CORRIDOR CONVEYING

Where the conveyor path enters the racetrack turnaround zone, it is sunken into an open channel with concrete culvert bridge sections at spacing matching the reclaiming bridge conveyor tracks.

Two corridor conveyors are installed in the centre of the racetrack, the full length of the stack. One is feeding PEM from the overland conveyor, and this feeding corridor conveyor is referred to as the stacking corridor conveyor. A tripping conveyor installation operates on rail tracks along/over the stacking corridor conveyor, with the tripping mechanism unloading the PEM from the stacking corridor conveyor at the required leach pad cell. The stacking corridor conveyer will be level or have a constant slope for this system, and earthworks create this corridor as part of the pad construction.

The second corridor conveyor is for reclaiming PEM from the heap to the waste disposal area and this conveyor is referred to, as the reclaim corridor conveyor. No tripping mechanism is required for the reclaim corridor conveyor, as it is loaded by the reclaim conveyors. This conveyor runs from south to north down the slope towards the tailings facility.

17.2.2.2 LEACH PAD STACKING

The stacking system design and equipment selection considers the following criteria:

- A robust easily maintainable design due to altitude and remoteness of location;
- High degree of standardisation for mechanical parts; and
- Overall low weight and point loading to minimise compaction.

A track mounted bridge mobile stacking conveyor is specified that meets the key criteria. Additionally, the stacking system is self-levelling.

One of the more important features of the stacking system is that it must maintain minimal ground bearing pressures during operation. This is due to the fact that the entire pad is covered by a HDPE geo-membrane and also a network of perforated collection piping. The stacker incorporates a bridge conveyor that gradually retreats from the face of the leach pile as it stacks. The minimal slow movement speeds of the proposed solution minimises the risk of liner perforation due to surface activity.

17.2.2.3 WASHING

Once the PEM has been leached, it is washed with water from the recycle pond to reduce the sulfuric acid concentration prior to disposal to tailings. The washing water operates in a closed loop from the pond, over the heap and back to the pond, with losses replenished using raw water makeup. Left untreated, the wash water circuit pH will lower as it rinses the acid from the PEM. In order to ensure it remains effective as a washing agent, water is treated with lime in a transfer tank prior to it being pumped back to the heap. Any gypsum precipitate would be filtered from the water as it permeates through the heap and would be disposed of with the spent PEM.

17.2.2.4 LEACH PAD RECLAIM

After the stacked PEM is leached and washed, the reclaiming operation begins. A racetrack system operates in a circular motion with a working area that moves through the heap reclaiming spent PEM and stacking fresh PEM in its place. There is a 4 day gap between the reclaimer and stacker. The spatial control obtained through using a track mounted bucket wheel excavator should remove the need to replace the expensive overliner on the heap base. However, any pad maintenance required may be undertaken in the gap time. A bucket wheel reclaimer is selected as the recommended engineering and operational solution. Several budget cost estimates were sourced from main suppliers with RACHO being chosen on the basis of price only.

17.2.3 LEACH

The chemistry of the acid leach is a process of uranium being dissolved in dilute sulfuric acid to form uranyl sulfate, as follows:

$$UO_2^{2+}{}_{(aq)} + SO_4^{2-} \longrightarrow UO_2SO_{4(aq)}$$

Operational heap cells are paired during leaching depending on the amount of time the cell has been under irrigation. Fresh PEM is irrigated firstly by intermediate leach solution (ILS) and then by barren leach solution (BLS). The ILS irrigating the newly stacked cells is collected in the pregnant leach solution (PLS) pond ready to be pumped to the recovery plant.

17.2.3.1 LEACH PAD SELECTION

There are two main pad leaching methodologies; on/off and single use/permanent. The single use pad was considered only in concept for a number of reasons.

Single use pads can operate in many configurations. They are commonly located on both high gradient terrain such as a valley side, but also on relatively flat topography. They have cost benefits related to single-handling of the PEM and common leach and tailings site. The topography at the proposed site may be suitable for this system. However, single use pads require detailed geo-mechanical data to design, and are more generally suited to slower leaching PEM. The project's uranium PEM is relatively fast leaching and, in addition, a level of process control greater than that typically available in single use pads is required for maximum recovery. The key geo-mechanical constraint is the maximum lift height obtainable before compacting and/or instability occurs. If multiple lifts were to be implemented, the effect of the upper layers bearing down upon the lower lift layer(s), HDPE liner material, and drainage piping must be considered, which will limit overall heap height. In addition, entire pad stability, particularly in seismic regions, is a major area of risk that can have catastrophic operational and environmental consequences.

Single use leach pads, utilising multiple lifts over a greater surface area or in a valley leach arrangement may have a cost benefit due the lack of any reclaiming operation. However, costs are adversely affected by the need to build new leach pad sites with expensive HDPE liner, piping and earthworks. Whether or not a single use pad is feasible depends on factors including availability of suitable surface area and, of critical importance, the maximum overall heap height the PEM will tolerate. The maximum heap height possible is heavily dependent on individual PEM characteristics and therefore further testwork would be required. These uncertainties led to the decision not to further consider a single use pad in this PEA.

The on/off pad requires PEM to be double-handled and disposed of in a different location to the leach site. However, it has the advantage of minimising the required pad surface area. In addition to the tighter process controllability, it also only requires PEM/leach mechanics/kinetics of a single lift to be considered, making the design relatively independent of the characteristics of the waste PEM, which are currently unknown. This proposed concept has been selected as it has the greatest design certainty of the available heap leach options. Although it still has many caveats regarding its design certainty it will provide an adequate basis for costing purposes.

17.2.3.2 LEACH PAD DESCRIPTION

The proposed design involves an on/off leach pad that will operate in a racetrack configuration. This requires falls in grade perpendicular to the corridor to promote cell drainage collection as well as fall the length of the entire pad to drain the flows to the ponds. In addition, earthworks and civil construction are required to level the turnaround area for the bridge conveyor and construct the channel depression for the product and tailings overland conveyors.

A known advantage of locating the pad on a suitably naturally sloping site can be a reduction in the earthworks required in construction. Whilst there is a natural average slope at the proposed site, due to the large area of the civil footprint, there are still considerable earthworks associated with levelling and shaping the site to create the appropriate drainage of both the pad and the channels that deliver flows to the ponds.

The leach pad base is made up of various layers of fill, aggregate and impervious geo-membrane layers. The civil works must create appropriate gradients in each cell of the pad to cause the irrigated leach solution to be contained within each cell until it is collected by the collection piping and drained out to the splitter boxes and channels.

Two layers of geo-membrane will be laid on the pad, separated by a sand filled leak detection layer. An HDPE liner, of 2 mm thickness, has been selected as the geo-membrane material for costing purposes. There are possible variations to the exact specification of the HDPE, and these will be determined in a detail design exercise as the project progresses. Once the collection piping has been laid over the upper geo-membrane layer, there will be a thick layer of aggregate (approximately 800 mm to 1 000 mm) to facilitate drainage and protect the geo-membrane and collection piping from damage by the stacking and reclaiming vehicles.

17.2.3.3 LEACH PAD PIPING - COLLECTION

The collection piping would be laid in a herring bone (or similar) arrangement. The branch collection lines are of NPS 100 corrugated perforated HDPE pipe and these run into a main backbone of NPS 250 pipe of the same perforated type. These are the pipes that the leach solution enters after percolating down through the heap, and which then collect the solution and drain it back to a pond. These collection pipes are sized considering drainage angles available and flowrates required. Heap height and the mass of vehicles on the heap are also considered during the pipe selection.

At the base of each cell, the collection piping becomes non-perforated, passes through the pad bund wall and into a splitter box. The splitter box incorporates plug/dart style shut-off valves, which are used to select the appropriate channel (and hence pond) that the cell will drain into.

17.2.3.4 LEACH PAD PIPING - IRRIGATION

The irrigation piping supplies leach solution from the various ponds up to the top of the heap. The piping and valves are arranged such that fluid from any pond can be used to irrigate any leach pad cell as required. The leach solution pumps feed transfer lines from the ponds to the edge of the leach pad. These lines are installed permanently during the pad construction, and are of NPS 550 HDPE pipes. Each transfer line tees onto the irrigation piping of each pad cell, with isolation butterfly valves installed to allow flow control to the cell as required.

After a pad cell has been stacked with PEM, the cell's irrigation pipes are laid manually over the top of the stack. A line of NPS 150 mm PVC lay-flat hose is used as a supply line, and the many branches of the dripper irrigation lines are fed from this main line. Lay flat hose is used to facilitate quick and easy installation and joining operations. The irrigation lines selected utilise a proprietary dripper system designed for leaching applications. These drippers are installed inside the pipe and are designed to provide a constant flow-rate for a given pressure without blocking over their period of use. This system is less prone to damage when stripping off the irrigation pipes, compared to an externally mounted dripper, and it also allows the drippers to be re-used for typically three leach cycles. The cost of replacement has been factored into the OPEX calculations.

The irrigation dripper pattern is varied by two factors; the centre to centre distance between the parallel laid pipes and the dripper intervals within the lines. A closely spaced dripper pattern will wet/leach the PEM very effectively, however piping costs increase accordingly. A sparse dripper pattern will wet/leach less effectively, particularly the upper regions of the heap, (although still achieving the required flow-rate), due to the drippers being more spaced out. The optimal wetting arrangement has been selected which consists of pipes spaced at approximately 0.5 m, with drippers installed every 0.5 m, resulting in a half metre square dripper grid at approximately 8 litres/hour/m^2.

17.2.3.5 LEACH SOLUTION STORAGE PONDS

Of the three storage ponds required for the site, two are required for the storage of leach solution. A further pond is required for the wash water, which is also irrigated over the pad. The ponds are referred to as the pregnant leach solution (PLS) pond, the barren leach solution (BLS) pond (BLS is also sometimes called Raffinate), and finally the recycle pond.

The pond liquid level will be below all the pads, allowing gravity to drain the various solutions from the pads to the ponds as required through channels lined with HDPE geo-membrane. The ponds are sized to provide a nominal process capacity, plus freeboard sufficient to contain seven days of a maximum 24 hour rain event and a permanent power outage (zero pumping discharge, whilst flows continue from the heap and precipitation). In addition, the pond liquid levels are stepped down along the row, so that in the event of any of the leach solution ponds being over filled, the contents will flow down to the neighbouring pond. The cascading is arranged in order of uranium concentration, to best contain the uranium product within the process.

The last pond in the cascade (recycle pond) must be sized such that it will never overflow. Currently, for costing purposes a preliminary design of the recycle pond is used, however the actual required size/volume of the recycle pond must be carefully calculated, and will result from detailed meteorological data, review of the operational criteria and a detailed site water balance. The final design criteria of the ponds, is yet to be determined and must be revisited as the required data becomes available.

17.2.3.6 LEACH SOLUTION PUMPS

Each pond has its own pumping station to transfer solution to leach pad irrigation or the process facility as required.

The current pond arrangement allows for the placement of pumps at a level near the bottom of the pond, with a suction line protruding through the wall. This allows the design to utilise standard centrifugal pumps, and negate the costly constraint of the low NPSH available at the high altitude site.

The pump suction strainers should be located to minimise ingress of solids and also ensure the maximum amount of liquid may be extracted from the pond if required.

17.2.4 RECOVERY

The pregnant liquor collected from the heaps under irrigation is fed to an ion exchange plant where the dissolved uranium is loaded onto a cationic resin. The ion exchange plant is a continuous loading and stripping process that upgrades the uranium by approximately 20 fold.

Recovery of the leached uranium is achieved via its adsorption onto the ion exchange resin followed by stripping with a concentrated sulfuric acid solution. Twelve fixed bed ion exchange columns are effectively sequentially rotated through the continuous process of loading, washing, stripping and regeneration. The static columns are effectively rotated by opening and closing valves to define the duty of the column at specific times in the processing schedule. The upgraded liquor from the IX plant is then processed by solvent extraction to enable the recycling of the concentrated acid required in the IX stripping process. The upgraded liquor from the SX plant is stripped using ammonium sulfate and the ammonium uranyl sulfate produced then undergoes precipitation. The precipitation reaction shows that the ammonium diuranate is precipitated from solution using ammonium hydroxide in a series of precipitators.

$$2(NH_4)_2UO_2(SO_4)_2 + 6NH_4OH \rightarrow (NH_4)_2U_2O_7 \downarrow + 4(NH_4)_2SO_4 + 3H_2O$$

Further testwork is required to better define the total residence time required at precipitation, which is expected to be in the order of two to four hours. The ammonium diuranate crystals are thickened and filtered to dewater the precipitate in preparation for kiln drying to drive off remaining water. The dry ammonium diuranate yellowcake product is then allowed to cool and is stored in a silo before packaging.

17.2.4.1 RECOVERY TECHNOLOGY SELECTION

There are several competing technologies for uranium recovery from pregnant leach solutions. After investigation and a number of assumptions, ion exchange (IX) has been identified and selected for costing purposes, as the most appropriate recovery technology for the Macusani operation. It must be understood, however, that this preliminary selection is based upon some very significant assumptions that require further work to confirm.

To determine the most appropriate recovery method for plant costing purposes, it has been necessary to assume that the testwork completed is representative of the leach kinetics and product characteristics. This results in a PLS concentration estimated as approximately 1 555 ppm U_3O_8, for the proposed flowsheet. Further testwork is required to confirm the assumptions behind this estimation. At this concentration, the IX process will be the most cost effective, benefiting from comparatively low capital expenditure and a relatively low operating cost.

It is also noted that chlorides may suppress uranium extraction in the IX process. This is a variable that needs confirmation by testing the water supply for the Macusani site, as an SX plant could be required (instead of IX) if the PLS were to contain high levels of chloride ions. It has been assumed that the water supply will contain below 5 g/L chlorides (or other competing ions), allowing IX to be utilised.

Stripping of the IX resin is achieved using a strong acid solution as the eluant. A subsequent solvent extraction stage is required to recover the excess acid used in IX stripping. An organic liquid phase (the solvent) will adsorb the uranyl sulfate from the strong acid solution. The strong acid solution will then be recycled back to IX as barren eluant. The loaded solvent will be contacted with an aqueous phase of dilute ammonium sulfate to recover the uranyl sulfate into solution.

The loaded ammonium sulfate solution is then transferred to precipitation, where the addition of ammonium hydroxide causes the precipitation of ammonium diuranate from solution. The precipitate is thickened, filtered and dried to produce a yellowcake ready for cooling, storage and packaging. The barren ammonium sulfate solution is then returned to solvent extraction for further loading.

Further testwork is required to make the final decision on the recovery technology and reagents. Should the recovery technology change from IX-SX as is currently assumed, it will have a significant effect on both the capital and operating costs associated with the plant.

17.2.4.2 IX PLANT VESSELS

For this PEA, an ion exchange plant (IX) is used to recover the uranium from the PLS. The IX system selected consists of approximately 12 stationary IX vessels that are rotated through the load-strip-rinse IX cycle, as shown in Figure 17-2. The position of each vessel in the continuous process will rotate, and rather than move the vessels themselves, as is the case in some IX plants, the rotating process is controlled by opening and closing selected valves. This manipulation of valves, combined with a supporting pipe network, facilitates sequential alternation of the liquids being pumped through a specific vessel thereby modifying a batch process into a continuous process.

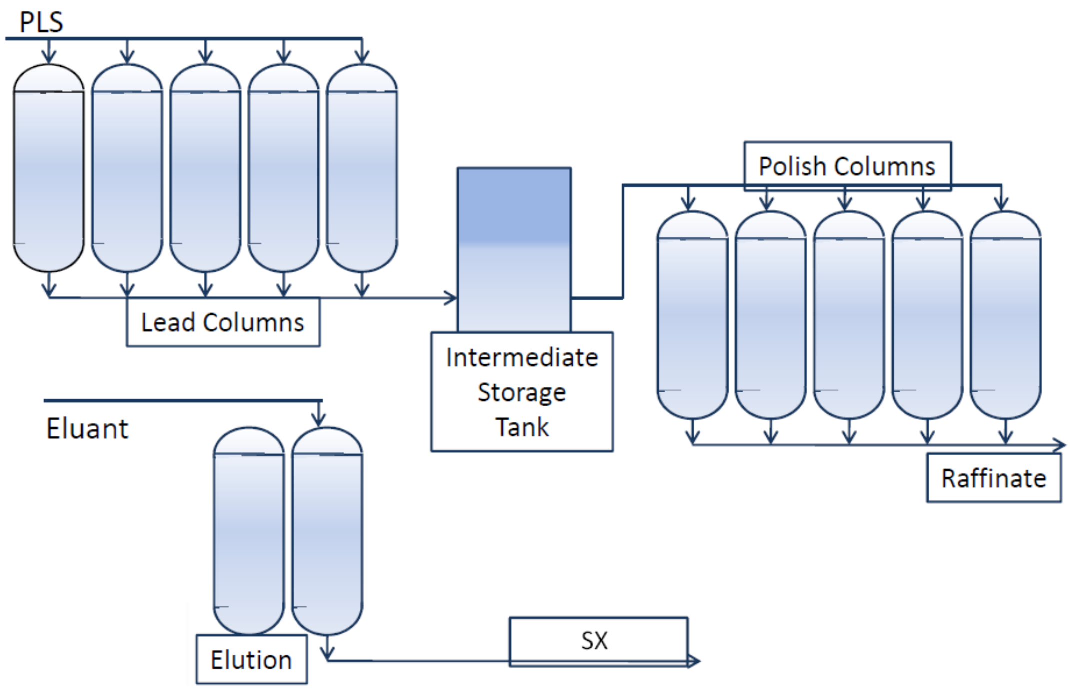


Figure 17-2: Ion Exchange Flowsheet (79)

17.2.4.3 IX RESIN LOADING

The equipment surrounding the IX process may be described by following the process sequence. First, the PLS is pumped by the pond pumps, from the PLS pond to PLS feed tank. This tank acts as a buffer between the on/off pond pumps and IX plant pumps. The pregnant solution passes through the IX vessels in a two stage adsorption process that loads the resin with uranium. The PLS is pumped from the PLS feed tank, through five IX lead columns (in parallel) by a centrifugal pump with a variable speed drive (VSD) installed to allow control of the pump and thereby the IX process. The solution leaving the lead columns is of lower tenor and is pumped into an intermediate storage tank. The solution from the intermediate storage tank is the pumped through five polishing columns (in parallel), which adsorb the remaining uranyl sulfate from the leach solution. The solution leaving the polishing columns is now BLS, which is then piped back to the BLS pond.

All pumps driving the IX process will have VSDs for process control purposes. The transfer piping will typically be HDPE pipe, although other materials may be used within the IX plant itself.

17.2.4.4 IX RESIN STRIPPING

Two loaded IX vessels at a time will then be put into a stripping stage. This stage is also referred to as elution since uranium is being eluted from the loaded resin. The loaded IX resin is stripped of uranium, which is accomplished by pumping a strong sulfuric acid solution through the vessel. The strong acid solution is referred to as the eluant and is between two and four molarity in concentration. The concentrated sulfuric acid is dosed from the reagents area by a peristaltic pump, into a mixing tank where it is mixed with the process solution returning from the downstream SX unit. The addition of acid will make-up the solution to the target acid concentration. From the mixing tank VSD controlled centrifugal pumps will pump the solution through the appropriate IX vessels according to the prevailing valve arrangement. The acid displaces the uranium, stripping it off the resin and back into solution. Optimum target acid concentration will be determined from IX testwork.

After the elution stage is complete, the IX columns are ready to be returned to the loading stage, where they will first adopt polishing duty before progressing to lead duty. This sequence allows the maximum recovery of uranium from the solution in the intermediate storage tank.

17.2.4.5 SOLVENT EXTRACTION

Loaded eluant from IX is pumped to the solvent extraction (SX) feed tank. The uranium is adsorbed into the liquid organic phase (the solvent) leaving the barren acidic eluant to be recycled back to IX. The loaded solvent is stripped of uranium through contact with the aqueous ammonium sulfate stripping solution. The loaded ammonium sulfate solution proceeds to precipitation, whereas the barren solvent is recycled for further extraction.

17.2.4.6 PRECIPITATION

The product from the SX plant is a concentrated solution containing dissolved uranium. This uranium is then recovered by adding ammonium hydroxide to precipitate the uranium, as ammonium diuranate, out of solution. The precipitation is completed in several precipitation vessels to balance size and resonance time. Agitators are installed to ensure the precipitated solids remain in suspension. The flow will launder between the precipitation tanks, and to facilitate the launders the tanks may be installed on a stepped foundation. Caustic is also added via peristaltic pump at this stage as the process of precipitation requires pH trimming to maintain precipitation under optimal conditions.

Once precipitation has occurred the mixture is laundered into a small thickener where the solids fall out of suspension. Depending on testwork results, a cone bottomed tank (without a rake) should fulfil the requirements for this thickening application and it has been included in the costing. The uranium salt is relatively dense and will drop out of solution at a high rate. Peristaltic hose pumps (duty + standby) are installed on the thickener underflow, pumping the underflow to a drum filter. The drum filter removes the excess liquids before the solids are fed to a kiln dryer.

All vessels and equipment in contact with the flow in this area of the plant will be constructed of corrosion resistant materials. For example, structures could be largely of duplex stainless steel such as LDX 2101, rubber lined carbon steel or other equivalent material. This will include the precipitation tanks, agitators, launders, thickener, drum filter and feeder.

17.2.4.7 DRYING

The filter cake is a wet yellowcake, and to dry the yellowcake ready for packaging, the liquids will be driven off in a rotary kiln stage. To accomplish this, the filter cake is heated to approximately 120 °C in a rotary kiln.

The kiln heater element will be of an electric type, which should be more efficient compared to a gas fired system at high altitude. Again, the kiln will be of corrosion resistant materials. Flow through the kiln will be controlled by a feeder. The hot gas from the kiln will be passed through a dust collection and filtration system to ensure no particulates or pollutants leave the system. The yellowcake will be allowed to cool before being packaged.

17.2.5 PACKAGING OF YELLOWCAKE

It is intended that yellowcake from the kiln dryer will fill a storage bin and as required, batches will be packaged into lined 44 gallon drums, using a feeder, ready for transit, after which the drums may be palletised and/or loaded into a container.

The packaging operation is straight forward although it will need to be contained within a negative pressure sealed building due to the nature of the product. The building would have an HVAC (heating, ventilation and air conditioning) system incorporating dust extraction and air filtration facilities to ensure no hazardous particulates or other emissions occur. Suspended powder would be extracted from the air and returned to the process via wet scrubbers. Typically the operators will also utilise specialised personal protection equipment, some of which (e.g. the outer layer) may remain in the building to ensure no contaminates are carried during entry/exit of the facility.

Dust reduction measures typical in packaging areas are as follows:

- Duct air intakes as required to contain/prevent dust from entering the workplace;
- Ventilate or enclose the drum filling operations and other operations handling yellowcake;

- Consider installing a pressure relief valve to the screw conveyor to vent dust generated during the transfer operations;
- Enclose, wherever possible, conveying systems or ventilate them at transfer points; and
- Isolate through physical separation, by walling off operations in one area of negative pressure, from the rest of the plant.

Dust reduction and control measures required to be employed for the handling of yellowcake are similar to those techniques typically in current industrial use for the handling of other powdered heavy metals. For example, lead dust used as a paint ingredient at paint factories and/or other similar applications.

17.3 PROCESS MATERIALS REQUIREMENTS

17.3.1 PROCESS MATERIALS / REAGENTS

17.3.1.1 LEACHING – SULFURIC ACID

The BLS pond would be dosed with concentrated sulfuric acid to replace the acid consumed during PEM leaching and ion exchange. A concentrated sulfuric acid storage facility would be installed adjacent to this pond. Peristaltic pumps dose a controlled volume of concentrated acid via a short dosing pipeline to the pond. The acid would be dosed at the point at which the return channels enter the pond, which would in turn be located separately to the pump inlets so as to allow for mixing of the acid. Sulfuric acid mixing with water is an exothermic reaction producing substantial quantities of heat energy. This energy can be dangerous due to the heat release causing high liquid temperatures. To aid mixing and to reduce this hazard, concentrated sulfuric acid would be dosed to the water in a region of high fluid flow into a pond which has a large thermal mass, rather than dosing water into the concentrated sulfuric acid.

Based on a weighted sulfuric acid consumption rate, an average of approximately 185 m^3/day of concentrated sulfuric acid would be consumed by the leach operation. Three 1 000 m^3 storage tanks, holding approximately 5.5 days acid reserve when full, have been proposed. As steel is not entirely impervious to concentrated sulfuric acid, the storage tank structure would be of steel construction, with thickness specified to allow for corrosion. Steel is known to be an acceptable material if care is taken in the design of the tank and water (dilute acid) is not allowed into the tank. If required, galvanic protection or a liner may be installed to increase the steel tank life. Typically, steel tanks are less costly than constructing a tank from a more expensive fully impervious material. These tanks would be situated in a bunded area with truck access for refilling, and a pumping and piping network connecting the pond to the acid storage facility.

17.3.1.2 LEACHING – LIME

In the ponds area, lime powder would be added via a bag breaker in a batched process to an agitated mixing tank. Once adequate mixing had occurred, the solution would be transferred to the caustic storage tank ready for use in the process. Tanks have been sized such that a single batch sufficient for 24 hours operation could be prepared when the storage tank had been depleted to 50 %. The storage tank would also be agitated to maintain the lime in solution prior to its application to the recycle pond via a peristaltic pump.

17.3.1.3 RECOVERY – SULFURIC ACID

A secondary closed loop of sulfuric acid is proposed to be used as eluant for the IX process through an SX plant. Whilst the SX plant provides a high recovery, a sulfuric acid make up tank would be located in the vicinity to replenish any losses.

All considerations regarding delivery, handling and storage mentioned in Section 17.3.1.1 are applicable in this situation.

17.3.1.4 PRECIPITATION – AMMONIUM HYDROXIDE

Ammonia gas would be delivered to site in a liquefied state via tanker and transferred to site storage or ISO tank on a use and return rotation.

The gas would be bubbled through water in an ammonia solution plant to form ammonium hydroxide, which would be transferred to an ammonium hydroxide storage tank. Approximately 50 kg/h of ammonia gas would be used to make a 30 % solution of ammonium hydroxide. Approximately 180 kg/h of ammonium hydroxide solution would be delivered to stripping and precipitation via peristaltic pumps at a rate determined via pH control.

17.3.1.5 PRECIPITATION – CAUSTIC (SODIUM HYDROXIDE)

In the reagents area, sodium hydroxide pellets would be added via a bag breaker in a batched process to an agitated mixing tank. Once adequate mixing had occurred, this solution would be transferred to the caustic storage tank. Tanks have been sized such that a single batch sufficient for 24 hours operation could be prepared when the storage tank was depleted to 50 %.

The caustic tanks will be located in a separate bunded area as any contact with acid solutions can cause an explosive chemical reaction.

17.3.2 HIGH GRADE OPTION

The heap leach option will follow the same recovery method as outlined in section 17. Through the limited testwork to date a trend has been established that PEM at a higher grade will consume a reduced amount of acid per tonne. Therefore for all four cases the acid consumed will be reduced to 8.5 kg/t.

The tank leach processing option includes crushing, milling and leaching followed by dissolved uranium recovery, precipitation, drying and packaging.

The PEM is mined, crushed and conveyed to the coarse PEM stockpile. The coarse PEM enters the milling circuit for primary grinding, secondary grinding and hydrocyclone classification to produce a tank leach feed having a density of approximately 45 % solids and a P80 of approximately 150 µm.

The PEM pulp is acid-leached in a short train of leach tanks for up to four hours. The slurry of pregnant leach solution and barren solids enters the resin-in-pulp (RIP) circuit. Ionic resin is moved counter-currently to the pulp in a train of approximately six RIP tanks. The resin adsorbs the uranyl sulphate ions from the solution to recover the uranium, leaving the barren pulp ready for thickening, filtration and disposal in the tailings management facility.

The loaded resin is stripped in an ion exchange (IX) circuit. The stripped resin is regenerated before return to the RIP circuit. The pregnant eluate from the IX circuit feeds to the precipitation circuit where ammonium diuranate is precipitated out by the addition of ammonium hydroxide. The precipitate is filtered to yield a yellowcake ready for drying, packing and dispatch.

Metallurgical testwork suggests a processing recovery rate of 93 %, resulting in an average annual production of 5.1 M lb U_3O_8.

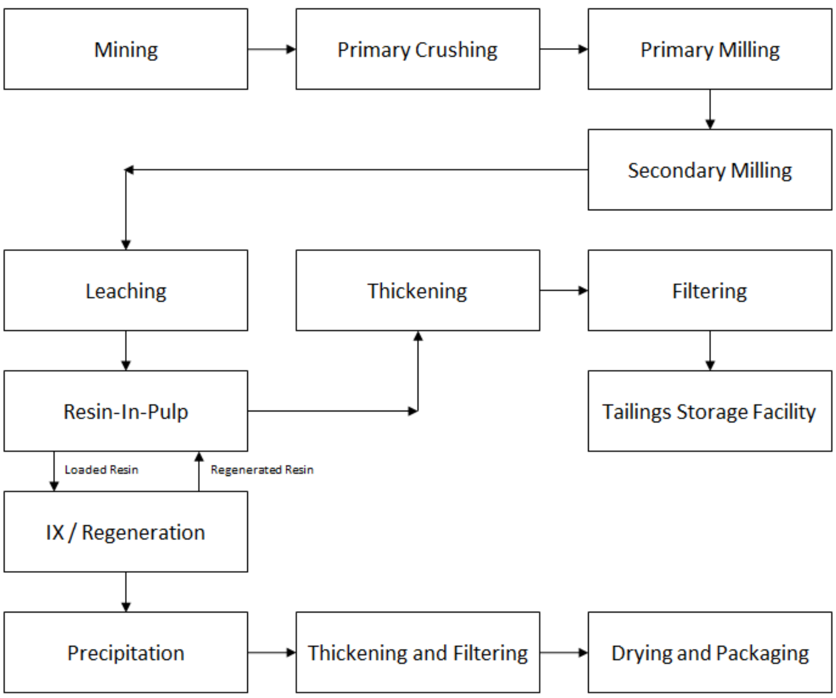


Figure 17-3 Resin in Pulp Typical Process

SECTION 18 PROJECT INFRASTRUCTURE

18.1 WATER DISTRIBUTION

The raw water distribution system designed includes eight pumps, raw water surge tank, and a duty and standby raw water transfer pump. It is proposed for the raw water to be pumped from the water source in the valley up to a raw water dam via HDPE piping, where possible, so as to maintain consistency with other systems on site.

18.1.1 WATER TREATMENT

Water treatment facilities would be required on site and capital expenditure and operating costs have been estimated and included.

The raw water would generally be run through a sand filter (or equivalent) with sterilisation for potable water. Water softening and/or demineralisation may also be required for some process streams depending on the water quality. The raw water supply is assumed to be of good quality such that no expensive treatment will be required.

18.2 POWER DISTRIBUTION

18.2.1 GRID CONNECTION

The San Gaban power line runs near the proposed power plant location and it has been assumed that the power line is at 138 kV. It is reported that the hydroelectric power installed capacity is 110 MW.

In order for a grid connection to be made an extension of the power line will be required to reach the project site and any connection will be subject to negotiation with the supply authority. These matters will need to be taken into account as the project progresses.

A 33/11 kV substation has been included at the site perimeter to connect to the grid.

18.2.2 DESCRIPTION OF PLANT SUPPLY AND DISTRIBUTION

The proposed distribution system consists of a 138/33 kV substation that would connect to the San Gaban powerline with a 33 kV aerial cable used to connect the substation to the 15 MVA 33/11 kV main transformer. This transformer would connect into a medium voltage (MV) switchgear at 11 kV (or other suitable MV level) which would distribute power throughout the site.

It is proposed that process area consumers would be fed by an MV cable with distribution transformers and a low voltage (LV) switchgear. Skid mounted containerised switch rooms could be used to decrease the installation time.

MV cabling to field areas would be direct buried to reduce access restrictions around the site, with the exception of the MV cabling to the bridge stacking and reclaim unit and the mobile crusher locations. The cabling to the bridge reclaim unit would be a flexible trailing cable that winds in and out on a cable drum to allow the bridge reclaim unit to move along the length of the leach pad. The cabling to the mobile crusher locations would be run along cable supports mounted on the overland conveyor structures. Allowance has been made in the conveyor design for the extra weight of the cables.

It is proposed that each substation consist of a containerised switch room with a transformer, switchgear and MCC. Low voltage motors would be supplied from the low voltage switchgear, while medium voltage would be used for motors generally larger than 250 kW.

Roadway lighting would also be required for 24 hour operation.

18.2.3 EMERGENCY POWER

Two 1.3 MW diesel powered generators for the provision of emergency power have been included in the estimate. Whilst the system is designed to be failsafe from an environmental and process point of view, the operational impact of a greater than 24 hour power failure would be significant due to the long leach start up cycle. The backup generators are only capable of supplying the critical loads for the operation. Loosely, these critical loads are:

- Emergency lighting and services;
- Leaching pumps; and
- IX and recovery.

The supply from the backup power generators is interlocked with the supply from the 33 kV overhead lines to remove the potential for the unsynchronised power sources to be connected together. Changeover to the backup power generators on loss of power from the normal power supply would be manually handled by the plant operators.

18.3 TRANSPORTATION AND LOGISTICS

Most equipment and materials will arrive containerised from overseas at Callao (Peru's main port, located near Lima) or a more southern port such as Ilo with suitable handling facilities. Containers would be offloaded at site using site based lifting equipment.

Containers would be loaded on trucks at port, and driven to site. Route surveys from port(s) to site, with accurate height and width measurements for any low bridges and abnormal load clearances, should be undertaken to determine the best road transit route. The route is likely to include access via the Interoceanica Highway, from the city of Juliaca to the town of Macusani (about 200 km).

18.3.1 ROADS

18.3.1.1 ACCESS ROADS

The connecting roads between the highway and the project site require significant upgrade and even perhaps rerouting to handle the proposed project generated traffic. The cost of these upgrades has been estimated at a desktop study level. A civil engineering surveyor, preferably a local contractor, will be required to assess the cost of these upgrades in greater detail. Two roads allow access to the site from the Interoceanica Highway which is intended to allow for future flexibility such as a one way system; heavy vehicle / light vehicle separation; or primary road with back up road in the event of road blockage such as a landslide.

18.3.1.2 INTERNAL SITE ROADS

In addition to standard access roads provided around the pumping areas on the plateau, a one-way light vehicle maintenance access road would be provided down the valley to the liquid tailings dam/IX area. The return solution pipeline would utilise the civil preparation of this road and follow it back up to the plateau.

There would be road access to the ammonia solution plant, including an adjacent truck bay, as the ISO tank is topped up 8 to 10 times daily on average, assuming 30 t tanker truck loads of liquefied ammonia delivered from the ammonia supplier.

There would also be road access to the sulfuric acid tanks including an adjacent truck bay as the tanks would be required to be topped up 11 to 12 times daily on average assuming 30 tonne tankers of concentrated acid delivered per day.

18.3.1.3 TRIUNFADOR 1 HAUL ROAD

There is an existing road between the Triunfador 1 mining concession and the proposed process facility. For the purposes of this PEA it has been assumed that the existing road could be modified to accommodate infrequent haulage from the Isivilla Complex. The PEM would be transported to the ROM Pad at the Colibri Complex for crushing.

18.3.2 CONVEYORS

18.3.2.1 RECLAIM CONVEYORS

The reclaim operation is intended to link to the tailings management facility via an overland conveyor that would receive the waste PEM from the reclaim corridor conveyor at the edge of the designed heap leach pad turnaround area.

The point of discharge of tails is proposed to be centrally located at the edge of the tailings area where the PEM would be stacked into the shallow wide valley, forward advancing using grasshopper link conveyors as the valley fills up.

18.3.2.2 KIHITIAN CONVEYING

The Kihitian mining area is approximately 8.4 km east-southeast from the processing area and due to the nature of the terrain in the region, a desktop analysis was performed to determine the most economic method of transporting PEM from the Kihitian property to the processing area.

The terrain along the plateau between the Quebrada Blanca deposit and the processing area has low elevation variation and therefore, based on previous experience it was considered that conventional overland conveying would be the most appropriate solution to transport PEM along the valley up to the process area. However between the valley and the Chilcuno Chico underground mine there is mountainous terrain and thus the following options were reviewed:

1. Primary crushing plant at Quebrada Blanca.
    a. Owned haul fleet.
    b. Leased haul fleet.
2. Primary crushing plant at edge of the Chilcuno Chico underground mine.
    a. Conventional covered conveyor.
    b. Pipe conveyor.
    c. RopeCon® conveyor.

It was determined that positioning the crushing station on the plateau and working in conjunction with the conventional conveyor to transport PEM to the processing facility would be the simpliest and most cost effective. The station has been positioned between the Isivilla and Kihitian complexes, and as the mine is developed the waste materials will be used to develop the haul roads and reduce the impact on haul distances.

18.4 SITE SERVICES

18.4.1 FUEL SUPPLY, STORAGE AND DISTRIBUTION

Fuel storage and dispensing facilities would be required to supply primarily diesel to various mobile equipment associated with the processing plant.

The fuel consumption of the vehicles associated with the process facility would typically be much less than that of the mining fleet, and the fuel facilities would typically be shared. A supply contract would normally be negotiated with a fuel company and such a contract could include equipment supply and installation.

A single storage facility with double walled steel fuel tanks in a bunded area would be used to store fuel. Two dispensing facilities are allowed for, one suitable for the mining fleet, and a second smaller facility suitable for the plant operations. Fuel delivery tankers (30 m³ in size) would supply fuel on an as required basis to meet the project consumption requirements.

Oil is intended to be stored in the drums in which it is delivered and dispensed as required near the storage facility, or mobilised on a utility vehicle to service the loaders in the field.

Due to the current basis that the mine will by contractor all open pit mining fleet facility will be part of the contractors scope.

18.4.2 COMPRESSED AIR

A compressed air system has been included. The system includes a duty and standby oil lubricated screw compressor, each with approximately 100 L/s capacity operating at 7.5 bar(g). The system includes an air filter, dryer, condensate removal valve and receiver tank. Note that oil lubricated compressors have been specified as the cost of oil-free compressors would be approximately double. The system has been sized based on a previous project for costing purposes only. Whilst the high atmosphere is a de-rating factor, the plant is not assumed to be an abnormally high compressed air consumer.

18.4.3 WORKSHOPS AND WAREHOUSES

Fixed plant and mobile plant mechanical workshops are included in the estimate. The mobile plant workshop would be suitable for servicing the mining fleet, skid steers etc., whilst the fixed plant workshop would be suitable for other plant equipment such as pumps, gearboxes etc.

A warehouse and storage facility is included to receive and store all incoming goods for the site, except for bulk deliveries of some plant consumables such as sulfuric acid and fuel. A secure, fenced lay down area for containers or other goods that are suitable for outdoor storage would be located around the warehouse building.

18.4.4 COMMUNICATIONS

It is assumed that Plateau Uranium would acquire the necessary local permits for communications connections to site. These permits would normally be obtained from the local telecoms authority, or cell-phone provider.

If the area is on the cellular grid, the best solution may be communications (including internet) via the cell phone provider. The cost for this option depends on the terrain, which dictates the equipment required and the rates/costs of the local communications provider who may use a microwave link.

Another option is to run a fibre optical cable along the power line. This would be done in agreement with the communications provider, since this line would link into their equipment, and they would have to be responsible for the line and its maintenance. From site to the town of Macusani, the distance for the communications link would be approximately 20 km to 30 km, depending on the route selected.

If the above options are not available, a satellite phone and internet link-up may be the only other option. This can be costly, but it does enable the site to be independent of local infrastructure and this can sometimes justify the cost.

The existing project office in the nearby village of Isivilla is connected with broadband internet and telephone systems. Therefore it is assumed no significant new communications infrastructure would be required in the capital cost estimate and the connection fees are included as an operating cost.

18.4.5 OFFICE FACILITIES

An administration building is included in the estimate. Internal furnishings and facilities costs have been allowed on a per metre squared turn-key cost basis.

18.4.6 EMPLOYEE HOUSING

A camp for project operations staff has been allowed for in the costing, to be located on the Macusani plateau, likely at site, but perhaps near the villages of Isivilla or Tantamaco. It would be suitable for 330 people.

A camp would be required as neighbouring villages, such as Isivilla and Tantamaco, are currently too small and lack sufficient facilities. Commuting from the larger town of Macusani would become more feasible after the required road upgrades connecting the site to the Interoceanica Highway were completed. It is envisaged that the camp would be constructed quite early in the project to support the project capital construction works teams. Ablution blocks, central kitchen, mess hall and changing room facilities have been included in the cost estimate.

18.5 HUMAN RESOURCES

An organisation structure for the project operations staff has been developed to support the project operating cost estimate as shown in Figure 18-1. This organisation structure does not show mining staff. It is anticipated that a total staff of between 367 and 577 would be required for the project including at maximum 276 mining operations staff.

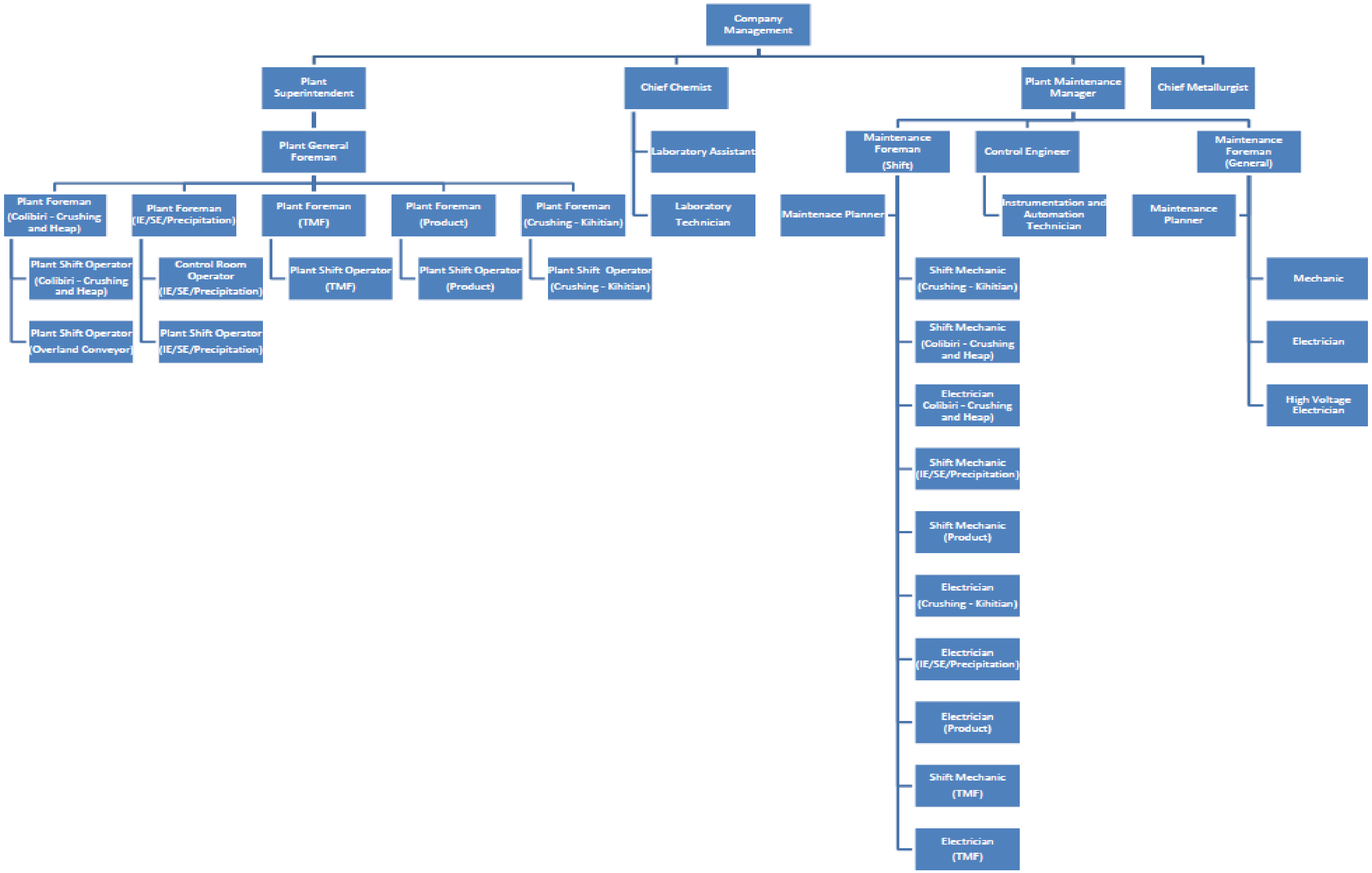


Figure 18-1: Operations Organisation Chart

SECTION 19 MARKET STUDIES AND CONTRACTS

The Qualified Person has reviewed all documentation referred to in this section. The Qualified Person is not aware of contracts in force with Plateau Uranium or any associated companies. Peru is a signatory of the IAEA and so contracts will need to conform to the IAEA standards.

19.1 INTRODUCTION

It is noted that yellowcake is not a high volume product requiring custom logistics chains. The product shipments will be able to use existing logistics chains. Therefore, subject to confirmation studies, the product would likely be marketable worldwide to any potential customer.

Primary supply at the mine level has been notably below demand levels for years and is forecast to continue in 2015 (156 M lbs. of production vs. 179 M lbs. of demand). Despite the primary supply deficit, the pipeline of new uranium mines that can be developed within the next five years is very small since the low uranium price environment has not incentivized much uranium exploration or development (80).

19.2 INTERNATIONAL SUPPLY AND DEMAND

With 438 reactors in operation around the world, the total generating capacity of nuclear energy reached 376.2 gigawatts (electrical) (GW(e)) at the end of 2014. During the year, five reactors were connected to the grid, one was permanently shut down and construction started on three reactors. Asia remained the centre of near and long term growth prospects, accounting for 46 of the 70 reactors under construction.

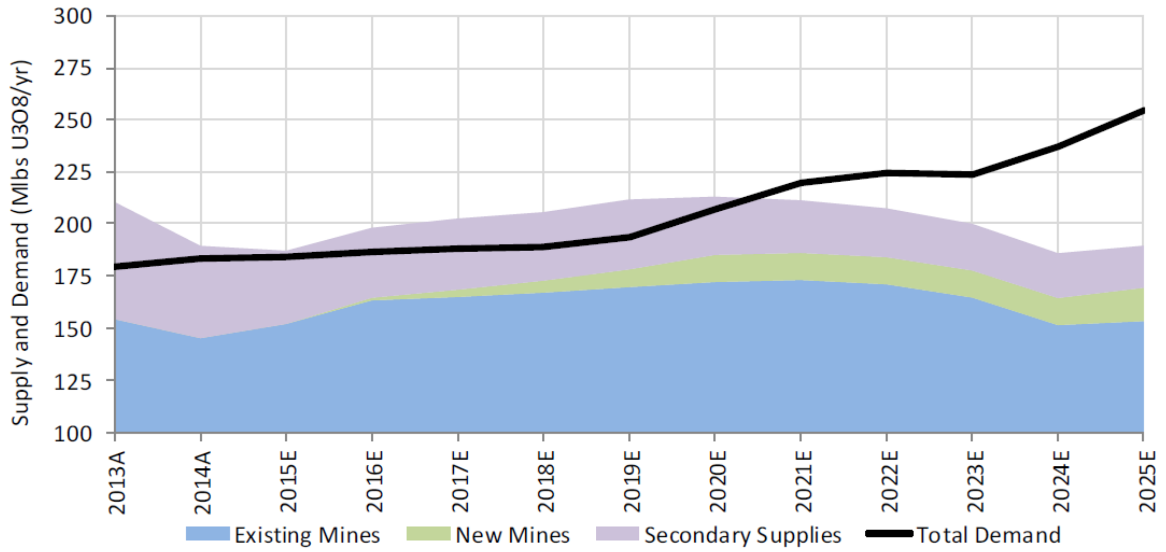
The IAEA's projections for 2030 showed an increase in global nuclear power capacity of 8 % in the low case and 88 % in the high case scenario. These projections were lower than those of 2013, mainly due to earlier than anticipated retirements of plants and a reduction in the number of planned new plants in some countries. Nevertheless, interest in nuclear power remains strong in some regions, particularly in countries with fast growing energy needs (81).

Over the last two decades, global electricity demand has grown 3 % per annum, with nuclear power now supplying some 11 % of global electricity generation and 21 % in OECD countries (82). France (75 %) and Slovakia (54 %) lead Belgium, Ukraine, Hungary, Sweden, Slovenia, Switzerland, Czech Republic, Finland, Bulgaria, and the Republic of Korea, which depend on nuclear power for more than 30 % of their electricity needs. The United States has the most reactors of any country and its 104 reactors provide some 19 % of its national electricity generation capacity (81).

Key factors that could impact the semi-balanced market outlook include disruption and delays to existing and new projects, and the uncertainty over the quantity and commercial terms for the supply of highly enriched uranium (HEU) from decommissioned nuclear weapons beyond 2013. These declining secondary HEU supplies are anticipated to see mine production accounting for some 85 % of global uranium production by 2015.

Kazakhstan, Canada and Australia account for about 64 % of the world's current primary U_3O_8 production capacity with Niger and Namibia third and fourth rank producers, respectively (82). During 2010 and 2011, there was substantial consolidation and further concentration of U_3O_8 supply as downstream nuclear energy companies and current U_3O_8 producers sought to secure additional U_3O_8 assets in the exploration and development stage.

The U_3O_8 demand and supply dynamics for the last two years, and a forecast for the next ten years, are illustrated in Figure 19-1.



Source: Raymond James Ltd., WNA, IEA, UxC, NIW, Bloomberg, company reports

Figure 19-1: U_3O_8 Supply and Demand Dynamics (83)

19.3 U_3O_8 PRICING AND MARKET OUTLOOK

Most U_3O_8 is traded through long-term bilateral agreements between suppliers (principally mining companies) and users (principally nuclear power utilities), seeking security of supply. There is no terminal market for U_3O_8; with spot and long term prices being published by a small number of independent companies based on market events. The spot market price is the most widely quoted, although in recent years spot market transactions have accounted for only approximately 20 % of the total market. Conversion, enrichment and fabrication services are typically purchased separately by the nuclear fuel end user.

Subsequent to the Fukushima disaster in Japan, spot prices fell 32 % year-on-year to 49.00 USD/lb U_3O_8 subsequently stabilizing in the current range of 34.50 USD/lb U_3O_8 to 43.80 USD/lb U_3O_8 (80) as shown in Figure 19-2.

Metals Price Forecast	2014	2015	2016	2017	2018	2019	2020+
U_3O_8 Spot (US$/lb U_3O_8)	32.0	39.5	53.0	63.75	67.5	70.0	70.0
U_3O_8 Long-Term (US$/lb U_3O_8)	49.0	57.5	65.0	70.0	75.0	75.0	75.0

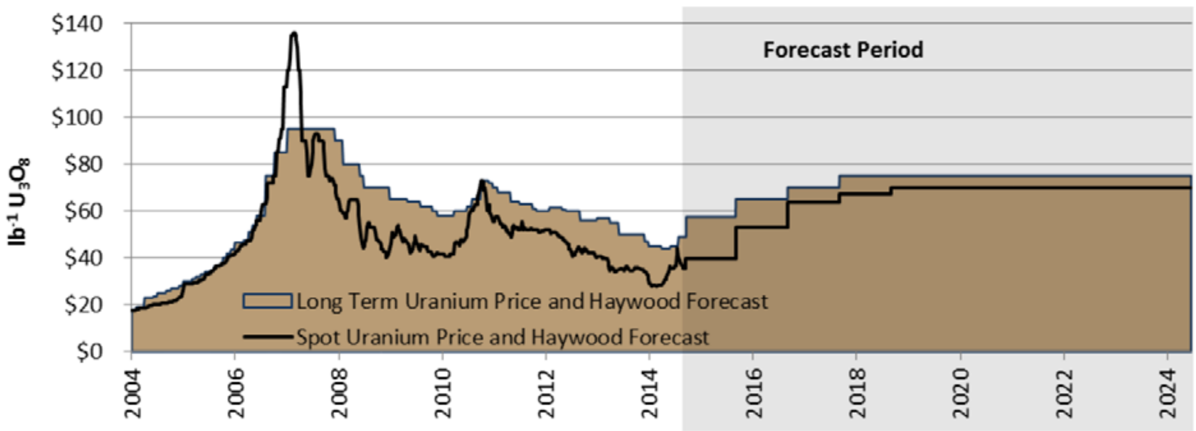


Figure 19-2 Past and Forecast Pricing per lb U_3O_8

SECTION 20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

Detailed environmental permitting and social impact considerations are not within the scope of a PEA. The design outlined in this section is preliminary and gaining more accurate costs will involve substantial further engineering definition, environmental assessment and cost analysis.

Much of the following information was sourced directly from the American Bar Association (84) and the Peruvian Institute of Nuclear Energy (85).

Due to the nature of the product, tailings management and mine closure may be a potentially significant item in terms of long term cost.

20.1 ENVIRONMENTAL STUDIES

20.1.1 ENVIRONMENTAL SETTING

20.1.1.1 BIOPHYSICAL (86)

The project is located within the ecoregion of Puna y Altos Andes. The environmental conditions of the region require morphological and physiological adaptations. Most species reproduce at the end of the dry season or summer, when it is less cold and with higher precipitation, allowing species to find more food. The vegetation consists of grasslands, semi-deserts, *Polylepis*[6], bushes and plants. Noteworthy species include the Chachacomo Tree, *Puya Raimondii*, Puna grasses, Tarhui and Chocho. Also, domesticated species such as potato, maca, cañihua, quinoa and amaranth are native to the area. Some native fauna found within the ecoregion that may or may not be present in the project area are classified by the International Union for Conservation of Nature as “vulnerable”, including the Andean Flamingo and North Andean Deer; “near threatened” including the Andean Condor, Andean Ostrich and Pampus Cat; and “critically endangered”, including the Junin Grebe (*Podiceps taczanowskii*) (87).

The project may or may not be located within the MEM Restricted Mining Areas and this should be verified with detailed cadastre (88).

A weather station located on site has recorded various physical parameters including wind temperature and precipitation (7).

[6] Genus of shrub and tree species endemic to the mid and high elevation regions of the tropical Andes

20.1.1.2 ARCHAEOLOGY AND CULTURAL HERITAGE

20.1.1.2.1 PRE INCA ROCK ART (89)

Approximately 100 sites of ancient rock art, some unique to the region are found within the Macusani and Corani Districts. This includes paintings and petroglyphs from various peoples spanning up to an estimated 10 000 years ago, (listed chronologically):

- Preceramic – hunting of camelids and / or deers;
- Kaluyo – Alpaca ancestors;
- Colla; and
- Inca – geometric shapes.

Collectively these cave paintings constitute the largest concentration of art from the Archaic period in the Americas and, in 2005, these sites were designated national cultural patrimony by the Peruvian government (RDN No. 2658/INC-2005). The specific locations of these artworks should be determined to confirm their proximity to the project. It is known some are located in close proximity to Isivilla as it forms a minor tourist attraction.

20.1.1.2.2 MINING (13)

The project site is located in proximity to four inactive or abandoned mines as listed below:

- Cerrochichirancane;
- Tanta Maco;
- Accopocro; and
- Chillumochico.

As well as reflecting recent cultural heritage in the area these mines have been identified as potential causes of local water contamination. A future study should locate the exact location and nature of these mines to determine whether they have any potential impact on the project.

20.1.1.3 SOCIO-ECONOMIC (90)

The project is located within the Macusani and Corani Districts of the Carabaya Province. The population of Peru is 29.8 million as of a July 2013 estimate with the population of the Corani District and Macusani District's being 3 581 and 10 950 respectively. The nearby village of Isivilla has a secondary school with 136 pupils. The official language of Peru is Spanish, however around 13 % of the population are Quechan speakers. It is likely that the people living near the project primarily speak Quechan de Cusco or Quechan de Puno dialects. Poverty is high in rural areas, with up to 55 % of the population living in poverty, and approximately a quarter to a third of Peruvian children aged 6 to 14 work.

The main sources of revenue in the Carabaya region are associated with raising alpacas, llamas and sheep and marketing of their products (grain, wool, meat), or revenue from mining activities. The Macusani and Corani Districts are renowned for their alpacas and llamas with the Macusani District having 10 % of the national population of the species and Corani being home to around 42 730 alpacas. Peru has more than 4 000 varieties of potatoes. Typically the local villagers are struggling small holding farmers. Very limited tourism occurs within proximity to the project.

20.1.2 PERUVIAN ENVIRONMENTAL STUDY REQUIREMENTS

Generally, the Ministry of Energy and Mines of Peru (MEM) requires exploration and mining companies to prepare an environmental evaluation, an EIA, an Environmental Management Plan (EMP), and a closure plan as shown in Figure 20-1. Mining companies are also subject to annual environmental audits of operations. Additionally, laws specifically relating to the mining of uranium (91) (92) (93) apply and include conditions of the Peruvian agreement with the International Atomic Energy Agency (IAEA) and the North Atlantic Treaty Organisation’s (NATO) regulations.

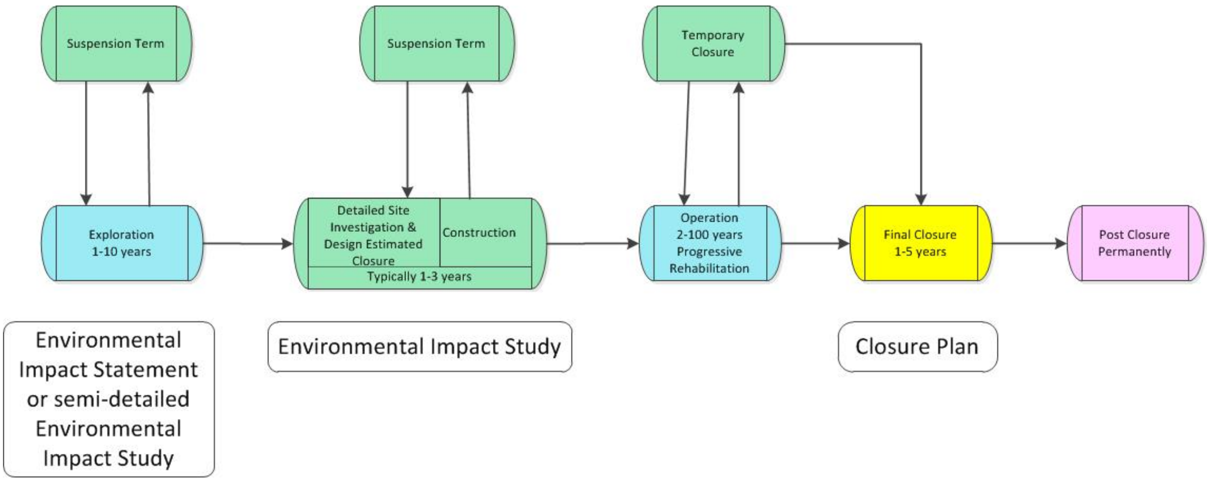


Figure 20-1: Life of Mine Legislated Required Environmental Studies (94)

20.1.2.1 ENVIRONMENTAL REQUIREMENTS FOR EXPLORATION ACTIVITIES

The Peruvian Supreme Decree 037-98-EM, established a tiered system imposing environmental requirements for exploration activities. A company must file an affidavit with an environmental evaluation for the following categories of exploration activities:

- Category A: Exploration activities with minimal or no surface alterations, such as geological studies, topography, and recollection of samples. The applicant must file an affidavit describing the activities.

- Category B: Exploration activities that result in discharges or waste disposal that could impact the area up to no more than 10 hectares. The company must file an application form attaching a description of the activities, a description of mitigation activities and reclamation plans.
- Category C: Exploration activities that result in discharges or waste disposal, that could impact the area and those exploration activities that require more than 20 platforms or construction of more than 50 m of tunnels. The latter require an environmental evaluation. MEM published a guidance document describing the contents of an environmental evaluation.

20.1.2.2 IMPACT ASSESSMENT STUDIES

As of April 1993, a mining company that has completed its exploration stage must submit an EIA when applying for a new mining or processing concession, or to increase the size of its existing processing operations by more than 50 %; or to execute any other mining project. The duty to prepare an EIA falls upon the concession holder and the EIA must be prepared by a consultant duly certified by the Ministry. Each EIA must include an analysis of the cost benefit and any alternatives. The EIA must include plans for expenditures on an environmental program representing no less than 1 % of annual sales. The impacts and control sections of the EIA must generally describe with specificity the measures to control air and water pollution, suppress noise, protect sensitive ecosystems and properly dispose of waste, which may then be incorporated into the permits issued by other authorities. With respect to mitigation, a description of air emissions controls, water treatment and reclamation is also required.

20.1.2.3 EMP AND CLOSURE MANAGEMENT PLAN

An EMP must be submitted to the MEM for permitting of an operation.

Additionally, a closure management plan for each component of the operation must be submitted. The closure plan must outline what measures will be taken to protect the environment over the short, medium and long term from solids, liquids and gases generated by the mining operation.

20.2 ENVIRONMENTAL PERMITTING

Various permits will be required for the Macusani operation. An investigation of the specific permitting requirements will be required as the project is developed further. The following references are provided:

- National Institute of Concessions and Mining Cadastre: this Peruvian institution is responsible for minerals permitting;
- The MEM: an entity of the Peruvian government responsible for managing the energy and mining sectors of Peru;
- As a uranium mine, regulations may be imposed by the IAEA;

- Law No. 28611: General Environmental Law;
- Law No. 27446 and Supreme Decree No 019-2009-MINAM: Law of the National System of Environmental Impact Assessment and Regulations: Supreme Decree No. 016-1993-EM and technical changes (DS-EM 059-93 and 058-99 ds-em): Environmental Regulation Protection of Mining and Metallurgical Activities and Modifications;
- Law No 28090 and Supreme Decree No. 033-2005-EM: Mine Closure Regulations;
- Supreme Decree No. 020-2008-EM: Environmental Regulations for Mining Exploration Activities;
- Supreme Decree No. 028-2008-EM: Regulation of Citizen Participation in Mining Subsector;
- Law No 27651: Supreme Decree No. 013-2002 Supreme Decree No 051-2009-EM: Formalising and Promotion Act of Small-Scale Mining and Artisanal Mining Regulations;
- Ministerial Resolution No 167-2008-MEM/DM: Approval of Terms of Reference for Common Mining Exploration Activities Category I and Category II;
- Ministerial Resolution No. 304-2008-MEM/DM: Adoption of Rules Governing the Citizen Participation Process in Subsector Mining;
- Law No. 28271. S.D. No-EM 059-205 and Modification (DS 003-2009 No-EM: Environmental Law) Mining Activities Regulation and Modifications; and
- Supreme Decree No: 078-2009-EM.

The requirements of these laws are as summarised in Table 20-1.

Table 20-1: Environmental Permitting Requirements (94)

Requirements	Large & Medium Mining (GMM)				General		
	Exploration		Exploitation	Remediation	Closure		
	Environmental Impact Statement (EIS)	Semi-Detailed Environmental Impact Study (EIAsd)	Environmental Impact Study (EIA)	Remediation Plan (PRA)	Termination Plan (Process / Installation)	Closure Plan Environmental Liabilities	Mine Closure Plan
General Requirements							
According Format Request	✓	✓	✓	✓	✓	✓	✓
Printed & Digital Copies	All for Environmental Assessment System Online (SEAL)			2	2	5	5
Executive Summary	✓	✓	2	2	2	5	5
Authenticated or certified copy evidencing the Legal Representation	✓	✓	✓	Demonstrate to tax department that bond cannot be withdrawn	Submit special bond report to the tax department	✓	✓
Submit to DGIA (MINAG)	-	-	✓	-	-	-	-
Submit to SERNANP	Only if you are in a protected National area						
List of professionals involved in the preparation of studies, including; full name, qualification and company	✓	✓	Registration of consultancy enabled	Registration of Consultant enabled by PC	✓	✓	If properly registered Consultant GMM enabled
Payment	30% UIT	40% UIT	100% UIT	100% UIT	100% UIT	GMM 100%, PPM 80% and PMA 70% UIT	
Requirements: D.S. 028-2008-EM y R.M 304-2008 – MEM/DM (Citizen Partnership)							
Summary of actions	✓	✓	-	-	-	-	-
Citizen Participation Plan	-	-	2	-	-	-	-
List of local authorities and surface land holders	✓	✓	-	-	-	-	-
Previous Workshop	✓	✓	Minimum 1 before and 1 during the development of the study	-	-	-	-
Copy of documentation supporting the completion of at least one workshop including records, photos, videos or other	✓	✓	✓	-	-	-	-
Protocol relationship	✓	✓	✓	-	-	-	-
Submit to DRAM	✓	✓	After approved Citizen Participation Plan and Executive Summary	✓	✓	Just to Feasibility Level	
Submit to Provincial and District Municipality	✓	✓		✓	-	-	-
Submit to Local Communities	✓	✓		✓	-	-	-
Requirements S.D. 020-2008-EM and RM 167-2008-MEM/DM (Exploration Activities of Large and Medium Mining)							
Program Summary (Annex III)	✓	✓	-	-	-	-	-
Affidavit format Law No. 29060	EIS will only be for automatic approval	-	-	-	-	-	-
Letter signed by the professional responsible for environmental management and legal representative	✓	✓	-	-	-	-	-

20.2.1 LICENCE TO EXTRACT URANIUM

As a uranium operation, the project is classified as Category 1 under the Radiation Safety Regulations (92). Therefore, a licence to extract uranium is required where the uranium has a specific activity greater than or equal to 1 BQ/g. NATO must be advised of various stages of the operation for approval under these regulations, including but not limited to notification of:

- Means of product transport;
- EIA and proposed closure plans;
- Emergency Action Plans;
- Production; and
- Sales (NATO authorised parties).

Additionally, for operations some personnel may require NATO licences to perform their specific tasks (91).

20.3 SOCIAL, COMMUNITY AND ENVIRONMENTAL IMPACTS

20.3.1 STAKEHOLDER ENGAGEMENT

In January 2001, MEM published guidance on the management of relations between companies and local communities ("Guía de Relaciones Comunitarias"). It describes the Social Impact Study ("Estudio de Impacto Social") now required as a part of the EIA. The Social Impact Study consists of an analysis of the impacts on persons, interpersonal relationships, economy and culture in the communities living in the area of influence, resulting from the mining operation. The plan also includes mitigation measures to reduce such impacts.

Regarding public participation in the EIA approval process, the law requires a single public hearing to take place and makes the EIA a public document, meaning the applicant must make it available to the public.

20.3.2 IMPACTS – SOCIAL AND ENVIRONMENTAL

An environmental study is required to be completed to fully understand the potential social and environmental impacts due to the implementation of the project. Details of some potential impacts are briefly described in this section.

20.3.2.1 POSITIVE IMPACTS OF PLATEAU URANIUM IN THE MACUSANI REGION

Plateau Uranium is working to engage and develop the local community to facilitate this process and have undertaken various community programs over the years including:

- Twice yearly medical campaign;
- Employment of local community members (40 from Isivilla, Tantamaco and Corani);
- Hygiene programs (water sanitation);
- Monthly madre leche contribution; and
- Schools programs sponsorship.

20.3.2.2 HEALTH OF WORKERS (95)

20.3.2.2.1 RADIOACTIVITY

Uranium mining and processing operations should be undertaken under an appropriate recognised code of practice for radiation protection and waste management.

While uranium itself is only slightly radioactive, radon, a radioactive inert gas, would be released into the atmosphere in very small quantities if the rock was mined and crushed. Radon is one of the decay products of uranium and radium. It occurs naturally in most rocks and minute traces of it are present in the air.

Gamma radiation may also be a hazard to those working close to mineralised material containing a high concentration of uranium. It comes principally from radium in the rock, so exposure to this is regulated as required. In particular, dust is suppressed, since this represents the main potential exposure to alpha radiation as well as a gamma radiation hazard.

At the concentrations associated with uranium (and some mineral sands) mining, radon is a potential health hazard, as is dust. Precautions taken during the mining and processing of uranium to protect the health of the workers include:

- Good forced ventilation systems in underground mines to ensure that exposure to radon gas and its radioactive daughter products are as low as possible and do not exceed established safety levels;
- Efficient dust control, because the dust may contain radioactive constituents and emit radon gas;
- Limiting the radiation exposure of workers in mine, processing and tailings areas so that it is as low as possible, and, in any event, does not exceed the allowable dose limits set by the authorities. For example, in Canada, mining in mineralised material containing very high concentrations of uranium is undertaken solely by remote control techniques and by fully containing that material where practicable;
- The use of radiation detection equipment in all mines and plants; and
- Imposition of strict personal hygiene standards for workers handling ammonium diuranate concentrate.

At any mine, designated employees (those likely to be exposed to radiation or radioactive materials) are monitored for alpha radiation contamination and personal dosimeters are worn to measure exposure to gamma radiation. Routine monitoring of air, dust and surface contamination is undertaken.

If ammonium diuranate is ingested it has a chemical toxicity similar to that of lead oxide. Similar hygiene precautions to those in a lead smelter should therefore be taken when handling uranium and / or yellowcake in the drying and packing areas of the plant.

20.3.2.2.2 ALTITUDE

Altitude sickness is a health concern for persons at the given project altitudes. It is anticipated that locally sourced labour or those living at high altitude will require minimal if any acclimatisation. However personnel living at low altitudes who stay at low altitudes for significant time between rotations will require acclimatisation for each period spent at site.

20.3.2.3 ACID ROCK DRAINAGE AND METAL LEACHING

To date no testing that would inform the risk of acid rock drainage or metal leaching has been undertaken. A suite of testwork would enable the risk presented by overburden, waste rock, sub-economic rock and tailings to be determined. It is recommended that the majority of this testwork be completed as part of an environmental study.

20.4 WASTE MANAGEMENT

20.4.1 SOLID TAILINGS

In this PEA the solid tailings are considered to be the spent PEM which is deposited via conveyors to the proposed tailings management facility. It has been assumed that the tailings will retain 6.5 % moisture content; up from an assumed PEM moisture content at feed of 5 %.

20.4.2 TAILINGS AND RADON

Solid waste products will be composed of most of the original PEM and, therefore, will also contain most of the radioactive elements within the PEM. In particular, solid tails contain the radium that is present in the original PEM.

When radium undergoes radioactive decay one of the products is radon gas. Radon and its decay products (daughters) are radioactive and as tailings material is on the surface, measures would need to be taken to minimise the potential emission of radon gas. The radon gas emanates from the tailings as the radium or thorium decays. It then decays itself to (solid) radon daughters. Radon occurs in most rocks and traces of it are present in air. However, at high concentrations it is a serious health hazard. It is proposed that a monitoring station be sited at the tailings management facility.

Clay or other suitable capping material may be placed to cap the tailings pile as it advances or at certain stages during LOM in an attempt to reduce radiation levels to near those normally experienced in the region of the PEM body. Capping of the solid tailings dump(s) with a two metre deep layer of fill is included in the cost estimate.

20.4.3 LIQUID TAILS

The process facility will produce liquid tailings but in an effort to minimise the liquid tailings, it is envisaged that the leaching solution circuit would contain a bleed stream to be drawn off post IX loading (raffinate stream). This process bleed is assumed to require a treatment plant, potentially an IX plant, with the outflowing fluid reporting back to the BLS pond to prevent build-up of impurities in the process liquids. As detailed analysis of barren solution has not been completed, the concentration or makeup of impurities is unknown. It is likely that this treatment plant will require specialised resin different to that of the main IX, and potentially other methods of treatment may be required.

Further environmental analysis will determine if the assumed process scenario will maintain pond levels through evaporation alone, and will require makeup water. For the purposes of the estimate, it has been assumed that there are no outflows from the process. Any treated water will be pumped into the BLS pond as process makeup water.

In addition, water flows through and from the tailings heap are proposed to be collected in a liquid tailings dam constructed at the lowest elevation area of the site, below the tailings heap. Further environmental analysis may determine that this dam will maintain a level through evaporation alone. However, for the purposes of the estimate it has been assumed that a pumping facility would be required to deliver a volume of 5 m^3/h up to the BLS pond on the plateau. It is likely the fluid will be heavily diluted and would be progressively cleaned via the means outlined. It has been assumed that manual methods to clean the liquids pond of settlement build up would be undertaken and disposed of on the main tailings management facility.

20.4.4 TAILINGS MANAGEMENT FACILITIES

All tailings management facilities would be lined with an impervious layer to minimise environmental discharge from the site over time. For costing purposes, this has been assumed as a clay layer with a 2 mm HDPE rough liner. Impermeable clay in the form of weathered/compressed rhyolite from the plateau is considered to be suitable. However, testwork and sampling will be required to confirm quality and quantity for future studies. To meet the ground conditions required for laying the liner, site preparation will be necessary to clear and smooth the entire area suitable for loading. Typically this may entail ripping/blasting the rocky outcrops and grading the area. Local rock porosity, slip resistance and shear will need to be considered in future studies to determine the suitability of this lining option.

It is proposed that the tailings management facility will be located to the west of the site in an adjacent valley and is estimated to be able to accommodate a heaped pile of 99 Mm^3 (118.7 Mt). The tailings management facility is intended to be designed in a heap fashion, as shown in Figure 20-2, which is anticipated to be sufficient for the life of mine, assuming the final heap leach remains a permanent feature.

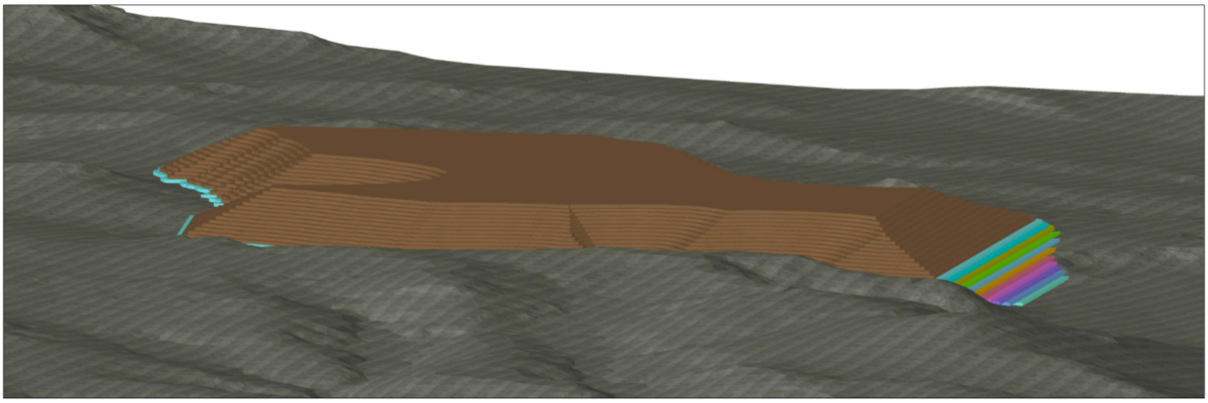


Figure 20-2: Tailings Heap Model

20.5 REHABILITATION AND CLOSURE

Due to the nature of a heap leach operation there will be a significant area of land that requires reclamation to some degree. A preliminary closure management plan should be prepared in the future to better delineate estimated costs of rehabilitation and closure. The following discussion describes considerations that were used for cost estimation of rehabilitation and closure.

- Currently an on/off leach pad is proposed, so it would be possible to remove spent PEM to a long-term storage facility – the tailings management facility described in Section 20.4 – during the normal operation of the plant. The tailings management facility is intended as a permanent long term feature.
- Dust control measures would be incorporated in the project design. Any potential dust on and around the site may require attention during site rehabilitation.
- The leach solution ponds and any remaining liquid effluents will require a rehabilitation plan. Depending on the recovery process selected, volumes of non-evaporative liquid may require disposal.
- For the mine/tails areas, at the conclusion of operations, allowance has been made to cover the solid tailings with enough clay to reduce both gamma radiation levels and radon emanation rates to levels near those occurring naturally in the region.
- It is anticipated that the access roads from the Interoceanica highway to the site would remain in place for local community use. All other access and haul roads would be ripped and re-graded, where required, to blend in with local topography. Safety berms and drainage infrastructure will be removed or graded, where applicable.
- For the purpose of this PEA, it is assumed that open pits, underground mines and waste rock dumps will remain as permanent features with egress routes maintained in the event of entry and stormwater diversions maintained around these features. All plant, infrastructure or

facilities are to be removed and ingress blocked. Limited re-grading has been allowed for to promote re-vegetation.

- A known issue associated with uranium mining is that all equipment in contact with the uranium becomes contaminated. For example, small amounts of uranium deposit on the inside of piping and pumps. This typically means all equipment must remain on site and be disposed of in an appropriate manner. In some cases equipment may be removed after being subject to a cleaning/treatment process. It is the usual practice to bury the used plant equipment at the tailings dump site and, therefore, this should be investigated and confirmed.
- Heap leach pad, ponds, culverts or similar are intended to be ripped and re-graded where required to blend in with local topography.
- Passive re-vegetation is proposed to promote soil stability and return of local species.

20.5.1 POST-CLOSURE MONITORING AND MAINTENANCE

An allowance for routine monitoring including personnel costs and laboratory feeds is included within the estimate. A closure management plan, including monitoring plan and time frame, will allow for improved accuracy of the rehabilitation and closure estimate. The monitoring system will also provide an early warning system to identify unforeseen issues post-closure.

20.6 DISCUSSION

It is once again noted that the above information is not representative of an exhaustive study of the environmental considerations for the project, and is provided for information only. The points raised are only those that have been noted during the PEA level design efforts associated with the various sections of the mineral processing facility. An environmental study is an outstanding requirement needed to fully define the environmental considerations for the project and the resultant costs that will be incurred.

SECTION 21 CAPITAL AND OPERATING COSTS

21.1 BASIS OF COST ESTIMATE

The cost estimate was generated from supporting engineering quantities and cost information derived from the following sources:

- Historical cost information sourced from in-house and commercial databases;
- Quotations from equipment suppliers;
- Rates from local service providers; and
- Client derived data from existing local operations.

Both capital and operating cost estimates were prepared in mixed currencies and reported in United States dollars (USD). The currency exchange rates used for the cost estimate are presented in Table 21-1. These rates were baselined using the three month average between 17 May and 14 August 2015.

Table 21-1: Currency Exchange Rates

Currency	Rate
AUD (Australian Dollar)	1.3200
CAD (Canadian Dollar)	1.2611
EGP (Egyptian Pound)	7.6974
EUR (Euro)	0.9017
GBP (Great British Pound)	0.6428
PEN (Peruvian Soles)	3.1323
USD (United States Dollar)	1.0000
ZAR (South African Rand)	12.3395

21.1.1 ESTIMATE CLASSIFICATION

The prepared estimate is classified by GBM as a Class 4 estimate with a +30 % / -30 % accuracy. The respective range of the GBM Class 4 estimate with respect to other classification systems is presented in Table 21-2 for comparison.

Table 21-2: Estimate Classification Comparison

<table>
<tr><th>GBM [based on AACE]</th><th>ANSI Standard Z94.0</th><th>Association of Cost Engineers (UK) [ACostE]</th><th>American Society of Professional Estimators [ASPE]</th></tr>
<tr><td rowspan="2">Class 4
-30 % / +30 %</td><td rowspan="2">Budget Estimate
-15 % / +30 %</td><td rowspan="2">Study Estimate Class III
-20 % / +20 %</td><td>Level 1</td></tr>
<tr><td>Level 2</td></tr>
</table>

21.1.2 ASSUMPTIONS

The project currently assumes additional land acquisition and surface rights will be obtained in the future to accommodate proposed infrastructure such as access roads and conveyors as well as the processing facilities themselves. The potential costs of such an acquisition are not included within the estimate.

The fleet requirements were developed using the operating time factors shown in Table 21-3: Cost Sheet Assumptions

A total of 5 538 operating hours were planned per year.

Table 21-3: Cost Sheet Assumptions

Assumption	Value	Assumption	Value
Operating Days	365	Operator Efficiency	85 %
Hours/Day	24	Equipment Utilisation	85 %
Meetings/Breaks (h)	-1.5	Shifts	2
		Crew Pattern	2

21.1.3 EXCLUSIONS

This estimate has been prepared beginning from the point of project approval. Therefore, costs incurred during project development are excluded. Major project development components are:

- Metallurgical testwork;
- EIA;
- PFS / BFS engineering;
- Exploration drilling; and
- Permits, licences or legal and administrative costs associated with government mining and environmental regulations. This includes reporting requirements during operation and related administrative costs.

Additionally, no allowance has been made for:

- Cost escalation;
- Currency fluctuations;
- Insurance;
- Container demurrage costs;
- Product (yellowcake uranium) transportation or transportation security;
- Monitoring during operation;

- Containment, monitoring or treatment of waste rock in the event that acid rock drainage or metal leaching are applicable; and
- Hydrogeological monitoring, dewatering or stormwater control measures.

21.1.4 CONTINGENCY

21.1.4.1 CAPEX

Contingency is a cost element of the estimate used to cover the uncertainty and variability associated with unforeseeable elements not defined in the project scope. A preliminary risk identification and assessment process was conducted and the following four types of risks were identified for the project:

- Country / Political Risks – political instability and in country politics.
- Technical Risks – rehabilitation and closure design and reserve risks.
- Construction and Operating Risks – skills availability and working at high altitudes.
- Performance Risks – power and water supply.

A 30 % contingency has been allocated to the direct costs.

21.1.4.2 OPEX

A contingency of 0 % has been applied to the project operating costs due to the level of scope definition.

21.2 CAPITAL COST ESTIMATE

Table 21-4 shows the estimated initial capital costs for the project, including the cost breakdown for each project area.

Table 21-4: Initial Capital Expenditure

Cost Centre	TOTAL [USD]	General [USD]	Mining [USD]	Processing [USD]	Waste Management [USD]	Infrastructure [USD]
Total Capital Investment	299 890 000	132 811 000	0	115 320 000	16 786 000	34 974 000
FIXED CAPITAL TOTAL	250 619 000	83 540 000	0	115 320 000	16 786 000	34 974 000
DIRECT TOTAL	167 079 000	0	0	115 320 000	16 786 000	34 974 000
Earthwork	35 728 000	0	0	21 807 000	6 844 000	7 078 000
Civil	28 342 000	0	0	23 287 000	4 733 000	322 000
Structural	14 841 000	0	0	724 000	0	14 117 000
Mechanical	55 245 000	0	0	50 887 000	4 358 000	0
Mobile Equipment	2 349 000	0	0	350 000	750 000	1 249 000
Electrical	12 763 000	0	0	4 138 000	101 000	8 525 000
Control and Instrumentation	2 364 000	0	0	0	0	2 364 000
Piping	13 738 000	0	0	12 418 000	0	1 320 000
Platework	1 709 000	0	0	1 709 000	0	0
INDIRECT TOTAL	83 540 000	83 540 000	0	0	0	0
EPCM	25 062 000	25 062 000	0	0	0	0
Field	8 354 000	8 354 000	0	0	0	0
Contingency	50 124 000	50 124 000	0	0	0	0
WORKING CAPITAL	49 271 000	49 271 000	0	0	0	0

Table 21-5 shows the estimated initial capital costs for the project including cost breakdown for each project area.

Table 21-5: LOM Project CAPEX Estimate

Cost Centre	TOTAL	Initial [USD]	Phased Expansion [USD]	Closure [USD]
Total Capital Investment	358 473 000	299 890 000	43 911 000	14 672 000
FIXED CAPITAL TOTAL	309 202 000	2506 19 000	43 911 000	14 672 000
DIRECT TOTAL	214 228 000	167 079 000	34 049 000	13 100 000
General	0	0	0	0
Mining	23 730 000	0	23 730 000	0
Processing	117 871 000	115 320 000	2 551 000	0
Waste Management	36 682 000	16 786 000	6 796 000	13 100 000
Product Handling	0	0	0	0
infrastructure	35 945 000	34 974 000	971 000	0
INDIRECT TOTAL	94 974 000	83 540 000	9 862 000	1 572 000
EPCM	26 178 000	25 062 000	1 116 000	0
Field	9 520 000	8 354 000	904 000	262 000
Contingency	592 75 000	50 124 000	7 842 000	1 310 000
WORKING CAPITAL	49 271 000	49 271 000	0	0

21.2.1 INDIRECT CAPITAL COSTS

21.2.1.1 EPCM

The Engineering Procurement and Construction Management (EPCM) costs have been estimated at USD 25.12 million, the magnitude of which was verified against other similar GBM projects and is considered within the industry accepted range for a Class 4 estimate. Estimated construction costs are not included within the EPCM estimate but rather are accounted for as part of the direct CAPEX costs. EPCM was selected as the method of the project delivery due to the current level of scope definition. It should be noted that the project could be executed as an Engineering Procurement Construction (EPC) project with Plateau Uranium or their representative acting as a Project Management Contractor (PMC). Assuming all other things are equal, if the project delivery method chosen is an EPC / PMC model it may be more expensive when compared to an EPCM due to the profit margins of the EPC contractor.

21.2.1.2 FIELD INDIRECT COSTS

Field costs total 5 % of the total direct cost. This estimate includes allowances for:

- IT hardware, software and training;
- Construction power;
- Construction camp costs;
- Client project team;
- Field Indirect costs;
- Mobilisation costs; and
- Independent project consultants used for verification such as engineers, geotechnical consultants and surveyors.

This value is considered low by industry standards, primarily because a number of areas have not been included as they are outside of the scope of this PEA. These areas include:

- Land acquisition and right of way costs;
- Currency hedging;
- Regulatory approvals and government project development fees; and
- Environmental bonds.

21.2.2 WORKING CAPITAL

The working capital has been estimated as approximately 4.5 months of an averaged year's operating cost. Preliminary calculations suggest this is the minimum number.

21.2.3 MINING CAPITAL COST ESTIMATE

The CAPEX estimate for the open pit operation is based on using contractor operated fleet for all surface mining operations. Therefore no initial or sustaining capital is required for operations.

An owner-operating open pit cost model has been generated for comparison. The underground mining has been based on an owner-operated cost model. The CAPEX estimate for the open pit operation is based on the scheduled plant throughput in the conceptual production schedule. The underground equipment CAPEX (including sustaining and replacement costs) required to achieve the target processing rate is summarised in Table 21-6.

No pre-stripping requirements were identified for the Macusani complexes; as such, no pre-stripping CAPEX has been estimated.

Table 21-6: Underground Equipment CAPEX Summary

Item	Unit Cost [USD M]	Initial Units	Total Cost [USD M]
Continuous Miner	3.52	1	3.52
Bridge Conveyor	0.20	1	0.20
Belt Storage Unit	0.28	1	0.28
Feeder Breaker	0.39	1	0.39
Conveyor 1	2.66	1	2.66
Wheel Loader	1.98	1	1.98
Light Vehicles	0.03	5	0.15
Service Truck	0.09	1	0.09
Jumbo	0.83	1	0.83
Charge Wagon	0.68	1	0.68
Pre PEM Production Equipment Total			**10.8**
Pre PEM Production Ancillary Equipment			**0.97**
Post PEM Production Total			**12.0**
TOTAL			**28.0**

21.2.4 CAPITAL COST ESTIMATE – PROCESSING

The direct processing costs, in Table 21-5, give the breakdown of the processing costs per project area. As can be seen, the major costs are in the crushing and leach pad area costs.

Due to the challenging terrain, civil works to construct a flat and evenly sloped pad surface have contributed significantly to the cost of the project. Even though the pad placement was numerically optimised, cuts of up to 15 m were required to attain an even slope on the pad surface. Significant geotechnical surveying and a more detailed design is required to further develop the most economic location for pad placement.

Where not included, installation costs have been applied on a percentage basis derived from historical data ranging from between 9 % and 21 %. Allowance has been made for the fact that some equipment/plant is priced on a turn-key basis.

For piping costs (including piping, valves and fittings) in areas of significant cost such as leaching, quantities for each type of material were derived from general arrangement drawings and unit quantity rates based on approximate pipe run lengths.

The costs of the buildings that form part of the process facility (including leaching pump houses) have been calculated or obtained on a turn-key basis. The result of estimating costs on a turn-key package basis is that there are lower figures reported in each of the minor cost centres that typically make up the item cost. For example, platework or civil costs for the IX/recovery building are part of a turn-key package and are not disaggregated into itemised units.

21.2.5 CAPITAL COST ESTIMATE – TAILINGS MANAGEMENT

The use of a dynamic (on/off) pad means that the leaching area must be cleared and cannot be used for tailings disposal. Detailed tailings design is not appropriate for this level of study. However, ravine locations near the leach pad have been identified as having suitable capacity and an estimate of costs is provided in Table 21-4 for reference. These areas would need to be prepared and lined prior to the placement of spent PEM, and these costs have been estimated based on a preliminary design. There is also an allowance for a small liquid tailings dam to prevent environmental discharge of liquid tails, and equipment to treat and redirect this fluid into the process. The entire facility needs to be reviewed as more detailed information comes to hand. Significant changes in CAPEX are possible based on future results of baseline studies, environmental assessment and legislation.

Materials handling will be via grasshopper link conveyors and mobile stacker. Detailed design may mean that the mobile conveyors estimated can be reduced in size by periodically installing an overland conveyor as the heap advances.

21.2.6 CAPITAL COST ESTIMATE – INFRASTRUCTURE

The bulk of the infrastructure costs are associated with the site electrical systems and accommodation. The camp-based accommodation costs are based on approximately 370 operations staff, with provision for sewerage, power, laundry, mess and some recreational facilities.

The budget cost of a 33 kV high voltage line from the San Gaban power line to the proposed plant location would be in the order of 200 000 USD/km and the costs for substations at each end of the high voltage line are estimated to be approximately USD 500 000 each. The high voltage line required is approximately 7 km long. It has been assumed that the supply authority will levy this charge as a minor increase in the rate charge, thus moving costs to OPEX. The costs are indicative only and only the 33/11 kV substation has been included in the project cost estimate.

Access road costs include the construction of a main access road which links the site to the Interoceanica Highway as well as an access road around the process facility and haul road between the Triunfador 1 mining concession (Kihitian Complex) and the process facility. Heavy vehicle traffic, which will occur during construction and operation, will be serviced by the main access road. 3D software has been used to calculate the cut and fill quantities of the 8 m wide access roads. Road design and costing is dependent on detailed site surveying and geotechnical analysis which has not been undertaken and subsequently the cost of the road is considered to have a low level of accuracy.

The costs of the buildings which form part of the administration and maintenance areas have been calculated on a square metre rate turn-key basis at local rates for buildings of similar uses.

As an initial investment, only a small allowance for environmental capital expenditure has been allowed for in the way of site vehicles for environmental staff. It is likely further purchases will be defined in the environmental study, though there is a possibility that environmental monitoring may be performed by a consultant and therefore this expenditure will be reclassified as OPEX.

21.2.7 PHASED COSTS DISCUSSION

Initial capital costs have been spread across the construction period of the project, assuming a two year construction period. A 30 % deposit has been assumed for all equipment, and is reflected in year 1.

Annual sustaining capital costs include an allowance for semi-mobile link conveyors for tailings. Once the detailed tailings design is completed, it is recommended that periodic extension of an overland conveyor over the tailings heap be investigated.

An allowance for rehabilitation and closure of operations is included at the end of the project including capping of tailings and monitoring. This has been inputted as a single year figure, however, it should be noted that monitoring would likely be an ongoing cost post mine closure rather than a lump sum in the final year of operation. The total cost of the decommissioning of the project needs to be re-evaluated once an environmental study and closure management plan has been developed.

The project has been designed with the intention that both the Kihitian and Isivilla Complexes can utilise the same crushing facility.

Future studies that include greater project definition will allow for more accurate phasing of sustaining capital costs. Sustaining capital has been considered at a high level only to better define the financial model, the results of which are described in Section 22.

The service life of the underground mining fleet has been estimated and where an extension of service life is intended for use, a major overhaul of 40 % of the original equipment capital amount has been planned approximately half way through the life of an item. A total overhaul cost for the project has been estimated at USD 12.0 million. The underground equipment will be purchased in year 1 of the project and be operational in year 2 and operate for the remaining life of mine.

21.3 OPERATING COST ESTIMATE

The average operating costs estimate was prepared based on the LOM costs. This enabled the estimation of unit rates for the operating costs per tonne of PEM or per pound of yellowcake produced. This approach was used as the majority of operating costs were variable rather than fixed. Fixed costs included general and administration costs and labour costs with the remainder of costs being variable. The variable costs for the mining operation are driven by the total amount of material mined, including waste. The variable costs for the processing plant are a function of either the PEM processed or the uranium extracted.

The total operating costs are estimated as detailed in Table 21-7.

Table 21-7: Project OPEX Estimate

Area	Description	Average Annual Cost [USD]	Percentage of total OPEX	Average cost per tonne of PEM [USD/t]	LOM average cost per lb of U_3O_8 produced [USD/lb]
TOTAL*		103 062 707	100.0%	9.46	16.90
000	GENERAL and ADMINISTRATION COSTS	4 185 000	4.16%	0.38	0.69
100	MINING	67 242 100	66.8%	6.17	11.02
	Weighted average of underground and open pit	69 613 280	67.5%	6.39	11.41
200	PROCESS	26 937 917	26.1%	2.47	4.42
	Reagents	14 149 639	13.7%	1.3	2.32
	Personnel	2 737 428	2.66%	0.25	0.45
	Fuel and Lubricants	75 100	0.07%	0.01	0.01
	Electricity	6 370 331	6.2%	0.58	1.04
	Maintenance and Operating Spares	3 605 418	3.5%	0.33	0.59
300	WASTE MANAGEMENT	1 044 031	1.0%	0.1	0.17
	Fuel and Lubricants	105 000	0.1%	0.01	0.02
	Electricity	939 030	0.9%	0.09	0.15
400	PRODUCT HANDLING	306 787	0.3%	0.03	0.05
500	INFRASTRUCTURE	975 691	0.95%	0.09	0.16
	Fuel and Lubricants	102 400	0.1%	0.01	0.02
	Consumables	873 291	0.85%	0.08	0.14

* Royalties excluded

Visual breakdown of the annual average operating costs can be seen in Figure 21-1.

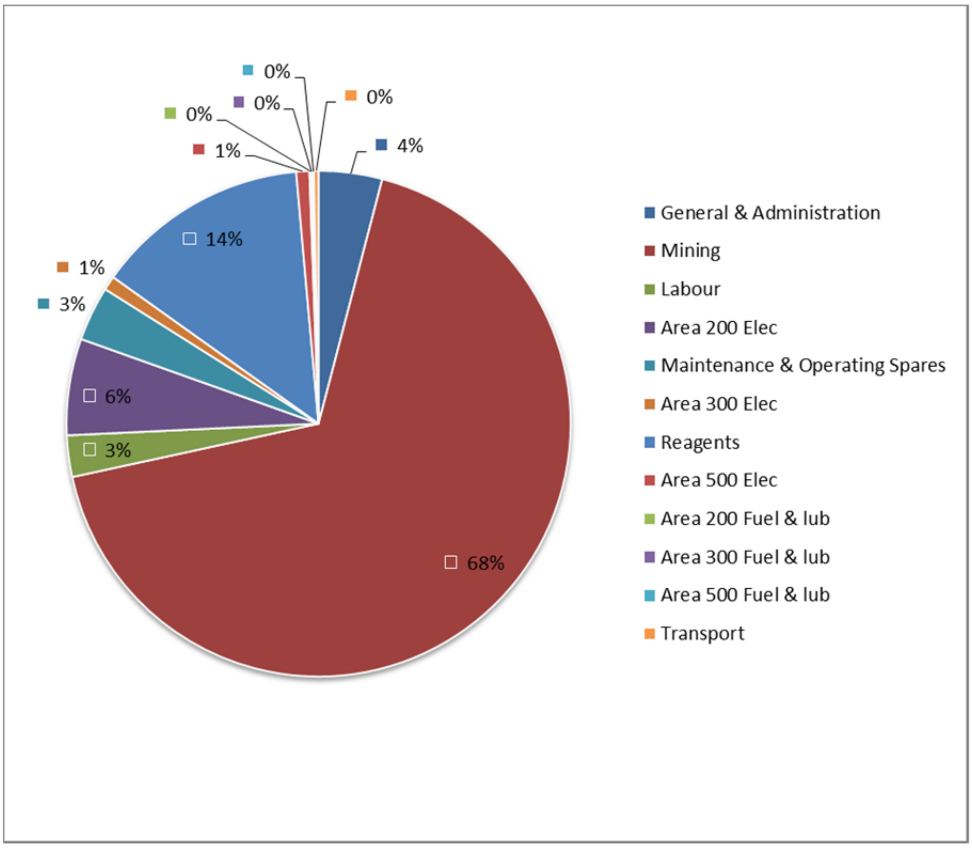


Figure 21-1: Average Annual OPEX Cost Breakdown by Percentage

21.3.1 GENERAL AND ADMINISTRATION

General and administration costs have been estimated for labour as approximately USD 15 000 per employee as well as an allowance for environmental monitoring.

21.3.2 MINING

The open pit mining activities for the Macusani complexes have been assumed to run as a contractor-operated operation for the optimisation, but an owner-operated engineering model has been presented for comparison. The cost estimate was built from first principles, input from Plateau Uranium, as well as WAI experience of similar sized open pit operations. Equipment efficiency was estimated based on site conditions. Local labour rates (for operating, maintenance, supervision and technical personnel) and estimates on diesel pricing (0.77 USD/l) and explosive pricing (2.24 USD/kg) were taken into consideration for the mining cost estimate.

The open pit mining costs were calculated for both mineralised material and waste mining, where variations in haulage profiles and equipment selection were taken into account in the cost estimate. Open pit mining costs for this preliminary assessment were estimated to be 1.04 USD/t waste mined and 1.71 USD/t PEM mined. The average cost equated to $ 1.85 USD/t of rock, which included a contractor margin also.

The mining cost estimates encompass open pit and dump operations, road maintenance, mine supervision and technical services. This also includes transportation of underground PEM to the primary crusher.

The pit optimisations as outlined in section 15 were conducted using a preliminary mining contract rate of $2.40 USD/t of rock. The final estimate of $1.85 USD/t rock. The effect of this on the project results has been considered, and deemed conservative and therefore acceptable for this PEA. Any future work will include re-optimisation based on this revised parameter.

The breakdown of open pit operating costs is shown below in Table 21-8.

Table 21-8: Open Pit Equipment Operating Cost Summary

Cost Centre	Unit	Overall (LOM)
Drilling	USD/t	0.03
Blasting	USD/t	0.25
Dozing and Grading	USD/t	0.04
Loading and Stockpiling	USD/t	0.19
Hauling	USD/t	0.78
Engineering / Geology	USD/t	0.25
General Mine Maintenance	USD/t	0.06
Supervision and Technical	USD/t	0.03
Other	USD/t	0.00
Total	**USD/t**	**1.65**

The underground mining costs have only included mineralised PEM panels for mining. Underground mining costs are estimated to be 7.77 USD/t PEM mined. The breakdown is shown in Table 21-9.

Table 21-9: Underground Equipment Operating Cost Summary

Cost Centre	Unit	Overall (LOM)
Continuous Miner	$/t	1.38
Conveyors	$/t	0.69
Development	$/t	1.94
Explosives	$/t	0.63
Auxiliary Fleet	$/t	0.91
Ventilation	$/t	0.08
Engineering/Geology	$/t	0.83
General Mine Maintenance	$/t	0.87
Supervision	$/t	0.44
Total	**$/t**	**7.77**

21.3.2.1 LABOUR RATES

Mining costs for both the open pit and underground have been based on the following labour rates within the subsequent models. The costs have been supplied by Plateau Uranium and are shown in Table 21-10.

Table 21-10: Labour Grades and Cost

Category	Grade	Leave/ Absenteeism Correction	Company Cost per Person (US$/yr)	Company Cost per Person (PEN/yr)
Mine Manager	A	1.00	75,000	234,923
Maintenance Superintendent	B	1.00	72,000	225,526
Chief Technical	C	1.00	41,987	131,516
Senior Technical	D	1.15	24,930	78,088
Engineers / Geologists	E	1.15	17,755	55,614
Operations Supervisor	F	1.15	19,244	60,278
Operations	G	1.15	9,937	31,126
Labour/Admin/Grade Control	H	1.15	8,747	27,398

21.3.3 PROCESSING, INFRASTRUCTURE AND WASTE MANAGEMENT

The processing, infrastructure and waste management costs are captured under the following sub-areas:

- Labour;
- Electrical power;

- Reagents;
- Operating spares and maintenance; and
- Fuel and lubricants.

A breakdown of these costs is presented in this section.

21.3.3.1 LABOUR

The labour requirements for the processing facility are broken down into management, technical services, supervision, operations and maintenance. The costs presented in Table 21-11 are the total costs to the employer for each employee. Back to back rotations have been assumed for these calculations, as have two shifts of 12 hours per day.

Table 21-11: Labour (Processing Facility)

Position	Quantity	Annual Salary [USD/person]	Annual Cost [USD/a]
Plant Superintendent	1	75 000.00	75 000.00
Plant Maintenance Superintendent	1	72 000.00	72 000.00
Chief Metallurgist	1	48 000.00	48 000.00
Chief Chemist	1	45 000.00	45 000.00
Maintenance Planner	2	39 362.61	78 725.22
Control Engineer	1	24 929.65	24 929.65
Plant General Foreman	1	33 000.00	33 000.00
Plant Foreman	16	30 000.00	480 000.00
Maintenance Foreman	5	26 241.74	131 208.70
Control Room Operator	24	9 936.87	238 484.93
Plant Operator / Plant Shift Operator	64	9 936.87	635 959.80
Mechanic / Shift Mechanic	32	9 936.87	317 979.90
Instrumentation and Automation Technician	8	9 936.87	79 494.96
Electrician	32	9 936.87	317 979.84
High Voltage Electrician	2	9 936.87	19 873.74
Laboratory Assistant	4	9 936.87	39 747.48
Laboratory Technician	8	8 747.25	69 978.00

21.3.3.2 ELECTRICITY

Energy costs have been calculated using tariff rates published by Osinergmin (96), the office responsible for regulating the generation and distribution of electricity. As these tariffs are published by the regulatory body they can be taken as indicative (the calculations are based on the lowest tariff published for medium voltage connection for the Azángaro Rural/San Gabán areas). The tariffs used to determine electrical supply costs give in Table 21-12:

Table 21-12 Utility Rates

Utility	Value	Unit
Electricity – Generation charge	115.97	USD/kW/a
Electricity – Transmission charge	79.57	USD/kW/a
Electricity – Fixed monthly fee	52.49	USD/a
Electricity – Active energy cost	0.06	USD/kWh
Electricity – Reactive energy cost	0.01	USD/kVArh

The power factor at this site is expected to be quite poor due to the high altitude of the site and the large percentage of inductive loads. With the reactive energy tariff charged on any reactive power exceeding 30 % of the total energy used, any power factor less than 0.95 will attract charges. Power factor correction equipment is installed to increase the site power factor above 0.95 and avoid these additional tariffs.

21.3.3.3 REAGENTS

Sulfuric acid is the single largest reagent used in the operation in two roles; firstly, as a lixiviant to leach the uranium from the host PEM, and secondly, as an eluant in the IX plant to strip the loaded resin. The acid is produced locally and Peru is a net-exporter to the international market. Other reagents are used as part of the recovery IX/SX process.

21.3.3.4 OPERATING SPARES AND MAINTENANCE

The cost of operating spares and maintenance was estimated as 8 % of the direct capital costs of equipment and road and building maintenance was estimated as 3 % of the respective direct capital costs.

21.3.3.5 FUEL AND LUBRICANTS

The estimated consumption requirements for fuel and lubricants were calculated based on the mobile fleet and a diesel price of 0.77 USD/L.

21.4 HIGH GRADE OPTION

Both the operating and capital estimates were calculated utilising the key parameters identified as part of the base case. To establish the capital estimate for the high grade heap leach, fixed infrastructure items remained the same but processing equipment and building were scaled accordingly. The tank leach capital estimate was based upon preliminary sizing major equipment and using typical industry factors to scale and factor in house pricing.

The operating estimate for the high grade heap leach remained the same per tonne of ROM. The tank leach operating estimate was adjusted where necessary. This included evaluation of energy consumed and acid consumption.

The mining costs remained the same as per the base case, however there may be opportunity to further reduce the mining contract rate.

Table 21-13 Capital Estimates

Case	Initial Capital (000's)	LOM Capital (000's)
Base Case	299.9	358.5
Case 1	247.5	279.4
Case 2	247.5	291.4
Case 3	267.4	299.3
Case 4	267.4	311.3

Table 21-14 Operating Estimates

Case	$/t ROM	$/lb U_3O_8
Base Case	9.6	17.28
Case 1	14.6	17.39
Case 2	13.6	15.95
Case 3	17.6	19.73
Case 4	17.0	18.81

SECTION 22 ECONOMIC ANALYSIS

The economic analysis presented in this section is preliminary in nature and includes Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. They do not have demonstrated economic viability and there is no certainty that the preliminary economic assessment will be realised. Caution should be used when interpreting the results presented.

22.1 ANALYSIS CRITERIA

The financial model prepared was a constant dollar type model which assumed that purchasing power did not change with time. This means the CAPEX, OPEX and revenue were considered constant through time in a like-for-like manner. The model was based on full equity financing.

The criteria inputted into the financial model included those parameters listed in Table 22-1.

Table 22-1: Financial Analyses Criteria

Criteria	Value	Reference
Discount rate	8 %	Plateau Uranium
Royalties	3 %	Plateau Uranium
Yellowcake	50 USD/lb	Plateau Uranium
Salvage at the end of Life of Mine	No	Assumed

The 3 % royalty is applied annually to the gross revenue. There may be further opportunity to reduce this, as it has been reported that other mining operations have been able to negotiate a 1-3 % royalty on the net revenue.

22.1.1 EXCLUSIONS

No allowance in the model was made for:

- Cost escalation;
- Currency fluctuations; and
- Required permits, licences or legal and administrative costs associated with government mining and environmental regulations. This includes reporting requirements during operation and related administrative costs.

22.1.2 ASSUMPTIONS

First year yellowcake production was assumed to be 64 % of nameplate capacity to account for the significant time required for start-up and attainment of steady-state of the heap leach.

22.2 FINANCIAL MODEL

The economic analysis was conducted over a range of discount rates and the discounted cash flow (DCF) results have been presented in Table 22-2. Sensitivity analysis was conducted on the base case, which demonstrates the sensitivity to the NPV if an increase or decrease is seen in the operating income, operating expenditure, capital expenditure, recovery and product price.

Table 22-2 DCF Results for Macusani

Discount Rate (%)	NPV (post tax USD)	NPV (pre tax USD)
6	713 620 376	1 001 402 010
8	603 118 762	852 754 239
10	510 022 044	727 444 336

The DCF model gave a robust result of a post tax NPV of USD 603.1 M and an IRR of 40.6 %.

The annual cashflow and cumulative cashflow are reported in Figure 22-1 and Figure 22-2, which also illustrates that the project has a payback period of 1.8 years.

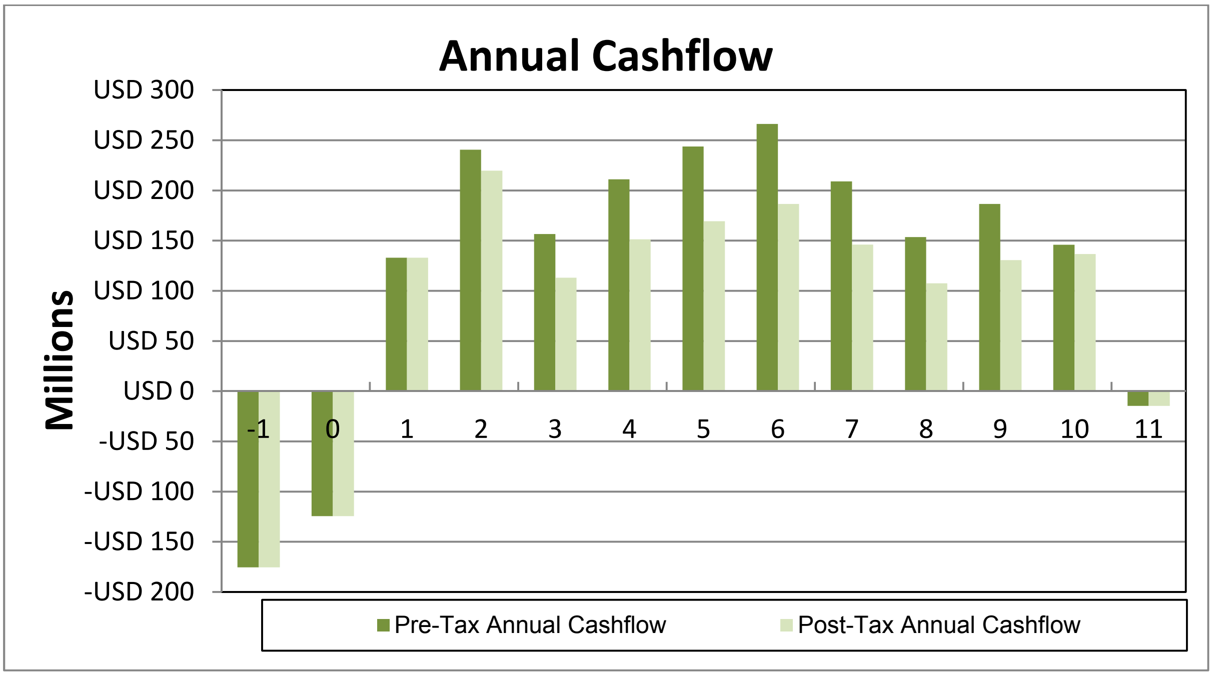


Figure 22-1: Annual Cash Flow

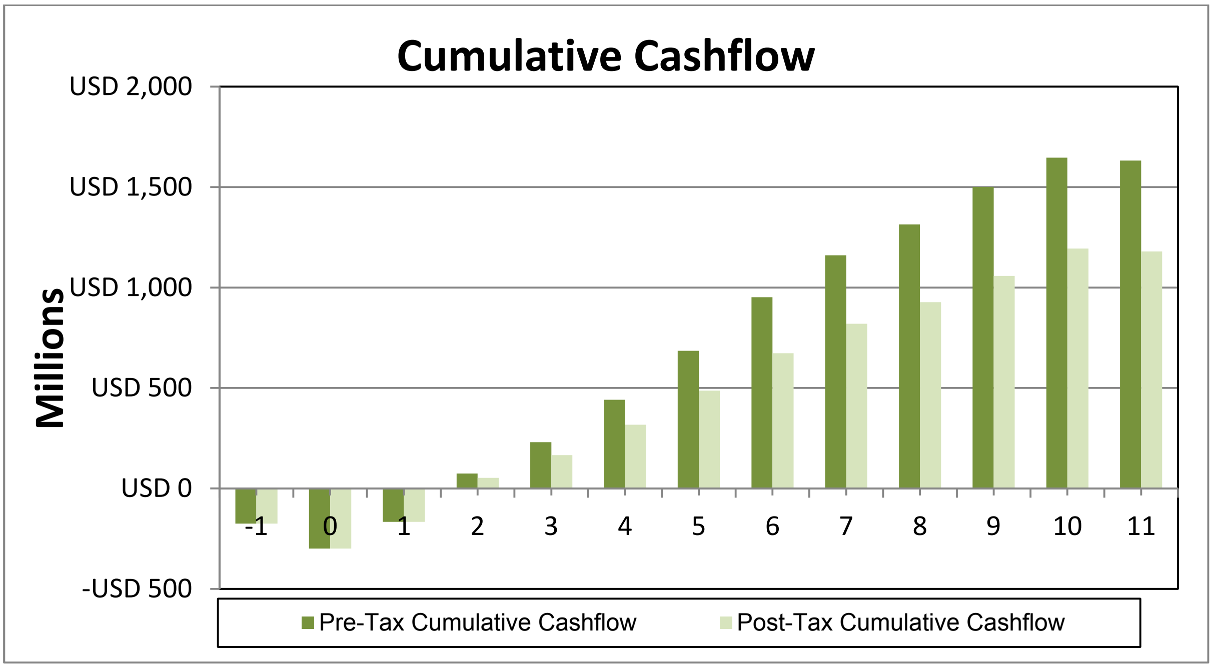


Figure 22-2: Pre-Tax Cumulative Annual Cash Flow

22.3 GOVERNMENT LEVIES

The Peruvian Government has various taxes, duties and levies that may or may not be applicable to future mining operations depending on the stability agreement (refer to Section 22.3.1) established at the time of exploitation and laws enforce at that time. A list of some taxes, duties and fees that could be applicable are listed in Table 22-3.

Table 22-3: Peruvian Taxation and Fees

Description	Value	Reference
Top rate of Corporation Income Tax (CIT)	30 %	(97)
Depreciation maximum	20 %	(97)
Depreciation Rate – Buildings	5 %	(97)
Depreciation Rate – Plant and Machinery	20 %	(97)
Depreciation Rate – Tailings Liner	10 %	Plateau Uranium
Government Royalty deductible from CIT calculation	Yes	(97)
Typical Import Duty	12 %	(98)
Typical Export Duty	0 %	(98)
Withholding Tax	None	(98)
Mining Right Fee	3.00 USD/ha/a	(97) / Section 4

Description	Value	Reference
Temporary Net Assets Tax	0.4 % Excess of PEN 1 000 000	(99)
VAT	18 %	Plateau Uranium
Mechanism to negate VAT	Yes	(98)
Potential tax holiday applied as part of stability agreement	Yes	(98)

22.3.1 STABILITY AGREEMENT (99)

Peruvian legislation includes stability agreements that provide guarantees regarding applicable taxes, tax reimbursements, tax holidays, free trade of minerals, free disposal of foreign currency received from exports, the availability of foreign currency for acquisitions or payments abroad, as well as the applicable exchange rate. The conditions of these stability agreements, including the duration of their applicability, may vary depending on many factors, such as:

- Proposed investment value;
- Proposed LOM;
- Laws in force at the time of agreement formation; and
- Mineral to be mined (currently no operating uranium mines in Peru).

Due to the uncertainty of the future terms of the stability agreement, the base case selected includes no applicable taxes and no deductions for government stipulated royalties. A potential stability agreement scenario, where some taxes are applicable is examined and is as described in Section 22.4.

22.4 SENSITIVITY AND SCENARIO ANALYSIS

Scenario analysis was conducted on the base case at 8 % discount rate, the variables listed in Table 22-4 were used. At this level of study it is difficult to be aware of the range of variation for each key input, therefore where information was not available a standard range of -10 % to +30 % was used.

Table 22-4: Sensitivity Variables

Operating Cost	Base	Low	High
Total capital cost	0 %	-15 %	+30 %
Process operating cost	0 %	-15 %	+30 %
Mining operating cost	0 %	-15 %	+30 %
Grade – Open pit	0 %	-6 %	+6 %
Uranium recovery	88.0 %	82.0 %	94.0 %
U_30_8 price (USD/lb)	50	35	65

The results of this analysis can be seen in Figure 22-3. The project is most sensitive to both price and recovery, and gives encouragement for future development since the base case has used an aggressive unit price. The second most influential factor is the mining operating cost. To increase confidence in the rate used, further work needs to be completed in understanding the mining methodology and factors used. It should be noted that the mining open pit contract rate used has been calculated in-house based on historical projects and that a quote from the market is recommended. There is opportunity to reduce the capital and process operating expenditure however the impact on project financials is limited.

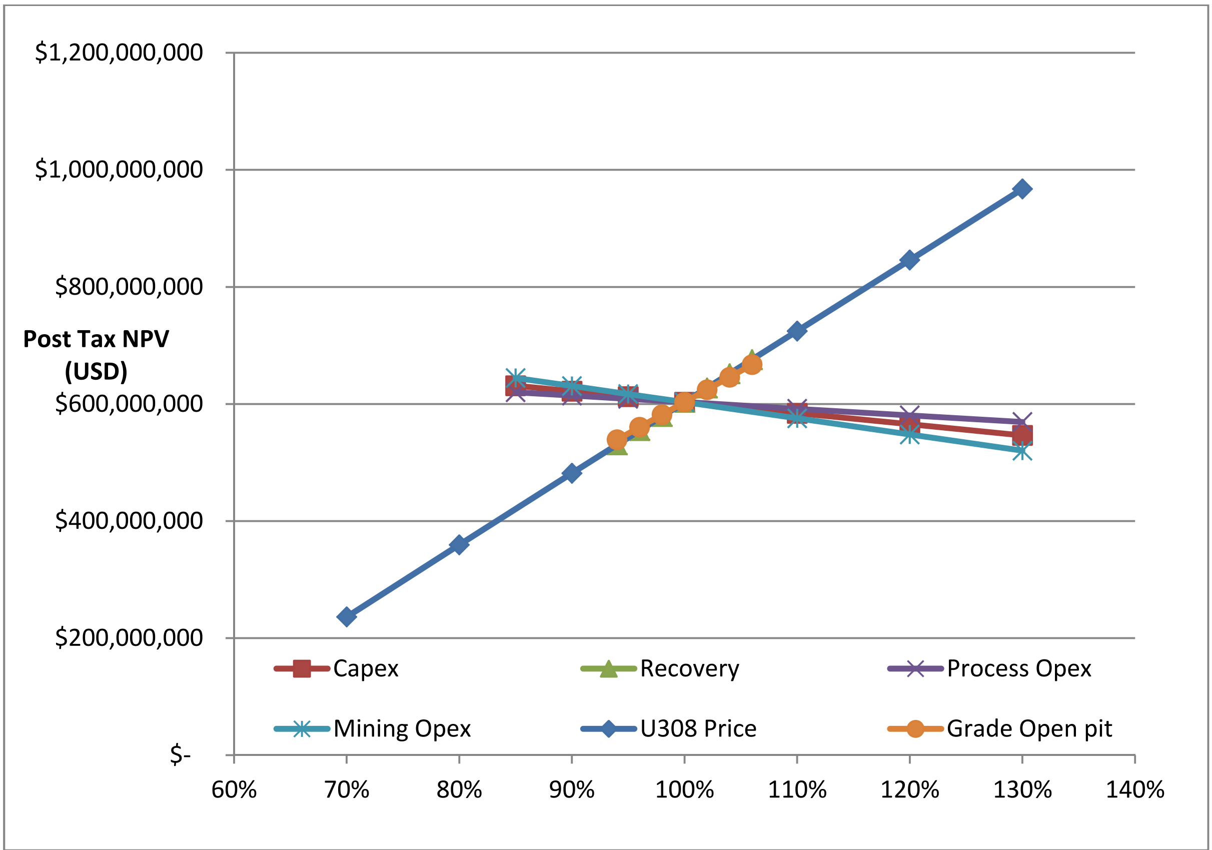


Figure 22-3: Post-tax Sensitivity Analysis

22.5 HIGH GRADE OPTION

The resulting pre and post tax NPV and IRR's for all four high grade cases have been compiled in Table 22-5, additionally the base case results are detailed to enable comparison.

Table 22-5 Financial Results

	Recovered U_3O_8	Pre-Tax		Post-Tax	
	M lb / a	NPV	IRR	NPV	IRR
Base Case	6.08	852.8	47.6	603.1	40.6
Case 1	4.26	544.4	41.2	417.4	37.3
Case 2	5.01	733.5	49.4	550.9	43.7
Case 3	4.50	510.2	36.8	397.2	33.9
Case 4	5.30	679.9	43.2	516.1	38.9

The base case gives the highest NPV for both the pre and post tax cases, however case 2 does have a higher IRR. It can also be seen that the underground mining contribution is significant to overall performance of the operation, approximately increasing IRR by 5 %.

SECTION 23 ADJACENT PROPERTIES

After the consolidation of concessions owned by Minergia and Global Gold, under Plateau Uranium, the only other significant holder of concessions in the area adjacent to the project is Fission Energy Corp, as shown in Figure 23-1.

23.1 FISSION ENERGY CORP

Fission Energy Corp is a Canadian based uranium exploration and development company with an international property in Peru. The company's shares were spun-out as a distribution by Strathmore Minerals Corporation in 2007 to Fission Energy Corp. In April 2013, Fission Energy Corp announced the spin-out of certain assets into a new company, Fission Uranium Corporation. Fission Uranium Corporation holds the rights to nine claim blocks encompassing 51 km^2, and surface rights over some of the areas with known uranium mineralisation in the project area. Within the area controlled by Fission Energy Corporation, the stratigraphy was reported from www.fission-energy.com/s/macusai.asp to be dominated by the sub-horizontal Pliocene Quenamari Formation, which is mainly composed of ignimbrite layers. Uranium anomalies occur on plateaus that are composed of the Upper Yapamayo Member of the Quenamari Formation.

Sampling to date has shown that the most significant uranium anomalies appear to be restricted to this assemblage. Mineralisation within the area is dominated by very high grade autunite veins along 'enriched fault planes', with lesser disseminated mineralisation. The significant fault planes can be up to 2 m thick, while multiple enriched fault planes occur in shear zones up to 150 m across.

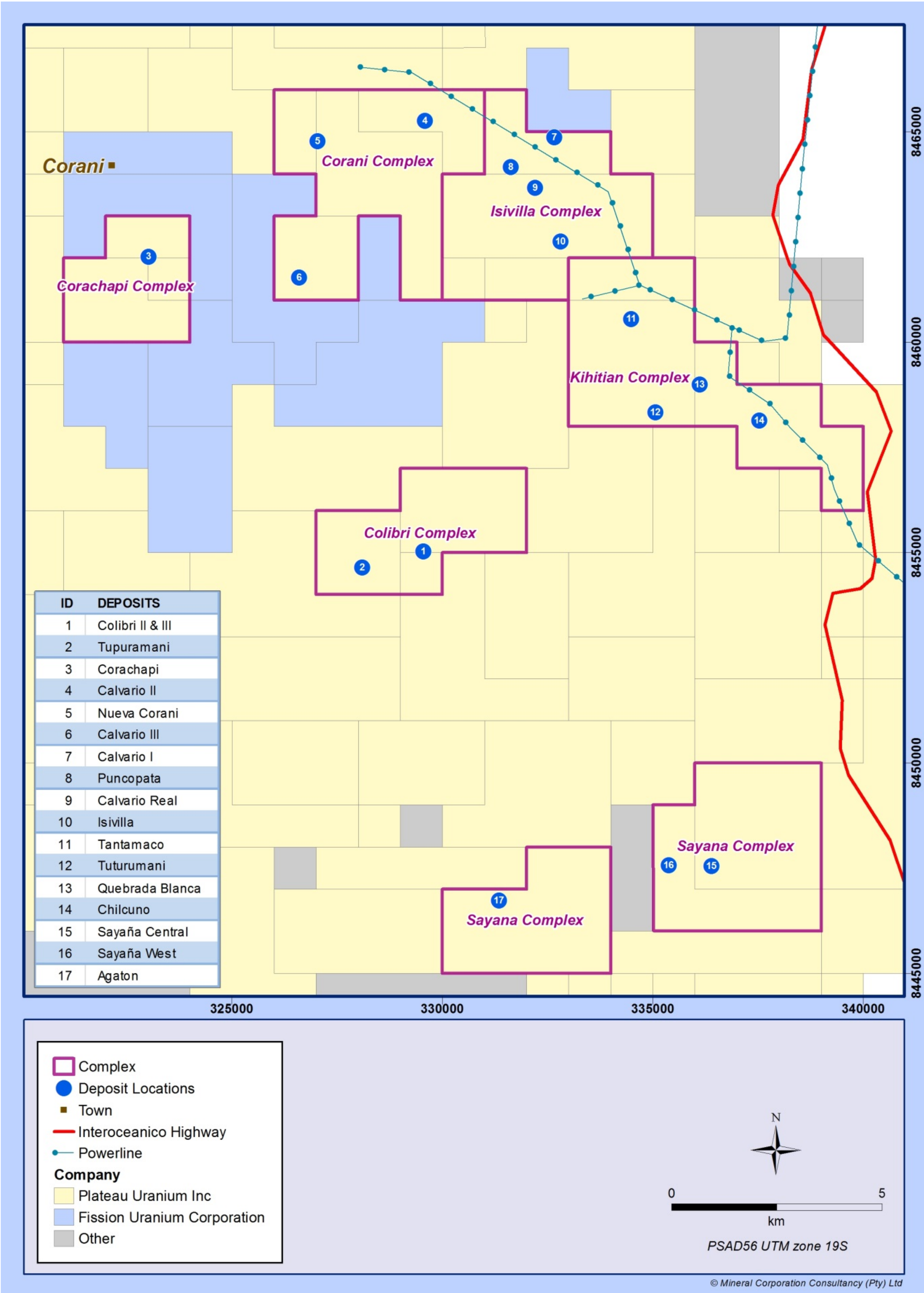


Figure 23-1: Adjacent Properties as at March 2015

SECTION 24 OTHER RELEVANT DATA AND INFORMATION

There is no other known relevant data or information not covered elsewhere in this report.

SECTION 25 INTERPRETATION AND CONCLUSIONS

25.1 MINERAL RESOURCES

The consolidation of the Minergia, Global Gold and Solex exploration data, within the Kihitian, Isivilla and Corani Complexes has resulted in an improved level of understanding and confidence in the continuity of the mineralisation within the project. Access to the expanded dataset which covers these three Complexes, as well as the Corachapi and Colibri Complexes, has allowed TMC to continue to develop the evaluation methodology, initially developed at the Colibri Complex, and apply it to the other Complexes. This has resulted in a set of block models which have been estimated in a similar manner, and which will be suitable for mine planning.

TMC is of the view that the use of MIK as the estimation methodology, has resulted in a more robust representation of the likely distribution of the uranium mineralisation, than a tightly constrained wireframe or grade envelope. In addition, the MIK models provide the opportunity to evaluate mining scenarios at different cut-off grades, and hence provide more options to optimise this very significant Mineral Resource.

25.1.1 KIHITIAN COMPLEX

The good continuity of the mineralisation, and the results of the QA/QC, has led to the definition of Indicated and Inferred Mineral Resources for the project. The updated block models have been prepared so that they can be used for mine planning.

25.1.2 ISIVILLA COMPLEX

The incorporation of new drilling information, particularly south of the Puncopata deposit, resulted in an extension of the Puncopata deposit to the south. Good continuity of uranium mineralisation has been demonstrated throughout the four deposits in this Complex, however, the absence of QA/QC information in the Calvario I, Puncopata and Calvario Real deposits has led to the declaration of Mineral Resources only in the Inferred category.

Inferred and Indicated Mineral Resources were defined for the Isivilla deposit based on data quality, geological and geostatistical confidence.

25.1.3 CORANI COMPLEX

While geological continuity is evident within the deposits of the Corani Complex, the limited information relating to sample preparation, security and analytical procedures for the data informing the Calvario II and Calvario III deposits limits the reporting of Mineral Resources to the Inferred classification. The Nueva Corani deposit data has been used for the reporting of Mineral Resources in the Indicated and Inferred classes.

25.1.4 SAYAÑA COMPLEX

Limited information on the sample preparation, security and analytical procedures undertaken by Solex, and which inform the Sayaña Complex is available. TMC is of the view that the analytical data would only be suitable for informing Mineral Resource estimates in the Inferred category, provided other evidence for the mineralisation can be identified. As there have not been any site visits by an independent Qualified Person to this Complex and no QA/QC data is available, these results cannot be employed to report Mineral Resources and/or Exploration Results.

25.2 MINING

25.2.1 GENERAL

WAI proposes that the Macusani complexes are amenable to be developed by the following methods:

- Colibri Complex - Open Pit
- Kihitian Complex - Open Pit and Underground
- Isivilla Complex - Open Pit

Mining of the complexes is planned to produce a total of 109 Mt of processing plant feed and 229 Mt of waste (2.1:1 overall strip ratio) over a ten-year mine production life. The project has been based on optimised pit shells that are not representative of mineable access to the deposits. Similarly, the underground designs have been limited to a maximum 10 m mining height.

The mine design has used a preliminary open pit mining contractor cost of 2.40 USD/t moved, which has been supplied by Plateau Uranium. WAI have provided an owner / operator estimate to benchmark the 2.40 USD/t contract rate. Subsequently an updated number has been used during financial analysis.Risks

The current understanding of the project is based upon limited and time sensitive information, such as metal and fuel pricing. Changes in the understanding of the project, the ability to convert Mineral Resources to Mineral Reserves and market conditions could affect the project’s economic viability.

A significant proportion (56 %) of the mining inventory Mineral Resources are classified as Inferred Mineral Resources. This material is considered too speculative geologically to be given a higher classification, particularly in relation to grade. The Mineral Resources do not have demonstrated economic viability and there is no certainty that the preliminary economic assessment will be realised.

Pit slope and underground geotechnical design has been based on information gathered during the 2013 site visit and supplied geotechnical testing information., Significant assumptions have been made in the process of deriving slope angles and stope widths. While the assumptions made are in WAI's opinion valid and appropriate, for further studies a comprehensive geotechnical investigation and study should be completed.

25.2.2 OPPORTUNITIES

With additional data, the project should be subjected to a series of strategic option reviews, WAI has conducted a high-level review of a high grade optimisation by open pit methods. This is an indicative review, but shows the project is highly amenable to cut-off grade optimisation.

The current PEA has considered three of six geological complexes. Currently the Kihitan, Isivilla and Colibri Complexes have been modelled, however, there are also the Corachapi and Corani Complexes within the project mining concessions that have modelled Mineral Resources. These two complexes were excluded to reduce project CAPEX, but as they have modelled Mineral Resources, there is an opportunity to consider the two complexes in an updated optimisation.

25.2.3 GEOTECHNICAL

The geotechnical design could possibly lead to up-side improvements or an increase in confidence in the currently used parameters. Currently, data is highly limited which has constrained the mine design for both the open pit and underground designs. To achieve a pre-feasibility level for the parameters, the following should be considered:

- Design and implement an orientated-core geotechnical drill program to log the rock mass and acquire intact rock and joint samples;
- Laboratory and field tests to adequately characterise the intact rock strength and joint properties of the main rock types;
- Underground, geotechnical mapping of the adits;
- Map and describe major faults, as viewed in drill core and rock outcrops within 200 m of the complex areas and integrate them with regional structural interpretation;
- Produce 3D digital wireframe models of lithology, alteration, and structures; and
- Characterise the rock mass using an appropriate rock mass rating system (for example RMR89) and map the geotechnical domains within a 3D model.

25.3 COST ESTIMATES

The capital cost estimate has been prepared to the level of a preliminary economic assessment study with an accuracy of -30% to +30 %.

25.3.1 CAPEX

The estimated initial investment was determined to be USD 299.8 million including a contingency of USD 50.1 million with a total capital cost estimate for the LOM of USD 358.5 million.

25.3.2 OPEX

The average operating cost calculated based on the LOM costs were USD 8.80 per tonne of Potentially Economic Material or USD 17.28 per pound of U_3O_8 produced.

The most significant operating costs were identified as:

- Contract Mining (60.0 %);
- Processing (27.0 %); and
- General administration (9.0 %).

Thus risks, such as the price of sulfuric acid (which accounts for a large portion of the reagents costs), and the limited geotechnical information (which significantly influences the mining costs), have the potential to impact upon the operational cost estimate.

25.4 FINANCIAL ANALYSIS

The financial analysis shows a positive result with respect to the project's economic performance. However, this does not imply economic feasibility of the project as the analysis included Inferred Mineral Resources. Since Inferred Mineral Resources cannot directly be converted into Mineral Reserves and due to their uncertainty of existence it cannot be assumed that any part of an Inferred Mineral Resource will ever be upgraded to a higher Mineral Resource category.

Further project definition and technical development will enable a more refined and accurate estimate of the project's economics. Particular attention must be paid to determination of the grade, process recovery rate and sale price given the sensitivity of the project's economics to these variables. The results of the Post-Tax scenario were an IRR of 40.6 %, an NPV of USD 603.0 million and a payback period of 1.8 years. A major uncertainty is the stability agreement to be entered into with the Peruvian Government that imposes taxation, tax holidays and tax reimbursement conditions on the project. It is important to note that the analysis conducted is based on a sale price 50 USD/lb, which if forecasts are correct, is a relatively conservative price and gives confidence in the project.

25.5 HIGH GRADE OPTION

All four cases provide positive results and demonstrate economically sound alternatives, and if a reduced initial capital is required options are available. By removing the underground mining the sustaining capital is reduced and potential risk in unknowns, however this does provide valuable recovered pounds of U_3O_8.

SECTION 26 RECOMMENDATIONS

Based on the results of the PEA it is recommended that a PFS be completed to give an indication of the project's feasibility. Further definition of the project is required to complete a PFS and therefore it is recommended that further drilling, studies and testwork be undertaken.

A project development schedule has been produced where the durations and key dependencies of the main tasks are outlined. Completion of these will be required prior to undertaking a PFS and, as shown in Figure 26-1, it is believed that the greatest risk to the timeframe and estimated cost is the infill drilling and metallurgical testwork.

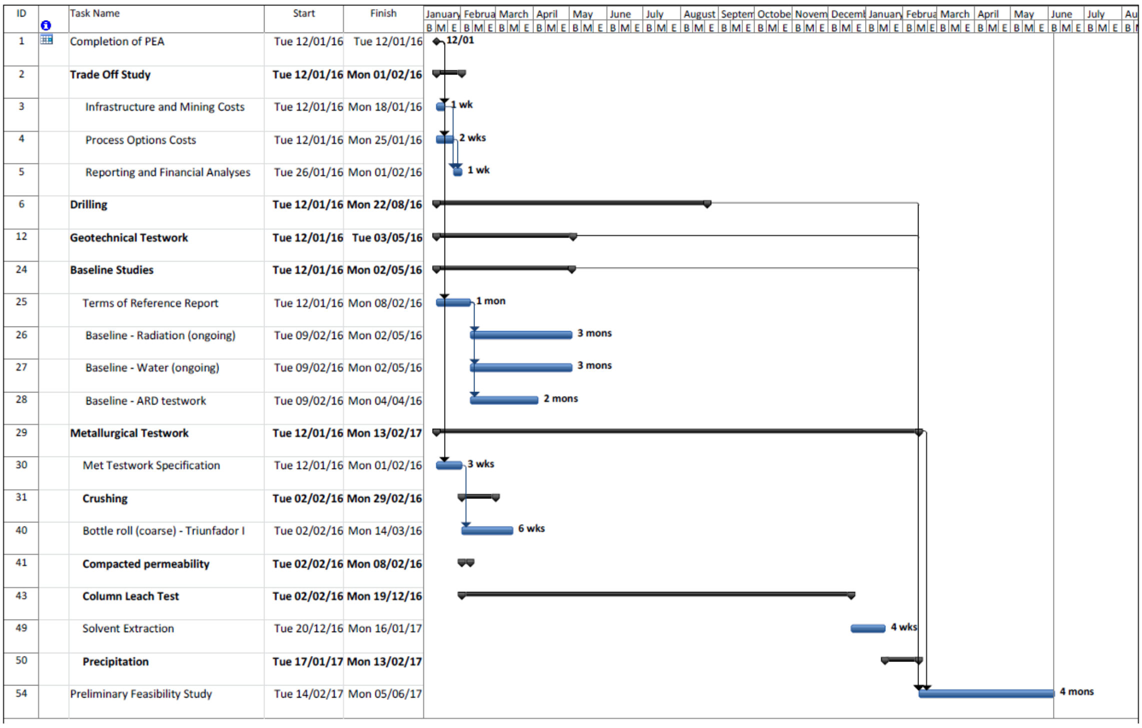


Figure 26-1: Recommended Project Development Schedule

It is estimated that the cost of undertaking this recommended additional work prior to completing a PFS is in the order of USD 4.0 million. A breakdown of this cost is as shown in Table 26-1.

Table 26-1: Preliminary Cost of Recommended Works

Testwork or Study	Timeframe	Cost [USD]
Metallurgical Testwork	14 months	600 000
Geotechnical Testwork	4 months	300 000
Environmental Studies	4 months	200 000
Infill and Sterilisation Drilling	8 months	2 500 000
Trade off studies	4 months	400 000

26.1 MACUSANI FURTHER ECONOMIC STUDIES

In 2008 a trade-off study was undertaken to determine the most appropriate processing method to complete a PEA. Two subsequent PEAs have been completed based on the results of that trade-off study. Since 2008 significant development of the project has been completed including:

- Metallurgical testwork;
- Further exploration drilling;
- Greater project definition increasing accuracy of costs; and
- Improved accuracy of costs from quotes received.

Some of the main drivers of the trade-off study were acid consumption rates, acid and power costs as well as the grade and size of the resource. Given the significant project development since 2008, an update of the trade-off study is recommended to verify that the processing method being pursued is still the most appropriate option for the current project Mineral Resources.

26.2 MINERAL RESOURCES

26.2.1 KIHITIAN COMPLEX

Should additional Mineral Resources within the Indicated category be required, TMC would recommend drilling immediately adjacent to the current Indicated Mineral Resource at the Tantamaco deposit, the Chilcuno Chico deposit or the Quebrada Blanca deposit. Additional drilling would not appear to be warranted at Tuturumani.

26.2.2 ISIVILLA COMPLEX

The following is proposed to address issues which have prevented the classification of the Mineral Resources for the Isivilla Complex in any category other than Inferred:

- Undertake a confirmatory drilling programme, consisting of drilling, logging, sampling and analysis of approximately 10 % of the drillholes drilled by Solex;
- Procure an extension to the high-resolution topography dataset which includes the north-eastern corner of Calvario I;
- Undertake a campaign with additional holes on a 200 x 200 m spacing, as proposed by Minergia, to appraise the north and east extensions of the Isivilla deposit.

26.2.3 CORANI COMPLEX

As with Isivilla, a confirmatory drilling programme would be required at the Corani Complex in order to validate the drillholes in the Solex database, for Calvario I and Calvario II. Approximately 10 % of the drillholes would need to be twinned.

26.2.4 SAYAÑA COMPLEX

TMC recommends that Plateau Uranium undertake a modest validation drilling programme in order to validate the dataset for the Sayaña Complex more comprehensively, and enable their use in Measured and/or Indicated Mineral Resource categories, as was completed for the Corachapi Complex.

26.3 ENVIRONMENTAL STUDIES

The project must be the subject of further study in support of both the project design and permitting procedures. Site studies are needed to define the project and process design criteria and environmental design. In particular the following is recommended:

- Baseline radiation studies;
- Continued data collection at the weather station;
- Hydrogeological study to determine water availability and confirm source location(s), quality and quantity, as well as groundwater flows/aquifers; and
- Metal leach / acid rock drainage testwork.

26.4 METALLURGICAL TESTWORK

Whilst there has been significant testwork completed to date, further testwork is required to support the level of accuracy required of a future PFS. To complement the existing testwork it is recommended the following testwork programme be implemented to enable further process definition and development in the engineering design and costing. It is recommended that the testwork is pursued concurrently with infill drilling activities. A proposed testwork scope obtained in 2014 to take the testwork to PFS level (100) was estimated to cost approximately USD 600 000.

26.4.1 PEM CRUSHING CHARACTERISTICS

Critical to the design of the heap leaching and crushing circuits is a full characterisation of the PEM competency and crushing characteristics. This type of testwork would determine:

- The Crusher Work Index of the PEM;
- The abrasivity of the PEM;
- The specific gravity and bulk density of the crushed PEM;
- The optimum crush size of the PEM for sizing of the crushing circuit; and
- The production of fines (size and quantity).

By extending the crushing characteristics testing to include milling work indices would be prudent. Further investigation of size by size liberation would also provide guidance on the inclusion of selective comminution in a tank leach milling circuit.

26.4.1.1 COMPACTED PERMEABILITY

Compacted permeability testwork completed on the optimum crushed PEM size will enable determination of whether the selected heap height is technically feasible for the design irrigation rate.

26.4.2 BOTTLE ROLL TESTWORK

If test work results for Isivilla cannot be obtained, additional bottle roll testwork will be required. Additional supporting BRT for deposits within the properties will be required and selected based on the production schedule and existing data.

26.4.3 COLUMN LEACH TESTWORK

To support the current column leach testwork, additional testwork is required using samples representative of the Mineral Resource to improve the confidence in the reported results. Additionally, due to the variability in the PEM grade (101) (102), it may be prudent to perform a high and low grade test programme as the mining schedule may affect the grade being supplied for mineral processing which may in turn require distinct heap leach operating schedules. It is recommended that column leach testwork is completed for each deposit without blending of deposits within a concession so as to easily identify differences or anomalies as well as support future production schedules. Such a programme would:

- Examine the need for oxidant in leach solution;
- Determine washing and neutralisation requirements;
- Determine heap slumpage;
- Determine percolation rates;
- Determine drainage rates;
- Determine the tenor of the leach solution;
- Determine uranium content of leach residue; and
- Determine leach kinetics for cascade leaching (in large format columns) for leach cycle time and leach pad schedule calculation.

Extension of the phenomenological model, to include the newly acquired results, could also be undertaken in parallel with the PFS level testwork. This would allow more accurate simulation of the heap leaching process, provide focus for any outstanding testwork, and allow investigation of a range of scenarios without requiring a further suite of column leach tests.

26.4.4 IX / ADSORPTION

Tests to date suggest that IX is an appropriate technology to recover uranium from the pregnant leach solution and have identified a suitable IX resin. The testwork also suggests that the resin should be eluted by strong acid solution. To confirm and validate these results, additional laboratory tests are required at a range of acid solution molarities and utilising different eluents to determine their effectiveness. This will allow the confirmation of eluate selection and the determination of the maximum acid strength required for optimum elution (in the case of strong acid elution).

26.4.5 MINERALOGY AND ACID CONSUMPTION

Quantitative mineralogical assessment is required to reinforce uranium extraction limits, identify acid consuming minerals and determine size-by-size liberation (especially for tank leaching but also for heap leaching).

26.4.6 SOLVENT EXTRACTION

Solvent extraction is a proven technology and is a technically feasible choice for this project, however, IX is considered the best choice for lower pregnant leach solution tenors. Therefore, the use of SX is limited to the treatment of the IX eluate (rather than the PLS). This reduces acid consumption in the IX elution process. There are several possible combinations of eluate and solvent extraction solvents. Further testwork that focuses on the most likely combinations is required.

To support the solvent extraction tests conducted to date, additional testwork is required utilising representative leach solutions, to improve the confidence in the reported results. The potential use of strong acid elution in IX with SX for acid recovery dictates the requirement for additional tests of ammonium sulfate as a stripping agent in SX.

26.4.7 PRECIPITATION

A thorough precipitation testwork schedule has not yet been completed. Solvent extraction testwork to date has been followed by precipitation with magnesium oxide or hydrogen peroxide. The precipitation method required is dependent on the upstream uranium recovery technology (IX or SX) and the resin eluant employed. A precipitation reagent screening testwork programme should be conducted. Tests utilising ammonium hydroxide to precipitate ammonium diuranate from loaded ammonium sulfate strip solution should be included to verify the technical feasibility and improve confidence in a possible flow sheet.

26.4.8 WATER QUALITY

Water tests to date indicate that site water quality is very good. Potential project water sources should be better defined and the water subjected to a full spectrum of water quality tests, including suspended solids and metal analysis. Such a schedule will verify the water quality for use in the process (which could influence SX or IX selection) and for use as potable water at the mining and processing sites. The testwork will also set a bench mark for environmental measures.

26.5 MINING

26.5.1 MINING

Considerations should be given to progressing the project to an advanced engineering design for mining, which would allow more detailed assessments of mined tonnes for the economic and project analysis.

WAI recommends that a review of exploration drill holes and Mineral Resource classification in relation to grade and potential mining envelopes is conducted, with the aim to increase Mineral Resource classification while optimising exploration drilling.

Currently within the geological complexes independent areas are identified in the open pit optimisation, the complexes are hosted within larger extents of mineralisation, but are not currently included within an economic pit shell, it is recommend that infill drilling be conducted to expand potential areas included within an optimisation.

A cut-off grade assessment should be performed to identify the optimal combination of Mineral Resources scheduled for mining and the minimum acceptable operating margin that these Mineral Resources should return. WAI has conducted a high-level optimisation using a 200 U_3O_8 ppm cut-off to demonstrate the effect of using a mining cut-off on the project.

26.5.2 GEOTECHNICAL

A geotechnical drilling programme and subsequent analysis in conjunction with maximum utilisation of data from existing drill core samples is required to obtain a level of knowledge of the rock mass characteristics.

The data and subsequent analysis should be used to estimate and confirm slope angles and underground stoping dimensions.

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