AN UPDATED INTEGRATED DEVELOPMENT PLAN FOR THE MADAOUELA PROJECT, NIGER

Prepared For

GoviEx Niger Holdings Limited

Qualified Persons

Eur.Geol. Robert John Bowell PhD C.Chem. C.Geol - Corporate Consultant (Geochemistry)

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Effective Date – September 20, 2013

Amended Date – April 28, 2014

Amended Date – August 11, 2015

Report Prepared by

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UK6553

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Corporate Consultant (Geochemist)

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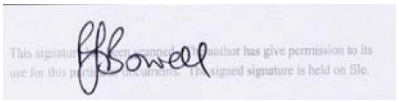
I, Robert J Bowell, a Chartered Professional Chemist, Chartered Geologist and a Certified Professional European Geologist, do hereby certify that:

1. I am currently employed as a consulting geochemist to the mining and mineral exploration industry, as a Corporate Consultant Geochemist with SRK Consulting (UK) Ltd, with an office address of 5th Floor Churchill House, 17 Churchill Way, Cardiff, CF10 2HH, UK.
2. I graduated with a Bachelors of Science Degree, First Class Honours in Geochemistry from Owen's College, Manchester University, Manchester UK, June 1988.
3. I graduated with a Doctorate in Geochemistry from Southampton University, Southampton, UK in June 1991.
4. I am a Chartered Chemist of the Royal Society of Chemistry, London, UK and have been since 1997. Membership number 332782.
5. I am a Chartered Geologist and Certified Professional European Geologist through the Geological Society of London since 1997 and European Association of Professional Geologists since 2000. Registration number 1007245.
6. I am a Fellow of the Institute of Mining, Metallurgy and Materials and have been since 2010.
7. I have been employed as a geochemist in the mining and mineral exploration business and in applied academia, for the past 25 years, since my graduation from university.
8. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. The Technical Report is based upon my personal review of the information provided by the issuer. My relevant experience for the purpose of the Technical Report is:
   - Geochemist, SRK Consulting from 1995 to date;
   - Exploration Geochemist with BHP Minerals, Hammersmith, London., 1991-1994;
   - Exploration Geologist, Ashanti Goldfields, Ghana, 1988
   - Uranium exploration experience as a geochemist and geometallurgical consultant, from 1998-1999; 2005-2006; 2007-current
   - Experience in the above positions working with and reviewing uranium mineralogy and geology, uranium analysis, resource estimation methodologies, geometallurgical testwork for uranium, uranium metallurgy, geochemical data quality, assurance and quality control in concert with resource estimation geologists and engineers.
   - As a consultant, I have been involved in several previous competent person's reports for

uranium projects including NI 43-101 Technical reports, 2002; 2006-2008

9. I am responsible for the preparation of the technical report titled “An Updated Integrated Development Plan for the Madaouela Project, Niger” and dated August 11, 2015 (the “Technical Report”) relating to GoviEx Niger Holdings Limited Madaouela Uranium Project; in particular the content of Sections 1 through 12 and Section 16. I have personally visited the Project in the field during January 2010, September 2011 and April 2012.
10. I have had prior involvement with the property and co-authored a previous 43-101 report and Preliminary Economic Assessment dated April 2011.
11. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
12. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, for which the omission to disclose would make the Technical Report misleading.
13. I am independent of the issuer applying all of the tests in Item 1.4 of National Instrument 43-101.
14. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
15. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible to the public, of the Technical Report.

Dated in Cardiff, United Kingdom, August 11, 2015



(“signed”) (“sealed”)

Eur.Geol. **Robert Bowell PhD C.Chem. C.Geol**
Corporate Consultant (Geochemist)
#332782, Chemist; #1007245, Geologist

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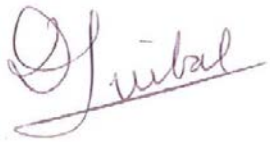
Email: dguibal@srk.com.au

I, Daniel Rene Guibal, FAusIMM(CP), do hereby certify that:

1. I am a Corporate Consultant (Geostatistics & Resources) with SRK (Australasia) Pty Ltd, trading as SRK Consulting, which is an international firm of consulting geologists and engineers, which has been practicing in this profession since 1974. I hold office at 10 Richardson Street, West Perth, WA 6005, Australia and have been employed as such since 1999.
2. I graduated from the Ecole Nationale Superieure de la Metallurgie et de l'Industrie des Mines de Nancy, France with an Engineering Degree (Ingenieur Civil des Mines de Nancy, 1971) and I have continually practiced my profession since that time.
3. I am a Fellow and Accredited Chartered Professional (Mining) of The Australasian Institute of Mining and Metallurgy, a Member of the Mineral Industry Consultants Association, Inc. and a Life Member of the Geostatistical Association of Australasia.
4. I have worked as a Mining Geostatistician for a total of 36 years since my graduation from University.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 of the Canadian Securities Administrators ("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 am a "qualified person" for the purposes of NI 43-101.
6. I am responsible for the preparation of Section 13 of the technical report titled "An Updated Integrated Development Plan for the Madaouela Project, Niger" and dated August 11, 2015 (the "Technical Report") relating to GoviEx Niger Holdings limited Madaouela Uranium Project. I have personally visited the Project in the field during the period January 12-16, 2010.
7. I have had prior involvement with the property and co-authored a previous 43-101 report and Preliminary Economic Assessment dated April 2011.
8. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
9. I am independent of the issuer applying the test in section 1.4 of NI 43-101.
10. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this August 11, 2015



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Timothy John McGurk

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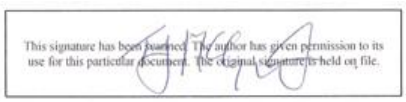
I, Tim McGurk, a Chartered Engineer, do hereby certify that:

1. I am currently employed as a consulting mining engineer to the mining and mineral exploration industry, as a Corporate Consultant Mining Engineering with SRK Consulting (UK) Ltd, with an office address of 5th Floor Churchill House, 17 Churchill Way, Cardiff, CF10 2HH, UK.
2. I graduated with a Bachelors of Engineering Degree, 2:1 Class Honours in Mining Engineering from the University of Nottingham, UK, in 1988.
3. I am a Chartered Engineer of the Engineering Council, UK since 1996. Membership number 487211.
4. I am a Fellow of the Institute of Mining, Metallurgy and Materials. Membership number 47795.
5. I have been employed as a mining engineer in the mining and mineral exploration business for the past 25 years, since my graduation from university.
6. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. The Technical Report is based upon my personal review of the information provided by the issuer. My relevant experience for the purpose of the Technical Report is:
    - Mining engineer, SRK Consulting from 2008 to date;
    - Principal adviser - mining engineering, Rio Tinto Technical Services, 2005 to 2008;
    - Mine manager, Lochaline Mine, 2003 to 2005;
    - Underground manager and mine extension project manager, Tara Mines, 2000 to 2003;
    - Mine superintendent, Black Swan Nickel Mine, 1996 to 2000;
    - Mining engineer production and design, Mount Isa Mines and Big Bell Mine, 1989 to 1996.
    - Experience in the above positions involved: design of underground mining operations; management of underground mine construction and mining operations including by room and pillar methods. Working with and reviewing assurance and quality control with owners, executive management, mine technical and operating staff, contractors and third party stakeholders.
    - As a consultant, I have managed and participated in the authoring of several multi disciplinary mining studies, including NI 43-101 Technical reports.
7. I have contributed to the preparation and review of the technical report titled "An Updated

Integrated Development Plan for the Madaouela Project, Niger" and dated August 11, 2015 (the "Technical Report") relating to GoviEx Niger Holdings Limited Madaouela Uranium Project; in particular the content of Sections 18, 20 and 21. I have personally visited the Project in the field during April 2012.

8. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
9. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, for which the omission to disclose would make the Technical Report misleading.
10. I am independent of the issuer applying all of the tests in Item 1.4 of National Instrument 43-101.
11. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible to the public, of the Technical Report.

Dated in Cardiff, United Kingdom, August 11, 2015.



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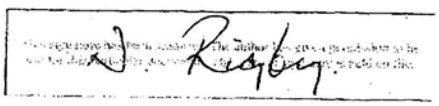
CERTIFICATE OF QUALIFIED PERSON

I, Neal Rigby, CEng, PhD, MIMMM do hereby certify that:

1. I am a Corporate Consultant of SRK Consulting (U.S.), Inc., 7175 W. Jefferson Ave, Suite 3000, Denver, CO, USA, 80235
2. This certificate applies to the technical report titled "An Updated Integrated Development Plan for the Madaouela Project, Niger, with an Effective Date of August 11th, 2015 (the "Technical Report").
3. I graduated with a BSc degree in Mineral Exploitation (Mining Engineering) with a first class honors in 1974 and a PhD in Mining Engineering in 1977 both from the University of Wales, UK. I am a member of the Institute of Materials, Mining and Metallurgy and a Chartered Engineer through the Council of Engineering Institutions. I have worked as a mining engineer for 39 years since my graduation from university.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I am responsible for peer review of the Technical Report in particular the content of Section 21.
6. I am an independent of the issuer applying all of the tests in section 1.5 of NI 43-101.
7. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement was lead QP for the PEA.
8. I have read NI 43-101 and Form 43-101-F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
9. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 11th Day of August 2015.



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Richard Ingram Skelton

Corporate Consultant (Mining Engineering)

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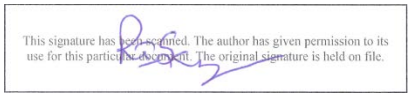
I, Rick Skelton, a Chartered Engineer, do hereby certify that:

1. I am currently employed as a consulting mining engineer to the mining and mineral exploration industry, as a Corporate Consultant Mining Engineering with SRK Consulting (UK) Ltd, with an office address of 5th Floor Churchill House, 17 Churchill Way, Cardiff, CF10 2HH, UK.
2. I graduated with a BSc (Mining) in 1969 and an MSc in 1974, both from the Royal School of Mines, Imperial College, London.
3. I am a Chartered Engineer of the Engineering Council, UK since 1974.
4. I am a Member of the Institute of Mining, Metallurgy and Materials since 1974. Membership number 43828.
5. I have been employed as a mining engineer in the mining and mineral exploration business for the past 43years, since my graduation from university.
6. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. The Technical Report is based upon my personal review of the information provided by the issuer. My relevant experience for the purpose of the Technical Report is:
    - Corporate Consultant / Principal Mining Engineer, SRK Consulting from 1996 to date;
    - Director and Marketing Manager, GEMCOM (Pty) Ltd. from 1990 to 1995
    - Director and Principal Mining Engineer, GEMCOM (Pty) Ltd. from 1984 to 1990
    - Senior Mining Engineer, Director and Department Head of Mining Planning, Steffen, Robertson and Kirsten Inc from 1977 to 1984
    - Senior Planning Engineer, Anglo American Corporation from 1975 to 1977
    - Open Pit Planning at Nchanga Consolidated Copper Mines Complex from 1969 to 1974
    - Experience in the above positions involved: design and planning of open pit mining operations; management of open pit mine construction projects, mining operations and open pit mine feasibility studies; working with and reviewing assurance and quality control with owners, executive management, mine technical and operating staff, contractors and third party stakeholders.
    - As a consultant, I have managed and participated in the authoring of several multi-disciplinary mining studies, including NI 43-101 Technical reports.
7. I have contributed to the senior review of the technical report titled "An Updated Integrated

Development Plan for the Madaouela Project, Niger" and dated August 11, 2015 (the "Technical Report") relating to GoviEx Niger Holdings Limited Madaouela Uranium Project; in particular the content of Section 15.

8. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
9. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, for which the omission to disclose would make the Technical Report misleading.
10. I am independent of the issuer applying all of the tests in Item 1.4 of National Instrument 43-101.
11. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible to the public, of the Technical Report.

Dated in Cardiff, United Kingdom, August 11, 2015



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CERTIFICATE OF AUTHOR

John Arthur

Principal Consultant (Geology)

SRK Consulting (UK) ltd

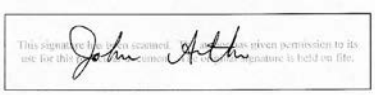
Email: jarthur@srk.co.uk

I, John Arthur, a Chartered Engineer do hereby certify that:

1) I am a Principal Consultant (Geology) with the firm of SRK Consulting (UK) Ltd ("SRK") with an office address of 5th Floor Churchill House, 17 Churchill Way, Cardiff, UK CF10 2HH;
2) I am a graduate of the University of Newcastle upon Tyne, UK in 1987 (BSc HONS Geology), the University of Leicester, UK in 1989 (MSc Mining Geology and Mineral Exploration) and and qualified with a PhD from Cardiff University, UK in 1994. I have practiced my profession continuously since 1994, initially as an exploration geologist based in South Africa in 1994 and 1995 before joining SRK in 1996 where I have worked extensively on gold exploration, evaluation and resource modelling projects. My main area of expertise is Mineral Resource Estimation techniques;
3) I am a Professional Geoscientist registered with the Geological Society of London as a Fellow (FGS) and have achieved chartered status as a CGeol (membership number 1005744);
4) I am a Professional Engineer registered with the Institution of Materials, Mining and Metallurgy as a Member and have achieved chartered status as a CEng (membership number 50132);
5) I have not personally inspected the subject project;
6) I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of National Instrument 43-101;
7) I am responsible for the preparation of the technical report titled "An Updated Integrated Development Plan for the Madaouela Project, Niger" and dated August 11, 2015 (the "Technical Report") relating to GoviEx Niger Holdings Limited Madaouela Uranium Project; in particular the content of Section 13;
8) I, as a qualified person, I am independent of the issuer as defined in Section 1.5 of National Instrument 43-101;
9) I have had no prior involvement with the subject property;
10) I have read National Instrument 43-101 and confirm that this technical report has been prepared in compliance therewith;
11) I have not received, nor do I expect to receive, any interest, directly or indirectly, in Madaouela Project or from GoviEx;

12) That, as of the date of this technical report, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading;

13) I consent to the filing of the technical report with any stock exchange and other regulatory authority and any publication for regulatory purposes, including electronic publication in the public company files on their websites accessible to the public of extracts from the technical report; and



Dr. John Arthur (CGeol FGS; MIMMM CEng)

Cardiff UK

August 11, 2015

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AN UPDATED INTEGRATED DEVELOPMENT PLAN FOR THE MADAOUELA PROJECT, NIGER – EXECUTIVE SUMMARY

1 SUMMARY (ITEM 1)

The Madaouela Uranium Project (the Project) is located near Arlit, in north central Niger, in one of the most significant areas of producing sandstone-hosted uranium deposits in the world. The Project is controlled 100 % by GoviEx Niger Holdings Ltd. ("GNH Ltd."), a wholly owned subsidiary of GoviEx Uranium Inc., a public company listed in Canada; collectively referred to herein as GoviEx.

The Project is an advanced stage property, for which five deposits have been developed to a high level of confidence (Marianne, Marilyn, Miriam, MSNE and Maryvonne). Current Project Mineral Resources are stated in Table ES 3. The study is based on detailed geological studies, metallurgical testing and processing options, mine design, infrastructure, rock mechanics, tailings and heap leach, hydrogeological and environmental impact.

Mining schedules have been developed including open pit mining at the Miriam deposit, and room and pillar mining with decline access at a combined 4,020 tonnes per day production rate from Marianne-Marilyn, MSNE and Maryvonne deposits. The base case mineral processing design is crushing, radiometric sorting, ablation and two-stage tank acid leaching, solvent extraction, with dry stack tailings. Utilizing Cyanex 600 resin allows both uranium oxide yellow cake and by-product molybdenum oxide to be produced as commercial products with acceptable quality for sale.

The Project development plan envisions an average 2.69 Mlb per year U_3O_8 yellowcake production rate over a twenty one year mine life, with a 93.7 % ultimate recovery of uranium. In addition it assumes recovery of on average approximately 592 t of MoO_2 per annum at a sales price of USD11.00 /lb, which revenue is used to offset processing costs. The base case project economics for this project at a long-term uranium price of USD 70 /lb U_3O_8 are positive, and indicate an after-tax NPV of USD 339 M (at 8 % discount rate) with an IRR of 23.5 % and a total life of mine ("LoM") net free cash of USD 1,126 M. Initial capital costs are estimated at USD 359 M, LoM capital costs at USD 676 M, and cash operating costs of USD 24.49 /lb U_3O_8 excluding royalties and including by-product credits. It is noted that Mo has not been included in the Mineral Resource model and hence is not considered to be at the same level of confidence as the U grades.

2 PROPERTY DESCRIPTION AND LOCATION

The Madaouela Uranium Project exploration property holdings extend over a large land area of approximately 1,596 km^2.

The property is located in the Agadez region (Arlit Department), in the Northern central part of Niger (Western Africa), southeast from the town of Arlit and west of the Aïr Mountain. The town of Arlit was founded in the 1960's, as a mining town, when the first uranium deposits were discovered by exploration department of the Commissariat à l'Energie Atomique

("CEA"). Arlit is located a distance of approximately 800 km by air from the capital city Niamey.

GoviEx acquired five mineral tenements in 2007, Madaouela I, II, III, IV, and Anou Melle, covering the historical resource areas drilled by CEA and PNC. Exploration efforts to-date by GoviEx have been concentrated the primarily on Madaouela I tenement.

The exploration permits were granted by the Government of Niger in exchange for a payment by GoviEx of € 25 M. GoviEx will pay an additional € 7 M, as a one-time payment, once any one of the five tenements, pursuant to the payment schedule agreed in 2007, is formally converted from an exploration permit to a mining permit.

The mineral exploration rights of the Madaouela property have been secured by GNH Ltd., after the signing on May 26, 2007 of the Madaouela I Mining Convention ("MIMC") together with its Side-Agreement ("MIMC-SA") between the State of Niger and GNH Ltd., as well as the Madaouela II, III, IV and Anou Melle Mining Conventions. The MIMC and MIMC-SA include and describe the respective rights and obligations of each of the signing parties for the management of the exploration program, and stipulate which further steps would apply should an ore deposit be discovered on the property.

The Exploration Title attached to the Madaouela I property is known as "Madaouela I" ("MIET"), and similarly for Madaouela II, III, IV, and Anou Melle tenements. Legal title to the property vested is GNH Ltd. on publication of Order No. 50/MME/DM dated June 4, 2007, published in the State of Niger official gazette.

In May 2010, the Minister of Mines and Industrial Development of Niger formally extended the expiry of the first validity period of the exploration licenses for the Madaouela I, II, III, IV and Anou Melle tenements to September 3, 2012 with no reduction in the area covered by the license. The extension was granted to compensate for interruptions to the Company's exploration activities at the Madaouela Project between August 2007 and November 2009 as a result of a government imposed state of alert.

On November 2, 2012, the Niger Ministry of Mines and Industrial Development granted a 36 months renewal of the Madaouela I, II, III, IV and Anou Melle tenements, reducing the original granted surface area by fifty percent.

On November 22, 2012 GNH Ltd. reapplied for the two halves of Madaouela I and IV that it had to relinquish as part of the renewal, which have been renamed Agaliouk and Eralrar. GNH Ltd. has been advised that the two applications are likely to be approved, but are awaited at the time of writing. Exploration rights apply primarily to uranium-bearing minerals but also encompass other related minerals that could be discovered on the property.

Taken together, the Mining Convention and the Exploration Title define the exploration and development rights of GoviEx for Mad I, II, III, and IV and Anou Melle tenements that comprise the Madaouela Uranium Project. The MIET is kept in good standing by conducting exploration work according to the work program agreed between GoviEx and the Republic of Niger.

On March 10, 2015, the Company completed and filed the Enviornmental and Social Impact Assessment ("ESIA") with the Nigerien Minister in charge of the Environment. This has been received by the ministry and has been approved.

The Mining Permit application was filed with the Minister in charge of mining on June 30, 2015 for the Madaouela I tenement, and renewal applications for Madaouela II, III, IV, and Anou

Melle tenements on July 1, 2015.

3 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES AND INFRASTRUCTRE

Access to Property

The main road (Arlit to Niamey via Agadez) heading south from Arlit, which is paved, goes through the tenements. The Madaouela camp is accessible through a dirt road branching off from the main road at the outskirt of Arlit. Off the main road, the use of 4x4 vehicles is required and access to all areas of the tenement is possible.

An airstrip belonging to Somaïr, a subsidiary of Areva, was constructed at the start of their mining operation. Subject to the owner's agreement and the payment of a landing fee the airstrip is used by charter companies. No commercial flights are available to Arlit. A recently renovated airstrip exists in Agadez, and is operational.

Topographic relief in the Project area is minimal, a few tens of meters from high to low. Areas of drilling are at approximately 430 m elevation (asl). Over much of the areas drilled, relief is less than 15 m from low ridges of sandstone outcrops to essentially flat sand covered plains. The nearest area of topographic significance is the Aïr Mountains, the west flank of which is located 15 to 20 km to the east of the Project area.

The Project is located in the Sahara desert climate zone and the average annual rainfall is expected to be less than 150 mm per year. Given the Sahara desert climate is fully arid, precipitation is less than evaporation and therefore water available for storage, runoff or recharge is minimal.

Average annual temperature ranges between approximately 15°C and 35°C, with a mean of 28°C. There are three seasons: a relatively cold season, September to February, a hot season, March to June, and a humid season, June to September. In the hot season temperatures can exceed 40°C and in the cold season nights are generally cool with temperatures below 20°C.

Local precipitation data for analysis has been obtained from two weather stations near the Project site named SOMAIR and Arlit. The average annual rainfall based on the data collected from Arlit between 1972 and 2010 is 56.5 mm. The average annual rainfall based on the data recorded at SOMAIR between 1981 and 2011 is 45.7 mm.

There is a clearly defined dry season between November and May with little or no rainfall. The wet season occurs between June and September, with the peak rainfall month being August. The dry season is characterised by contiguous periods of rainless months for all years whereas the wet season portrays variations within years in terms of rainfall distributions as well as the rainfall depth.

Physiography

Underground water resource are available and most of the water resources used in Niger come from fossil water tables, which are drawn down more and more while little, if any, replenishment takes place considering the precipitation regime.

In the vicinity or Arlit, carboniferous sandstone formations host underground water tables that have been exploited since the start of the uranium mining operations in the 1960's. The human concentration that has accumulated over the years in the Arlit area has contributed to reducing the underground water resources; however, water reserves in the region are

considered to be large.

Native vegetation is scarce and has almost completely disappeared due to the combined effect of human pressure, overgrazing and droughts. There is very little permanent vegetation on the Project area. However, at times, local herds use this territory as grazing land, mostly after the rainy season, which could turn the desert into a green countryside for just a few weeks.

The towns of Arlit and Akokan have a combined population of approximately 160,000, and were developed by Somair (Arlit, from 1968) and Cominak (Akokan, from 1975), to house the staff and workers required for the uranium mines of the two mining companies, which continue to operate today.

The proximity of the town of Arlit and Akokan are an asset for the Madaouela Uranium Project. The towns have electricity, potable water and two hospitals. Every administration is represented and the main banks have outlets. There are filling stations, bus transportation, repair shops; essentially all that is needed to sustain the community services that would provide for people associated with mine development.

Arlit is connected to the Southern part of Niger via the so-called “uranium road” through Agadez and Tahoua to Niamey, the Niger capital further south. This road is paved almost the entire way, but poor road maintenance has resulted in deterioration. Most of the goods and services that are necessary for people to live in Arlit and for the uranium industry to operate and produce come through that main access road.

A power line connects the town to the Sonichar coal-fired power station located north of Agadez.

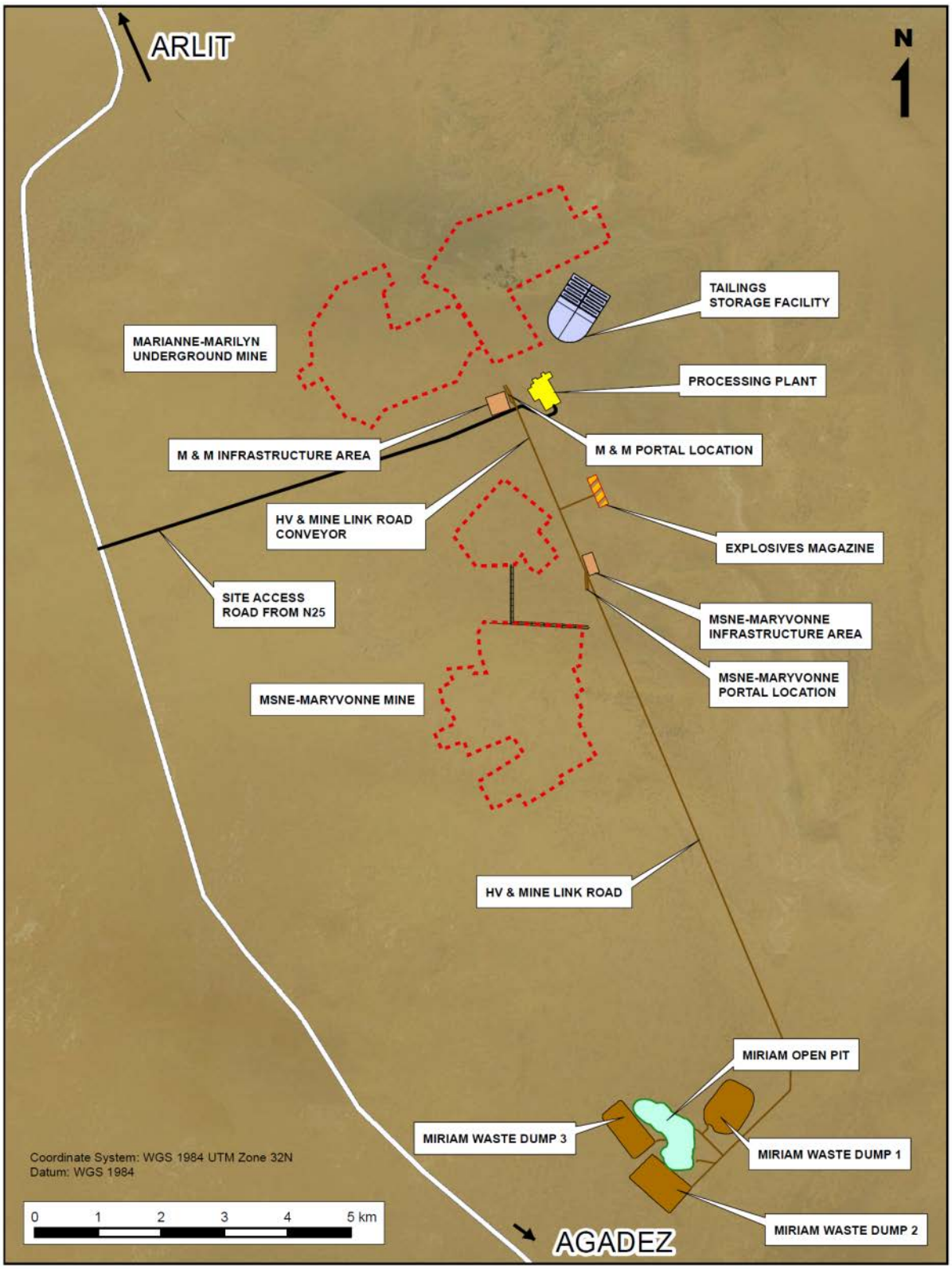

Figure ES 1: Site Layout Plan

4 HISTORY

The CEA conducted drilling operations using drilling grids of 800 m over large areas, and down to 100 m over two contiguous mineralized zones termed Marianne and Marilyn. The discovery of the Marilyn deposit was then drilled locally at 50 m and less spacing, and an underground mining test was implemented for detailed sampling mineralogical studies, processing tests and investigations into the global rock quality from a mining perspective. CEA also discovered the other deposits that are the current active mines in the area (the Somaïr and Cominak operations), and subsequently ceased exploration work on Madaouela in 1967.

The Japanese Power and Nuclear Fuel Development Corporation ("PNC") conducted additional uranium exploration work up to 1992 and produced a report on the feasibility of the Madaouela deposit in 1993, which was later updated in 1999. Historical mineral resources/reserves were stated at 5 to 15 Mlb U_3O_8, depending upon the cut-off used.

5 GEOLOGICAL SETTING AND MINERALIZATION

Regional Geology

The Madaouela Uranium Project properties are located in the Tim Mersoi Basin. This basin covers most of the western part of the Republic of Niger with extensions in Algeria, Mali, Benin and Nigeria. It opens and deepens toward the south and west. In early Paleozoic, an open gulf developed to the south of the Central Saharan Massif and fed continental sediments to the developing basin. During the Mesozoic and Tertiary, the area was mainly continental, periodically invaded by marine transgressions diminishing in thickness to the south and passing laterally into continental series. Uplift movements beginning in the Middle Eocene gave the basin its present aspect.

The Paleozoic sediments are outcropping between Arlit and the Algerian boundary. Pre-Carboniferous sedimentation consists of Cambro-Ordovician sandstones and graptolite shales. The Carboniferous formations are of major interest because they host the major reduced uranium deposit in the Arlit area. The stratigraphic sequence begins by the grey-black shale Talak Visean argillites.

This is followed by the Akokan unit ("UA") is a transitional term between the marine clay of Talak and the fluvio deltaic sedimentation of the Guezouman and Tarat. It consists of several lenticular fine grained silty clay units.

The Guezouman formation includes a lower and an upper member. It is composed of fine- to coarse-grained cross-bedded sandstone units with minor conglomerates (Teleflak) at the base. These contain quartz, phosphatic siltstone gravels, more or less deformed silty shale debris, metaquartzite, granite and rhyolite pebbles. The upper member, flowing southwest to south, consists of fine to medium grained sandstone, with minor siltstone and thin argillaceous intercalations.

Regional Geological Structures

The structure of the Tim Mersoi basin is marked by the westward dip of the units. The deformation of the sedimentary body resulted from basement fault activities located between the Air Massif and the Azaoua lineament. Several large faults systems cut the sediments and have played a major role during the sedimentation since the Upper Paleozoic.

Mineralisation

The Madaouela deposits exhibit classic characteristics of uranium sandstone deposits common the world over (Cuney and Kyser, 2009).

The mineralogy of uranium in the deposit is dominated by pitchblende and coffinite. The overall paragenesis could be divided in three stages: (1) early sulfides; (2) uranium on organic matter such as wood fragments; (3) carbonates and barite. The uranium minerals largely occur on the surface of minerals, or as infillings between the grains. Some brannerite may also occur in the deposit. Mineralogists never identified brannerite, and in the Ti-U diagram of SEM analysis, high values of uranium are independent of titanium whereas low values of uranium are associated with Ti minerals. The major carriers of uranium are therefore uranium oxide and not brannerite.

It is important to note that pyrite may have developed on large areas, but is now preserved only in the halo of the large redox front. Molybdenum is associated with pyrite as a trace element.

Geological Controls on Mineralization

The Guezouman sandstone at the Guezouman-Talak contact in the primary locus of mineralization, as controlled by the reducing environment and lesser permeability of the Talak argillites below mineralization, and the regional paleo-groundwater redox boundary in the Guezouman sandstone, down gradient from outcrops. Other relevant geological controls are the N70E structural, which represent older faults, and edges of paleo-channels. Low-amplitude domal features in the sedimentary units are related to the structural environment and are therefore relevant exploration guides.

Type, Character and Distribution of Mineralization

The uranium mineralization is all reduced uranium minerals (U (IV) minerals), uraninite and coffinite. The uranium minerals occur as disseminations in the matrix of the sandstone, with nearly all the mineralization occurring in one tabular horizon. At the favor of vertical “redox front”, uranium mineralization in the Guezouman may occur at several levels, as it is the case in the Miriam deposit. The Akouta “front” was the best example of this type of concentration. In the Miriam case a close relationship with structural features is very likely. Mineralization can sometimes be present at the contact of the Guezouman and the UA formation, in the Talak, and in the UA where the UA is preserved against a N70E fault; however, that mineralization is also relatively insignificant to the main basal Guezouman sandstone tabular lens of mineralization.

6 EXPLORATION

The main exploration tool used by GoviEx on the Madaouela Uranium Project has been by drilling on a defined grid pattern, and interpreting the presence of redox fronts or anomalous uranium intercepts to justify further drilling. Other exploration work completed on the Project includes; field mapping at MAD I in 2009-10, strip mapping along drill lines was completed at MAD I, MAD II and MAD III in 2011; MAD IV in 2012; and Anou Melle in 2014.

Remote sensing analysis by MIR Teledetection was completed over the whole project in 2009 and has greatly assisted in understanding the structural complexity of the area. This included obtaining quality satellite imagery SRTM for topography, Landsat and Aster imagery for spectral analysis and photo interpretations.

Mapping by GoviEx was carried out on the Marianne and Marilyn deposits. The mapping used preliminary photo interpretation, followed by field validation as access was restricted due to security concerns, associated with the military camp at the time. From field observations a number proposals were made for the mineralisation event and confirmed the structural complexity proposed by MIR Teledetection.

Between 2010 and 2012, strip mapping along regular spaced lines was completed by GoviEx over MAD I, MAD II, MAD III and MAD IV. The main purpose of the exercise was to validate the regional geological map data, followed by measuring the direction of paleo-flow to determine channel development and help in defining drill sites. The reading of radioactivity using the SPP2 spectrometer help identified potential target horizons. The geologist's primary task was the recording the lithology, stratigraphy, bedding orientation, presence of faults and fractures. Further to this once drilling was completed, it helped in the interpretation of drill sections.

In 2014, GoviEx completed field verifications on the Anou Melle licence. The main aim of the mapping programme was to confirm: the presence of faults interpreted by MIR Teledetection in 2009; previous work by CEA undertaken in the 1960's; and to confirm the stratigraphy and to verify a number of surface radiometric anomalies.

Extensive surface and sub-surface exploration has been conducted by GoviEx at Madouela using industry best practice for the style and extent of mineralisation, which occurs here. The detailed and regular spaced drilling has allowed the deposits to be outlined with a high degree of confidence and, coupled with the field mapping, structural, hydrographic and remote sensing analysis, has enabled the identification of additional potential.

7 DRILLING

The GoviEx exploration program commenced in August 2008, following property acquisition in 2007. Between 2008 and 2010 majority of the drilling undertaken was focused on the Mad I property, and was a combination of exploration and in-fill resource definition drilling. Drilling is by mud-rotary drill rigs that drill 120 mm diameter holes.

Table ES 1 summarises GoviEx's exploration drilling program for the period August 2008 to December 2013.

Table ES 1: Summary of Mad I drilling program metres for the period August 2008 to December 2013

	Year	2008	2009	2010	2011	2012	2013	Total
Deposits	**Type**	**(m)**	**(m)**	**(m)**	**(m)**	**(m)**	**(m)**	**(m)**
MAD 1 CUMUL	Water well	84	0	561		745	403	1,793
	RDH	57,162	90,204	96,717	59,637	153,690	57,186	514,596
	mixed RDH-DDH	316	179	0	1,189	6,344	3,424	11,452
	Reopening historical DH	5,086	15,452	7,592	4,488	5,049	0	37,667

Note: Mad I Cumul includes Madaouela I and Agaliouk

The program started in August 2008, after permission was granted to conduct drilling west of the Madaouela Army Base, and initially focussed on development drilling of Marianne on a 50 m grid with at total of 62,648 m drilled in 2008.

Drilling east of the army camp was authorized in April 2009, enabling drilling to commence the

50 m grid on Marilyn. Development drilling on Marianne and Marilyn has continued until March 2013 as the size of the deposit has increased and further in-fill definition drilling has been required.

In order to increase the global potential and define inferred resource in areas historically drilled on a large grid (800 m to locally 400 m), from 2009 drilling targets were identified in the area south of the Marianne deposit. The drilling targets were identified by similarity with the geological controls defined for the Marianne-Marilyn mineralization as previously discussed.

Known historical drill intercepts that were in the suspected redox boundary trend line were drilled on a 200 m grid basis.

One area was discovered in the north east (MSNE) and two smaller zones are currently being recognized further south (MSCE). Drilling since February 2010 in the southern part of the area has confirmed the newly discovered Miriam deposit, together with the discovery of a new area termed the MSEE (Mad South Extreme East) deposit, of similar thickness and grade to the MSNE deposit. In 2012 the Maryvonne deposit was discovered.

Between 2008 and the end of 2013 a total of 514,596 m were drilled on the Mad I and Agal licenses with 4,858 holes. Included in this drilling was the discovery of the La Banane deposit, which is the first deposit to be discovered in the Madaouela sandstones.

Madaouela II, MAD III, IV, Eral and Anou Melle

Initially exploration on these properties started in April 2010 and continued until 2013 except at Anou Melle where it ended in July 2010. The distance to the drilling camp and the poor state of available drilling rigs at the time, resulted in an earlier stop to this program.

The subsequent exploration activity then was concentrated east of the Madaouela fault on Mad II, Mad III, Mad IV and Eral from August 2010 to January 2013, following the purchase of three new rotary drills by GoviEx. The exploration was conducted at 3,200 m grid on sub EW profiles, and following redox interpretation profiles at a 1,600 m grid were drilled on the northern part of Mad IV and Mad III.

Table ES 2: Summary drilling activities by tenement to January 2013

	Metres (m)
Mad II	12,629
Mad III	16,716
Mad IV	25,272
Anou Melle	3,263
Total	**57,880**

Surveying

Surveying is done with precision, care and crosschecking in the field, using the DGPS equipment; thus providing collar locations to centimetre accuracy. Surveying uses a network of permanent survey monuments for base stations and is tied to real-world coordinates using WGS 84 as a format.

Logging

Logging is done using; three SEMM designed logging units, with probes modified for GoviEx. Internal QA/QC of intersections greater than 100 raw c/s (GM probe) is conducted using one Mount Sopris logging unit (GNSA owned) equipped with DHT27 Areva probe. For each drillhole logging unit, two probes are used;

- a resistivity and natural gamma (scintillation(PM)) probe,
- and a probe containing natural gamma (by Geiger tube GM) and by scintillation counter (PM)) and deviation (magnetic/inclinometer) instrumentation. GM logs are used to define in-situ uranium grades for the drillhole database.

The procedure used by GoviEx at Madaouela is to convert CPS per anomalous interval by means of a correlation curve developed by comparing core intervals with gamma-log intervals for the 46 core hole intervals drilled at Marianne. The process involves re-positioning the core pieces for the whole-core interval of mineralization, and determining the contacts and peak radiometric reading with a hand-held scintillometer on the core. This is then matched with the radiometric curve developed from a down-hole plot of CPS. The core is cut and analyzed for uranium content for the same interval as the radiometric indicate. A best fit line defines the relationship of GT as follows:

$$GT_{core} = U_{core} \times T_{core} = (Factor \times CPS \times T_{probe}) = GT_{probe}$$

The same can be done on composited grade (U%) versus (CPS) at a given composite interval for each; the relationships have been found to be similar to that for GT. The factor is then used to convert CPS to eU grade.

GoviEx has found that the coefficient of correlation between GT_{core} and GT_{probe} is 0.968, with a 2 sigma precision on the mean of 8.1 %; a relatively close clustering of data along a linear relationship.

8 MINERAL RESOURCE ESTIMATES

For the narrow flat lying mineralisation that occurs in most of Madaouela Uranium Project deposits (MSNE, MSCE, MSEE, and Maryvonne), a 2D approach has historically been used as an appropriate resource modelling technique for the potential Underground mining method. A more recent update of Marilyn/Marianne (M&M) has used a 3D unfolding approach allied with conventional block kriging, in order to smooth the process to mine scheduling. Both approaches are suitable for the flat lying mineralisation. For the potential Open Pit mining at Miriam a 3D recoverable resource approach (UC) was utilised.

Mineral resources in this report are defined using the convention of “eU” in kg/t for grade. The eU distinquishes concentration as being uranium concentration largely defined from gamma spectrometry with only a small portion of core chemically assayed. This does not compromise the integrity of the result but only seeks to demonstrate source of the information. This value can be related to eU_3O_8 concentration by a factor of 1.17924. Uranium concentrations determined by chemical assay are shown simply as U or U_3O_8.

The Classified Mineral Resources at a 0.4 kg/t eU cut-off (approximately 0.047 % eU_3O_8) and a minimum thickness of 0.4 m (except for Miriam, where a Mining selectivity unit (SMU) of 5 m x 5 m x 0.4 m is assumed) are summarised in Table ES 3.

Table ES 3: Summary of the classified mineral resources in accordance with CIM guidelines for Madaouela (cut-Off: 0.4 kg/t eU) as of April 20, 2015

Classification	Tons (Mt)	Grade (kg/t eU_3O_8)	eU_3O_8 (t)	eU_3O_8 (Mlb)
Marianne/Marilyn				
Measured	2.14	1.79	3,835	8.45
Indicated	14.72	1.43	21,000	46.30
Inferred	5.04	1.17	5,910	13.02
Miriam				
Measured	9.62	1.08	10,397	22.92
Indicated	2.68	0.79	2,112	4.66
Inferred	0.58	1.33	773	1.70
MSNE				
Indicated	5.05	1.61	8,111	17.88
Inferred	0.10	1.34	131	0.29
Maryvonne				
Indicated	1.23	1.79	2,195	4.84
Inferred	0.42	1.66	703	1.55
MSCE				
Inferred	0.72	1.81	1,308	2.88
MSEE				
Inferred	1.45	1.64	2,373	5.23
La Banane				
Indicated	1.57	1.64	2,589	5.71
Inferred	1.15	1.18	1,358	2.99
TOTAL MEASURED	**11.76**	**1.21**	**14,232**	**31.37**
TOTAL INDICATED	**25.25**	**1.43**	**36,007**	**79.39**
TOTAL INFERRED	**9.46**	**1.33**	**12,556**	**27.66**

**On November 22, 2012, GoviEx submitted to the Niger authorities a license application covering certain portions of the original Madaouela I and IV licences surface areas that were excluded from the renewed licences granted on November 2, 2012. GoviEx has been advised that the two applications for the excluded areas of Madaouela I and IV will be approved, but is awaited at the time of writing its applications, and hence has not adjusted its resources to account for any potential changes. However, it should be noted that resources associated with MSEE, and La Banane would be materially affected, and to a limited extent Miriam's resources would be affected should GoviEx not be successful in its application.*

The Company's mineral resources as at April 20, 2015 are classified in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum's "CIM Definition Standards - For Mineral Resources and Mineral Reserves" in accordance with the requirements of National Instrument 43-101 "Standards of Disclosure for Mineral Projects" (the Instrument). Mineral reserve and mineral resource estimates reflect the Company's reasonable expectation that all necessary permits and approvals will be obtained and maintained.

Mineral resources that are not mineral reserves do not have to demonstrate economic viability. Mineral resources are subject to infill drilling, permitting, mine planning, mining dilution and recovery losses, among other things, to be converted into mineral reserves. Due to the uncertainty associated with inferred mineral resources, it cannot be assumed that all or any part of an inferred mineral resource will ever be upgraded to indicated or measured mineral resources, including as a result of continued exploration.

The resource estimation performed at Madaouela confirms that both the 2D approach used by GoviEx, and the 3D approach for the larger and more complex deposits (MM and Miriam) are valid and that, more specifically, both the co-kriging and the Residuals method work very well. The choice between the methods is a matter of convenience, although SRK gave preference to co-kriging, because fewer assumptions are required.

The present resource update has ignored the small number of "secondary" intersections higher up in the sequence, the continuity of which is difficult to assess. Some of them might

give rise to minable ore in the future, depending on the mining method. In that sense the estimation can be considered as conservative. In addition, there is scope for additional mineralisation outside the currently recognised zones, particularly in the southern licenses and along the redox front.

The quality of the estimation is good where the drilling grid is at or below 50 m x 50 m, and reasonable at 100 m x 100 m, which allows a significant portion of the Mineral Resources to be classified as Indicated with selected areas of Marilyn-Marianne and Miriam to be classified as Measured where detailed infill drilling is available (30 m x 30 m).

The current estimation suggests that in-filling to at least 50 m x 50 m as a minimum, is necessary to better define the envelope of the mineralisation and where the geology appears complex closer spaced drilling is required. Miriam has been the subject of a very intensive drilling campaign, which has clarified the geological model and allowed SRK to upgrade the classification significantly.

The molybdenum grade of the deposit has been taken from testwork as the current resource model does not account for by-product elements. The range of molybdenum concentrations in the deposit is high from 300 ppm to more than 4,200 ppm based on several studies (CEA 1965, GoviEx data unpublished, SGS 2009 2013, and Mintek 2012). The molybdenum occurs in close proximity to uranium but rarely overlapping as it is formed in more reducing environments than where the uranium is precipitated hence molybdenum grade cannot be reliably deduced from knowledge of uranium grade.

For purposes of determining reagent consumption related to molybdenum and possible product generation the grade of 1,200 ppm was taken from multiple analysis of the 8 t bulk metallurgical sample that was mined from M&M and analysed at Mintek. In addition testwork at Ablation technologies on Miriam ore gave an average of 850 ppm. In order to provide a conservative approach the 850 ppm number was used. When diluted in the ore feed this grade decreases to 472 ppm.

9 MINERAL PROCESSING AND METALLURGY TESTING

Metallurgical testwork has been completed through SGS Lakefield, Canada and Mintek, RSA. This testwork has comprised mineralogical investigations, geometallurgy, bottle roll and column leaching involving acid and alkaline leaching, acid pugging, two stage acid leaching, pre-concentration by radiometric sorting and flotation, communition testwork, uranium recovery assessment by precipitation, ion exchange and solvent extraction as well as assaying of yellowcake product.

Cytec Industries (Cytec), Black Range Minerals Ltd and Australian Nuclear Science and Technology Organisation (ANSTO) have undertaken studies to improve the efficiency of several of the processing stages.

Cytec investigated the use of a new SX reagent (Cyanex 600) for an improved recovery of molybdenum and uranium from the pregnant leach solution producing both to acceptable saleable products.

Black Range Minerals and Ablation Technologies focused on optimisation of the comminution stage of the processing circuit by the addition of an ablation stage to reduce the amount of feed material requiring acid leaching, upgrading uranium and reducing carbonate content.

ANSTO assessed the potential to recover the various acids used in several parts of the

circuit.

Comprehensive geometallurgical work has defined uranium mineralization present as coffinite (60 %) and uraninite (40 %) with negligible other phases such as autinite and becquerlite at Miriam and rare silica-mix-TiO_2 minerals. Uranium minerals occur with interstitial clays and carbonate in the cement of the Guezoman and Madaouela sandstones. Grain size is bi-modal with coarse and fine grained uranium minerals. Uranium minerals are present as fine grained, typically less than 30 µm with occasional grains up to 100 µm in size. The most abundant uranium-bearing minerals in the sample are "pitchblende/silica-mix-TiO_2", "coffinite-mix-TiO_2" and coffinite. Uranite and autunite are less abundant. The "pitchblende/silica-mix-TiO_2" and "coffinite-mix-TiO_2" phase contribute 65 % of the total uranium. The uranium-bearing minerals in the high-grade ore mostly report to the 10-25 µm size fraction.

Molybdenum occurs in the ores largely as a trace element in coffinite and pyrite with only minor molybdenite identified. In samples from the ore stockpile from M&M, powellite ($CaMoO_4$) was also identified.

An 8 tonne metallurgical sample was collected in 2011 from existing high grade (4 t), low grade (2 t) and waste rock material (2 t). This simulates the potential feed from underground operations and overall grade was determined to be approximately 1.5-1.6 kg/t $U_3O_{8.}$ And approximately 1,200 ppm Mo. GoviEx staff collected the sample on site from stockpiles at M&M created during test mining in the 1960's. In addition approximately 450 kg of Miriam ore from drill core was evaluated at SGS Lakefield.

The sample consists of predominately quartz followed by K-feldspar (in the form of microcline, 3.7 mass %). Plagioclase was not detected in analysis one. The quantity of kaolinite is similar to K-feldspar (3.5-3.7 mass % and 5.2-5.5 mass % for analysis one and two respectively). Chlorite and muscovite were not detected in analysis one. Small quantities of Ca bearing phases (calcite, dolomite and gypsum) were noted together with: the presence (insignificant) of plagioclase, muscovite, chlorite and hematite.

The different quantities of kaolinite and muscovite (illite) in the XRD spectra, indicate that these clay minerals are not ubiquitous, and occur sporadically as alteration products, within microcline and plagioclase clasts in the sandstone in <5 mass %. The clay minerals kaolinite and illite are therefore not expected to have a major effect on the percolation of the lixiviant through the uraniferous sandstone particles, owing to their heterogeneous dispersal in the sample. The petrographic analysis shows that the matrix of the high-grade sample contains up to 2 % calcite, which will consume acid.

Bond ball work index ("BBWI") tests were carried out for the high grade ("HG"), low grade ("LG") and waste rock samples. The tests were conducted in a standard Bond tumbling ball mill with dimensions 305 mm x 305 mm with no lifters. Based on the typical classification table for BBWI the HG ore sample contained hard and very hard components, the LG ore sample contained medium hard and hard components and the waste rock sample contained hard components. The breakage characteristics of the HG ore and waste rock sample are approximately the same at 300 & 212 µm and 212 & 150 µm respectively

A composite sample (100 % passing 30 mm) comprising low grade (30 %), high grade (35 %) and waste rock (35 %) were prepared for the beneficiation testwork. In addition to the RADOS testwork (equivalent to radiometric sorting), Mintek also conducted scrubber and flotation testwork to assess the effectiveness of this process in upgradingthe concentrated uranium feed.

Radiometric sorting proved successful with 98.5 % of uranium recovered into a 56 % mass pull with rejection of 40 % of the material. Based on the operations of radiometric sorting at a uranium mine in Ukraine it is established that 93 % of uranium recovered in a mass pull of 40 % with resue ore or 98 % in a mass pull of 56 % in non-resue ore can be applied as potential targets for Madaouela ore. However due to collection of fines calcite was also concentrated by this approach.

The objective of the scrubber testwork was to recover the uranium that was indicated to occur with the fine interstitial cement and separate this from the coarser quartz particles in the sandstone. The purpose of this is to liberate and concentrate finer uranium bearing particles without concentrating gangue so as to reduce the mass of ore that requires leaching. Due to the hardness of the sample (as indicated in the BBWI testwork) the U_3O_8 could not be efficiently liberated from the GoviEx composite sample.

Six flotation tests were conducted using various test setups, various conditioning reagents and at various pH values. The optimum test condition was reverse uranium flotation on a deslimed sample with an ore pH of 7.5, using: yellow dextrin and fuel oil as conditioning reagents at P_{80} of 212 µm. The float resulted in a 96 % recovery of uranium, 53 % rejection of the carbonate material and 5 % reduction in leach feed mass.

Ablation works by combining the ore with water to form a slurry and impacting two slurry streams together to form a high energy impact zone that separates heavier uranium minerals from the host rock. Several different slurry concentrations were assessed. The optimum concentration was 20 %, although this will be refined during larger pilot trials. The 20 % TSS trial resulted in provisional recoveries of approximately 99 % of the contained ROM uranium into the carrier water and fines fraction (-149 µm) using a two stage ablation process. Based on further testwork on coarse fraction Ablation propose that this is possible to reach 95-99 % during the greater throughput pilot plant trial and full scale plant. The overall mass of the feed requiring milling and leaching is reduced by approximately 64 %. Importantly more than 60 % of the calcite is rejected prior to acid leaching.

Alkaline leaching even after 48 hours with high sodium carbonate concentration (30 g/L carbonates and sodium bicarbonate was added to a concentration of 10 g/L bicarbonate) produced a maximum extraction efficiency of 70 % uranium, however after 2 hours the extraction efficiency was already 60 %. The kinetics of the alkaline leach is somewhat comparable to the acid leach kinetics, however the final extraction efficiencies are significantly lower (particularly when compared to the acid pugging leach extraction efficiencies).

Acid leaching was demonstrated in diagnostic leach tests and followed up in column leaching, acid pugging and simulation of two stage acid leaching. The extent and kinetics of uranium dissolution were determined as a function of grind size, acidity, pre-treatment, and temperature and re-grinding. The results from these testworks are summarised in this section. There was no significant difference in the uranium extractions after 24 hours in relation to differing grind sizes (60 µm, 100 µm, 300 µm and 500 µm), however the rate of leaching increased with decreasing particle size.

The effect of altering the pH and residual sulfuric acid concentrations (pH 1.5, pH 1.85 and 25 g/l) during the acid leach showed no significant difference in the uranium extraction for the tests after 24 hours. It should be noted that the pH variation tests were carried out at a coarse grind (500 µm) and 60 ^{o}C. Acid curing also did not result in any significant effect on uranium extraction efficiency after 24 hours.

The highest uranium extraction efficiency was achieved by re-grinding the residue from a coarsely ground leached sample and re-leaching it. An increase in leach temperature (from 60 to 80 ^{o}C) increased leach kinetics and acid consumption (70 to 89 kg/t respectively) with no significant improvement in uranium extraction efficiency.

A series of tests was conducted by Mintek. These utilised a multi-stage leach approach using the filtrate from one leach to act as the lixiviant in the next leach. This was repeated for 3 stages and reported between 98 to 99 % extraction of uranium. Acid consumption in these tests was approximately 85 kg/t.

The extraction efficiencies obtained during this testwork for acid pugging was consistent with the previous testwork results and indicate that approximately 95 % are achievable using the acid pugging process. The extraction efficiency was proportional to duration of the acid curing, with the highest achieved after 9 days (96 %), however >90 % could be achieved with one day curing and with elevated temperature. However due to calcite content the acid consumption by this approach was high (100-120 kg/t).

For all sample types the oxidative steady and increased acid tests outperformed the non-oxidative steady acid test. The optimum conditions for maximum U_3O_8 dissolution for all the samples were oxidative state and increased acid feed. The extraction efficiency of 96 % is encouraging as the material only needs to be ground to <20 mm, however the efficiency of this process is dependent on the clay content of the ore.

The presence of high molybdenum in the pregnant leach solution required assessment of its removal by ion exchange process. Resin in pulp recovery was evaluated but due to the coarse grain size of the leached ore was not applicable.

Solvent extraction has been evaluated and found to be the most appropriate approach to uranium extraction from the pregnant leach solution and collection prior to precipitation of yellowcake. Assuming good pre-stripping or separation of molybdenum can be achieved; the final yellowcake product meets Cameco requirments for saleable yellowcake.

In addition to the above mentioned requirement for pre-stripping of molybdenum prior to the extraction of uranium, Cytec have conducted tests on a newly developed solvent Cyanex 600 that can be operated at low pH (less than 2) and can recover and separate both urnaium and molybdenum from the PLS with few other impurities and then through successive stripping stages can produce separate molybdenum and uranium products. This step results in the recovery of 99 % for uranium and 98 % for molybdenum into separate streams. Both products have acceptable quality for sale.

Overall metallurgical recovery of 93.7 % for uranium can be achieved by this approach.

10 OPEN PIT MINING

An open pit study was conducted on the Miriam deposit including the following:

- Pit optimisation to determine the optimal pit shape to carry forward to design;
- Mine design of the selected pit shell and waste dump designs;
- Life of mine schedule;
- Equipment and labour requirements; and
- Cost estimate.

Pit Optimisation

Pit optimisation using the Lerchs-Grossmann algorithm was carried out on the Miriam deposit to determine the optimal shape for an open pit, which is used as the basis for pit design. For the purposes of this study, SRK has considered that only Measured and Indicated Resources contribute revenue in the optimisations. The classification has been carried through to the mining schedule and the economic analysis. The pit optimisation results form the basis of the mine design and schedule.

Table ES 4: Selected pit shell results

Optimisation Results	Units	Value
Diluted Ore	(Mt)	7.9
	(kg/t U)	0.82
	(t U)	6,492
Quantities		
Total Rock	(Mt)	90.8
Waste	(Mt)	82.9
Stripping Ratio	(t:t)	10.5

Mine Design

The engineered final pit design for the Miriam deposit is based on the selected pit shell. Where possible, ramps are located where practical within the constraints of the location of the waste dumps and run of mine ("RoM") stockpile.

The RoM quantities within the final pit design are shown in Table ES 5. RoM has been separated into high grade ("HG") and low grade ("LG"). The final Miriam pit design is approximately 1,400 km long, 550 m wide and reaches a maximum depth of 120 m.

Table ES 5: Miriam final pit design quantities

Material	Tonnage (Mt)	Grade (kg/t eU)
High Grade (>0.6 kg/t U)	5.2	0.98
Low Grade (0.4 ≥ kg/t U ≤ 0.6)	2.3	0.49
Total RoM	**7.5**	**0.82**
Waste	91.8	
Total Movement	99.4	
Strip Ratio (t_{waste}/t_{ore})	12.2	

Mine Schedule

The open pit life of mine ("LoM") plan is based on the final pit design and three pushback pit shells. The key scheduling parameters were as follows:

- Plant feed target of 1,400 ktpa;
- Limit total material movement ex-pit to 19.5 Mtpa;
- Two years of pre-production waste stripping;

Loading and Hauling

The equipment fleet has been selected based on the outcomes of the LoM schedule, material movement profile, required selectivity of the orebody and mining space within the pit and designs and pushback shells.

The primary shovel selected for mining ore and waste are 5.7 m^3 bucket capacity excavators . A 44 t off-highway truck will be used as the primary truck.

All ore mined will be stockpiled at Miriam until scheduled as feed for the processing plant. A 3.5 m^3 front-end loader will load Run of Mine ("RoM") ore from the stockpile onto 24 t articulated trucks (secondary trucks) for haulage to the processing facility.

Drilling

All material above the 400 mRL production drilling will be undertaken on a 5.7 m by 4.9 m spacing on 10 m benches with a 171.5 mm blasthole diameter. All material below the 400 mRL 2.9 m by 2.5 m drill patterns will be used on 5 m benches with an 88.9 mm blasthole drill. Pre-split drilling will be undertaken with an 88.9 mm blasthole drill rig. The 171.5 mm drill rig will also be used for infill drilling on a 25 m by 25 m spacing for 10 m bench heights.

Equipment Requirements

The 8 year life of pit annual equipment requirements are shown in Table ES 6. (Years -2 and -1 represent the two year pre-production period). The ancillary equipment requirements have been based on the material movement profile, the primary equipment size and requirements, labour requirements and the number of working areas.

Table ES 6: Annual open pit selected equipment requirements

Equipment Requirements	Units	Maximum	-2	-1	1	2	3	4	5	6
Primary Shovel	(#)	4	3	4	4	4	4	3		
Primary Loader	(#)	1			1	1	1	1	1	1
Primary Truck	(#)	17	8	16	17	17	17	11		
Secondary Truck	(#)	11			10	11	11	11	11	10

Labour Requirements

The Arlit region has a history of mining and provides a workforce familiar with the techniques and skills required for the proposed operations. Niger has its own mining school and the experience of the existing mines is that it provides competent professionals ready for the industry. SRK has assumed that the Madaouela project can be staffed almost entirely with Nigerien labour for both technical and labour functions. Any requirement for expatriate assistance is assumed to be covered by the general and administrative costs.

The project schedule is based on four work crews working three 8-hr shifts per day. This shift structure complies with Nigerien legislation and removes any requirement for overtime payments. Office staff are assumed to work a standard 40-hr work week.

A summary of the estimated annual labour requirements is provided in Table ES 7.

Table ES 7: Annual open pit labour requirements

Labour Requirements	Units	Maximum	-2	-1	1	2	3	4	5	6
Mine Operations	(#)	187	91	134	182	187	185	158	48	44
Mine Maintenance	(#)	62	38	46	58	62	54	58	21	21
Technical Services	(#)	20	20	20	20	20	20	20		
Total Open Pit Staff	(#)	269	149	200	260	269	259	236	69	65

Capital Costs

The majority of the mining equipment capital costs for the Madaouela project have been sourced from the manufacturers. Replacement costs throughout the mine life are based on the estimated equipment life for each unit. All equipment assumed to be replaced at full purchase cost. No allowance for salvage values of the mining equipment has been considered in this study.

A summary of the estimated capital costs for open pit mining is provided in Table ES 8.

Table ES 8: Open pit capital cost estimate

Capital Costs	Units	Total
Equipment Capital	(USDm)	29.9
Equipment Replacements	(USDm)	0.3
Miscellaneous	(USDm)	0.8
Total	**(USDm)**	**31.1**

Operating Costs

Operating costs have been developed from first principles. The results from the cost estimate are presented in Table ES 9 split by category. The costs are inclusive of the RoM rehandle cost.

Table ES 9: Open pit operating costs

Category	Units	Unit Costs
Mine Staff	(USD/t ex-pit)	0.22
Loading	(USD/t ex-pit)	0.23
Hauling	(USD/t ex-pit)	1.04
Drilling	(USD/t ex-pit)	0.12
Ancillary	(USD/t ex-pit)	0.29
Blasting	(USD/t ex-pit)	0.56
Total	**(USD/t ex-pit)**	**2.47**

11 UNDERGROUND MINING

Marianne-Marilyn (M&M) and MSNE-Maryvonne deposits are to mined by room and pillar mining methods for the following reasons:

- Allows pillars to be left in areas of low grade;
- Allows height of mining to be adjusted to reflect the variable nature of the mineralisation;
- Flexibility to react to localised variations in the deposit;

- No specialised equipment required;
- Reduced training requirements as local workers already familiar with the mining technique; and
- Demonstrated as being an effective method in similar conditions.

A common approach to mining has been assumed for all underground deposits to allow the same skills and equipment to be applied across all operations. The modular nature of room and pillar mining in panels allows the inputs to production costs and productivities to remain relatively stable across all operations, simplifying the analysis and ultimate operations.

Due to the shallow depth of mineralisation, which is typically around 100 metres below the surface for the Madaouela deposits, decline access has been adopted. This has the following additional advantages;

- Enables the use of conveyors to bring the mineral ore to surface.
- Simplifies materials handling to bring spare parts and other consumables into the mine.
- Simplifies personnel access during shift and change-over periods.
- The processing plant is envisaged to be at a distance from the deposits meaning any lateral distance covered by a decline will reduce the transportation distances on the surface.

Vertical development requirements are limited to providing ventilation raises. The proximity of the mine workings to the surface enables primary ventilation of working areas by use of many small diameter raises regularly distributed throughout the orebody. This is a practicable alternative to fewer, more isolated, large diameter raises and an effective way to provide safe working conditions underground by quickly eliminating radon emmissions generated in the mine from working places. The requirements for vertical development are therefore anticipated to be relatively high.

Materials Handling

The mine has been laid out for conveyor haulage and consists of a central conveyor network to the surface that is fed by trucks hauling from the panels.

This approach greatly reduces the number of trucks required on the mine and improves the ventilation efficiency of the operation. The use of trucks to haul from the panels to the central conveyors is a very flexible approach, which allows ore production to transition between panels without the need to establish conveyor infrastructure. As high ventilation volumes are required to dilute radon emissions, this will also provide sufficient ventilation for diesel equipment operating in the panel. Where mining districts have relatively small tonnages, no conveyor is installed and a single end access into the panel is developed.

Loading

Faces are to be cleaned to stockpiles in central locations in each panel, from where ore and waste are to be rehandled by a larger loader into the trucks. The room and pillar method will create many stockpiling locations close to the load points and the trucking fleet is decoupled from the production fleet increasing their productivity and the flexibility of scheduling. The trucking fleet will also transfer between production panels and development/waste mining, without affecting the productivity of mining from the face (this assumes the trucking fleet productivity is larger than the low-profile loading).

Trucks will tip onto feeder breakers, which will load the conveyors. Feeder breakers are mobile to allow transfer points to be moved quickly and simply as production areas advance

and conveyors are extended, to minimise trucking distances. They can be used to load conveyors at any point along its length.

Ground Control

Scaling of roof and sidewall is required for all work places following blasting and subsequent check scaling to test the integrity of the ground over time. Scaling is assumed to be undertaken using manual methods due to the low-profile production areas limiting the application of mechanical scaling. The sub-horizontal nature of the sedimentary bedding should result in scaling being simple and effective.

A bolting pattern has been applied to underground mining using 1.8 m resin bolts with a bolt spacing of 1 m and 1.5 m between rings. For production areas, the resin bolt lengths are reduced to 1.5 m to account for the limited roof height. No surface support (mesh or shotcrete) is applied as the sub-horizontal bedding planes provide favourable roof conditions.

Equipment

Development Fleet

The central access, infrastructure and panel access development are treated separately from the production activities due to the higher mining heights used. This allows a conventional fleet of development equipment to be used. It is estimated that the maximum annual development requirements are in the order of 10,200 m per year (though a typical year would be closer to 7,500 m per year), which requires one development suite for the equipment fleet.

Production Fleet

The production fleet is constrained by the working height of the proposed excavations. As the majority of the mineralisation has a thickness less than the minimum mining height, production mining will be largely by low-profile equipment. In most cases, the minimum mining height will be determined by the equipment selection. Low profile mining equipment developed for and used extensively in South African platinum and chrome operations is specifically designed to reduce the total mining height and reduce waste dilution in low seam environments.

Ore production rates of 1,447,000 tpa (4,020 tpd) by conventional mining methods are planned. In total, four production fleets will be required to achieve this rate.

Materials Handling Fleet

The materials handling fleet comprises a rehandle loader and truck fleet and conveyors to transport RoM from the production face to the processing plant, and waste from the development face to the surface stockpile.

Underground Life of Mine Plan

The underground Life of mine plan (LoMP) combines production from Marianne-Marilyn and MSNE-Maryvonne into a single schedule. The schedule assumes one development crew in operation at any time. All panel and infrastructure development in Marianne-Marilyn is scheduled to be completed before the decline access to MSNE-Maryvonne commences. The underground LoMP is independent of the proposed open pit mining at Miriam.

Table ES 10: Summary of underground life of mine plan

Parameter	Units	
Production	Mt	21.945
Marianne-Marilyn	Mt	14.132
MSNE-Maryvonne	Mt	7.813
Grade	kg/t eU	0.776
Marianne-Marilyn	kg/t eU	0.787
MSNE-Maryvonne	kg/t eU	0.757
Years of Production	years	15.8
Marianne-Marilyn	years	10.0
MSNE-Maryvonne	years	5.8

Labour

The Arlit region has an extensive history of mining and provides a workforce familiar with the techniques and skills required for the proposed underground operations. In addition, the limited remaining mine life at the Akouta Mine may result in trained workers becoming available for the proposed operations prior to the commencement of any operations.

GoviEx and SRK have assumed that the Madaouela project can be staffed almost entirely with Nigerien labour for both technical and labour functions. Any requirement for expatriate assistance is assumed to be covered by the general and administrative costs.

Table ES 11: Technical staff requirements for the Underground Operations

Position	Shift Structure	Workers	Total Workers
Operational Staff	**Shift**	76 (per shift)	304
Technical Staff	**Office Hours** **Shift**	18 5 (per shift)	18 18
Grand Total			**340**

Capital

A summary of the estimated capital costs for underground mining is provided in Table ES 12, and includes the capital costs associated with equipment, conveyors and development.

Table ES 12: Estimated capital costs for underground mining

Branch	Capital in 1st 3 Years of Underground Mining (USDm)	Total Capital Cost in LoMP (USDm)
Marianne-Marilyn	92.5	181.5
MSNE-Maryvonne	0.0	120.8
Total	**92.5**	**302.3**

Operating Costs

Operating costs have been developed from first principles. Where possible, equipment and

consumable costs are sourced from suppliers, otherwise data is sourced from GoviEx, and SRK's own databases of similar operations. Operating costs are estimated for mining plan for inclusion in the TEM. The mining operating costs consist of three elements:

- Lateral development;
- Conveying to the processing plant; and
- Surface rehandling of waste.

SRKs estimate for the total mining costs (including conveying, labour and surface waste rehandling) at full production is USD28.6 /t_{RoM} at M&M and USD28.5 /t_{RoM} at MNSE-Maryvonne.

12 MINERAL RESERVE ESTIMATES

The Mineral Reserve estimate are based on those Mineral Resources classified as Measured and Indicated (as of May 20, 2015) and incorporate technical and economic studies that justify economic extraction on the information currently available. All Mineral Reserves are classified as Probable in accordance with international guidelines for classification.

The Mineral Reserve Statement reported for the M&M deposit is based on the Madaouela Mineral Resource Estimate by SRK (April 2015). The Mineral Reserve Statement for MSNE-Maryvonne and Miriam are based on the Madaouela Mineral Resource Estimate by SRK (March 2013).

SRK's Mineral Reserve Statement for M&M and MSNE-Maryvonne as at May 20, 2015, was prepared under the direction of Tim McGurk FIMMM who is a Qualified Person as defined by the CIM Code. SRK's Mineral Reserve Statement for Miriam as at May 20, 2015, was prepared under the direction of Rick Skelton MIMMM who is a Qualified Person as defined by the CIM Code.

The U_3O_8 equivalents have been calculated based on the formula presented below.

$$U_3O_8 \text{ equivalent} = \text{eU (uranium metal)} \times 1.17924$$

Table ES 13 shows the Mineral Reserve Estimate for the three deposits.

Table ES 13: Mineral reserve statement (May 20, 2015)

Deposit	Cut-Off Grade eU (kg/t)	RoM	Uranium Metal		Uranium Oxide	
		Tonnes (Mt)	Grade eU (kg/t)	Contained U (t)	Grade eU_3O_8 (kg/t)	Contained U_3O_8 (t)
Marianne-Marilyn (M&M)*						
Probable	0.48	14.1	0.79	11,164	0.93	13,165
MSNE-Maryvonne*						
Probable	0.48	7.8	0.76	5,938	0.89	7,002
Total Underground (Probable)	0.48	21.9	0.78	17,102	0.92	20,167
Miriam Open Pit**						
Probable	0.40	7.5	0.82	6,192	0.97	7,302

* *Underground Mineral Reserves for Marianne-Marilyn and MSNE-Maryvonne are reported at a cut-off grade of 0.48 kg/t eU. Cut-off grades are based on a price of USD 70 /lb of U_3O_8 (USD 154 /kg U_3O_8) and uranium recoveries of 83.0 %, without considering revenues from other metals. Note Mineral Reserves include both Measured and Indicated Resources.*

***Open Pit Mineral Reserves are reported within the MAD I licence and within a designed pit shell at a cut-off grade of 0.4 kg/t eU. Cut-off grades are based on a price of USD 70 /lb of U_3O_8 (USD 154 /kg U_3O_8) and uranium recoveries of 83%, without considering revenues from other metals. Mining modifying factors are 2% ore loss and 5% dilution at 0 kg/t grade. Note Mineral Reserves include both Measured and Indicated Resources.*

Most of the baseline data used for the Mineral Resource estimate has been obtained through gamma probe results that are potentially of lower confidence than assayed results from drill core. Consequently a portion of the intersections have had verified results from chemical assays and the database of probe results regressed against equivalent chemical assay grade.

The underground mine plan is based on the geological interpretation that mineralisation is located directly above the contact between the Guezouman Sandstone and the Talak Shale. The mine design and mining methods applied are based on the interpretation of this geological contact.

Three room and pillar mining methods applied to the underground deposits at varying orebody dips are planned to be mined in a similar fashion with additional waste development and larger pillars applied in the design to compensate for their differences. There is a risk that at steeper dips, mining becomes more complicated, which incurs more ore losses and higher mining costs than is allowed for in the Mineral Reserve estimate.

Limited metallurgical tests have been made on material from the Miriam deposit compared to M&M, however preliminary results suggest the physical and metallurgical properties of the Miriam deposit are similar to those in the M&M deposit, and these have been applied to the technical studies on Miriam. While considerable metallurgical tests have been completed there remains a risk that the results of future metallurgical development diverging from existing studies.

13 PROPOSED PROCESSING OPTIONS

The following section provides a brief description of the treatment process. The Selected Flowsheet Option: Crushing, ROS, Milling, Ablation, Two stage leach and Cyanex 600 SX.

Ore Receipt and Feed Preparation

Mined ore from Marianne and Marilyn is crushed underground (80 % < 306 mm) prior to transport and is received on surface at the transfer point at the discharge of the inclined conveyor from the mining operation. From here material is transferred to the ROM stockpile via the stockpile feed conveyor. Facility is also provided to receive primary crushed ore from other mining operations or from static stockpiles around the plant area into a feed receipt bin by road truck or front end loader (“FEL”).

Feed preparation effectively consists of two stage crushing with intermediate screening. Ore from the stockpile is reclaimed by 3 apron feeders and routed to a secondary jaw crusher operating in open circuit with the product screened and the oversize routed to a tertiary cone crusher with the product returned to the sizing screen. Product from the crushing circuit is nominally < 20 mm (80 % passing).

The primary crushed ore from the stockpile is routed to ROS where fines are screened out and routed directly to fine crushing with the coarse fractions (+20 mm) subject to preconcentration by ROS. The accept product only is routed to fine crushing while the rejects are routed to a waste stockpile.

Accept product from the ROS is secondary crushed and mixed with water to create 20%

slurry, which is fed into the Ablation stage, in which one or more set of two jets of slurry site opposite each other to create a high energy impact zone. The high energy impact zone causes a break down of the ore material into coarse and fine particles, which are separated by screening and gravity. The uranium is predominately in the fine particles and this fine material slurry is thickened prior to feeding into the acid leach tanks. The coarse material, which is predominately sand, is thickened and sent to the tailings dam. An additional recovery step involves gravity separation of residual uranium and heavy mineral particles from the coarse material that is then fed back into the Ablation product. This improves overall recovery to 99% and also results in carbonate reduction and thus reduces acid consumption for the two stage acid leach step.

The two stage leaching circuit consists of primary and secondary agitated leach tanks in recirculation with a leach thickener after the first stage and a belt filter after the second stage leach. In the first stage solution from the second stage belt filters is recirculated to leach the fresh feed after which the slurry is thickened with the thickener overflow routed to the uranium recovery circuit as pregnant leach solution (“PLS”). The thickener underflow is routed to the second stage leach, which uses fresh acid to further leach the milled solids. After the second stage the slurry is filtered and the filtered solids residue is washed with fresh solution and discarded to the tailings disposal system.

Leach tanks are agitated and aerated to allow the milled ore to react with sulfuric acid allowing dissolution of the contained uranium. The acid consumption (per ton of feed to the leach) in this case is forecast to be lower because of the rejection of the acid consuming constituents of the feed during flotation. The solution from leach (thickener overflow) is routed to uranium recovery.

Uranium Recovery

Based on laboratory testwork solvent extraction (“SX”) has been selected as the mos appropriate method for uranium recovery. Cytec has developed a new SX reagent that canallow recovery and separation of uranium and molybdenum from acidic solutions termed CYANEX® 600 extractant. CYANEX 600 is a highly selective, stable phosphinic acid based reagent and allows molybdenum and uranium recovery from acidic solutions by the same solvent. Based on the test results, the use of 10 Vol% CYANEX 600 could be used to selectively extract and separate Molybdenum and Uranium while achieving recoveries greater than 98 % for Mo and 99 % for U.

The overall circuit configuration would consist of two extraction steps, one stripping step for iron, two stripping steps for molybdenum, an ammonia washing step and two uranium stripping steps. A wash stage will be used after the Uranium strip to prevent phosphoric acid transfer. Phosphoric acid will be recovered through an evaporation step in order to reduce reagent requirements. Additional uranium recovery will come from this step through a bleed stream that would feed back into the solvent extraction process.

Operating and Capital Costs

Tenova Bateman have calculated forecast operating and capital costs for the processing option presented. These are summarised in Table ES 14. Further details are provided in the individual capital and operating cost sections.

Table ES 14: Overall summary of forecast overall operating and capital costs

		Selected Option
		Enhanced SX
Capital cost	USD m	234
Operating costs	USD/t ore to plant	23

Operating and Maintenance Labour

Tenova Bateman generated labour complements for the different flowsheets. In summary these included the following total personnel presented in Table ES 15.

Table ES 15: Summary labour complements

	Selected Processing Option
Management / Admin	72
Supervisors	42
Operators	212
Labour	114
TOTAL	**438**

14 PROJECT INFRASTRUCTURE

This section presents the surface infrastructure assets proposed at four separate operational areas:

- Marianne-Marilyn underground operation (“M&M”);
- MSNE-Maryvonne underground operation (“MNSE”);
- Miriam open pit operation (“Miriam”), and;
- Explosives Magazine (“Magazine”).

Surface infrastructure compounds will be located adjacent to the Marianne-Marilyn and MSNE underground portals that are linked by a 3 km long heavy vehicle road. This road runs parallel to a surface conveyor that transports ROM material from the mining operations to the process plant. It is anticipated that the mining and support workforce will all change at the change facility located at the Marianne-Marilyn compound before bus transportation to their place of work, including underground operations.

The proposed Marianne-Marilyn compound has been located approximately 100 m south of the Marianne-Marilyn portal and adjacent to the main site access road which links the process plant site to the existing Arlit-Tchirozine (N25) road. The proposed MSNE compound has been located approximately 100 m north of the MSNE portal and adjacent to the road linking the Marianne-Marilyn and MSNE portals. The proposed Miriam compound has been located approximately 100 m northeast of the Miriam pit, and adjacent to the road to MNSE.The explosives magazine area has been located just North of MNSE with a minimum 500 m clearance from all other assets.

15 TRANSPORT AND LOGISTICS

Potential logistics scenarios to transport the anticipated quantities of reagents and consumables required supporting the proposed Madaouela mine and process plant has been

assessed. This includes existing regional infrastructure to establish multi-modal solutions to transport reagents and consumables to Arlit, approximately 12 km north-west of the proposed development.

The assessment indicates that road transportation from Cotonou in Benin represents the most cost effective option to transport process reagents/consumables to Arlit. Bollore have provided a quotation and in accordance with the clients' instruction a transportation unit rate of USD 190 /t will be incorporated into the Technical Economic Model.

16 MARKET STUDIES AND CONTRACTS

The fundamental principles of the uranium market revolve around the sale of U_3O_8 is into the the market; transported; and transformed for use in nuclear reactors. As such the following elements are pertinent to understand:

- Understand the position and role of uranium within the nuclear fuel cycle;
- Analyse U_3O_8 demand with particular reference to the U_3O_8 requirements of the world's Reactors;
- Explain the transformation of U_3O_8 into UF_6 and the role of the Conversion Facilities who provide such a service;
- Summarise the requirements for transportation of U_3O_8 from GoviEx's Madaouela Uranium Project to the Conversion Facilities;
- Examine the contractual relationship between GoviEx as the Uranium Producer and the Conversion Facilities;
- Analyse the uranium market prices and pricing mechanisms;

GoviEx is predominantly an exploration and mine development company, and is therefore considering contracting out the logistics, marketing and sales of its U_3O_8 to a third party company, who have a long history and the required skills to complete these complex tasks.

The near-term and medium-term outlooks continue to have some residual inventories available to the market at a time when uranium demand seems to be static. However, demand improves in the long-term outlook as the scale and pace of Japanese reactor restarts should improve, with Chinese reactor growth as well. This could set up for increased price volatility beyond 2016, as a number of operating and planned projects could fall victim to low prices over the next couple of years. Based on history alone, uranium prices can make huge swings when future production levels are uncertain due to the long lead times required to bring new projects online

Additionally the uranium mining industry, without a marked increase in metal price, will not have sufficient projects whose economic hurdle rates are sufficient to fill the expanding supply gap.

Therefore from 2017 the market watchers believe that the uranium market will move into a deficit as existing supply cannot meet the growing demand requirements and the uranium price will need to be substantially higher to provide the motivation for producers to solve the impending supply gap. These two major factors are therefore expected provide the impetus to improved uranium prices from 2017 and beyond, if not before. Analysts are predicting spot to increase to USD 70 /lb to USD 80 /lb U_3O_8, and long term prices USD 60 /lb to USD 70 /lb with term contract prices extended to be at least USD 5 to 10 /lb higher than spot.

17 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

In accordance with the requirements of Niger legislation for Environmental Impact Assessments ("EIAs"), GoviEx contracted Legeni to undertake various baseline studies (including a socio-economic assessment, air quality monitoring, reconnaissance surveys of fauna and flora, soils and geomorphology and archaeology and cultural heritage) for its proposed Madaouela uranium project. SRK reviewed the specialist baseline studies as well as completion of hydrology and hydrogeological studies on the project concession. Legeni was also contracted to undertake stakeholder engagement in accordance with Niger in-country regulatory requirements and international good practice. Following completion of baseline, the final ESIA project report and information was submitted to the Ministry March 10, 2015, the ESIA has been approved by the Minister in charge of the environment.

The overall conclusion of this ESIA is the majority of potential negative impacts identified can be reduced to acceptable levels with effective management measures, which GoviEx is committed to implementing. There are a number of management controls requiring interaction with either Government officials or potentially affected communities, in particular the livestock herders using the Project area for grazing at certain times of the year. GoviEx will actively engage with these key stakeholders to present the proposed management controls and seekto find consensus on the way forward. GoviEx is also committed to continuing to undertake ongoing stakeholder engagement with the wider communities and other stakeholders potentially affected, positively and negatively, by the Project.

The impacts of the Project have been arranged in groups, as presented in Table ES 16, in order of importance for the decision-making process. The residual impact significance rating provided in the table assumes the management measures described within the ESIA report have been successfully implemented.

Table ES 16: Summary of impact significance ratings

Grouping	Identified impacts	Residual impact rating
Most important positive and negative impacts needing to be taken into account during decision making.	Water supply wellfield impacting on local aquifers and groundwater users	Medium
	Increased government revenue from fiscal and foreign exchange income	High +ive
	Direct and Indirect employment of Nigeriens residing in the Arlit Department	Medium +ive
Negative impacts having a high or medium significance without management and that require careful monitoring to ensure management measures are effective. If monitoring reveals additional management/remedial measures required, these must be implemented without delay.	Mine dewatering impacting on local aquifers and groundwater users	Medium
	Potential post-closure leaching of deleterious constituents from flooded underground mine workings to groundwater impacting on local aquifers and groundwater users	Medium
	Particulate matter from operations exacerbating health issues associated with the naturally high dust levels in the region	Low
	Land clearance for surface infrastructure causing loss, degradation and/or fragmentation of natural habitat and potential loss or disturbance of species of conservation value	Low
	Direct exposure to radiation through proximity to TSF and inhalation or ingestion of tailings material or process water	Low
	Increased demand on infrastructure and for services and goods	Low
	Change in community dynamics and social relationships	Medium

Grouping	Identified impacts	Residual impact rating
	Reduced access to the Project area for water and pasture	Low
	Increased infectious disease transmission between workers and the host community	Low
Impacts that can be managed readily through measures that are not difficult to implement and are known to be reliable.	Impacts of site infrastructure on storm water runoff patterns	Low
	Potential leaching of deleterious constituents from the waste facilities to groundwater impacting local aquifers and groundwater users	Low
	Seepage and discharge from waste water systems	Low
	Potential post-closure impacts of Miriam open pit on water users	Low
	Degradation and/or alteration of landscape resulting in loss of a soil resource and reduced land capability	Low
	Contamination of soils from solid or liquid waste or from aerial deposition	Low
	Gaseous emissions from operations exacerbating health issues in the region	Low
	Tailings dam forming pool during the rainy season and acting as an attractive nuisance to birds in the region	Low
	Direct inhalation of radon and radionuclides from the ventilation systems of the underground mines	Low
	Accidental spills of uranium concentrate during packaging and transport	Low
	Increased incidences of avoidable accidents and injuries	Low
	Accidental damage or loss of archaeological sites resulting from land clearance for construction of Project infrastructure and looting	Medium

Green shading indicates potential positive impacts, red and yellow shading represents potential medium and low negative impacts, respectively

Stakeholder engagement is required in accordance with Niger legislation and international good practice. To date several interactions have been undertaken with local stakeholders during the data collection for the social baseline studies. These informal meetings took place in the Communes of Arlit, Gougaram and Dannet and included discussions with groups of men, women and youths and background information on the project was presented. In addition meetings were held with traditional authorities, community leaders, technical experts and other key informants. A number of issues and concerns were raised during these meetings such as the existing negative legacy of mining activities in the area; the potential loss of local livelihoods due to competition for land and water resources, which will be exacerbated by population influx; possible pollution to the environment; potential infrastructure improvement and job creation; and the necessity for proper stakeholder consultation.

The conclusions drawn aim to assist decision-makers in determining if environmental approval of the project can be given. The impacts of the project have been arranged in groups, as presented in Table ES 16, in order of importance for the decision-making process.

The first group of impacts in Table ES 16 represent the most important impacts for decision makers. Positive impacts are denoted with a “+ive” after them and represent the main benefits created by the Project from an environmental and social perspective and may be considered to balance some or all of the negative impacts. These include increasing government revenue during construction and operation and the provision of employment and income for Nigerien people leading to reduced poverty. The only negative impacts with a high significance before management measures is related to the proposed wellfield, however by maximise re-use of the mine dewatering and implementing the other proposed mitigation measures the

significance of this can be reduced to medium.

The second group of impacts have a high or medium significance prior to the implementation of management measures but these measures reduce the significance of the impact to medium or low. These are the impacts requiring careful monitoring to check the management measures are effective and to ensure additional remedial measures are implemented if required. The final group of impacts are those that are either considered of low significance due to the nature of the impact or can be easily reduced to acceptable levels through management. The management measures are not difficult to implement and are known to be reliable.

Several impacts have already been managed through inherent measures incorporated into project design. There is further opportunity to avoid or reduce the severity of some impacts by continuing to consider environmental and social elements as final designs are confirmed. The robustness of the supporting management programmes, along with implementation, assurance and continual improvement functions of the ESMS ("Environmental and Social Management Strategy"), are fundamental to enabling the successful implementation of management measures by the GoviEx, its contractors and sub-contractors. A key part of the ESMP ("Environment and Social Mitigation Plan") is the ongoing monitoring to confirm whether the impacts identified in this ESIA materialise and if any additional measures are required.

18 CAPITAL AND OPERATING COSTS

The tables below summarises the capital and operating costs for the Madaouela Uranium Project. The detailed development of these individual costs are provided in the relevant sections.

Capital Expenditure

Total capital expenditure for the life of the operation are presented in Table ES 17. A two and half year construction period ahead of plant production is envisaged. Beneficiation plant capital includes a 10 % contingency.

Table ES 17: Capital expenditure

Parameter	Units	Total amount
Project Capital		
Capitalised operating costs (incl year 1)	(USDm)	66.1
Mining	(USDm)	29.7
Processing	(USDm)	234.1
Tailings	(USDm)	3.1
Infrastructure	(USDm)	20.4
Water Management	(USDm)	5.2
Total Project Capital	**(USDm)**	**358.5**
Sustaining capital		
Mining	(USDm)	303.7
Tailings	(USDm)	4.8
Infrastructure	(USDm)	5.4
Water Management	(USDm)	3.4
Total Sustaining Capital	**(USDm)**	**317.3**
Total Capital Expenditure	**(USDm)**	**675.8**

Operating Costs

Life of mine operating costs have been presented in Table ES 18. By-product credits for the molybdenum are not included in Table ES 18.

Table ES 18: LoM operating costs

Operating Cost Item	Total amount (USDm)	Unit Cost (USD/t ore)
Mining	813	27.61
Processing	694	23.57
Tailings	22	0.74
Site Infrastructure	16	0.54
Water management	15	0.50
G&A	105	3.56
Distribution cost	6	0.22
Closure Cost	15	0.51
Subtotal operating costs	**1,686**	**57.25**

19 ECONOMIC ANALYSIS

SRK completed a technical economic analysis on the Project. The various assumptions and inputs for the base case are described below:

- Mining sequence: Miriam open pit being mined first, with the focus on the high grade ore being processed first, followed by Marianne-Marilyn and MSNE-Maryvonne underground mines;
- Low grade ore from Miriam is fed to the mill during the underground mining stages; and
- Processing is based on radiometric ore sorting followed by ablation and two-stage leach with recovery of uranium and molybdenum oxides through the use of Cyanex 600 solvent

exchange process.

- Molybdenum is considered as a by-product from the processing plant, based on metallurgical testwork results. SRK notes that the molybdenum is not included in the Mineral Resource or Ore Reserve statements.

Inputs

The following assumptions have been applied to the life-of-mine plans and the economic analysis:

- Mineral Resources as defined in line with the CIM guidelines;
- Maximum plant feed of 1.47 Mtpa;
- Overall uranium (eU, determined largely by radiometric analysis) recovery of 93.7 % based on 0.78 kg/t U for underground and 0.82 kg/t U for Miriam , and molybdenum recovery of 67.0% based an average 0.47 kg/tgrade;
- A ROM molybdenum grade of 0.47 kg/t, resulting in approximately 592 t of MoO_2 per annum. The revenue generated by the molybdenum is presented in the below as a by-product credit to offset the operating costs. Operating and capital costs have been included in the processing plant estimates. SRK notes that Mo has not been modelled in the Mineral Resource estimate;
- A total project life of 24 years (3 construction, 21 revenue generating);
- A U_3O_8 price of USD 70.00 /lb, and MoO_2 price of USD 11.00 /lb;
- Operating costs and capital expenditures as defined in Section 26 above;
- A 30 % income tax rate applies after a three year tax holiday;
- Royalty rate based on the Niger Mining Code which states a sliding scale royalty rate from 5.5 % up to 12 % for higher profit to revenue ratios. The royalty is payable on both uranium and molybdenum revenue. SRK notes that the revenue associated with the molybdenum is presented as a by-product credit in the below, however for the royalty calculations is treated appropriately;
- A base case 8 % discount rate; and
- No provision for salvage value at closure is assumed in the analysis.

Total ore tonnages and grades for the three deposits are presented in Table ES 19.

Table ES 19: Technical mining inputs

Parameter	Units	Base Case
LoM (ore production)	(years)	21
Marianne-Marilyn	(years)	11
MSNE-Maryvonne	(years)	7
Miriam	(years)	5
RoM Ore	(kt)	29,454
Marianne-Marilyn	(kt)	14,132
MSNE-Maryvonne	(kt)	7,813
Miriam	(kt)	7,510
RoM U Grade	(kg/t)	0.79
Marianne-Marilyn	(kg/t)	0.79
MSNE-Maryvonne	(kg/t)	0.76
Miriam	(kg/t)	0.82
RoM Mo Grade[1]	(kg/t)	0.47
U Content	(kt)	23.2
Marianne-Marilyn	(kt)	11.1
MSNE-Maryvonne	(kt)	5.9
Miriam	(kt)	6.2

[1]*Mo grade is not JORC compliant and is based on metallurgical testwork and not a resource estimation*

The base case economic analysis results in a post-tax pre-finance NPV of USDm 340 (at 8 % discount rate), with an IRR of 23.5 %. Undiscounted payback will be in production year 3. Total LoM statistics are presented in Table ES 20. A year by year summary is presented in the technical economic model summary.

Table ES 20: Technical economic model summary and results

Parameter	Units	Base Case
Mining		
RoM Ore	(kt)	29,454
U Grade	(kg/t)	0.79
U Content	(kt)	23.2
Processing		
U Recovery	(%)	93.7%
U Recovered	(lb)	47,955,179
Revenue		
U_3O_8 sales	(lb)	56,550,835
U_3O_8 price	(USD/lb)	70
U_3O_8 revenue	(USDm)	3,959
Operating Expenditure		
Direct Operating Costs	(USDm)	1,686
By-product credits	(USDm)	(301)
Subtotal operating costs	(USDm)	**1,385**
Royalty	(USDm)	396
Total Operating Costs	**(USDm)**	**1,781**
Unit Operating Costs		
Subtotal operating costs	(USD/t ore)	47.01
	(USD/lb U)	28.88
	(USD/lb U_3O_8)	24.49
Royalty	(USD/t ore)	13.44
Total Operating Costs	(USD/t ore)	60.46
	(USD/lb U)	37.13
	(USD/lb U_3O_8)	31.49
Operating Profit – EBITDA	**(USDm)**	**2.178**
Corporate Profit Tax	**(USDm)**	**376**
Capital Expenditure		
Project	(USDm)	359
Sustaining	(USDm)	317
Total Capital Expenditure	**(USDm)**	**676**
Net Free Cash	**(USDm)**	**1,126**
NPV @ 8.00%	**(USDm)**	**339.8**
NPV @ 10.00%	**(USDm)**	**251.5**
IRR	(%)	23.5%
Break even price (NPV=0 @ 8%)	(USD/lb U_3O_8)	48

Table ES 21 presents NPV and IRR sensitivity results for changes in price and discount rate. A sensitivity to Mo grade (base case 472 ppm) is presented in Table ES 22 at the base U price of USD 70 /lb and 8% discount rate.

Table ES 21: NPV and IRR sensitivity to price and discount rate

Price (USD/lb U_3O_8)	Discount rate (%)	NPV (USDm)	IRR (%)
70	10%	251	23.5%
	8%	340	
75	8%	411	26.2%
65	8%	263	20.2%

Table ES 22: NPV and IRR sensitivity to price and discount rate accounting for Mo

Mo grade	NPV (USDm)	IRR (%)
300 ppm	310	22.3%
472 ppm (Base case)	340	23.5%
600 ppm	362	24.3%

The sensitivity of project value (NPV) to commodity price is +USD 99 M for a +10 % change and –USD 109 M for a -10 % change on the base price of USD 70 /lb U_3O_8. Sensitivity of project value (NPV) to operating costs and capital cost are broadly similar at around ±USD 41 M for a ± 10 % change. Sensitivity to ±10% change in molybdenum grade is ±USD 8 M.

20 ADJACENT PROPERTIES

There are currently two producing mines in the Arlit region that are on property positions immediately adjacent to the Madaouela Uranium Project:

Somair

Société des mines de l'Aïr (Somaïr, the mining company of the Aïr) was established in 1968. The company is operated by AREVA, which owns 63.4% of the share capital; the remaining 36.6% is held by Société du patrimoine des mines du Niger (Sopamin, the Nigerien national mining company). Somaïr has operated several uranium deposits near the town of Arlit since 1971. The ore is extracted from open pit mines and heap leached or processed mechanically at the front end of the Arlit mill. In both cases, the uranium solutions are processed at the back end part of the mill, whose capacity was raised to 3,000 metric tons of uranium per year in 2011 (7.8 million pounds of U_3O_8).

Cominak

Cominak (Compagnie Minière d'Akouta) is 34% owned by AREVA, which operates it. The other shareholders are Sopamin of Niger (31%), Ourd (25%), and Enusa Industrias Avanzadas SA of Spain (Enusa, 10%). The ore is extracted underground and is then processed in the site's mill, producing approximately 1,500 metric tons of uranium per year (3.9 million pounds of U_3O_8).

Imouraren

Located 80 kilometers south of Arlit, this deposit was discovered in 1966 and constitutes one of the largest deposits in the world today (174,196 metric tons of reserves). The feasibility study was completed in December 2007 and submitted in April 2008. AREVA received the mining permit for the deposit in early January 2009. The Imouraren SA mining company was

established, with AREVA NC Expansion (86.5% AREVA and 13.5% Kepco/KHNP) holding a 66.65% interest and Sopamin of Niger holding the remaining 33.35%. In view of market conditions construction work was suspended. The site, equipment and facilities are currently put under care and maintenance, and all demobilization operations should be completed in the first quarter of 2015.

Table ES 23: Reported production, mineral reserves and resources for the Areva mines in Niger

	Production	Total Reserves		Measured and Indicated Resources		Inferred Resources	
Mine	Total Uranium ("TU")	TU	Kg/t U	TU	Kg/t U	TU	Kg/t U
Somair	2331	7771	1.46	28919	1.41	21925	1.68
Cominak	1501	12100	3.62				
Imouraren		213772	0.70	62584	0.58	2879	0.66

Source: Areva – 2014 Reference Document

The above stated resources are provided for comparative purposes only. They have not been reviewed by a Qualified Person, are unknown as to compliance with or correlation to CIM definitions for mineral resources, and therefore should not be relied upon.

21 GEOTECHNICAL STUDIES

SRK undertook geotechnical site investigation work from September 2012 to March 2013 which included oriented core geotechnical drilling, logging, acoustic televiewer surveys, laboratory strength testing, structural interpretation from Quickbird imagery and surface mapping of structures where possible.

The Marianne-Marilyn deposit is the northernmost mineralised zone within the Madaouela Project. Four main geological formations are defined at Marianne-Marilyn. From surface these are, Tchinezogue, Guezouman, Unite d'Akokan ("UA") and Talak formations. The UA is present as two distinct channels between the Guezouman and Talak formations.

The base of the Guezouman sandstone, situated at depths ranging from 50 to 160 m from surface, hosts the uranium mineralization. The Marianne-Marilyn orebody spans approximately 5 km by 2 km in plan, and the average orebody thickness is 1 m. The mineralisation is elongate along a WSW-ENE trending structural axis. Structurally, the deposits are affected by two well-defined monoclinal structures that control the location of elongate residual lenses of UA sediments, and the basal channel structures of the Guezouman. These Akokan channels are interpreted to be bound along their southern side by a north-dipping syn-sedimentary fault zone that is restricted below the Guezouman sandstone. The west of the deposit is dominated by a dome, where the stratigraphy defines a radially gently-dipping zone of uplift. Stratigraphic dips are accentuated on the flanks of this structure but near-horizontal at the apex. The phreatic surface sits at approximately 390 mRL along the centre of the UA channel, and rises to between 405 mRL and 410 mRL to the north and south. The Guezouman will drain freely and elevated pore pressure will not influence mine design. A strong upward hydraulic gradient is observed between the underlying Talak and the Guezouman. Boreholes tapping the deeper aquifers beneath the UA channel show groundwater elevations of around 417 mRL. The elevated pore pressure in the Talak may be very slow to dissipate due to the low permeability of those units.

Marianne-Marilyn will be mined by room and pillar method involving development of mining panels on initial pillar layout driven from the central access drift towards the ventilation raise. The initial pillars, which are two times the size of the final pillar plus an opening, are split into four final pillars on retreat. Pillars will be a composite of either Guezouman Sandstone and Talak shales, or Guezouman Sandstone and UA material (where present). The mining height is 1.8 m without resuing.

The geotechnical analysis work undertaken includes interpretation of geotechnical data, characterization of rock mass behavior, and geotechnical design. Empirical analyses were conducted to determine acceptable range of excavation span, ground support, and pillar design. Then, 3D stress analysis and 2D pillar stability analysis were conducted using numerical modelling techniques to confirm and optimize the design.

The recommended geotechnical design for Marianne-Marilyn is 7 m room width with varying pillar dimensions, ranging from 4 m x 4 m to 5 m x 5 m, depending on the depth of cover, dip of mineralization, and presence of fault structures. The main ground support in the production panels is full column resin grouted 1.5 m long #6 rebar bolted on a 1.2 m by 1.2 m square pattern. Mining height at production panels is kept at 1.8 m with the use of low profile mining equipment. Barrier pillars of 15 m are to be left between adjacent mining panels, and secondary barrier pillars with a minimum 10 x 15 m dimension should be left every 100 m span within the panel. Minimum of 20 m spacing (25 m if depth > 100 m) should exist between the central access drift and production panels.

The MSNE and Maryvonne deposit exists in similar geological formations as Marianne-Marilyn, and will be mined by the same room and pillar methods as Marianne-Marilyn. The geotechnical designs for MSNE and Maryvonne are therefore the same as those for Marianne-Marilyn, with the exception that Maryvonne pillar dimensions are 6 m x 6 m, due to the lower level of technical information available, but could be revised with additional drilling.

Both Marianne-Marilyn and MSNE-Maryvonne are accessed by 700 m long declines that plunge at 8° to provide access to the underground. As the decline excavation is a LoM critical infrastructure, a higher level of ground support is recommended as follows:

- 2.4 m long #8 (25 mm) resin grouted rebar on 1.4 m centres through welded wire mesh in the back;
- Ribs to be supported with 1.5 m long #6 (20 mm) resin grouted rebar installed on 1.5 m centres through welded wire mesh that has been extended to within 1 m of the floor; and
- The maximum mesh size of the welded wire mesh should be 100 mm (4”) square, constructed from #8 gauge wire or thicker.
- In addition, smooth wall blasting techniques are strongly recommended in permanent infrastructure excavations

Miriam is located within the southern extent of the Madaouela license area. The mineralisation at Miriam is generally near surface and will be mined by open pit. Geology within the Miriam deposit consists of three main sedimentary units: Tchinezgoue siltstones, overlying Guezouman sandstone, overlying Talak shales. Miriam is not associated with any notable domes or monoclinal structures. There are relatively few structures that can be linked from the surface to offsets in stratigraphic markers. Geotechnical analysis of rock mass data suggests that lithology type and location of the water table will likely be the main influencing factors governing rock mass behaviour.

Based on modeling results, two Design Sectors are required for Miriam Pit:

- The Weathered Zone in upper 40 m;
- Unweathered Zone below 40 m depth

Slope configurations have been calculated and for a pit depth of 110 m with the overall slope angle of 48 °. The slope configurations are based on the following criteria:

- Target Factor of Safety for overall slope at PFS level is 1.3;
- Bench stack height is limited to 60 m on all walls. A geotechnical berm (or ramp) of 20 m minimum width is required every 60 m to de-couple the slope;
- Reduced quality rock mass exists in the Weathered Zone therefore a lower bench face angle is required; and
- Good pre-split blasting to minimise damage on the final walls.

22 HYDROGEOLOGY

The ore horizons at Madaouela are located at the base of the Guezouman, which is a regional aquifer and must be dewatered to facilitate extraction of the ore. Over-exploitation of groundwater is a major issue in the area and the impacts of dewatering must be thoroughly evaluated and mitigated. The project must also operate within the Mining Code and is therefore committed not to waste water (i.e. zero discharge). Careful planning of the dewatering programme is therefore essential to avoid wastage of groundwater, especially during mine development when process water demand is zero.

Groundwater inflow to the underground workings also has the potential to impact on operations due to the argillic nature of the underlying Talak formation, which is highly erodible and suffers from poor trafficability when wet. Over-abstraction of groundwater is a major concern in the Arlit area and a water association comprising the two local mining companies and local Government has been formed to ensure cooperation between all stakeholders. GoviEx has not joined this water association to date, principally due to the fact that its current rate of groundwater consumption is negligible compared to the existing operations.

There is no legislative requirement to pay for abstracted groundwater. It is also understood that the exploration and mining licences come with a right to abstract groundwater within that licence area. The right to abstract groundwater is however dependent on that abstraction having no adverse impact on existing groundwater users within or adjacent to that licence area.

Groundwater flow is primarily controlled by both lithology and structure, as is typical for sedimentary groundwater systems. The majority of groundwater flow occurs within the sedimentary sandstones. The major faults, including the Arlit and Madaouela Faults, result in discontinuity within the high permeability sandstone beds and form low permeability barriers within the regional groundwater system.

The interbedded sandstone and shale sequence leads to a highly anisotropic groundwater system with hydraulic conductivity that is several orders of magnitude higher parallel to bedding than it is perpendicular. The bedding planes are typically shallow dipping so this results high horizontal hydraulic conductivity and low vertical hydraulic conductivity. This permeability contrast gives rise to limited vertical groundwater flow and strong vertical hydraulic gradients. The absence of vertical connectivity also results in strong vertical water quality gradients with depth. There is limited throughput in the deeper groundwater units which subsequently show increased dissolved solids content due the longer resident times.

Groundwater recharge and discharge mechanisms in the Sahel are complex due to the fact that present day recharge rates are negligible, but discharge of fossil groundwater continues to occur. Groundwater discharge in the project area is now dominated by the activities of the Somair and Cominak operations and the associated water supplies to the towns of Arlit and Akokan. The majority of the abstraction is from the Tarat aquifer. Drawdown within the Tarat has also extended to the south and east of Arlit, extending into the Madaouela I concession area. Drawdown in the Tarat is estimated at 40 m immediately to the west of Miriam (compared to over 100 m in the vicinity of the Cominak mine). The extent of drawdown further south is yet to be confirmed. The impact on water levels east of the Madaouela Fault is minor although some leakage across the fault is occurring in the Miriam area due to drawdown within the Guezouman.

The drawdown within the Tarat is shown to result in downward seepage from the overlying Madaouela and Izeguandene deposits and increased rates of upward flow from the Guezouman to the Tarat. The influences of the abstractions in the Marilyn area are also evident within the Guezouman.

Drawdown within the Tarat will have led to increases in the rate of discharge from Guezouman into the overlying Tarat, and may be responsible for depletion of groundwater within the Guezouman. SRK is yet to obtain accurate baseline (i.e. 1960's) groundwater elevations for the Guezouman but current estimates suggest the Guezouman has experienced about 10 m of drawdown on average, and possibly around 20 m in the Marilyn area. Local abstractions (historical dewatering at Marilyn and the army camp water supply well) are also responsible for the observed drawdown but increased rates of leakage from the Guezouman into the overlying Tarat is also likely to have an influence.

The drawdown associated with the GoviEx wellfield results in a change in groundwater flow directions east of the Madaouela fault. The impact of dewatering from GoviEx drilling areas is also evident, although the dominant control on groundwater movement to the west is still the abstractions around Cominak and Somair.

Groundwater levels in the Marilyn area are heavily influenced by groundwater abstractions. The Marilyn deposit was dewatered for a period of approximately 12 months during the 1960's and the Guezouman is still used to provide a water supply for the army camp. This had resulted in a cone of depression that is concentrated along the UA channel in the Marilyn deposit. The phreatic surface sits at approximately 390 mRL along the centre of the UA channel, and rises to between 405 mRL and 410 mRL to the north and south. The eastern extreme of the Marilyn deposit is in close proximity to the dry limit of the Guezouman; the deepest section of the Marianne deposit is approximately 80 m below water level.

The phreatic surface ranges from approximately 405 to 410 mRL across the MSNE deposit, approximately 80 to 90 m above the elevation of the workings. Groundwater flow direction is from the southeast to the northwest in the MSNE and is influenced by the abstractions at Marilyn and the regional impact of dewatering at the Somair and Cominak mines to the northwest.

The Miriam open pit will be excavated through the Tchinezogue and Guezouman formations. Dewatering of the open pit will be required. The phreatic surface sits at approximately 415 mRL across the pit area, approximately 45 m below surface and 65 m above the final pit floor. The Tchinezogue formation comprises silts, fine sandstones and clays. The unit will yield very little groundwater although some nuisance inflows from the sandstone beds may be expected. The lower 25 m of the pit walls will comprise the Guezouman formation.

SRK has developed a 3D numerical groundwater flow model to support the dewatering and water supply evaluations. A finite-difference software package, Modflow2005 has been used for the model. The model has been used to predict LoM dewatering rates and the drawdown around each mine area through LoM, and the recovery period post-closure.

For the purposes of design it is assumed that an installed dewatering capacity of 200 m^3/hr is required to handle groundwater inflows at both Marianne-Marilyn and MSNE-Maryvonne. This means pumps capable of delivering 200 m^3/hr are installed in each area. For the purposes of OPEX calculations, average inflows of 100 m^3/hr are assumed for both M&M and MSNE-Maryvonne. An installed pumping capacity of 60 m^3/hr is assumed for the Miriam open pit operation. An average long-term inflow rate of 30 m^3/hr is assumed for OPEX calculations.

The dewatering strategy comprises a combination of dewatering boreholes (eighteen in total) and conventional centrifugal dewatering pumps for underground and open pit operations. In the medium to long-term all dewatering discharge will be pumped to the process plant for use in the process. All dewatering wells are assumed to be completed to 175 m depth and completed with 6" internal diameter borehole casing. The above-ground reticulation system has not been designed as dewatering hole locations have not been defined. A nominal 1 km of pipeline per well has been assumed. The total costs for the dewatering wellfield are calculated at USD 2.95 M. The Miriam open pit dewatering system has been calculated at USD 250,000 excluding power costs.

23 TAILINGS STORAGE FACILITY

SRK has developed preliminary designs for the options being considered for the disposal and storage of tailings and has provided high-level cost estimates for each for use in the project economic model. The designs have been completed based on the design criteria for the current project as developed by others, applicable national and international standards and, where appropriate, based on SRK's experience

For the Tailings Storage Facility ("TSF") option, SRK has selected a dry stacked TSF as the most appropriate concept. This concept is similar to the disposal methodology used at the nearby Somaïr and Cominak operations. The facility has been designed to store a total of 16.4 Mt, or 10.3 Mm^3 of tailings over 20 years, at a production rate of 2,250 tpd post Radiometric Sorting.

The cost estimates, based only on the scope of this report, and the TSF costings must be considered along with the appropriate process costings.

SRK has provided risks and opportunities related to the current design and a plan to progress the designs to the next stage. Based on the design criteria, the designs presented are technically feasible and will provide tailored solutions to the project requirements.

For the Base Case option based on the analyses undertaken, SRK has made the following conclusions:

- Dry stack tailings disposal is feasible;
- There is a net loss of water from the total system;
- Capex cost per tonne of tailings is USD 1.01 /t for dry stacked tailings; and
- Opex cost per tonne of tailings is USD 1.45 /t for dry stacked tailings.

24 VENTILATION

The underground uranium mines Marilyn-Marianne and MSNE-Maryvonne, which are part of the Madaouela Uranium Mine are designed to be room and pillar operations. An investigation into the ventilation requirements for Marilyn-Marianne and MSNE-Maryvonne at each stage of mine development has been undertaken. Modelling and simulation work uses CAE Mining's Ventsim Visual 3 software ("Ventsim") to determine the optimum ventilation network with respect to the standard ventilation requirements of uranium mines. The investigation covers the primary and secondary ventilation; intake and exhaust raise development; fan size and power requirements.

The use of negative pressure ventilation systems is preferable when efficiently and effectively ventilating underground uranium mines. Single pass ventilation is essential in order to minimise levels of radioactivity in working areas. Because the rate of decay of radon into its radioactive daughter elements ("radon daughters") increases with time, very small quantities of radon gas can become problematic. Because of this, single-pass ("primary") ventilation should be used where possible and local secondary ventilation of working areas is considered essential.

The results from the iterative analysis of the ventilation design show that the proposed ventilation network can support five actively extracting panels, at least two additional developed panels and multiple access development drives.

The exact volumetric flow into the mine will depend on the number and location of active panels and diesel machines. The design phases suggested in this study should be considered examples of possible stages and therefore act only as conceptual guides to show that ventilation of the mine is achievable for the given ventilation design.

Given the low expected residency times, single-pass primary ventilation system and airflow design contingency the proposed ventilation design for the Madaouela Project is considered technically feasible.

A primary ventilation system is proposed, where potential contaminated air is exhausted promptly through exhaust raises located within the panel developments minimising the risk of radiation and other contaminants leaking into the central access development.

The approach to ventilation contamination control at the Madaouela underground uranium project should prioritise the removal of radon gas and radon progeny at its source. This is achieved by numerous exhaust raises located within the ore panels, and preventing the recirculation of contaminated air. The secondary ventilation should be used as a means of diluting and directing potentially contaminated air to the exhaust system, and providing fresh air supply to development headings.

Reducing the ingress or groundwater will minimise the potential for water-borne progeny, as well as effective ventilation management and operating practices, and properly utilised and extensive real-time instrumentation monitoring.

The required airflow rates for each phase of the mine are dependent on the emissions of radon gases and the planned diesel-powered equipment operating at that stage. This study has taken into account the maximum instantaneous airflow requirements for diesel powered equipment, operating sumps and accounted for a ventilation contingency of 25%. Analysis determined a maximum total airflow of 1,075 m^3/s is required. The phased ventilation modelling shows that 882 m^3/s of airflow for Marilyn-Marianne and 1,150 m^3/s for MSNE-

Maryvonne is required.

Effective and robust scheduling of panels and development headings is critical in maintaining acceptable air velocities in haulage drives, conveyor drifts and working areas. Analysis found that each intake raise can only provide fresh air to three active ore panels simultaneously if connected using three drifts (two access, one conveyor), and only one panel can be ventilated if a single access or panel development drift connects to the raise.

25 CONCLUSIONS AND RECOMMENDATIONS

SRK's interpretations of the geology, mineral resources, and scoping level study of conceptual mining and processing options for the Madaouela Uranium Project are as follows:

SRK has completed technical studies to a pre-feasibility level of confidence for the project. The Project development plan envisions an average 2.69 Mlb per year U_3O_8 yellowcake production rate over a twenty one year mine life, with a 93.7 % ultimate recovery of uranium. In addition it assumes recovery of on average approximately 592 t of MoO_2 per annum at a sales price of USD11.00 /lb, which revenue is used to offset processing costs. The base case project economics for this project at a long-term uranium price of USD 70 /lb U_3O_8 are positive, and indicate an after-tax NPV of USD 339 M (at 8 % discount rate) with an IRR of 23.5 % and a total life of mine ("LoM") net free cash of USD 1,126 M. Initial capital costs are estimated at USD 359 M, LoM capital costs at USD 676 M, and cash operating costs of USD 24.49 /lb U_3O_8 excluding royalties and including by-product credits. It is noted that Mo has not been included in the Mineral Resource model and hence is not considered to be at the same level of confidence as the U grades.

The sensitivity of project value (NPV) to uranium price is +USD 99 M for a +10 % change and –USD 109 M for a -10 % change on the base price of USD 70 /lb U_3O_8. Sensitivity of project value (NPV) to operating costs and capital cost are broadly similar at around ±USD 41 M for a ± 10 % change. Sensitivity to ±10 % change in molybdenum grade is ±USD 8 M.

GoviEx geologists are confident that the exploration potential on all tenements has the possibility to significantly increase the current resources of the Madaouela Uranium Project. GoviEx exploration drilling has identified additional targets including La Banane that is included in the resource statement and potential new targets at Marianne northwest, MSNE, Maryvone and northwest Miriam as well as additional targets not as yet drilled in Mad I, Agal, Era and Mad IV.

SRK concurs that the entire property position warrants the added exploration effort and does indeed have the potential to significantly increase the current project resources and thus significantly improve the potential economic viability of the Madaouela Uranium Project; however, SRK cautions that an exploration potential cannot be relied upon until further drilling and sampling is done to properly assess that potential.

The Madaouela Uranium project is sufficiently attractive from a technical and economic perspective that it justifies pursuit by GoviEx toward further resource definition and steps to project development.

SRK recommends that GoviEx continue the process of moving the Madaouela Uranium Project forward with additional technical studies.

Studies still required include;

- Detailed optimization of open pit mining equipment and schedules

- Detailed process plant design
- Pilot plant design and operation including ROS, ablation and solvent extraction options

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AN UPDATED INTEGRATED DEVELOPMENT PLAN FOR THE MADAOUELA PROJECT, NIGER

1 INTRODUCTION (ITEM 2)

SRK Consulting (UK) Limited (SRK) has been commissioned by GoviEx Uranium Inc. ("GoviEx") to prepare a Development Plan for the Madaouela Uranium Project in the Republic of Niger, Africa. The Madaouela Uranium Project (the Project) is located near Arlit, in north central Niger, in one of the most significant areas of producing sandstone-hosted uranium deposits in the world. The Project is controlled 100 % by GoviEx Niger Holdings Ltd. ("GNH Ltd."), a wholly owned subsidiary of GoviEx Uranium Inc., a public company listed in Canada; collectively referred to herein as GoviEx.

SRK has prepared three previous technical reports for GoviEx, in NI 43-101 format, titled "NI 43-101 Technical Report on Resources, Madaouela Uranium Project", dated March 17, 2010, *"NI 43-101 Preliminary Economic Assessment Madaouela Uranium Project*' dated March 08, 2011, and "An Intergrated Development Plan for the Madaouela Project, Niger", dated June 14, 2013.

This report provides an updated opinion for the Project subsequent to work carried out during the IDP, the Environmental and Social Impact Assessment (ESIA) and work conducted to complete the mining permit application. The ESIA and mining permit application have been submitted to the Nigerien government, the ESIA has been approved. There are changes and updates to the report in every section. This document provides a Technical Report on the results of mining and processing studies and a preliminary feasibility study of the potential project development, as well as an update to the overall project resources. The resources in this document are correct and verified by SRK as of May 20 2015 and the reserves as of 20 May 2015 and are compliant with NI 43-101 guidelines. This report is prepared according to NI 43-101 guidelines. Form NI 43-101F1 was used as the format for this report. The purpose of this Technical Report is to provide the reader with a brief review of the historical and current exploration activities conducted at the Madaouela Uranium Project, a review of the most recent resource estimate, and a discussion of the elements of the conceptual design for mining and processing, including a study of the project's potential economic viability.

The Project is an advanced stage property, for which five deposits have been developed to a high level of confidence (Marianne, Marilyn, Miriam, MSNE and Maryvonne). Current Project Mineral Resources are stated in Table ES3. The study is based on detailed geological studies, metallurgical testing and processing options, mine design, infrastructure, rock mechanics, tailings and heap leach, hydrogeological and environmental impact.

Mining schedules have been developed including open pit mining at the Miriam deposit, and room and pillar mining with decline access at a combined 4,020 tonnes per day production rate from Marianne-Marilyn, MSNE and Maryvonne deposits. The base case mineral processing design is crushing, radiometric sorting, ablation and two-stage tank acid leaching, solvent extraction, with dry stack tailings. Utilizing Cyanex 600 resin allows both uranium oxide yellow cake and by-product molybdenum oxide to be produced as commercial products

with acceptable quality for sale.

The Project development plan envisions an average 2.69 Mlb per year U_3O_8 yellowcake production rate over a twenty one year mine life, with an 93.7 % ultimate recovery of uranium. In addition it assumes recovery of on average approximately 592 t of MoO_2 per annum at a sales price of USD11.00 /lb, which revenue is used to offset processing costs. The base case project economics for this project at a long-term uranium price of USD 70 /lb U_3O_8 are positive, and indicate an after-tax NPV of USD 339 M (at 8 % discount rate) with an IRR of 23.5 % and a total life of mine ("LoM") net free cash of USD 1,126 M. Initial capital costs are estimated at USD 359 M, LoM capital costs at USD 676 M, and cash operating costs of USD 31.49 /lb U_3O_8 including royalties and by-product credits. It is noted that Mo has not been included in the Mineral Resource model and hence is not considered to be at the same level of confidence as the U grades.

The purpose of this technical report is to provide a review of the historical and current exploration activities conducted at the Madaouela Uranium Project, a review of the most recent resource estimate, and a discussion of the elements of the conceptual design for mining and processing, including a preliminary feasibility study of the project's potential economic viability.

The exploration data and the resource estimates presented in this report are prepared using the industry-accepted Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") "Best Practices and Reporting Guidelines" for disclosing mineral exploration information, the Canadian Securities Administrators revised regulations in NI 43-101 (Standards of Disclosure For Mineral Projects) and Companion Policy 43-101CP, and CIM Definition Standards for Mineral Resources and Mineral Reserves (November 27, 2010).

2 RELIANCE ON OTHER EXPERTS (ITEM 3)

The Qualified Persons ("QP's") for this technical report, Ryan Freeman, Robert Bowell, Daniel Guibal, Rick Skelton, Tim McGurk and Neal Rigby have examined the historical and current data for the Madaouela Uranium Project provided by GoviEx with respect to resources, metallurgical test work, and other project information, and have relied upon that basic data to support the statements and opinions presented in this Technical Report. Several GoviEx staff are also contributors of information sections of this report; those contributions have been supervised and reviewed by the Qualified Persons. In the opinion of the authors, the project data is present in sufficient detail to provide an accurate representation of the Madaouela Uranium Project.

It is the opinion of the QP's that there are no material gaps in the information for the project. Sufficient information is available to prepare this report, and any statements in this report related to deficiency of information are directed at information, which in the opinion of the authors, should be sought as the project progresses.

This report includes technical information, which requires subsequent calculations to derive subtotals, totals, and weighted averages. Such calculations inherently involve a degree of rounding and consequently can introduce a margin of error. Where these rounding errors occur, SRK does not consider them material.

The results of this Technical Report are not dependent upon any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings between GoviEx, SRK, and the authors. SRK will be paid a fee for its work in accordance with normal professional consulting practice.

2.1 Qualifications of Consultants (SRK)

Robert J. Bowell, BSc PhD C.Chem. C.Geol FGS, E.Geol.FIMMM

Robert Bowell is a Principal Geochemist with SRK, with 21 years' experience in applied geochemistry, data analysis and qualification, exploration, exploration management, and mining project evaluation. He has had four years direct experience with uranium exploration, geochemical analysis, mineralogy and evaluation of uranium deposits for project development. He is a registered professional geologist with the Geological Society of London and with the European Geological Association. He is a Qualified Person for this report and is in particular is responsible for the discussion on geology, exploration and mineralization in Sections 3 through 10 and the mineralogy and metallurgical testing program presented in Section 12 of this report, and for the Processing portion of Section 16. In addition, he is responsible for sections 1, 2, 11, 18, 22 and 25 through 28 and is QP for the overall report.

Dr. Neal Rigby, PhD CEng, MIMMM, PhD

Dr. Neal Rigby is a Principal Mining Engineer with SRK and has 35 years' experience in the international mining industry. Neal has performed mining engineering, project management and management consulting for a wide range of uranium projects. He has held a variety of positions in production, academia, business and consulting. He is Chartered Engineer, a member of the Institute of Materials, Mining and Metallurgy, member of the Institute South Wales Institute of Engineers, Corporate Member of SME and AIME, and holds his PhD in Mining Engineering from the University of Wales, UK. He is a Qualified Person for this report and in particular the Mining and Technical Economic sub-sections of Section 21 of this report.

Daniel Guibal, MSc FAusIMM, CP

Daniel Guibal is SRK's Corporate Consultant (Geostatistics and Resources), with SRK-Australia in Perth, has over 37 years' experience with geostatistics as applied to mineral resources in the mining industry. His particular fields of expertise include resource estimation (including non-linear geostatistics, MIK, uniform conditioning, and conditional simulation) resource classification (JORC), sampling theory, open pit optimization, and training of professionals in statistics, geostatistics, sampling and grade control. He has experience with a broad range of commodities including uranium. He is a Qualified Person for this report and is responsible for the discussion on mineral resources and the mineral resource estimates presented in Section 13 of this report.

John Arthur (PhD, FGS CGeol; MIMMM CEng) is a Principal Consultant (Resource Geology) with some 26 years' experience in the international minerals industry and is a Qualified Person/Competent Person according to the guidelines of the JORC, IoM3 and CIM (43-101) Mineral Resource classification schemes. His experience includes planning and execution of exploration projects, geological modelling and Mineral Resource estimation associated with mining studies from scoping level through to full Bankable feasibility studies, reserve estimation and open pit optimization, and technical and economic review work associated with financial audits of mining operations. Worldwide experience of various deposit types and styles includes Archaean shear zone gold, iron ore, nickel laterites, diamonds (offshore and onshore), copper (porphyry and copperbelt styles). He has worked extensively in West, Central and Southern Africa, South America, Russia and the CIS and the Far East. Key skills include practical application of geostatistical estimation techniques allied with extensive knowledge of the Gems, Surpac and Isatis modelling and geostatistical software packages. He is a Qualified Person for this report and is responsible for the discussion on mineral resources and the mineral resource estimates presented in Section 13 of this report.

Rick Skelton MSc CEng MIMMM MSAIMM ARSM

Rick Skelton has over 35 years of mining experience, specialising in open pit mine and quarry planning, with particular emphasis on the implementation and use of software in resource / reserve evaluation, mine planning and strategic business management. Rick joined SRK in South Africa in 1977 and was based in SRK's Johannesburg offices until 1996, when he moved to SRK (UK) in Cardiff. Whilst with SRK in South Africa, he established and spent 10 years with Gemcom, then an SRK Group company, where he was responsible for the development and marketing of Gemcom's mining software, and for marketing and supporting Mincom's mining and business software systems. In total, Rick has worked on over 150 international open pit projects which have included operating mines, feasibility studies, audits / reviews and general mine computerisation projects. These projects have been located

throughout Africa, Brazil, Scandinavia, Chile, India, Australia, Ukraine, Kazakhstan, Russia and other CIS countries. He is a Qualified Person for this report and is responsible for senior review of the report, particularly Section 14 on reserves and Open Pit Mining sections of Section 15 of this report.

Tim McGurk BEng (Hons), CEng, FIMMM

Tim McGurk has twenty five years experience in the mining industry. Tim worked on gold, diamonds, lead/zinc, nickel and silica sand underground operations in South Africa, Australia, Ireland and Scotland in a variety of engineering and management roles. His experience includes mine engineering design, mine construction project management, mine management and management of scoping and pre-feasibility studies. Since joining SRK in 2008, Tim has managed a number of large, multi-disciplinary technical studies including the Zanaga Iron Ore Project for Xstrata and is currently leading the mining studies for Rio Tinto's underground Jadar project. Tim holds First class mine manager's certificates of competency for Queensland and Western Australia and is a Chartered Engineer. He is a Qualified Person for this report and is responsible for senior review of the report, and in particular the sub-sections of Sections 18, Capital and Operating costs in Section 20 and the Economic analysis of Section 21 of this report.

2.1.1 Site Visit

SRK has completed several site visits during the IDP execution and has had staff on site continuously from September 2012 to March 2013 for purpose of geotechnical and hydrogeological supervision. Specific field tasks undertaken include:

- Collection of bulk metallurgical sample, March 2011
- Collection of mineralogical samples, June to July 2012
- Collection of quarterly water quality samples from July 2012 to June 2013
- Supervision of geotechnical and hydrogeological drilling, July 2012, September 2012 to March 2013
- Social and environmental assessment, April 2012 and May 2013
- Project orientation study, April 2012

3 PROPERTY DESCRIPTION AND LOCATION (ITEM 4)

3.1 Property Description

The Madaouela Uranium Project exploration property holdings extend over a large land area of approximately 1,596 km^2.

The property is located in the Agadez region (Arlit Department), in the Northern central part of Niger (Western Africa), southeast from the town of Arlit and west of the Aïr Mountain. The town of Arlit was founded in the 1960's, as a mining town, when the first uranium deposits were discovered by exploration department of the Commissariat à l'Energie Atomique ("CEA"). Arlit is located a distance of approximately 800 km by air from the capital city Niamey (Figure 3-1).

3.2 Mineral Titles

The Madaouela Uranium Project consists of five tenements, Madaouela I, II, III, IV, and Anou Melle (Figure 3-2), held 100 % by GoviEx Niger Holdings Ltd, and Agaliouk ("Agal"), Eralrar ("Era") for which GoviEx submitted prospecting applications on November 22, 2012.

Exploration efforts to-date by GoviEx has been concentrated on the Madaouela I ("Mad I") and Agal tenements, which combined are approximately 486.5 km^2 in area. The historical resources were identified on the Madaouela I and Agal tenements, as are the current resources stated by GoviEx. Similar Decrees, Mining Conventions, and Exploration Permits are current and held 100 % by GoviEx Niger Holdings Ltd for Madaouela II, III, IV, and Anou Melle properties.

The mineral exploration rights of the Madaouela I property were secured by GoviEx Niger Holdings Ltd. ("GNH Ltd.") after the signing on May 26, 2007 of the Madaouela I Mining Convention ("MIMC") together with its Side-Agreement ("MIMC-SA") between the State of Niger and GNH Ltd. MIMC and MIMC-SA include and describe the respective rights and obligations of each of the signing Parties for the management of the exploration program, and also stipulates which further steps would apply should an ore body be discovered on the property. It also gives all necessary descriptions regarding the geographic identification and location of the property itself. MIMC and MIMC-SA were given legal status on May 26, 2007 after the promulgation of the corresponding Decree, referenced as no. 2007-186/PRN/MME and dated May 25, 2007, in the State of Niger gazette No.7 special edition dated June 8, 2007.

The Exploration Title attached to that property is known as "Madaouela I" ("MIET"). Its legal status was recognized by the Order no. 50/MME/DM dated June 4, 2007, published in the same State of Niger gazette as cited previously. Exploration rights apply primarily to uranium-bearing minerals but also encompass other related minerals that could be discovered on the property.

The two documents, MIMC and MIET go side by side. MIET must be kept in good standing and exploration work conducted according to the obligations of and commitments made by GNH Ltd. as set out in MIMC and MIET. Should the Exploration Title holder deliberately fail to meet its obligations, Exploration Title could be withdrawn. The Exploration Title was renewed on November 2, 2012. A Mining Permit application was filled with Minster in charge of Mining on June 30, 2015 for the Madaouela I tenement and renewal applications for Madaouela II, III, IV and Anou Melle tenements on July 1, 2015.

3.3 Madaouela I Mining Convention and its Side-Agreement

The MIMC was agreed upon on the premise that obligations of both Parties, as stipulated in the agreement, will be met up until the convention term. Specific dispositions belong to these two agreements and are consistent with the Niger Mining Law, which was modified by the Order no. 93-16 dated March 2, 1993, to give better incentives for investors on the one hand, and to satisfy the West African Economic and Monetary Union international and shared dispositions regarding mining activities in the subcontinent area on the other hand.

State of Niger Guarantees

- General conditions (legal, administrative, economic, financial, fiscal, etc.) as committed to by the State of Niger and stipulated in MIMC and MIMC-SA remain in force until the end of the MIMC (Article 21); and
- Stability of administrative conditions makes the exploration business much more predictable, reduce the administrative risk and therefore allow the operator to concentrate on its core exploration business.

Overall Goal of the Mining Convention

The Mining Convention is aimed at facilitating the exploration work and thus increasing the probability of an ore body discovery and its subsequent exploitation. As stated in the MIMC-SA for the Madaouela I tenement and in view of the former geological information from past exploration work, GNH Ltd. shall aim at identifying additional uranium resources and at producing a feasibility study in line with the Nigerien government requirements.

Duration

Article 7 of MIMC stipulates a 20 year term, starting on May 26, 2007 and the convention is renewable thereafter. The MIMC-SA repeats the main agreement terms.

Scope for Exploration Work and Expenditures

The MIMC describes the content of the first three years exploration program as proposed by GNH Ltd., including work scope and expenditures that are adjustable over the years. It requires that GNH Ltd. update its work program and expenditure forecasts accordingly. GoviEx has completed USD 52.6 M in expenditure from May 2007 to December 2014.

Reporting

MIMC requires that quarterly and annual reports on exploration technical results and expenditures be submitted to the Ministry of Mines and Energy and comply with the report format as defined by the Ministry.

Surface Lease Fee

An annual surface lease fee must be paid according to a fee schedule as stipulated in the MIMC. Surface lease fee rates increase every time the Exploration Title is renewed. Current annual surface lease fees are USD 4,517.

Fixed Fees

An annual fixed contribution of USD 10,000 is payable for each exploration permit to assist the Ministry in the training of its agents. Fixed fees must be paid every time there is a change in the status of the Exploration Title such as renewal, extension, sales, transfer, etc.

GoviEx is in compliance of its reporting and payment requirements under the MIMC.

Property Ownership of the State of Niger at the Exploitation Phase

Article 8 of the Mining Law Ordinance (dated March 2, 1993 and its modification law dated August 9, 2006) provides that the award of an exploitation license by the State allows it to hold 10 % in the share capital of the exploitation company during exploitation. Those 10 % holdings are "free of charge, and may not be diluted". Article 8 also provides that the State is entitled to contribute in cash or assets to the share capital of the company in consideration for an additional 30 %, either directly, or through a public (State-owned) entity.

3.4 Madaouela I Exploration Title ("MIET")

Duration and Extension

Article 5 of MIET stipulates a term of three years, renewable twice for the same term (subject to required reductions of 50 % of the applicable permit area). Additionally, the exploration title may be extended for one year for technical reasons in order to complete a feasibility study in line with the Nigerien government requirements.

The Corporation's mining concession for Madaouela I entered into force on June 4, 2007. In May 2010, the Minister of Mines and Industrial Development of Niger formally extended the expiry of the first validity period of the exploration licenses for the Madaouela I tenement to September 3, 2012 with no reduction in the area covered by the license. The extension was granted to compensate for interruptions to the Company's exploration activities at the Madaouela Project between August 2007 and November 2009 as a result of a government imposed state of alert. On November 2, 2012, the Niger Ministry of Mines and Industrial Development granted a 36 months renewal of the Madaouela I license, reducing the original granted surface area by fifty percent. A Mining Permit application was filed with the Minister in charge of Mining on June 30, 2015 for the Madaouela I tenement.

3.5 Madaouela II, III, IV and Anou Melle Tenements

As for Madaouela I, similar Decrees were signed by the President of the Republic of Niger, and dated May 26, 2007 for the Madaouela II, III, IV, and Anou Melle tenements. Mining Conventions were entered into between the Republic of Niger and GoviEx dated May 26, 2007 for the exploration, development and production of uranium and related minerals for all tenements. In addition, Exploration Permits were awarded by the Minister of Mines and Energy for all tenements on June 4, 2007. In May 2010, the Minister of Mines and Industrial Development of Niger formally extended the expiry of the first validity period of the exploration licenses for the Madaouela II, III, IV and Anou Mellé tenements to September 3, 2012 with no reduction in the area covered by the license. The extensions were granted to compensate for interruptions to the Company's exploration activities at the Madaouela Project between August 2007 and November 2009 as a result of a government imposed state of alert. On November 2, 2012, the Niger Ministry of Mines and Industrial Development granted a 36 months renewal of each of the four licenses, for each license, reducing the original granted surface area by fifty percent. GoviEx has applied (as of July 1 2015) for renewal of the Madaouela II, III, IV and Anou Melle tenements.

In connection with each of those permits, the Corporation agreed to complete an exploration work program and to meet certain milestones set forth in the Mining Conventions for each permit. Since May 2007 GoviEx has spent a combined USD 26.3 M on these licences up to December 2014.

3.6 Agaliouk and Eralrar Tenements

On November 22, 2012, the Company submitted to the Niger authorities an application covering certain portions of the original Madaouela I and Madaouela IV licences surface areas that were excluded from the renewed licences granted on November 2, 2012, now known as Agaliouk and Eralrar. Approval of this application is still outstanding. GoviEx has been advised by the Niger Government that they will be re-issued; however, there can be no assurance that this will be the case. On March 23, 2014, the Company received written confirmation via email from the Nigerien Director General of Mines and Geology that the application was in order and that the formal issuance of the licenses should follow shortly. Separately, the Company has been in direct discussions with both the Nigerien President and Prime Minister, who have both indicated that that there is no reason to expect that the licenses will be refused.

Table 3-1 lists the geographic locations of corner points of all seven tenements.

Table 3-1: Geographic boundaries of GoviEx land holdings

Tenement	Corner	East Longitude	North Latitude
Madaouela I	A	7° 25' 00"	18° 45' 00"
	E	7° 30' 45"	18° 45' 00"
	F	7° 30' 45"	18° 32' 00"
	D	7° 25' 00"	18° 32' 00"
Agaliouk	E	7° 30' 45"	18° 45' 00"
	B	7° 36' 30"	18° 45' 00"
	C	7° 36' 30"	18° 32' 00"
	F	7° 30' 45"	18° 32' 00"
Madaouela II	A	7° 36' 30"	18° 45' 00"
	E	7° 42' 15"	18° 45' 00"
	F	7° 42' 15"	18° 32' 45"
	D	7° 36' 30"	18° 32' 45"
Madaouela III	A	7° 36' 30"	18° 32' 45"
	E	7° 42' 15"	18° 32' 45"
	F	7° 42' 15"	18° 20' 00"
	D	7° 36' 30"	18° 20' 00"
Madaouela IV	A	7° 25' 00"	18° 32' 00"
	E	7° 30' 45"	18° 32' 00"
	F	7° 30' 45"	18° 20' 00"
	D	7° 25' 00"	18° 20' 00"
Eralrar	E	7° 30' 45"	18° 32' 00"
	B	7° 36' 30"	18° 32' 00"
	C	7° 36' 30"	18° 20' 00"
	F	7° 30' 45"	18° 20' 00"
Anou Melle	A	7° 15' 24"	18° 27' 56"
	B	7° 18' 19"	18° 27' 56"
	C	7° 18' 19"	18° 25' 39"
	D	7° 25' 00"	18° 25' 39"
	E	7° 25' 00"	18° 20' 00"
	F	7° 15' 24"	18° 20' 00"

3.7 Financial Conditions

The exploration permits were granted by the Government of Niger in exchange for a payment by GoviEx of € 25 M. GoviEx will pay an additional € 7 M, as a one-time payment, once any one of the five tenements, pursuant to the payment schedule agreed in 2007, is formally converted from an exploration permit to a mining permit.

3.8 Location of Mineralization

As of March 2013, the mineralized deposits on the Madaouela I and Agal tenements are identified on Figure 3-2. The deposits on the Madaouela I tenement targeted for developing uranium resources and their estimation, and for inclusion in the study are: Marilyn, Marianne, MSNE, Maryvonne and Miriam. The deposits of MSCE and MSEE have inferred resources defined and La Banane has indicated and inferred. These are excluded from the economic assessment subject to further work. There are other documented occurrences on the other tenements but these have not yet been explored. There is good potential for additional resources.

3.9 Environmental Liabilities and Permitting

Environmental and Social Impact Assessment (ESIA)

An ESIA was conducted for the Project during 2014, on March 10, 2015, the Company completed and filed the Environmental and Social Impact Assessment ("ESIA") with the Nigerien Minister in charge of the environment. The ESIA was prepared by SRK with assistance from Legeni S.A. on the field work and stakeholder engagement. The ESIA has been approved by the Minister in charge of the environment on July 31, 2015.

CEA Old Exploration Camp

CEA started exploring in the 1960's and identified two main mineralized targets, which they named Marilyn and Marianne. They constructed a campsite to accommodate the CEA exploration team. Traditional buildings, mostly small structures made of "banko", a mixture of clay and thatch, still remain in place and are currently used by the Niger military. A new Niger military camp is to be constructed, to be situated outside the northwest corner of the Madaouela I license area. Planning for this development has already started with a special committee setup that includes the Niger Government ministries of Defence, Finance and Mining.

CEA Old Drillholes

CEA drilled a large number of drillholes, generally marked on the ground surface by markers, however not difficult to retrieve 50 years later. Most of the old CEA drillholes, where they can be found, have been reopened, reamed and relogged as part of the GoviEx exploration work.

CEA Old Mining Workings

CEA implemented some test mining work to further explore the Marilyn area. They sunk a shaft (67 m deep) and excavated a network of galleries (330 m) to sample the mineralization and better understand the mineralization at different locations. CEA stockpiled two ore grade categories and waste rock. Stockpiles remain enclosed within old CEA camp perimeter. The quantity of mineralized material was estimated be more than 2,000 t in total (some has been collected previously by OURD to realize their feasibility study). Waste rock quantities have not been evaluated.

SRK visited the stockpiles of mineralization. One pile was hand sampled by GoviEx on one end for metallurgical samples, and it was noted that essentially no oxidation of mineralization is present, due in part to the very limited rainfall annual in the region. In 2010 GoviEx did carry out bulk sampling, by extracting 5 trenches of 1 to 1.2 m section, and 10 to 15 m apart, using a shovel and classical sampling practice (one shovel out of ten). 24 tonnes were drummed corresponding to 2 x 12 tonnes identical lots. One lot was flown to Mintek in the Republic of South Africa ("RSA"), to carry out under SRK supervision, processing and ore sorting tests. These piles of uranium mineralized material will eventually need to be removed and / or processed, but they pose little environmental concern at present, as there is no residence or dwelling within 150 m.

Underground Water Quality

Underground water is still being pumped from two water holes and the old shaft and is used by the military camp. However, the underground water quality at the camp is not likely to meet the drinking water quality standards, due to bacterial and chemical contamination during several years when the shaft was collecting used water from the camp.

Domestic Garbage Dumps

The Madaouela I tenement, being in close proximity to the town of Arlit, has been littered with small piles of domestic garbage originating from former drilling activities, from town dwellers, and/or from the military camp. An inventory of such dumps has been mapped in order to establish a baseline for environmental monitoring purposes.

Cuttings from Drillholes

The drilling operation produces cuttings and mud that are collected in small pits on surface and examined by geologists to document the geology and the geophysical logs. When drilling through mineralized sections, the drilling operation produces uranium-bearing materials. Although their radioactivity is rather modest, such materials are thereafter disposed of and buried in small pits at each drillhole location in a manner to reach radioactivity readings equivalent to the general background radioactivity level on surface.

3.9.1 Encumbrances

As stated in Section 3.2 "Property Ownership of the State of Niger at the Exploitation Phase" an exploitation license by the State allows it to hold 10 % in the share capital of the exploitation company during exploitation. Those 10 % holdings are "free of charge, and may not be diluted". The State is entitled to contribute in cash or assets to the share capital of the company in consideration for an additional 30 %, either directly, or through a public (State-owned) entity. In addition, there is a 5.5 % to 12 % sliding scale royalty payable to the State based on the commercial value of exported minerals. These are the only financial encumbrances (or royalties) on the property.

There have historically been some physical restrictions associated with operations on the Madaouela I exploration permit area:

Security

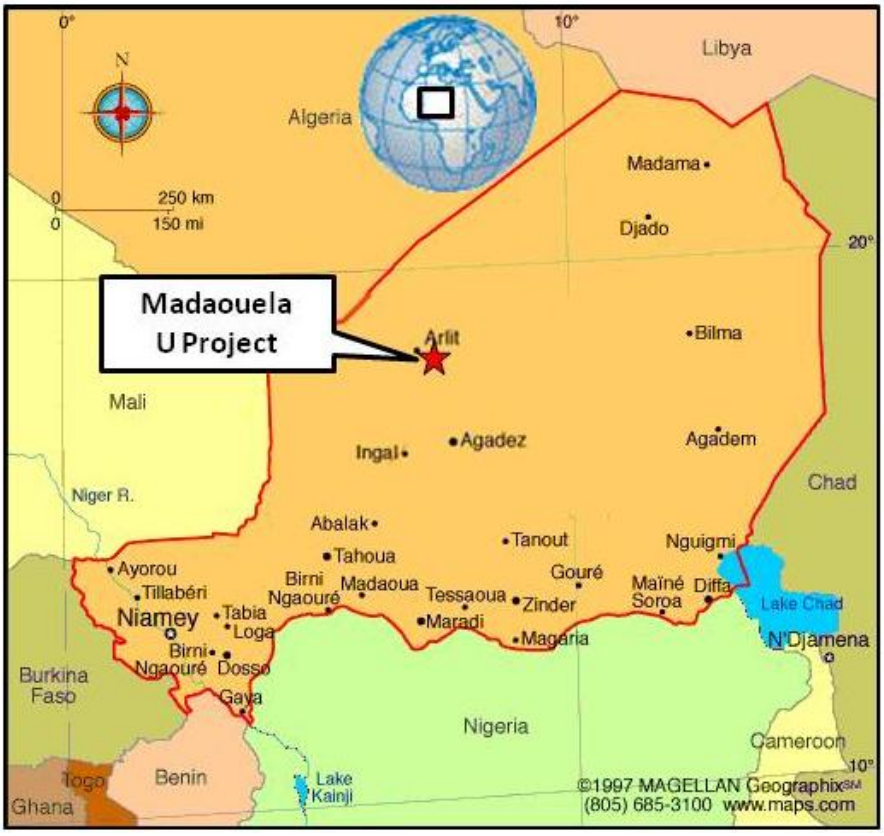
Road access from Agadez to Arlit was previously under military control, with convoys (including export shipments of uranium) being accompanied by military escort. Local Tuareg groups historically opposed to the Government have reached peace agreements with the central Government, there is still the potential for theft of trucks and other equipment,

especially as exploration moves further from the existing military camp. As a result, GoviEx has arranged for security personnel to accompany GoviEx teams as they move to more distant drilling targets.

GoviEx, and most other exploration and mining companies in Niger, reported interruptions to operations for a 27 month period during 2007 to 2009, over which time the Niger Government declared a "mise en garde" (“state of alert”) in the Arlit-Agadez region due to local separatist activity. The mise ′en garde was lifted in November 2009, and GoviEx reported no security incidents against the company or its personnel since this period.

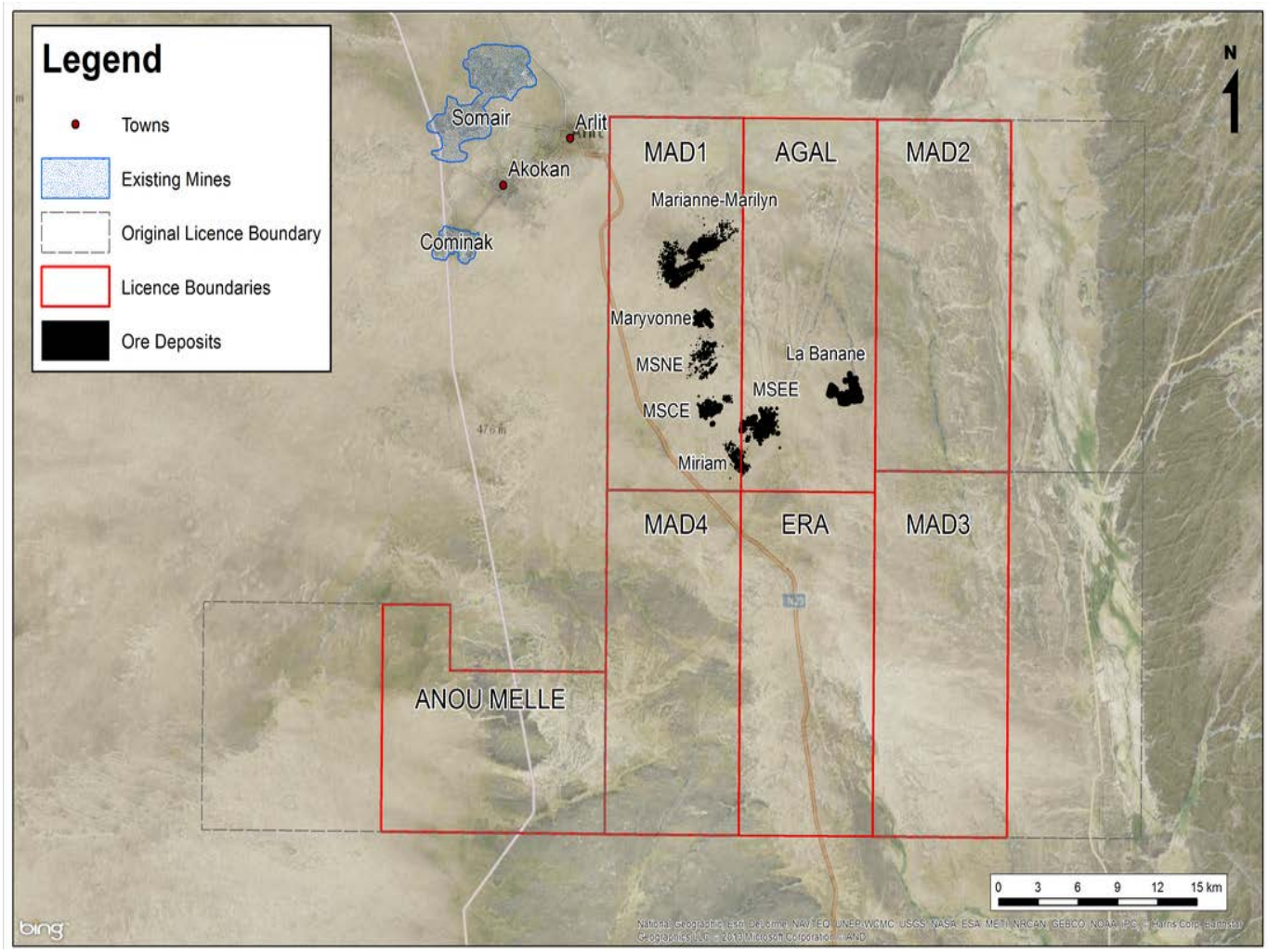
Madaouela Military Camp and Access to Exploration Land

Although the State of Niger is committed to facilitating access to every part of the tenement land, according to guarantees stated in the MIMC, the military camp is still present but GoviEx received permission to operate within the boundaries of the camp. The presence of the camp has helped secure the work around it. The GoviEx exploration team’s primary access to site is via the dirt road leading to the military camp and GoviEx is therefore committed to its maintenance.



Source: Map base by Magellan Geographix, 1997

Figure 3-1: Niger location map – Madaouela uranium project



Source: SRK, 2013

Figure 3-2: GoviEx land holdings – Madaouela Uranium Project

GoviEx has 100% ownership of Mad I, II, III and IV, and applications filed November 22, 2012 for Agal and Eral are pending. GoviEx has been advised by senior government officials that Agal and Eral can expect the applications to be approved.

4 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY (ITEM 5)

4.1 Access to Property

The Madaouela Uranium Project is situated southeast of the mining town of Arlit, as shown on the map below, and the Marianne-Marilyn deposit is located approximately 9 km from Arlit. The main road (Arlit to Niamey via Agadez) heading south from Arlit, which is paved, goes through the tenements. The Madaouela camp is accessible through a dirt road branching off from the main road at the outskirt of Arlit. Off the main road, the use of 4x4 vehicles is required and access to all areas of the tenement is possible.

An airstrip belonging to Somaïr, a subsidiary of Areva, was constructed at the start of their mining operation. Subject to the owner's agreement and the payment of a landing fee the airstrip is used by charter companies. No commercial flights are available to Arlit. A recently renovated airstrip exists in Agadez, and is operational.

4.2 Topography, Elevation and Vegetation

Topographic relief in the Project area is minimal, a few tens of meters from high to low. Areas of drilling are at approximately 430 m elevations (asl). Over much of the areas drilled, relief is less than 15 m from low ridges of sandstone outcrops to essentially flat sand covered plains. The nearest area of topographic of significance is the Aïr Mountains, the west flank of which is located 15 to 20 km to the east of the Project area.

4.3 Climate and Length of Operating Season

4.3.1 Climate

Regional climate

Figure 4-1 below illustrates the four climate zones in Niger reported by the Second National Communication of Niger under the United Nations Framework Convention on Climate Change (UNFCCC, 2009). These four zones are described as follows:

- The Sudano- Sahelian zone - representing about 1% of the country's total area with average rainfall ranges from 600 to 800 mm.
- The Sahelian zone - covering approximately 10% of the country and receiving 350 to 600 mm of rain per year.
- The Sahel- Saharan Africa - accounting for 12% of the land area and where rainfall range is from 150 to 350 mm average.
- The Sahara desert - covering 77% of the country where the average annual rainfall does not exceed 150 mm and within which the Project is located.

The Project is located in the Sahara desert climate zone and as described, the average annual rainfall is expected to be less than 150 mm per year. Given the Sahara desert climate is fully arid, precipitation is less than evaporation and therefore water available for storage, runoff or recharge is minimal.

Average annual temperature ranges between approximately 15°C and 35°C, with a mean of 28°C. There are three seasons: a relatively cold season, September to February, a hot season, March to June, and a humid season, June to September. In the hot season temperatures can exceed 40°C and in the cold season nights are generally cool with temperatures below 20°C.

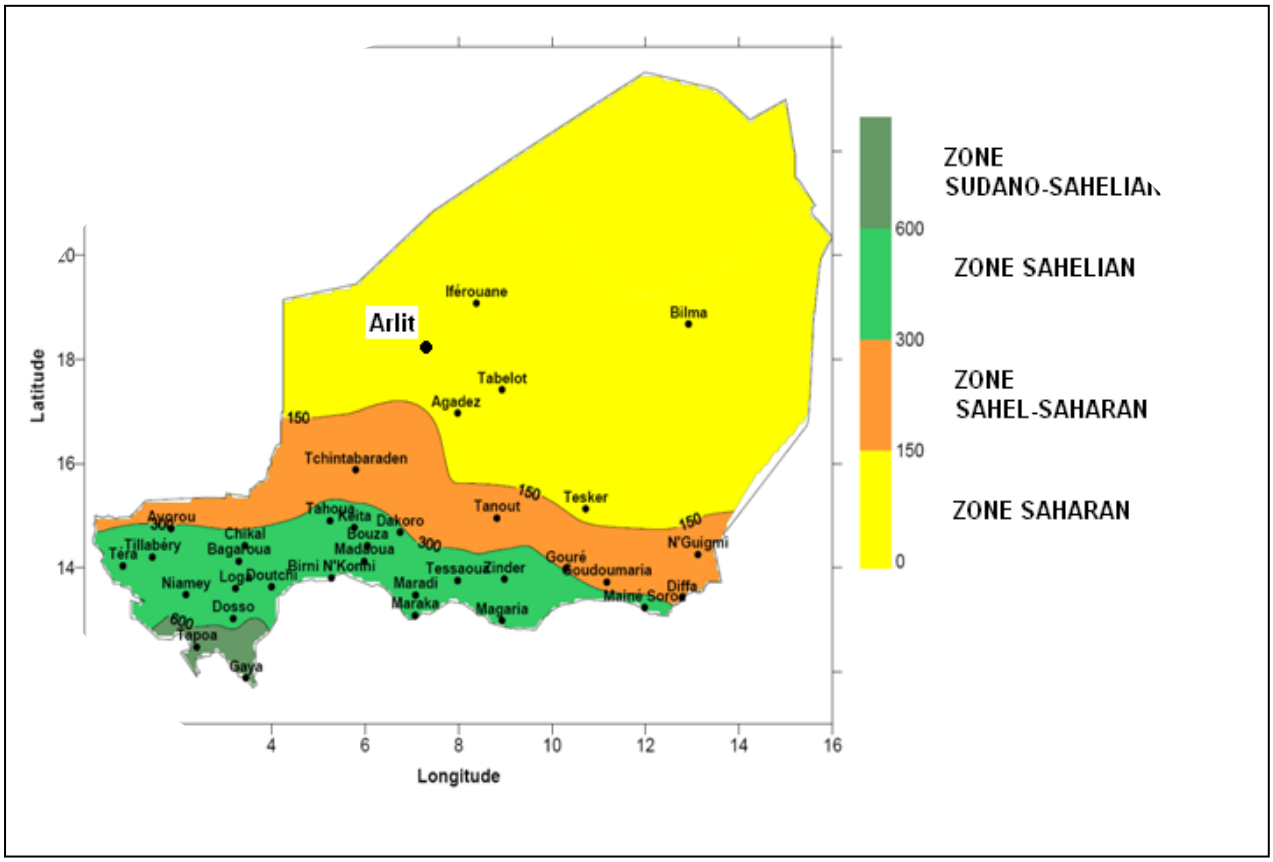


Figure 4-1: Principal climate zones of Niger (Modified from UNFCCC, 2009)

Precipitation

Local precipitation data for analysis has been obtained from two weather stations near the Project site named SOMAIR and Arlit. Data from Arlit station are provided by the National weather service and SOMAIR by the SOMAIR operation. The positions of these stations in relation to the Project site are shown in Figure 4-2.

The average annual rainfall based on the data collected from Arlit between 1972 and 2010 is 56.5 mm. The highest annual rainfall recorded for the period of 190.8 mm (2010) and the lowest of 0.5 mm (1972), highlight the annual variability. The average annual rainfall based on the data recorded at SOMAIR between 1981 and 2011 is 45.7 mm. The highest annual rainfall recorded for the period was 114.30 mm (1997) and the lowest 3.5 mm (1983).

There is a clearly defined dry season between November and May with little or no rainfall. The wet season occurs between June and September, with the peak rainfall month being August. There is some variation in the seasonal pattern evident for May and October when assessing data from the full record, with significantly greater quantities of rainfall recorded during these months, which would indicate an extended wet season during some years from May to October.

Table 4-1 summarises the maximum and average monthly rainfall value based on the data from the two weather stations for the period 1972-2010. All months have experienced at least a year with zero rainfall recorded. The dry season is characterised by contiguous periods of rainless months for all years whereas the wet season portrays variations within years in terms of rainfall distributions as well as the rainfall depth.

Table 4-1: Monthly averages and maximum rainfall (mm) at Arlit station (1972-2010) and SOMAIR station (1981-2011)

Station		Jan	Feb	Mar	Apr	May	Jun	July	Aug	Sep	Oct	Nov	Dec
Arlit	**Ave**	0.0	0.0	0.0	0.0	1.2	5.2	15.5	25.9	7.8	0.9	0.0	0.0
	Max	1	0	0	0	26.9	48.8	75	78.9	62	17	0.0	0.0
SOMAIR	**Ave**	0.0	0.0	0.2	0.0	1.7	5.3	10.0	21.4	6.0	1.1	0.0	0.0
	Max	0.2	0.0	5.6	0.8	15.9	40.8	30.8	85.6	33.3	13.2	0.0	0.0

A comparison of rainfall data from both stations for the period 1981-2005 is shown in Figure 4-3. Seasonal trends are common to both locations although the rainfall amount at Arlit is greater throughout the wet season than at SOMAIR.

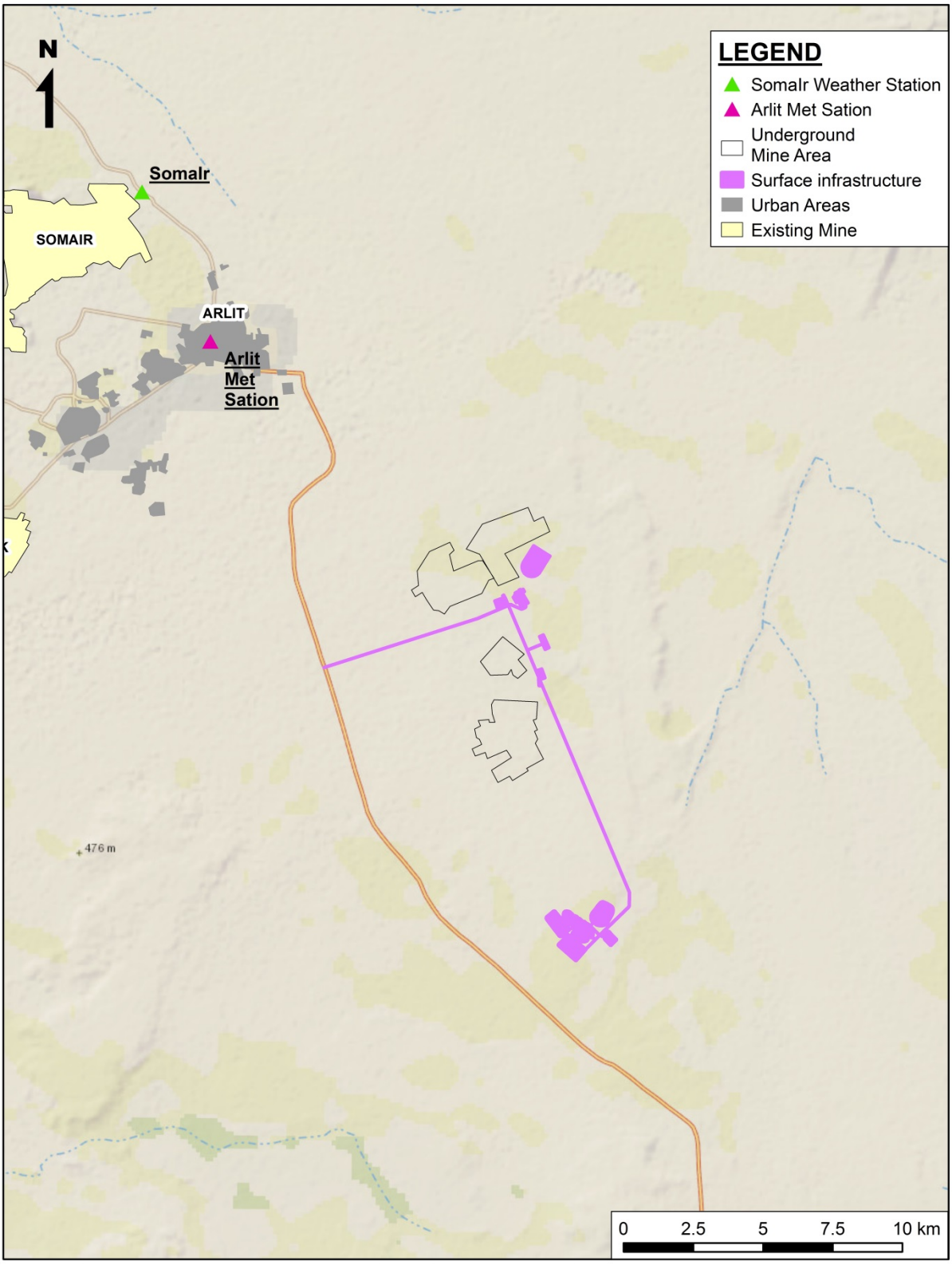


Figure 4-2: Met station locations

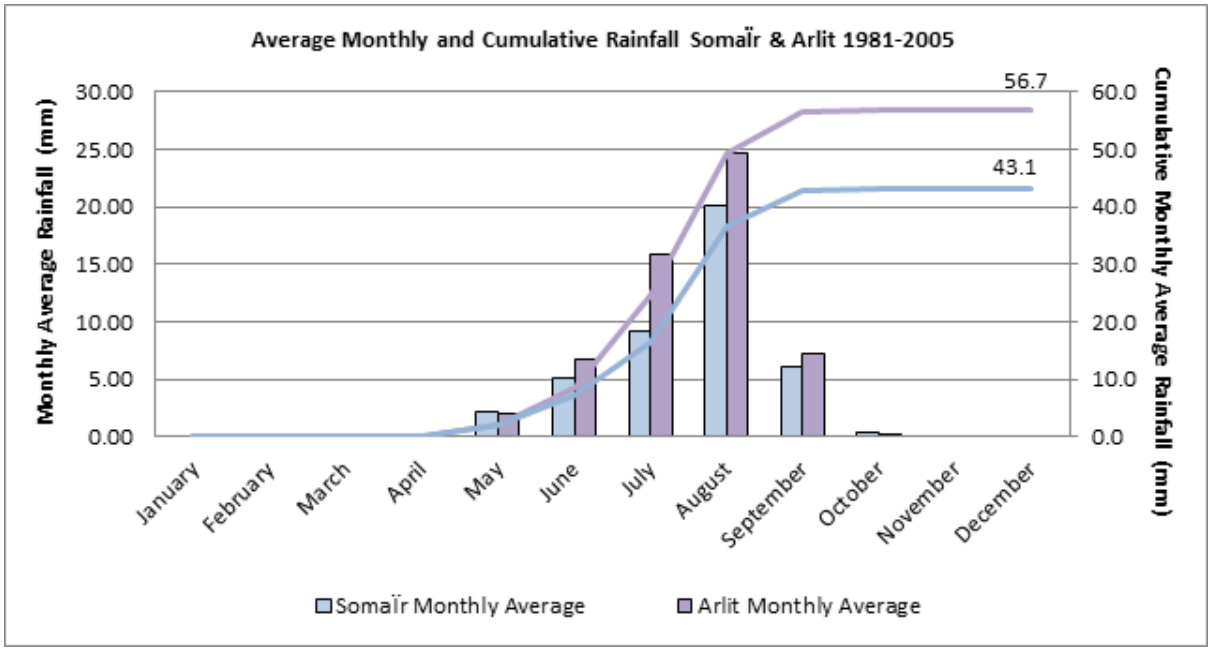


Figure 4-3: Average monthly and cumulative rainfall at SOMAIR & Arlit 1981-2005

Daily precipitation-storm pattern

Storm duration and intensity comprises short, intense peaks of heavy rain lasting 15 minutes and less intensive rainfall close to the main downpour (shower) lasting in total between 2 and 3 hours. The highest recorded 15 minute rainfall was 7.8 mm on the 17 June 2005. The highest 24-hour rainfall was 25 mm on the 7 August 2007.

There is only one record of two consecutive days with rainfall greater than 10 mm. This occurred on the 6 and 7 August 2007. This 48 hour rainfall comprises two separate rainfall events and a total of 38 mm.

Peak rainfall estimation

Rainfall data obtained at SOMAIR operation between 1977 and 2011 were analysed to estimate the 24h extreme rainfall. The largest recorded daily precipitation event was 43 mm in 1997. The ranked maximum annual rainfall series of this period has been utilised to estimate storm magnitude on a range of return periods using a variety of extreme event distributions, including Weibull, Gringorton, Cunnane and Hazen. The results of this analysis are summarised in Table 4-2 below and show narrow differences within the event distributions methods.

Table 4-2: Maximum 24 hour rainfall for different returns periods using Weibull, Gringorton, Cunnane and Hazen method with observed data from SOMAIR (1977 to 2011).

Return Period T		2	5	10	20	50	100	200	500	1000
Rainfall (mm)	**Weibull**	10.5	22.6	30.6	38.3	48.2	55.6	63.1	72.8	80.2
	Gringorton	10.8	21.6	28.7	35.5	44.4	51.0	57.6	66.3	72.9
	Cunnane	10.8	21.7	28.9	35.8	44.8	51.5	58.2	67.0	73.7
	Hazen	10.9	21.4	28.4	35.1	43.8	50.3	56.7	65.3	71.7

Evaporation data

Daily average evaporation data per month and year between 1981 and 2005 has been provided by the SOMAIR site. The average annual total evaporation based on the data collated at SOMAIR between 1981 and 2005 is 4,456 mm/y. The highest average monthly evaporation is 472 mm and occurs in May and the lowest 252 mm in January.

Figure 4-4 shows monthly averages of evaporation and temperature within this period. Average monthly temperatures range from 19.5°C to 35.3°C and it has an obvious effect in the evaporation as both evaporation and temperature fluctuate along the year comparably. The average temperature was 28.9°C, which is characteristic of Arid Climates.

Evaporation is greater than rainfall all year round, which is a typical of the Saharan Climate.

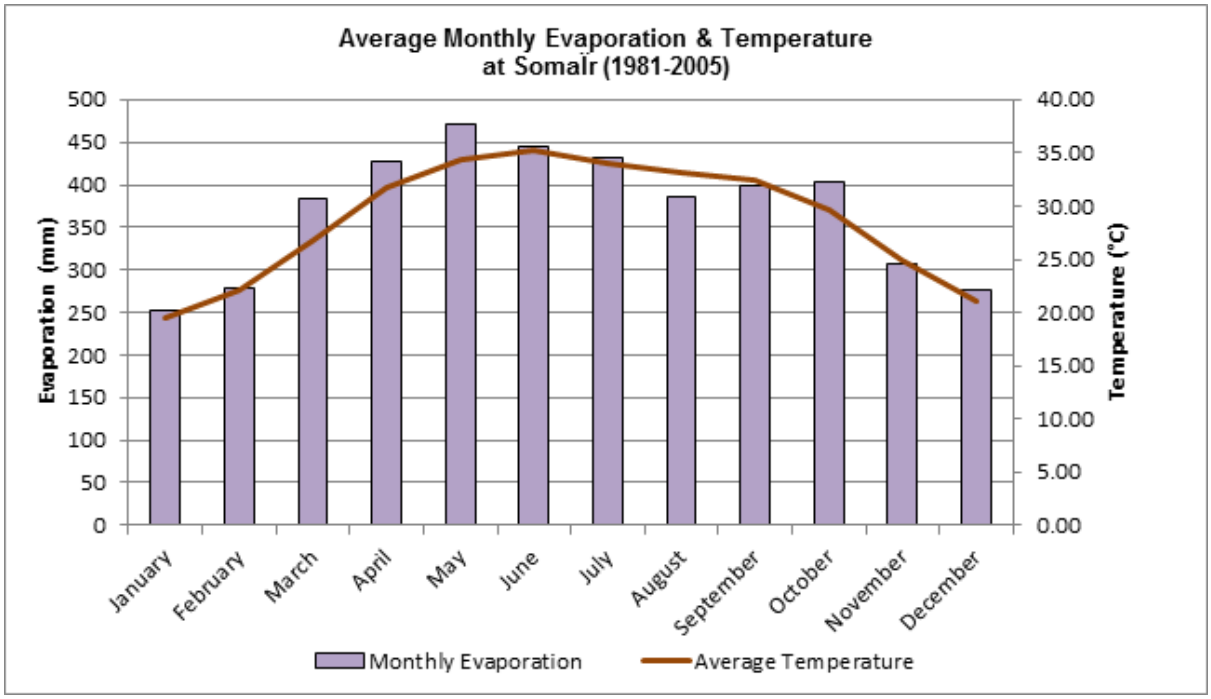


Figure 4-4: Average monthly evaporation & temperature at SOMAIR (1981-2005).

Wind

The wind regime in the Project area was established using the outputs of the 5th generation mesoscale model ("MM5"). Two year data output (2012-2013) was analyzed to establish the annual and seasonal wind rose for the project site. The main wind direction in the year is easterly, however southerly winds are also observed in high frequency. On an annual basis the distribution of the wind directions (i.e. blowing "from") are as follows:

- North – 4.31%;
- Northeast sector– 6.20%;
- East sector – 11.80%;
- Southeast sector – 5.83%;
- South sector – 10.60%;
- Southwest sector– 5.45%;
- West – 4.53% and
- Northwest -1.69%

The northeasterly winds are dominant in January and February. From February onwards the southwesterly winds increase in frequency until June, when the southwesterly winds convert to southerly winds. In the meantime northeasterly winds convert to easterly winds and diminish after June. From June onwards and until October, the southerly winds are the dominant wind direction. Starting in October southerly winds diminished in significance and easterly winds dominate the region until end of December.

Long term wind frequency data covering the period 1981 to 2004 collected by SOMAIR indicates that the northeasterly and southeasterly winds are dominant in the region with frequencies above 15% and 10%, respectively.

4.4 Physiography

The highest mountain, Mount Bagzane (elevation 2,022 m), is located in the Aïr Mountain to the east. Apart from the Niger River flowing through the southwestern tip of the country, and adjacent to the capital city of Niamey, there are no other free flowing perennial streams in Niger.

Underground water resource are available and most of the water resources used in Niger come from fossil water tables, which are drawn down more and more while little, if any, replenishment takes place considering the precipitation regime.

In the vicinity or Arlit, carboniferous sandstone formations host underground water tables that have been exploited since the start of the uranium mining operations in the 1960's. The human concentration that has accumulated over the years in the Arlit area has contributed to reducing the underground water resources; however, water reserves in the region are considered to be large.

Underground water sourced from older (Ordovician) sandstone is often artesian or confined, and this particular source locally is salty water that is not fit for human consumption.

Native vegetation is scarce and has almost completely disappeared due to the combined effect of human pressure, overgrazing and droughts. There is very little permanent vegetation on the Project area. However, at times, local herds use this territory as grazing land, mostly after the rainy season, which could turn the desert into a green countryside for just a few weeks.

4.5 Local Resources and Infrastructure

The towns of Arlit and Akokan have a combined population of approximately 160,000, and were developed by Somair (Arlit, from 1968) and Cominak (Akokan, from 1975), to house the staff and workers required for the uranium mines of the two mining companies, which continue to operate today.

The proximity of the town of Arlit and Akokan are an asset for the Madaouela Uranium Project. The towns have electricity, potable water and two hospitals. Every administration is represented and the main banks have outlets. There are filling stations, bus transportation, repair shops; essentially all that is needed to sustain the community services that would provide for people associated with mine development.

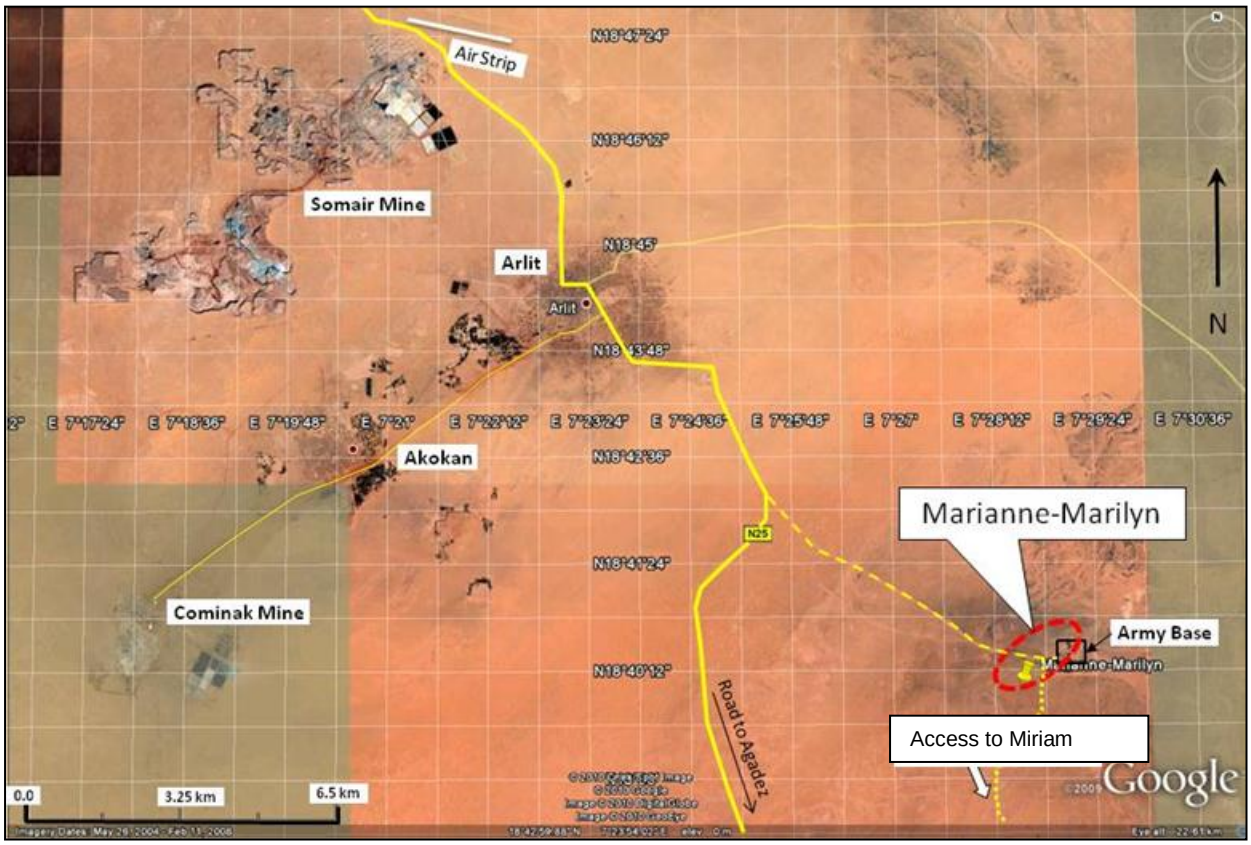
Arlit is connected to the Southern part of Niger via the so-called "uranium road" through Agadez and Tahoua to Niamey, the Niger capital further south. This road is paved almost the entire way, but poor road maintenance has resulted in deterioration. Most of the goods and services that are necessary for people to live in Arlit and for the uranium industry to operate

and produce come through that main access road. The Northern link to Algeria, a poorly maintained dirt road, is not practicable for major transportation of equipment and supplies, but is still used by most local transporters for petrol and food.

A power line connects the town to the Sonichar coal-fired power station located north of Agadez. Extensions are reportedly planned to the power station to accommodate the needs of Imouraren and possibly any new mining operation. An intensive drilling program apparently has demonstrated mineable new zones of coal in the vicinity of the power station. However any future mines development would have to seek guarantees from Sonichar to provide sufficient supply.

Land use related to any future exploration or/and mine development scheme is allowed under the MIMC provisions, including some flexibility to use any portion of the tenement land and/or any of neighbouring lands. Considering the essentially flat landscape and terrain of the Madaouela tenement, there should be no issues in identifying the surface areas that are necessary for any mine layout requirements for future facilities (waste rock and mineralization stock piles, tailings management facility, processing plant, etc.).

GoviEx completed construction of an exploration camp consisting of several small buildings to facilitate drilling activities in the southern part of the Project area. Manpower requirements would be sourced as unskilled labour from the Arlit area, as well as locally from a workforce that has been locally trained for mining related skills.



Source: Map base by Google Earth 2010, modified by SRK, 2010

Figure 4-5: Madaouela uranium project - access map

5 HISTORY (ITEM 6)

5.1 Past Exploration and Development

5.1.1 CEA (1963 to 1965)

CEA conducted drilling operations using drilling grids of 800 m over large areas and down to 100 m, two contiguous mineralized zones termed Marianne and Marilyn. The discovery of the Marilyn deposit was then drilled locally at 50 m and less spacing in the areas where a mining test was implemented with a view to sampling uranium mineralization and investigating the global rock quality underground.

CEA sunk a 67 m deep shaft and drove 330 m of drifts within the mineralized formation. The simultaneous discovery of the Somaïr uranium deposits, west of Madaouela, was responsible for the decision to stop all works at the Madaouela site. The aim of the Marilyn mining test was threefold:

- To sample the mineralization and carry out tests for processing and recovery, and to study the mineralogical characters of the mineralization and get a global understanding of the mineralized zone geology;
- To establish and calibrate procedures for grade estimation and grade control (grade/radioactivity correlations, U/Ra equilibrium, etc.); and
- To evaluate mining schemes for future mining possibilities.

Results of Madaouela CEA mining test are summarized in a report published in 1967, but is not summarized here as it does not specifically relate to this technical report.

5.1.2 PNC (1980 to 2000)

PNC, the Japanese Pacific Reactor and Nuclear Fuel Development Corporation, took over the CEA tenement in 1980 in association with ONAREM, a Niger State-Owned Organization. In 1981, PNC resumed uranium prospection across the Sekiret exploration tenement and in 1988 across the Tessili tenement. Both tenements were then defined with a much larger surface area than the current Madaouela I tenement.

PNC conducted additional uranium exploration work up to 1992 and produced a report on the feasibility of the Madaouela deposit in 1993, which was later updated (1999) by the Japan Nuclear Cycle Development Institute, the new company name for PNC. Very few drillholes (less than 20) were drilled by PNC in the area of what is now Madaouela I; all were located either on the south or the east, none on Marianne or Marilyn. The feasibility study was fully based on CEA drillholes and ore sourced from the CEA stockpiles. The 1993 feasibility study looked at open pit and underground mining schemes for the Marianne and Marilyn deposits, based on mineral resources of 6,199.7 t U, and their economic assessments. Its main conclusions were as follows:

- Open-pit mining would not be economic, even when considering the uranium price of USD 60 /lbU_3O_8; and
- Underground mining would not be cost-effective if the uranium price is below USD 40 /lbU_30_8.

In 1999, the economic assumptions and data for the 1993 feasibility study were updated and a new run concluded that;

- Underground mining could be cost-effective, provided the uranium price stabilizes at around USD 27-28 /lb U_30_8 level.

However, the open-pit mining scheme remained uneconomical under the same assumptions as those of the 1993 study.

A Chinese company (CNUC) took over Madaouela and held the land from 2003 to 2006. They drilled a limited number of holes and departed, apparently because they could not access the CEA original data. Little exploration work seems to have been carried out until GoviEx Niger Holdings Ltd. ("GNH Ltd.") took over the property in 2007. Note that GoviEx was in the same situation as far as access to old data, and decided to reopen as many former holes as necessary to sustain its drilling program.

5.2 Historic Mineral Resource and Reserve Estimates

The 1993 PNC feasibility study quotes in pages 9-10 of the report the uranium resources as 3,263,000 t at 0.19 % uranium or 6,199.7 t uranium. In 1999, the updated PNC study reported the mineable mineral resources shown in Table 5-1, after simulating the underground operation on Marianne and Marilyn, using a cut-off grade of 0.1 %.

Table 5-1: Historical mineable mineral resources

Deposit	Mineable (t)	Grade (% U)	Contained U (t)
Marilyn	1,642,660	0.17	2,777
Marianne	1,016,059	0.16	1,569
Total	2,658,719	0.17	4,346

Source: PNC, 1999

At 0.15 % cut-off grade, the overall mineable reserves were computed at 1,887 t U, equally distributed between Marianne and Marilyn.

These historical reserves are being presented as part of the historical record only, and demonstrate significant historical effort to evaluate the uranium mineralization at the Madaouela Uranium Project. These historical reserves have not been evaluated by a Qualified Person for compliance with CIM resource/reserve classification, and therefore should not be relied upon. In addition, GoviEx is not presenting these historical numbers as current resources or reserves for the Madaouela I tenement. GoviEx has completed additional drilling to the mineralization at Marianne and Marilyn, and other areas on the Property, as is presenting current CIM compliant resources in 13.9 of this study.

5.3 Historic Production

Historic production of uranium at the Madaouela Uranium Project is limited to the few thousand tonnes of material extracted from the CEA constructed test underground mine. Commercial mine production has not happened. Historic production from the two mining operations, Cominak and Somaïr, in the Arlit District are estimated to be over 250 Mlb U_3O_8.

6 GEOLOGICAL SETTING AND MINERALIZATION (ITEM 7)

6.1 Regional Geology

The following description of the geology of the area was compiled from descriptions by Bigotte and Obelliane (1968), Valsardieu (1971), Black et al., (1973), Elhamet (1983), Forbes, (1989), Capus et al, (2004), Gerbaud (2006) and the written notices accompanying the geological maps for the Tegama quadrangle (BRGM, 1980).

The Madaouela Uranium Project properties are located in the Tim Mersoi Basin, a sub-basin of the Phanerozoic Iullemeden Basin developed on the Proterozoic West-African shield basement. This basin covers most of the western part of the Republic of Niger with extensions in Algeria, Mali, Benin and Nigeria. It opens and deepens toward the south and west. In early Paleozoic, an open gulf developed to the south of the Central Saharan Massif and fed continental sediments to the developing basin. During the Mesozoic and Tertiary, the area was mainly continental, periodically invaded by marine transgressions diminishing in thickness to the south and passing laterally into continental series. Uplift movements beginning in the Middle Eocene gave the basin its present aspect. It was subsequently filled with continental fluvial and lacustrine sediments.

In the Madaouela Uranium Project area, the total thickness of the sediments could reach up to 1,500 m. Five main periods could be distinguished:

1. The Proterozoic basement of the Air Mountains.
2. The large deltas with reduced sedimentation associated with low temperate climate during the Lower Carboniferous.
3. The Gondwana deserts associated with the warmer climate during the Upper Carboniferous and Permian.
4. The large fluvial system and volcanism during the Jurassic.
5. The clay lake and fluvial system during the Cretaceous.

6.1.1 Proterozoic Basement of Air

The basement of the Air Mountains is composed of Upper Proterozoic highly metamorphosed rocks, crosscut by numerous granite and alkaline intrusions. Seven lithologies have been recognized, including biotite gneiss, leptynites, diopside-hornblende leptynitic gneiss and micaceous shale, with some quartzite, amphibolite and limestone. To the west, they are thrust on to epimetamorphic rocks, probably middle Proterozoic in age.

Several generations of granite have been recognized, such as early leucocrate granites, resulting from melting of the sediment, syntectonic calkalcaline granites, and late tardi-tectonic alkaline granites. A second generation of alkaline granites crosscut all the formation. The basement is strongly folded and crosscut by numerous N-S, N20 and N140 faults.

During the Mesozoic, large annular massifs were emplaced and constitute the northern extension of the Younger granites in Nigeria. Tertiary trachytes and some basalts have been also described.

6.1.2 Paleozoic Formations

The Paleozoic sediments are outcropping between Arlit and the Algerian boundary. Pre-Carboniferous sedimentation consists of Cambro-Ordovician sandstones and graptolite shales. Several Devonian sandstone units are deposited from lower to Upper Devonian

(Idikel, Touaret, and Akara). During the Lower Dinantien, the Farazekat sandstone marked a glacial sedimentation, followed by varved argillites.

The Carboniferous formations are of major interest because they host the major reduced uranium deposit in the Arlit area. The stratigraphic sequence begins by the grey-black Talak Visean argillites. It consists of black shale with brachiopods, productoids and spiriferoids. The upper unit is a phosphatic siltstone, coarsening upward, which represents the continental shelf deposition.

The Akokan unit ("UA") is a transitional term between the marine clay of Talak and the fluvio deltaic sedimentation of the Guezouman and Tarat. It consists of several lenticular fine grained silty clay units, which could not be mapped individually. The units seem to have deposited conformably on the Talak argillites. Locally sediments of the Akokan unit have been accumulated and preserved of erosion within monoclinal structures. These structures could correspond to extensional half-grabens.

The Guezouman formation includes a lower and an upper member. The lower member deposited on a paleo surface dominated by N70E ridges, and the flow direction is generally in that direction. It is composed of fine- to coarse-grained cross-bedded sandstone units with minor conglomerates (Teleflak) at the base. These contain quartz, phosphatic siltstone gravels, more or less deformed silty shale debris, metaquartzite, granite and rhyolite pebbles. This conglomerate corresponds to channel lag deposition and contains reworked debris of extensive silcrete formed during the emersion of the Talak-UA ensemble. The upper member, flowing southwest to south, consists of fine to medium grained sandstone, with minor siltstone and thin argillaceous intercalations. The Tchinezogue Namurian argillite and silty to fine sandstone follow and are capped by the Unite Terminale, which may consist of argillaceous siltstone and local fine grained sandstone. Several subunits are distinguished in the Tchinezogue, based on their colour and their silt or clay dominant composition (gris bleu, blanchatre, bleu-vert).

The late Carboniferous megasequence, the Tarat Formation, is of Westphalian age; it consists mainly of sandstones interbedded with organic- and pyrite- rich mudstones. It represents the filling of large fluviatile channels. The lateral variations are large, ranging from coarse sandstone, without internal preserved structures to fine sandstones and siltstones. The Tarat sedimentation ends with reduced argillaceous siltstone (Tarat argileux), with micro-ripple laminations

The Madaouela formation consists of coarse arkose and arkosic sandstone unit intercalated with laminated silty clays. The formation has not been recognized west of the Madaouela flexure, it reappears west of the Arlit in Azzaoua Permo-Triassic Formations

During the Permian time, another sandstone unit, the Izegouandane is deposited. It is more arkosic, with carbonate cement, showing numerous evidences for an oxidized environment. The sandy argillite of Tejia caps the series. Continental sedimentation continues during the Triassic, including the Tamamait sandstones, the Moradi red argillites and the sandstone of the Aokare unit.

6.1.3 Jurassic Formations

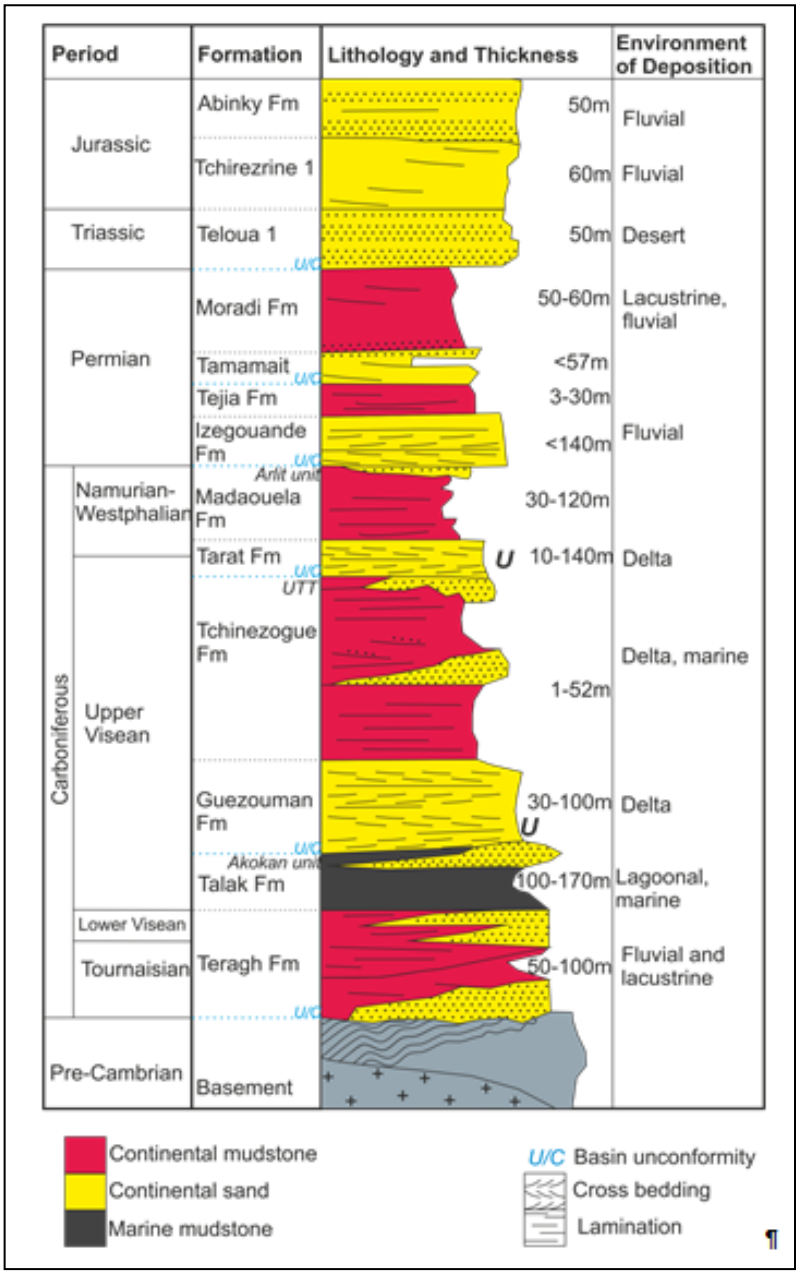
The Jurassic formations are largely distributed and cover most of the district. They have been collectively named the Agadez sandstone, but comprise several units (Teloua, Tchirezrine) where some analcimolite of volcanic origin have been described (Abinky). The upper Tchirezrine is composed of feldspathic sandstone with oblique stratifications; it hosts the large

Imouraren uranium deposit, which is located south of the Madaouela Uranium Project.

6.1.4 Cretaceous to Present

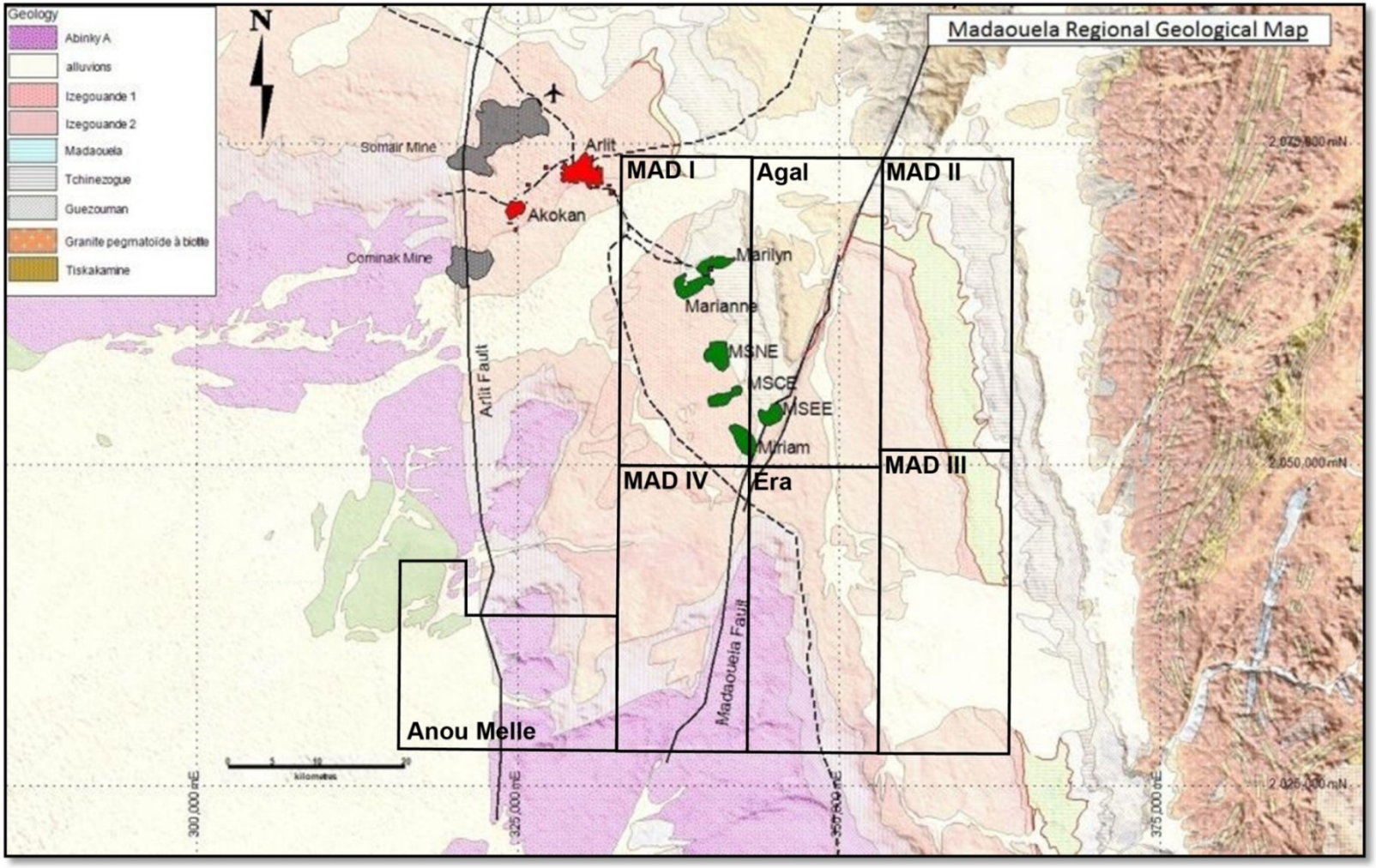
During the Cretaceous time, argillites and sandstones were deposited, especially near paleo-reliefs. The main unit is the Irhazer argillites and silts, with marl, limestone, silexite and fine sandstones, and is followed by the Tegama sandstones and argillites that show dinosaur-rich beds.

The continental sedimentation is still active along the palaeochannels, sedimentary detritus cones resulting from the erosion of the Aïr Massif.



Source: Stratigraphy of the Madaouela District based on GoviEx information, drillhole geological logs and published materials (Bigotte and Obelliane 1968; Coquel et al. 1995; Tabore et al. 2011)

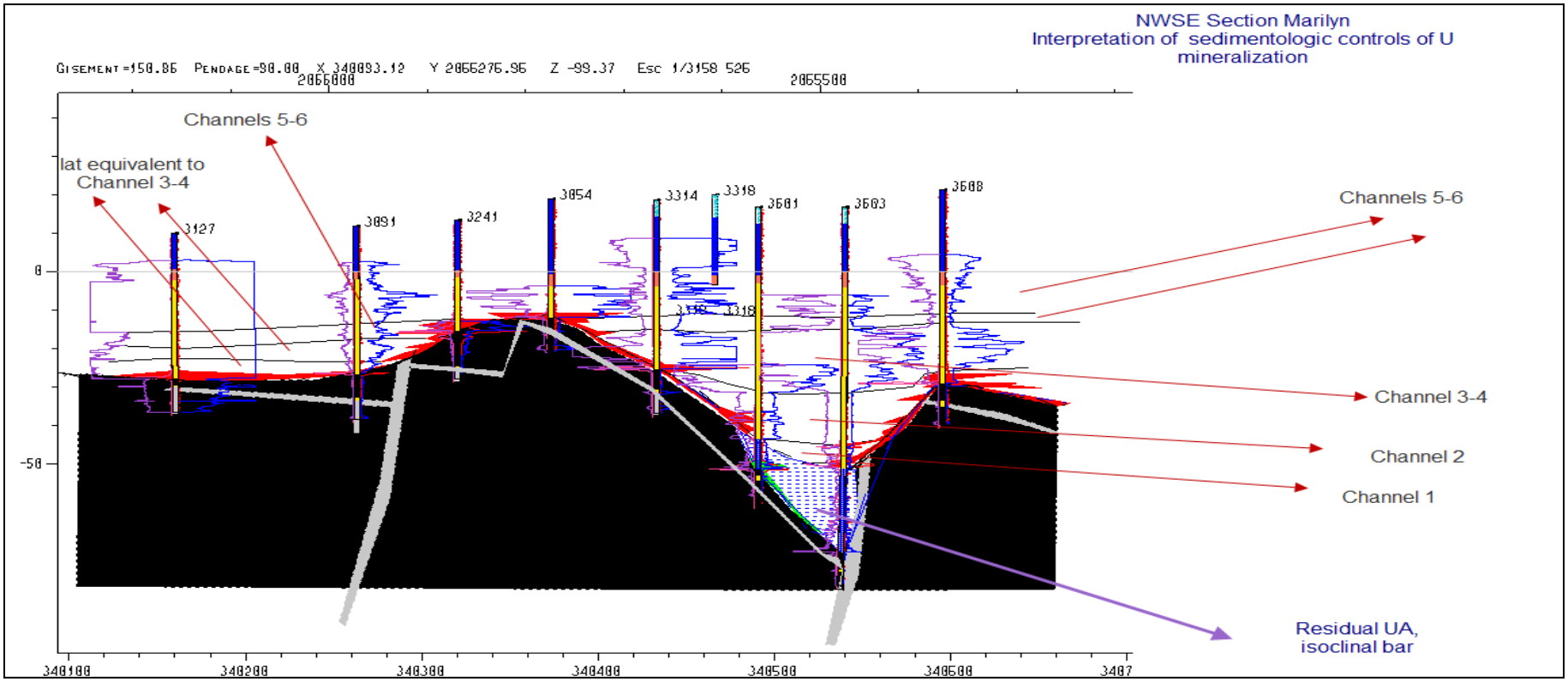
Figure 6-1: Stratigraphic column



Source: GoviEx, 2011 modified by SRK 2013

Figure 6-2: Project geological map

GoviEx has 100% ownership of Mad I, II, III and IV, and applications filed November 22, 2012 for Agal and Eral are pending. GoviEx has been advised by senior government officials that Agal and Eral can expect the applications to be approved.



Source: GoviEx, 2009; modified by SRK, 2010

Figure 6-3: Representative cross section A-A'

6.2 Regional Geological Structures

The structure of the Tim Mersoi basin is marked by the westward dip of the units. The deformation of the sedimentary body resulted from basement fault activities located between the Air Massif and the Azaoua lineament. Several large faults systems cut the sediments and have played a major role during the sedimentation since the Upper Paleozoic.

Gerbaud (2006) have proposed a structural evolution including an E-W Jurassic extension, and a NW-SE Cretaceous compression. The main structure is the NW to N20 Arlit fault-flexure, on the In Azaoua continental lineament. It has not been active during the sedimentation of the Guezouman, but active during the Tarat sedimentation, and has been reactivated several times. It shows a complex organization, with mainly sinistral, reverse and normal faults and folds, dome structures. All the major uranium deposits are located immediately to the east of the fault.

The Madaouela fault, NE-SW (N30) shows similar characters; with a large uplift of the northwestern compartment (Qrt-de-brie). It is also probably inherited from a basement fault.

The ENE-WSW (N70) faults display a regular set with mainly dextral offset; faults show popup structures related to trans-pressional relay and trans-tensional graben, both of them could be mineralized (Azelik, Daly).

A set of NW-SE (N150) sinistrial faults is less apparent, but could have played a role during the sedimentation.

6.3 Localised Geology of the Madaouela Uranium Project Deposits

6.3.1 Marianne-Marilyn

Marianne-Marilyn deposit is the northern most deposit in the project area.

Geology

Four main sub-horizontal geological formations exist in the Marianne-Marilyn project area. From surface, Tchinezogue, Guezouman, UA and Talak sedimentary units exist and are described as follows from youngest to oldest:

- Tchinezogue is a fluvio-deltaic sedimentary unit with thin sub-horizontal bedding. A degree of heterogeneity exists, and areas near the surface are weakened due to weathering effects;
- Guezouman is a massive, relatively competent fine to coarse grained sandstone unit with a relatively low density. The base of this unit, situated at depths ranging from 50 to 160 m from surface, hosts the uranium mineralization;
- UA exists in channels between the Guezouman and Talak, and is characterized as thinly bedded mudstones and siltstones. The two NE-SW running central drives are planned directly above the UA channels; and
- Talak unit is fine grained claystone and mudstone lying at the bottom of the project area. It is weak and relatively isotropic.

Isometric view of the project area and an N-S cross section through the lithology is shown in Figure 6-4.

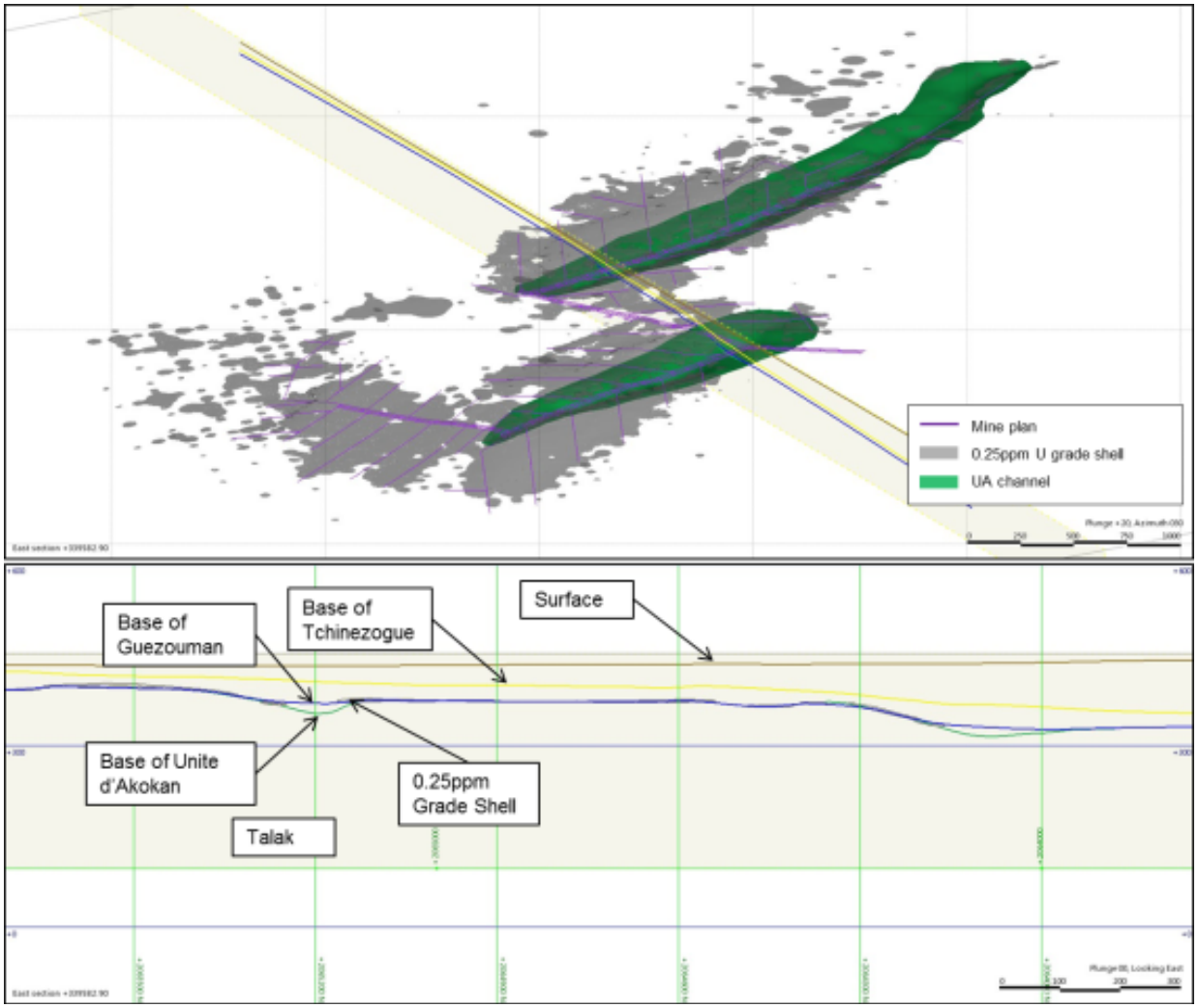


Figure 6-4: Isometric view and Cross section view of Marianne-Marilyn geology

Structural Geology (Marianne-Marilyn)

Structural Setting

The Marianne-Marilyn deposits are the northernmost mineralised zones within the Madaouela Project. The mineralisation is elongate along a WSW-ENE trending structural axis that comprise of a series of domes, faults and monoclines (Figure 6-5).

The major structures affecting the deposit and their potential implications for the geotechnical evaluation are outlined below.

Major Structures

Monoclines

The Marianne-Marilyn deposits are affected by two well-defined monoclinal structures that control the location of the UA sediments (Figure 6-5); Yahaya & Lang, 2000). Based on subtle gradient changes in the base of the Guezouman/Top Talak horizon a further, smaller, monoclinal-type structure has been interpreted to the north of the large southwestern Akokan-bearing structure.

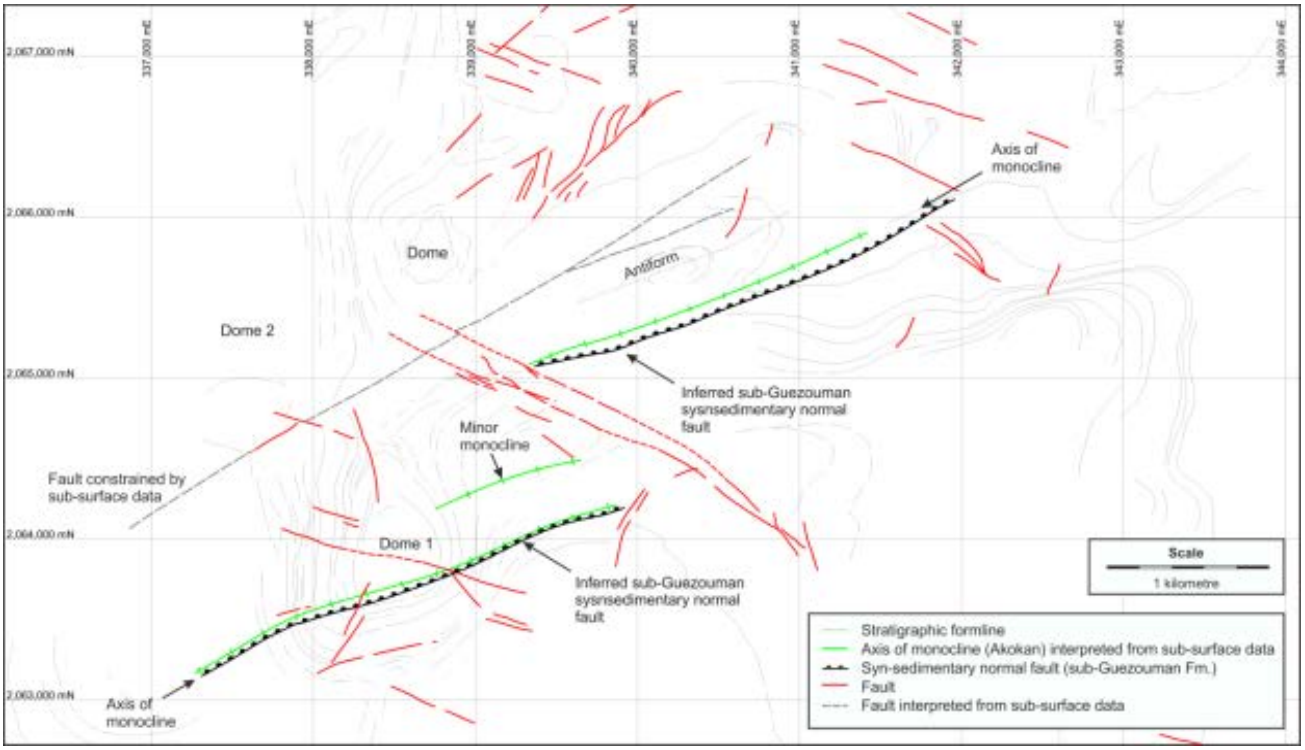


Figure 6-5: Plan view of stratigraphic formlines and structures interpreted at Marianne-Marilyn.

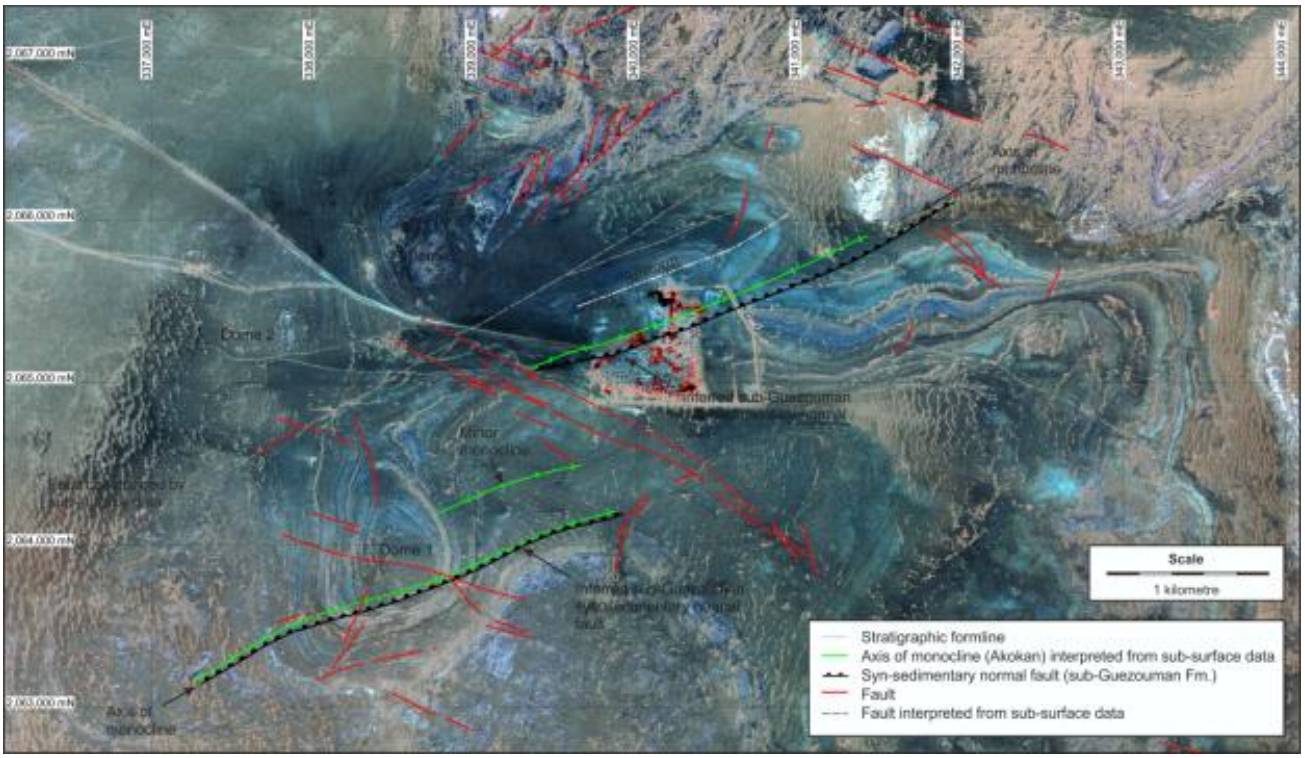


Figure 6-6: Plan view of stratigraphic formlines and structures interpreted at Marianne-Marilyn relative to the Quickbird imagery

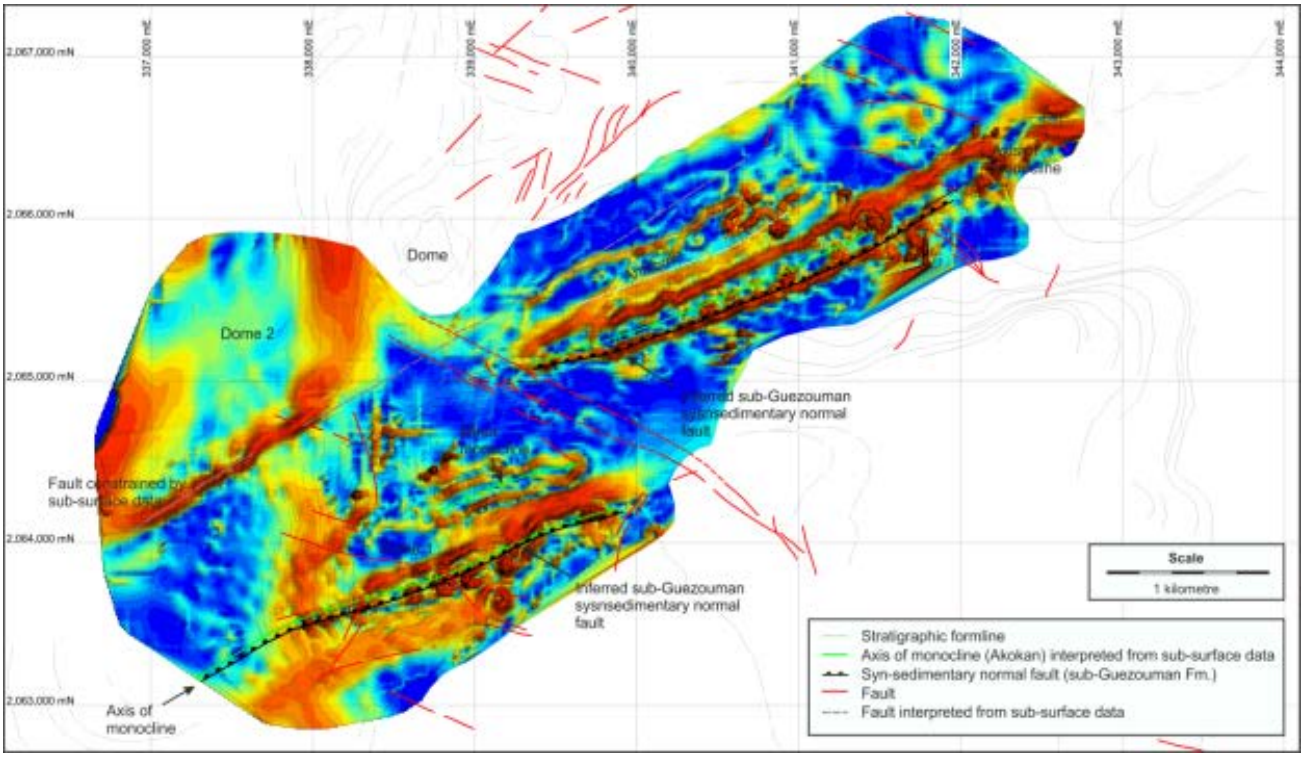


Figure 6-7: Plan view of stratigraphic formlines and structures interpreted at Marianne-Marilyn relative to a dip map of the top Talak/base Guezouman

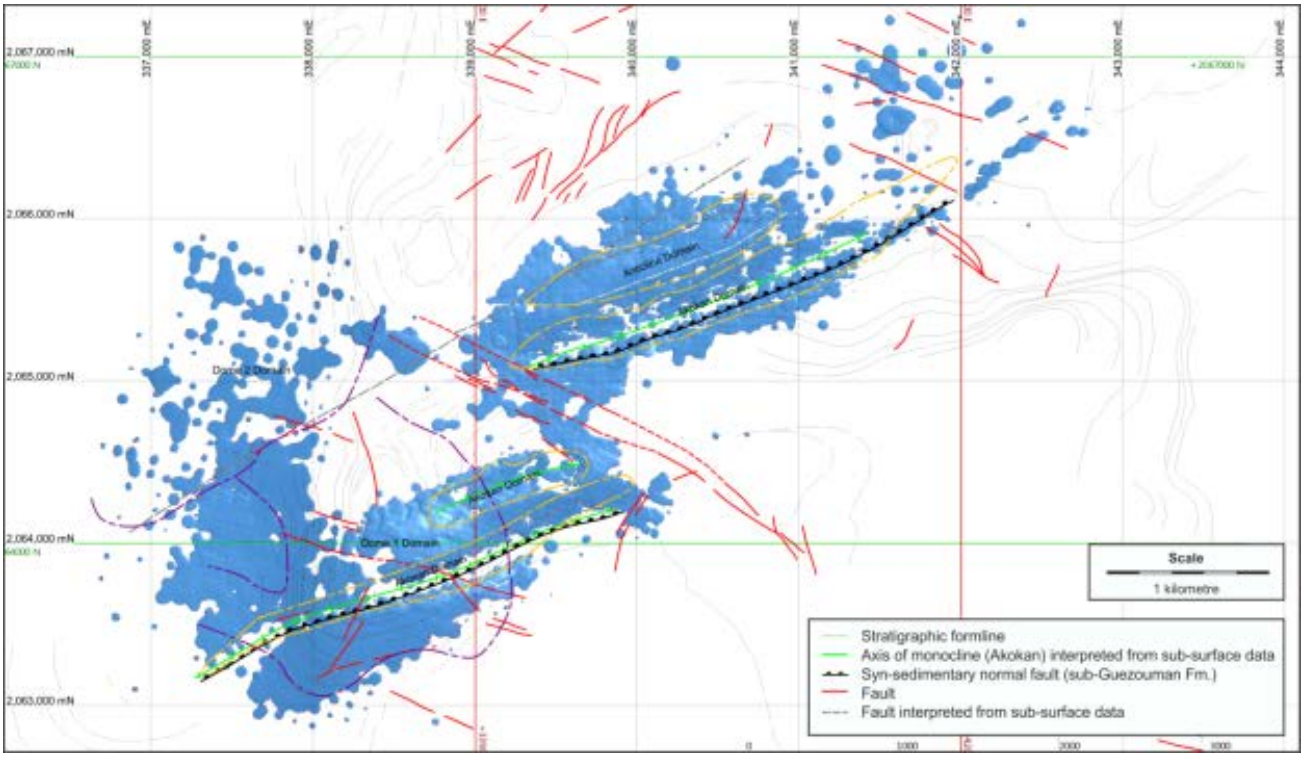


Figure 6-8: Plan view of structural domains relative to the Leapfrog 0.25 kg/t eU grade shell for Marianne-Marilyn.

Domes

The west of the Marianne-Marilyn deposit is dominated by a dome, where the stratigraphy defines a radially gently-dipping zone of uplift (Figure 6-5 and Figure 6-6). Stratigraphic dips are accentuated on the flanks of this structure but near-horizontal at the apex. In general the stratigraphic dips on the flanks are approximately 5° or less.

The origin of the domes is not fully understood but is likely to be related to contraction over basement-controlled structures, related to broadly N-S or NW-SE oriented shortening during the Upper Cretaceous and Tertiary (Gerbeaud, 2006).

Faults

In total eight faults have been interpreted to be through-going structures. The majority of these traces are associated with small displacements of the stratigraphic traces on the surface. In addition to these interpreted through-going fault structures are suspected zones of syn-sedimentary faulting, which control the positions of the Akokan channels.

Broadly radial to the dome are a set of minor brittle faults that radiate out from the centre of the dome. Where stratigraphy permits the displacement to be constrained, these faults accommodate 1-2 m of vertical displacement and/or 5-15 m of strike-slip displacement. They are therefore considered to be minor faults.

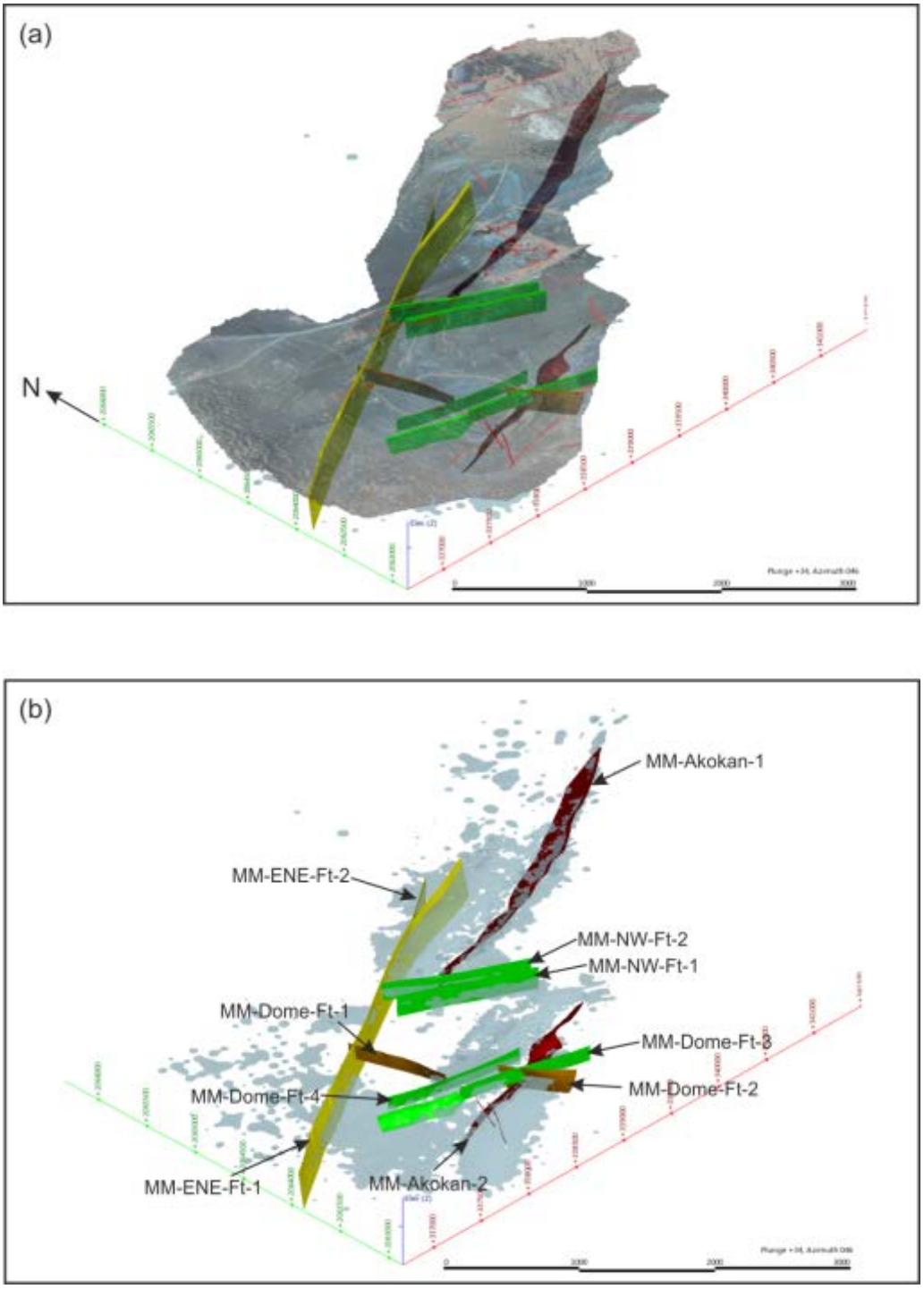


Figure 6-9: 3D view of interpreted faults at Marianne-Marilyn: (a) relative to topography with interpreted Quickbird image drape; (b) relative to Leapfrog 0.4 kg/t eU grade shell.

Stratigraphic Dip

Stratigraphic dips are known from both average dip of horizons between drillholes and direct determinations of bedding dip from acoustic televiewer data and oriented drill core. Generally bed dips are low, <5 °, outside of the monocline and dome structures. Importantly from an underground mining perspective, bed dips from drillholes generally show good agreement with the 3D horizons modelled from horizon intercepts. This supports the interpretation that the horizons are generally flat and not affected by short-wavelength dip changes that may have affected the elevation of the ore horizon. It is anticipated that few problems will be encountered staying on the ore during mining associated with dip variations.

Drill on Marilyn in mid-ground, Army Camp in background.
Source: SRK, 2010

Figure 6-10: Photo looking southeast from Guezouman outcrops

Sandstone (background) and Unite d' Akokan (foreground)
Source: SRK, 2010

Figure 6-11: Typical outcrop of Guezouman

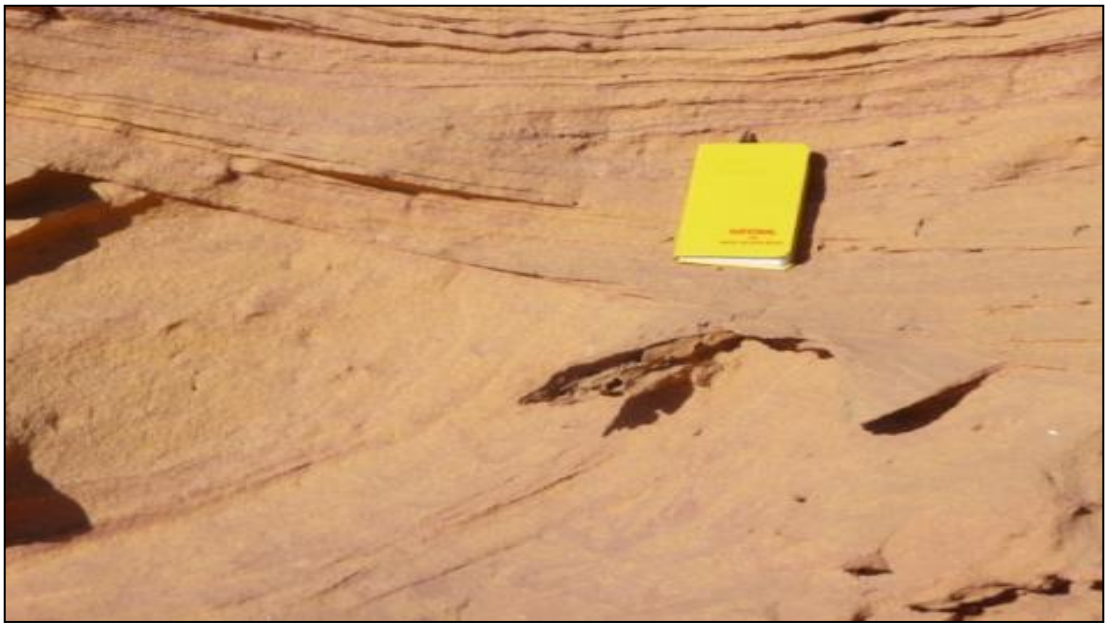


Figure 6-12: Cross bedding in Guezouman Sandstone

6.3.2 MSNE

The MSNE deposit is approximately 4 km south of Marianne-Marilyn and is planned be mined by conventional room and pillar method. Ore depth ranges from 100 m to 160 m across the deposit.

Geology

Exploration and geotechnical drilling suggests that the geology at MSNE comprises Guezouman sandstone overlying UA channels overlying Talak mudstone formation. The Guezouman dips sub-horizontally across the whole deposit and forms the hanging wall and host rock. The UA formation exists in the southern area of the deposit and will form the footwall. To the north, where UA is absent, the Talak unit will form the footwall. Figure 6-13 shows isometric view of MSNE project area and an N-S cross section through MSNE project area with the geology wireframes provided by GoviEx.

Characteristics of each formation are considered to be similar to those described in Section 3.2 (Geology of Marianne-Marilyn).

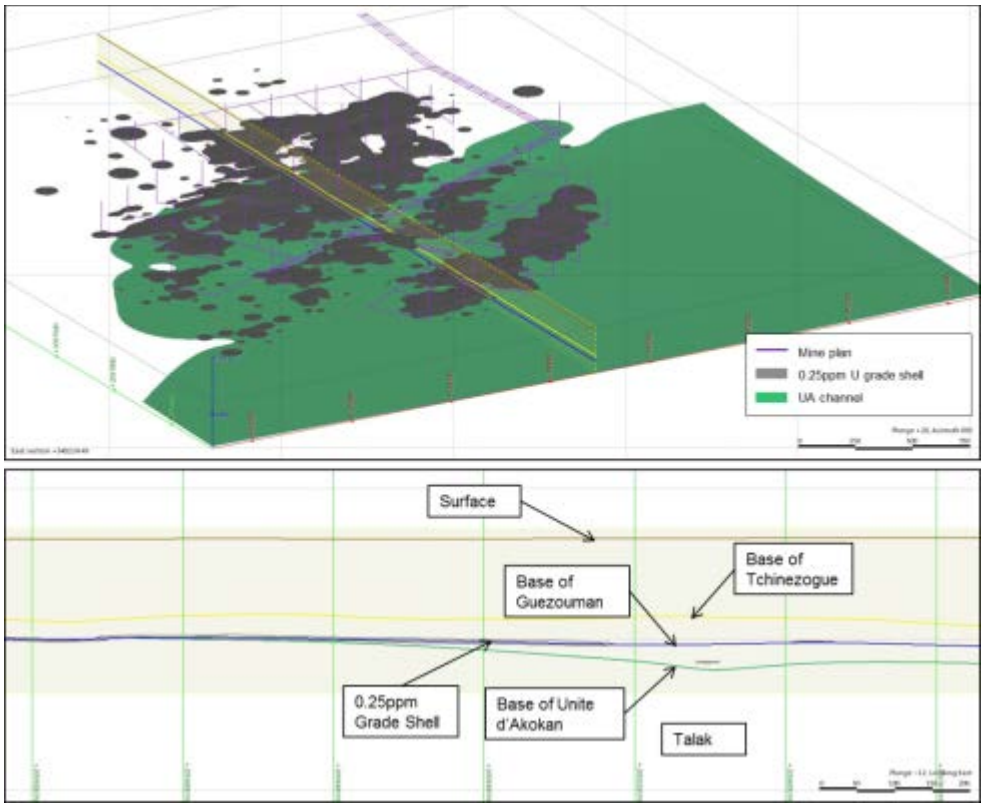


Figure 6-13: Isometric view and Cross section view of MSNE geology

Structural Geology

Structural Setting

The structural setting of the MSNE deposit is only partially understood because of the very poor exposure across the deposit area.

Major Structures

Monoclines

A single WSW-ENE trending monocline is interpreted from the depressed floor of the Talak Formation (Figure 6-14 and Figure 6-15) and the presence of an Akokan *barre* (Yahaya and Lang, 2000).

Faults

In total six faults have been modelled to be through-going structures at MSNE. All of the structures interpreted at MSNE have some surface manifestation.

MSNE-NE-Ft-1 appears to belong to a relatively significant NE-SW fault trend. This fault trace appears to bound the deposit area to the west and accommodates a significant west-side-down dip-slip displacement that juxtaposes Tchinozogue Formation on the west against Tarat Formation on the east. Fault displacement is interpreted to be transferred to an adjacent segment to its south over a broad zone of fault overlap, which may constitute a relay zone.

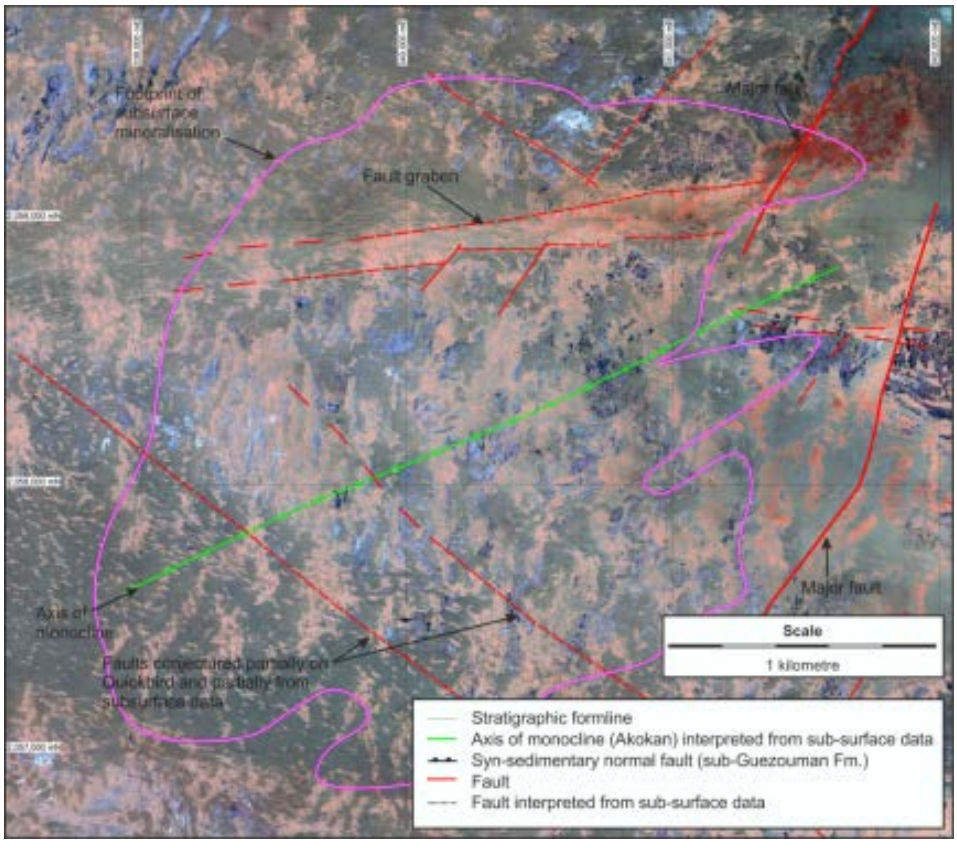


Figure 6-14: Plan view of stratigraphic formlines and structures interpreted at MSNE relative to the Quickbird imagery.

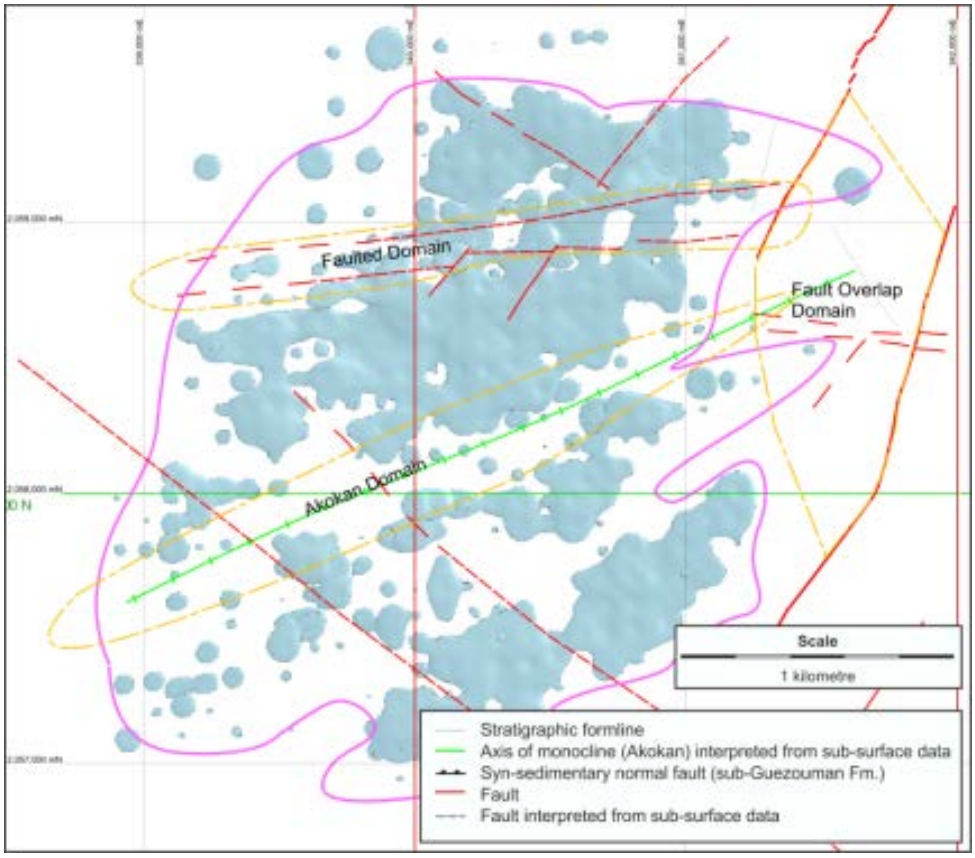


Figure 6-15: Plan view of structural domains relative to the Leapfrog 0.25 kg/t eU grade shell for MSNE.

Stratigraphic Dip

The stratigraphic dips in the MSNE deposit area are relatively subdued away from significant structures. Dips calculated on the basis of horizon intercepts are commonly 0-3 ° and rarely exceed 5 °. Dip increases up to approximately 10 ° locally around fault MSNE-NW-FT-1. Undulations in the footwall away from the monocline and other structures are not anticipated.

6.3.3 Maryvonne Geotechnical Design

Maryvonne deposit will be mined by conventional room and pillar with access from the south via MSNE.

Geology

Geology across Maryvonne is interpreted to be similar to the northern part of MSNE. The hangingwall is expected to consist of Guezouman Sandstone and the footwall will be within Talak shales. UA formation was identified in a small portion of boreholes trending NE-SW. This may represent a small UA channel with a vertical thickness of 10-30 m. Stratigraphy is generally sub-horizontal (dip less than 6 °) across the deposit.

Structural Geology

At surface, Maryvonne is located in a relatively sandy area with interrupted exposure of the Izegouande Formation, similar to MSNE.

Structurally the deposit is located 6 km west of the regional NNE-SSW striking Madaouela Fault. Unlike Marianne-Marilyn, to the north, domes do not affect the Maryvonne deposit area. However, the precise nature of the structural geology of the deposit is not particularly well understood due to poor surface exposure and widely spaced exploration drillholes.

The principal structures that have been outlined at Maryvonne are based solely on the topography of stratigraphic horizons in 3D. Towards the south of the deposit, a broad linear depression in the top Talak horizon with several drilled intervals of Akokan Unit is interpreted to represent an ENE-trending monocline, similar to those at Marianne-Marilyn.

Two loosely constrained structures have been interpreted which comprise an oppose-dipping set of moderately SSE and NNW dipping faults. The nature of these faults are unknown, but are conjectured to have similar characteristics to the Akokan faults at Marianne-Marilyn.

A second pair of faults has been interpreted in the central part of Maryvonne, based similarly on a broad linear zone of depression, but without any intervals of the Akokan Unit. These structures are very tentative and may only accommodate 5-10 m of displacement. They are therefore likely to be relatively minor faults in nature, but with similar characteristics to the Akokan-bounding faults at Marianne-Marilyn.

6.3.4 Miriam Geotechnical Design

Mineralization at Miriam is shallower, lower grade and thicker than the mineralization at Marianne-Marilyn. The mineralisation is generally near surface (upper 110 m) and will be mined by open pit.

Geology

Geology within the Miriam deposit consists of three main sub-horizontal sedimentary units. From shallowest to deepest, the main units are Tchinezogue, Guezouman and Talak and can be described as follows:

Tchinezogue is a fluvio-deltaic sedimentary unit. Bedding is generally thin and sub-horizontal. In upper 30-40 m this unit is weathered and oxidised, as a result of exposure caused by retreat of the groundwater table (Figure 6-16).

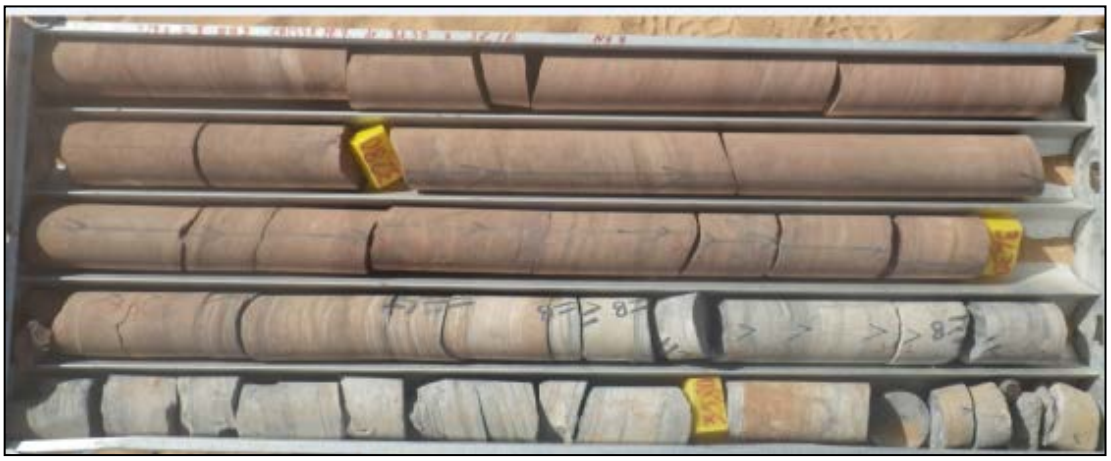


Figure 6-16: Oxidation and weakening of the Tchinezogue unit due to exposure above the water table. Below the water table (approximately 35 m depth) the rock is generally unweathered.

Guezouman formation (Figure 6-17) is a coarse grained sandstone mainly composed of quartz with K-feldspar, plagioclase, and clay minerals. It is relatively massive with low fracture frequency, although some “disking” is occasionally observed due to bedding planes being opening by drilling. It is equigranular and has a low density due to the relatively large percentage of pores between grains. Some fine-grained sub-formations exist within the Guezouman Sandstone.

Figure 6-17: Guezouman sandstone showing a zone of relatively closely spaced bedding planes

Talak is a grey-black fine grained shale/argillite unit (Figure 6-18) and forms the base of the stratigraphic sequence at Miriam. The majority of mineralisation sits within the Guezouman Sandstone immediately above the Talak argillite. The Talak will form the floor of Miriam pit.

Figure 6-18: Example of the Talak argillite at Miriam (this will form the floor of the pit)

Structural Setting

Miriam is located within the southern extent of the Madaouela license area. Structurally the deposit is located directly to the west of the regional NNE-SSW striking Madaouela fault, which accommodates approximately 300 m of vertical down-to-the-east displacement. The precise nature of the structural geology of the deposit is not particularly well understood as surface exposure in the area of the Miriam planned open pit is very poor.

Major Structures

Faults

Unlike the other deposits considered herein, Miriam is not associated with any notable domes or monoclinal structures. Only two faults have been explicitly modelled in the area of the Miriam deposit. These structures have been interpreted primarily on the basis of linear traces on the Quickbird image.

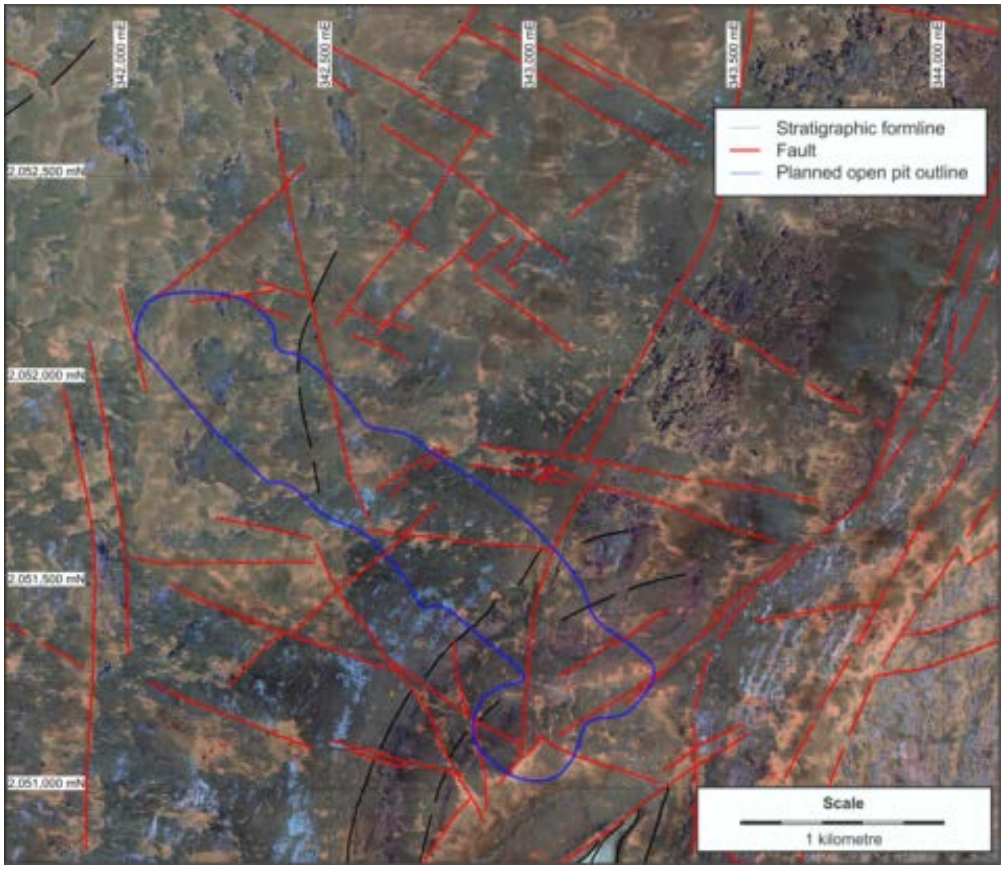


Figure 6-19: Plan view of stratigraphic formlines and structures interpreted at Miriam relative to the Quickbird imagery

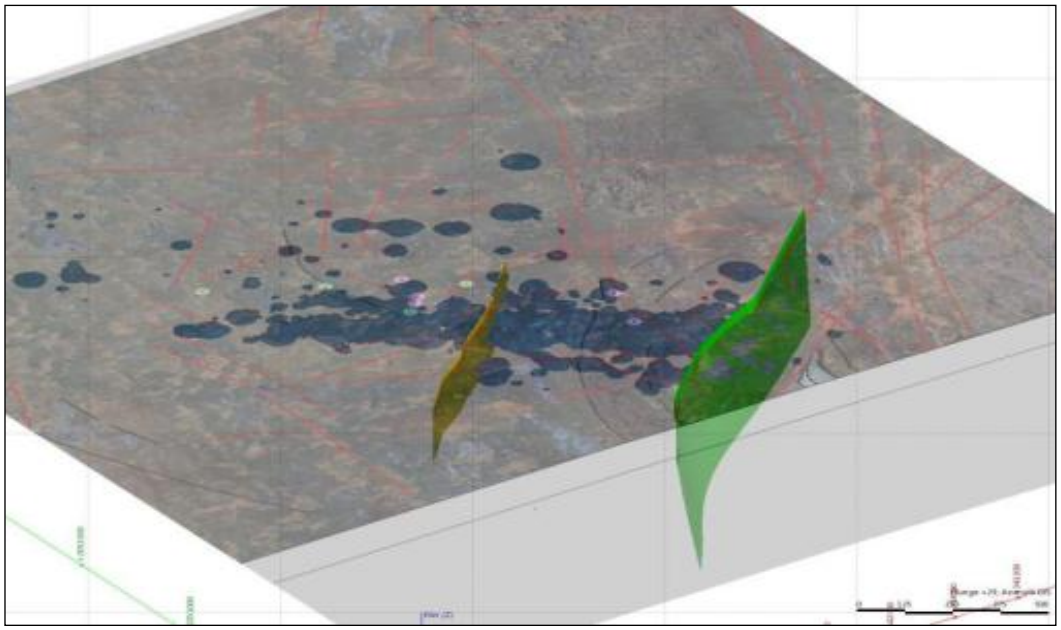


Figure 6-20: Miriam faults shown relative to topography with Quickbird drape and Leapfrog 0.4 kg/t eU grade shell. (green fault - MI-NE-Ft-1; yellow fault - MI-NE-Ft-2)

Stratigraphic Dip

Stratigraphic dip based on drillhole interval elevations are low, predominantly 0-4 ° over the main pit area. Towards the south of the planned open pit, there is a minor decrease in the average dip and dip variability.

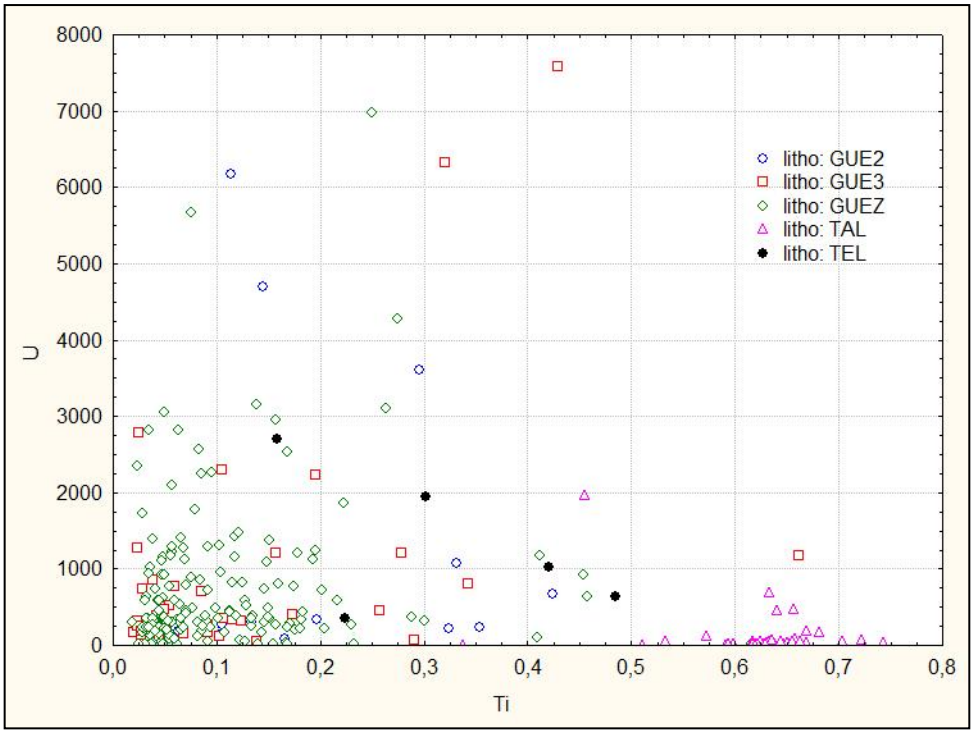
6.4 Mineralization

6.4.1 Geochemistry and Mineralogy of the Madaouela Uranium Project

The Madaouela deposits exhibit classic characteristics of uranium sandstone deposits common the world over (Cuney and Kyser, 2009).

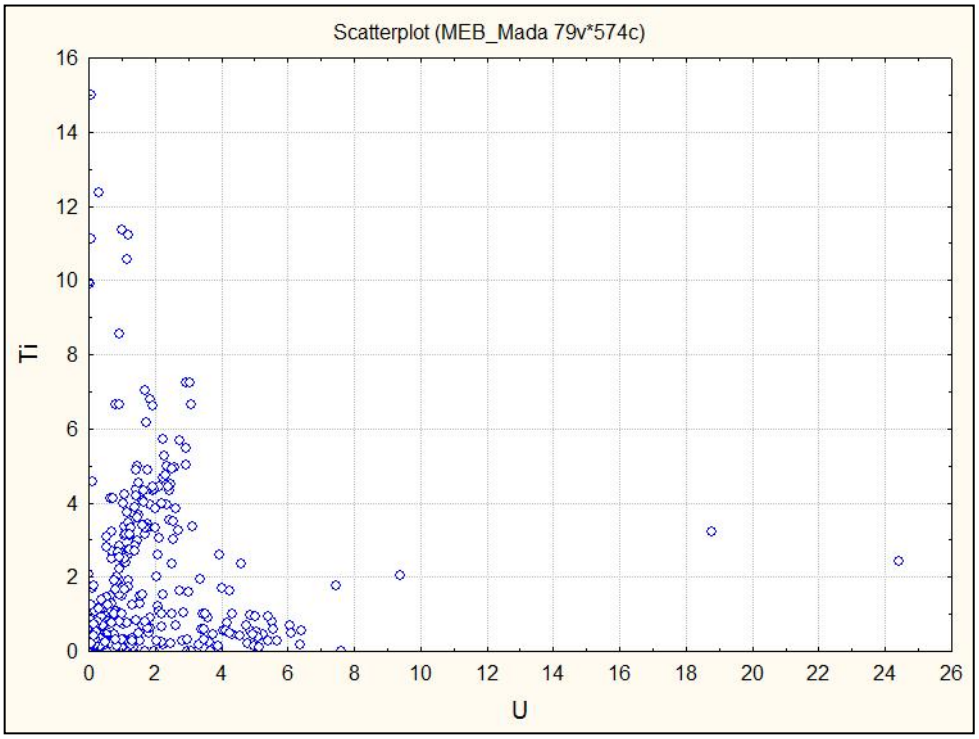
The mineralogy of uranium in the deposit is dominated by pitchblende and coffinite. The overall paragenesis could be divided in three stages: (1) early sulfides; (2) uranium on organic matter such as wood fragments; (3) carbonates and barite. The uranium minerals largely occur on the surface of minerals, or as infillings between the grains. In SRK's opinion, some brannerite may also occur in the deposit. Mineralogists never identified brannerite, and in the Ti-U diagram of SEM analysis, high values of uranium are independent of titanium whereas low values of uranium are associated with Ti minerals. The major carriers of uranium are therefore uranium oxide and not brannerite.

Figure 6-21 shows the X-Y plot of uranium and titanium for several lithologies, and Figure 6-22 shows the U-Ti relationship in SEM scans. This relationship has been described in other deposits, with uranium "adsorbed" on corrugations at the surface of Ti oxides



Source; GoviEx, 2009

Figure 6-21: X-Y plot of uranium versus titanium



Note: Analysis of grains from the Madaouela deposit by SEM: U and Ti show two types of relations; high U phase have low Ti content, and high Ti shows low U.

Source: GoviEx, 2009

Figure 6-22: U-Ti relations in SEM scans

In the nearby Akouta uranium mine, the paragenesis is composed of three associations: (1) Pre-ore minerals: wood fragments replaced by pyrite, quartz overgrowths, barite; (2) Ore-stage minerals: Uraninite, coffinite, with calcite and sphalerite, radiogenic galena, jordisite, marcasite; and (3) Post-ore minerals, with kaolinite as feldspar replacements. The Madaouela deposit seems to follow the same pattern.

It is important to note that pyrite may have developed on large areas, but is now preserved only in the halo of large redox front. Molybdenum as a trace element is associated with pyrite as a trace element.

6.4.2 Mineralized Zones (by SRK)

GoviEx drilling has established connections between the mineralization at Marianne and Marilyn, once separate deposits. The Marianne-Marilyn deposit is a nearly flat tabular body of mineralization that spans approximately 5 km (N70E direction) by 2 km across in plan, and the deposit thickness varies from 0.2 to over 2 m (average thickness of about 1 m).

The general dimensions of the extent of mineralization at Marianne-Marilyn are described above. Drilling in the southern part of Madaouela I has thus far defined three individual pods of mineralization in the same stratigraphic setting and with similar grade, depth (100 m to 140 m), and thickness of mineralization to that at Marianne-Marilyn. The main body of the largest of the three pods of mineralization, Madaouela south northeast (“MSNE”), extends approximately 4 km east-west, by approximately 6 km north-south. MSNE is located approximately 5 km south from Marianne-Marilyn. MSCE and MSSE are located approximately 3 km and 6 km south of MSNE, respectively (Figure 6-23). The overall trend of mineralization on the Mad I tenement extends about 17 km south from Marianne-Marilyn to the furthest south historical drillhole of interest on this one redox boundary trend line; other redox boundary trends may exist.

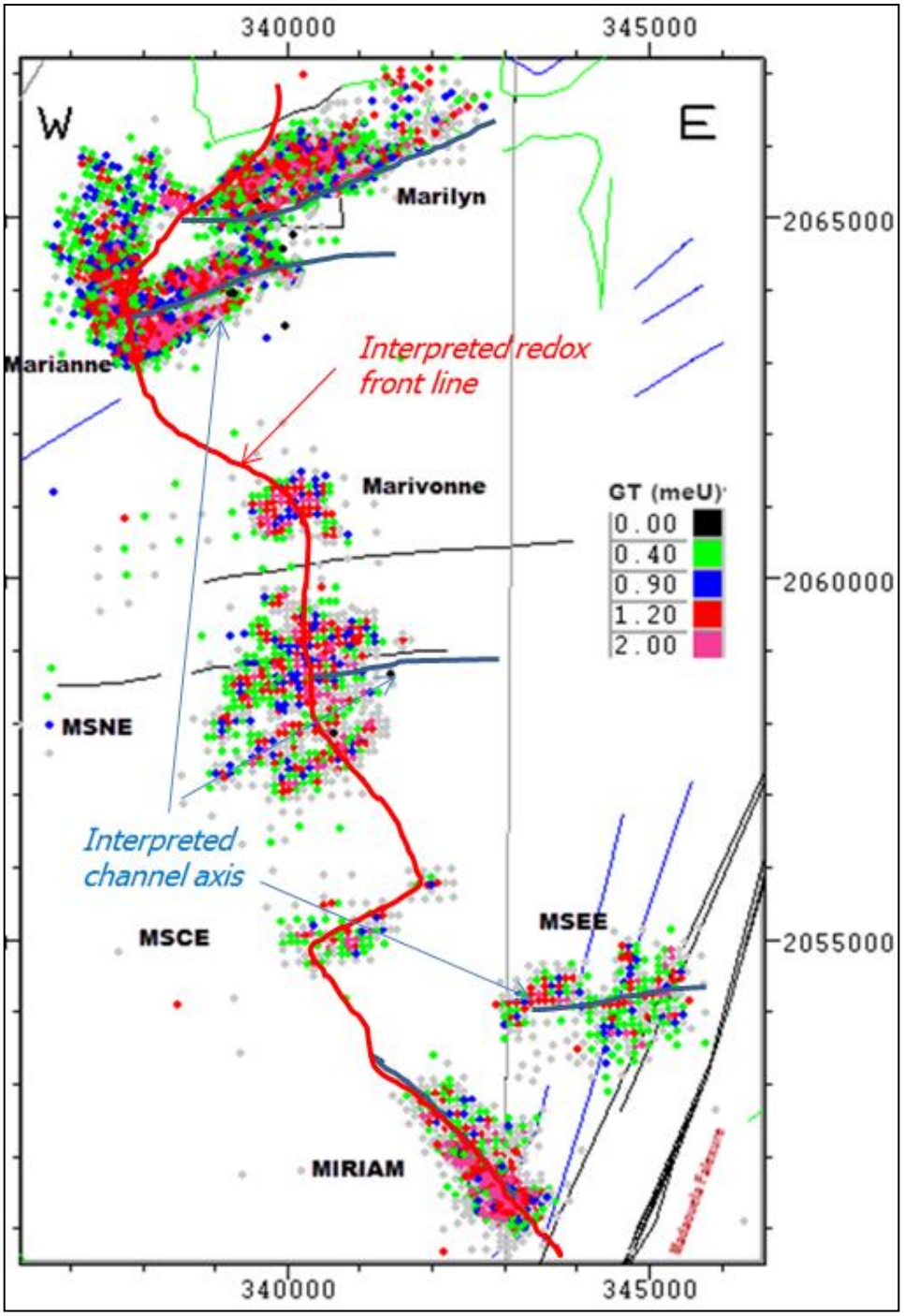
The mineralization occurs at depths from about 30 m on the eastern end of Marilyn, to approximately 60 m in depth in the middle of the Marianne-Marilyn deposit, up to 120 m in depths on the west extensions of Marianne; below the relatively flat topographic surface. The stratigraphy gently dips to the southwest, with local minor domal features. Mineralization at MSNE is similar in depth (about 100 to 140 m deep), grade, and thickness, although not yet drill-defined as large as the Marianne-Marilyn deposit is in plan.

The MSNE deposit occurs to the south of Marianne-Marilyn and is approximately 140 m deep. It occurs on the same redox front as Marianne-Marilyn (Figure 6-23) and the deposit is approximately 2.7 km long and 2.3 km.

Maryvonne deposit is located to the north of MSNE deposit and is approximately 100 m deep. The length of the deposit is 1.2 km and the width is 0.75 km

Miriam deposit is situated 8 km to the south of MSNE deposit and is dipping from 65 m to 120 m deep. The deposit is 2.4 km in length and 0.8 km wide.

In 2012, a new deposit was discovered to the east of Miriam, La Banane. This deposit is included in the resource determination for this project but not in the reserves due to minimal information being available at the time of writing. The deposit shows some mineralogical differences and occurs in the Madaouela formation above the Guezoman. It represents a potentially new exploration target distinct from mineralization in the Guezoman.



Source: GoviEx 2012 modified by SRK 2013

Figure 6-23: Relationship of MSSE to Marianne-Marilyn and redox front, Mad I

Surrounding Rock Types

The lithological unit below the Guezouman-hosted uranium mineralization is the Talak argillite/siltstone. The lithology immediately above the mineralized horizon is un-mineralized Guezouman sandstone. The argillite and silt of the Tchinezogue constitute a reduced capping which had probably played an important role in preserving the uranium mineralization in the

Guezouman

Geological Controls on Mineralization

The Guezouman sandstone at the Guezouman-Talak contact in the primary locus of mineralization, as controlled by the reducing environment and lesser permeability of the Talak argillites below mineralization, and the regional paleo-groundwater redox boundary in the Guezouman sandstone, down gradient from outcrops. Other relevant geological controls are the N70E structural, which represent older faults, and edges of paleo-channels. Low-amplitude domal features in the sedimentary units are related to the structural environment and are therefore relevant exploration guides.

Type, Character and Distribution of Mineralization

The uranium mineralization is all reduced uranium minerals (U (IV) minerals), uraninite and coffinite. The uranium minerals occur as disseminations in the matrix of the sandstone, with nearly all the mineralization occurring in one tabular horizon. At the favor of vertical “redox front”, uranium mineralization in the Guezouman may occur at several levels, as it is the case in the Miriam deposit. The Akouta “front” was the best example of this type of concentration. In the Miriam case a close relationship with structural features is very likely. Mineralization can sometimes be present at the contact of the Guezouman and the UA formation, in the Talak, and in the UA where the UA is preserved against a N70E fault; however, that mineralization is also relatively insignificant to the main basal Guezouman sandstone tabular lens of mineralization.

7 DEPOSIT TYPES (ITEM 8)

Sandstone-hosted uranium deposits are defined as epigenetic concentrations of uranium minerals occurring as impregnations and replacements primarily in fluvial, lacustrine, and deltaic sandstone formations. They occur in permeable medium- to coarse grained sandstone, usually deposited in continental fluvial or marginal marine sedimentary environments. Impermeable shale or mudstone are inter-bedded in the sedimentary sequence, and often occur above and below the mineralization.

The source of uranium is usually igneous or volcanic rocks (alkaline tuffs, granitic intrusion) either in close proximity to or inter-bedded with the sandstone units. The uranium mineralization typically precipitates from oxidizing fluids, under reducing conditions caused by a variety of reducing agents including: carbonaceous material (detrital plant debris and amorphous humate), sulfides accompanying organic matter decay, hydrocarbon, and inter-bedded mafic volcanic rock with abundant ferro-magnesian minerals. The reducing agent for Madaouela is most likely in-situ organic material (lignite) or hydrocarbons transported along major fault.

The main primary uranium minerals are uraninite and coffinite with minor secondary uranium minerals being noted in exposed (weathered) mineralization.

Sandstone deposits are an important source of uranium representing approximately 30 % of the world's known uranium resources and accounting for over 18 % of the African uranium deposits in 2007. This style of uranium deposit typically yields small to medium size deposits (10,000 to approximately 50,000 t of U_3O_8) characterized by low to medium grade (0.05 to 0.5 % U_3O_8). The deposits typically occur in clusters within a broad redox front.

Major sandstone-hosted uranium deposit provinces worldwide include the Powder River Basin in Wyoming, the Colorado Plateau and the Gulf Coast Plain in south Texas in the United States, the Tim Mersoi Basin of Niger, the Franceville Basin of Gabon, Cretaceous basins in Kazakhstan and Uzbekistan, the Frome Embayment of South Australia, and the Karoo Basin of RSA, and Tanzania. The sedimentary basins occur mainly in rock from Carboniferous to Tertiary age, since the development of the continental vegetation, and therefore the formation and preservation fossil carbonaceous material that establish a reducing environment in continental sandstone.

Four main types of sandstone deposits have been recognized world-wide: (1) Basal-type deposits in paleo-valleys incised in basement rock; (2) tabular deposits; (3) roll front deposits and (4) structural deposits, within sandstone adjacent to a permeable fault zone.

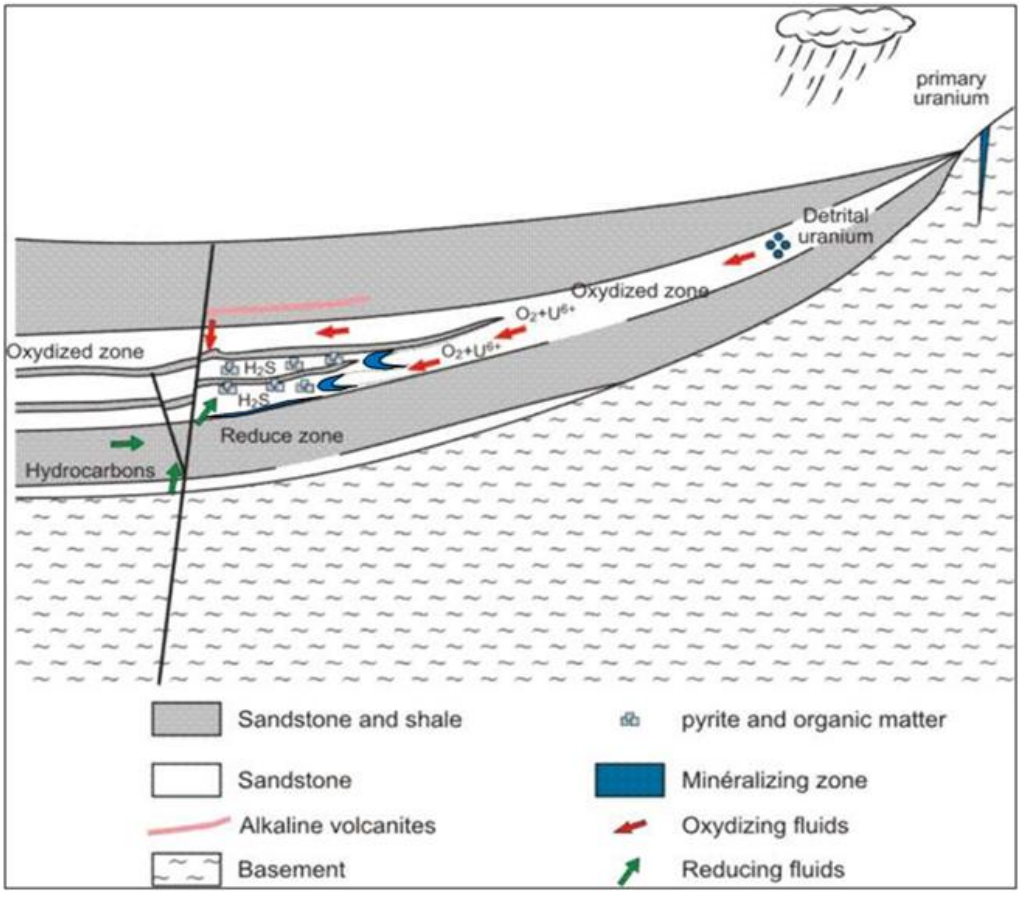
In Niger, including the Madaouela Uranium Project, the uranium deposits belong to the tabular and roll front deposit types. Deposits are epigenetic (Pagel et al, 2008). Mining operations at Arlit (SOMAÏR) and Akouta (COMINAK) ranked amongst the ten largest uranium producers delivering in 2005 approximately 7.5 % of the world supply, and making Niger the fourth largest uranium producing country in the world. Areva and the Niger Government have announced in 2015 their decision to delay the development of the Imouraren deposit, located south of Madaouela. (http://www.areva.com/EN/news-8192).

Table 7-1: Reported production, reserves and resources for Areva mines in Niger

	Production	Total Reserves		Measured and Indicated Resources		Inferred Resources	
Mine	TU	TU	Kg/t U	TU	Kg/t U	TU	Kg/t U
Somair	2,331	7,771	1.46	28,919	1.41	21,925	1.68
Cominak	1,501	12,100	3.62				
Imouraren		213,772	0.70	62,584	0.58	2,879	0.66

Source: Areva – 2014 Reference Document

Five mineralizing phases have been distinguished by Forbes (1989) in the Akouta deposit: (1) an early replacement of wood fragments by pyrite and formation of quartz overgrowths on detrital quartz grains; (2) a sulfate phase with formation of barite; (3) the U-Mo-V deposition phase; (4) the alteration of feldspars and formation of kaolinite on the west side; and, (5) the formation of iron oxides and manganese-rich cements. The uranium mineralization in the Carboniferous and Tarat deposits consists of pitchblende (enriched in ZrO_2) and coffinite, associated with V-bearing chlorite, corrensite, jordisite, montroseite, pyrite, molybdenite, marcasite, sphalerite and dolomite.



Source: GoviEx, 2009; modified by SRK, 2010

Figure 7-1: Schematic cross-section of a fluvial basin and conceptual formation of uranium deposits

8 EXPLORATION (ITEM 9)

8.1 Introduction

The main exploration tool used by GoviEx on the Madaouela Uranium Project has been by drilling on a defined grid pattern, and interpreting the presence of redox fronts or anomalous uranium intercepts to justify further drilling. This is described in Chapter 9 Drilling. Other exploration work completed on the Project includes; field mapping at MAD I in 2009-10, strip mapping along drill lines was completed at MAD I, MAD II and MAD III in 2011; MAD IV in 2012; and Anou Melle in 2014. Remote sensing by MIR Teledetection was completed over the whole project in 2009 and has greatly assisted in understanding the structural complexity of the area.

8.1.1 2009

MIR Teledetection were contracted to carry out remote sensing analysis of the Madaouela licences. This included obtaining quality satellite imagery SRTM for topography, Landsat and Aster imagery for spectral analysis and photo interpretations.

8.2 Principle Component and Spectral Analysis

MIR Teledetection used the Aster and the Landsat data to carry out the spectral analysis.

- ASTER (Advanced Spaceborne Thermal Emission and Reflection Radiometer) provides high-resolution images of the planet Earth in 14 different bands of the electromagnetic spectrum, ranging from visible to thermal infrared light.
- Landsat a joint initiative between the U.S. Geological Survey (USGS) and NASA represents the world's longest continuously acquired collection of space-based moderate-resolution land remote sensing data. Landsat collects data over 7 channels in the visible, near infrared, and shortwave infrared wavelength regions as well as a panchromatic band.

The various channels were enhanced to increase the contrasts, followed by the application of filters to accentuate the peaks and spectral decorrelation was applied to each input channels. A number of composite maps were generated using the combination of visible, near-infrared and mid-infrared. The focus was on Landsat channels 4,3 and 2, which visually highlights the contrasts in minerals contents. Figure 8-1 below shows the contrasts between the various lithological packages.

By analysing the spectral component of each pixel it is possible to identify mineral signature information for a given terrain. MIR identified 11 classes of information, which they linked with ground information. Hence a thematic map could be created which matches with the regional geological map (Figure 8-2).

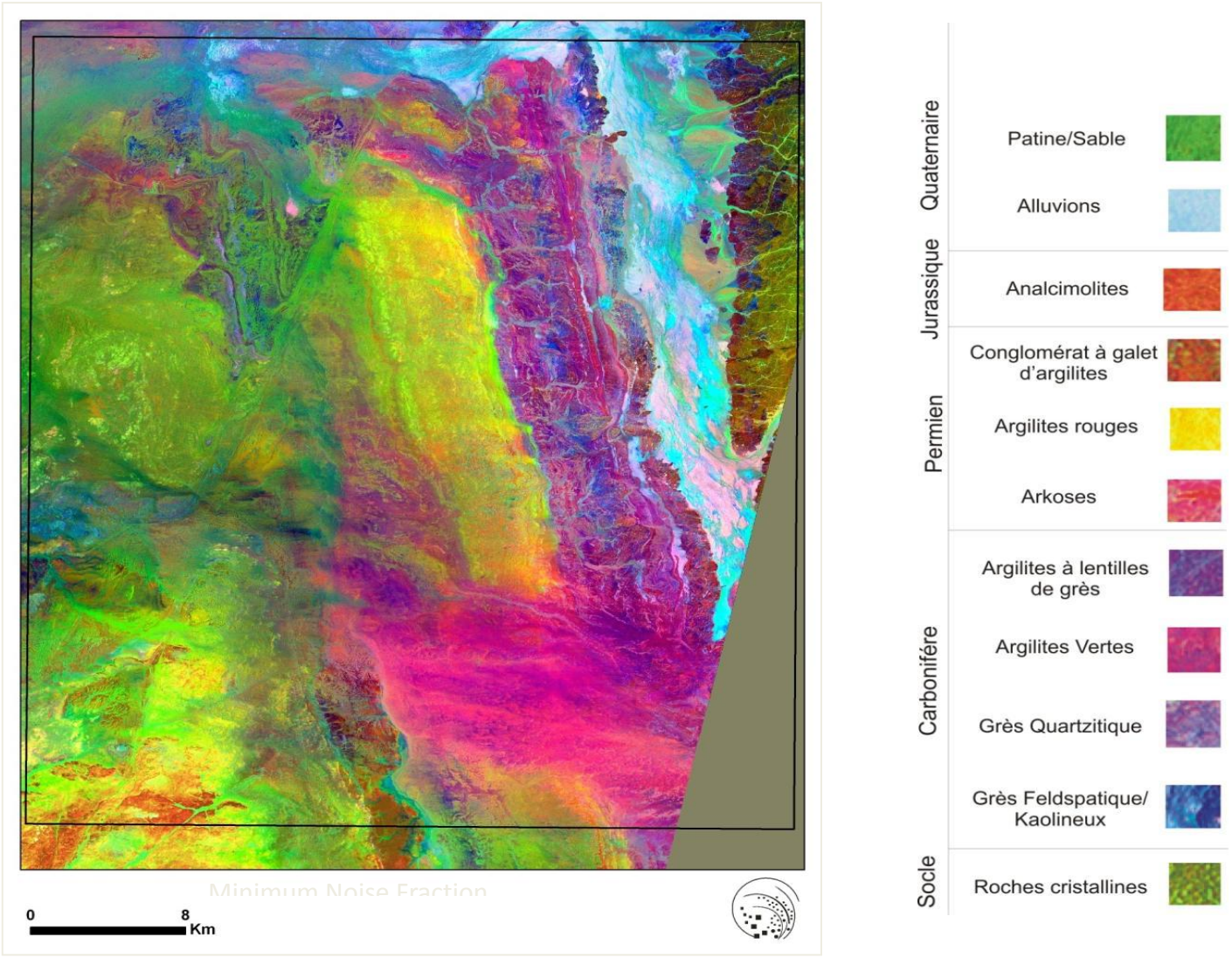


Figure 8-1: ASTER RGB (Red-Green-Blue) representation of principle components of channels 4,3 and 2 respectively. This choice of channels highlights the lithological packages and matches with the regional geological map.

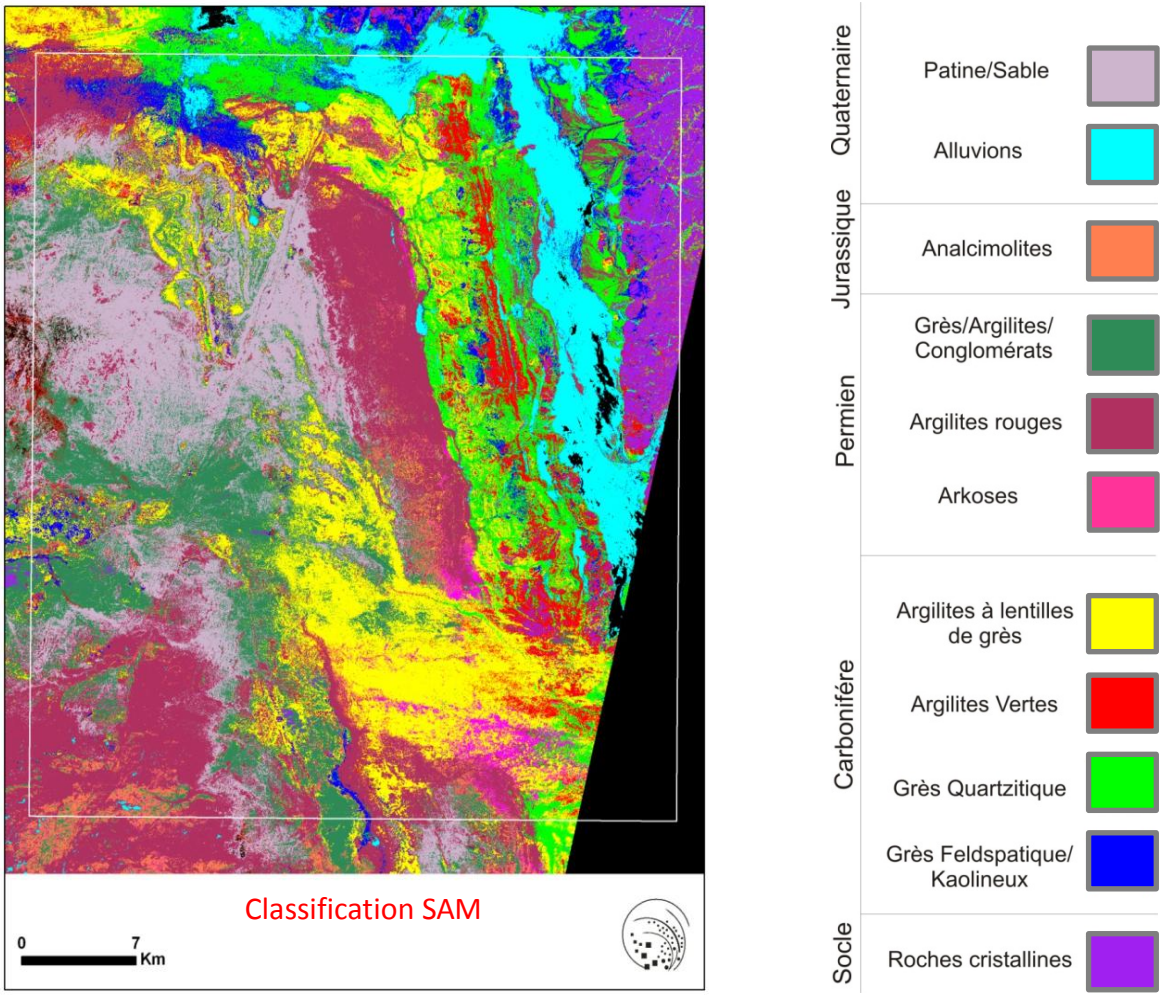


Figure 8-2: Classification of pixels based on 11 mineral signatures determined by analyzing the Aster data set. It provides a better distinction between the various rock units, and also surface alteration of the rocks.

8.3 Structural Analysis

Structural analysis was carried out by analyzing the ASTER as well as the SRTM data.

- The **Shuttle Radar Topography Mission** (**SRTM**) is an international research effort that obtained digital elevation models on a near-global scale to generate the most complete high-resolution digital topographic database of Earth prior to the release of the ASTER GDEM in 2009.

The lineaments such as faults and fractures were identified, as well as bedding structures, Figure 8-3 and Figure 8-4. The movement along the faults was also identified based on the interpreted rock relation, Figure 8-5A, and the identification of dome structures, which are believed to play an important role in the mineralisation of uranium, with a suggested relationship of faults and domes - Figure 8-5B.

8.4 Hydrography

Automated identification of hydrographical features (drainage) was carried out using the SRTM data, based on the theory that current drainage patterns may reflect historical patterns, Figure 8-5C.

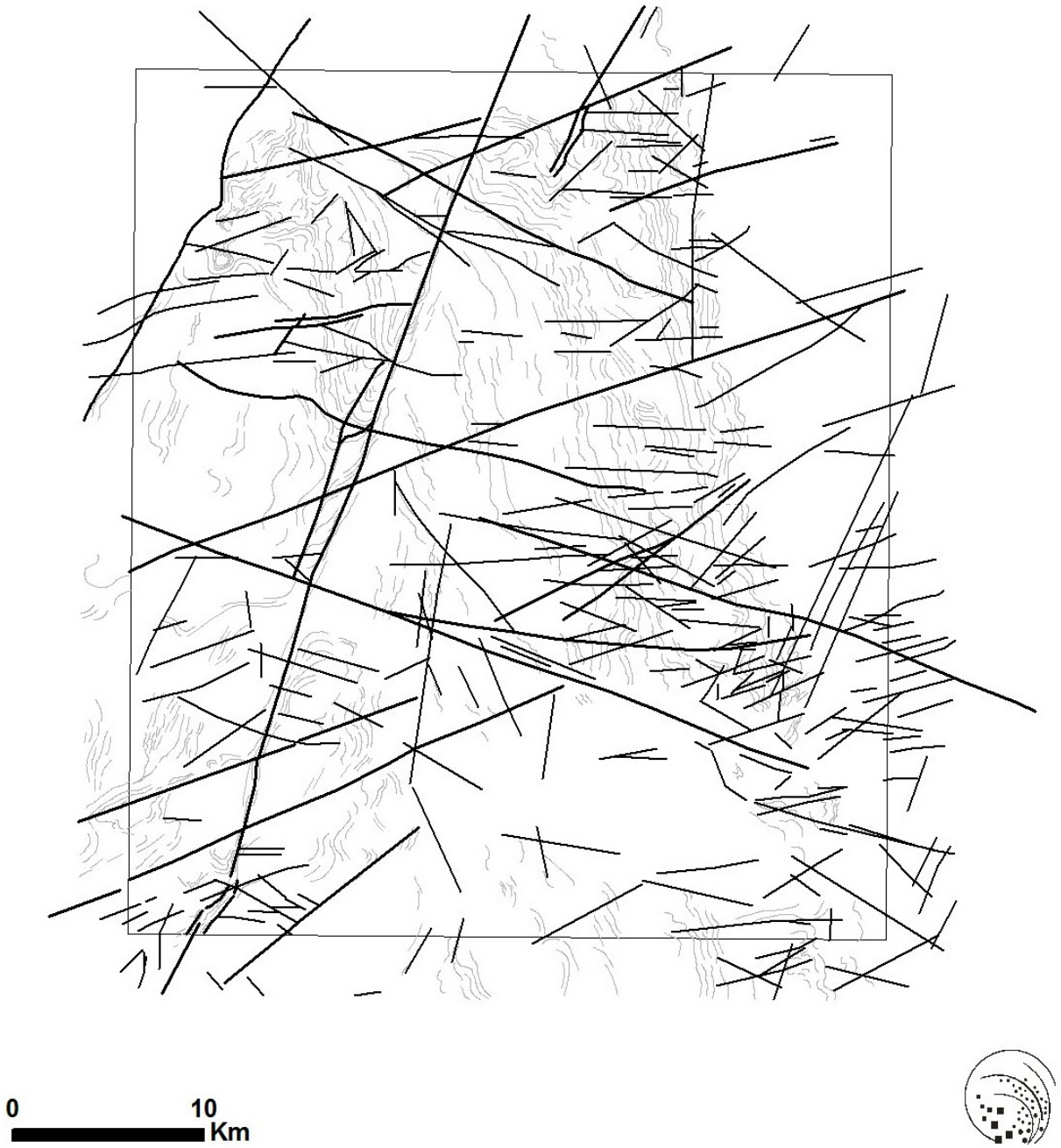


Figure 8-3: Structural analysis - major fractures and bedding patterns were interpreted from the Aster and SRTM data sets at various scales.

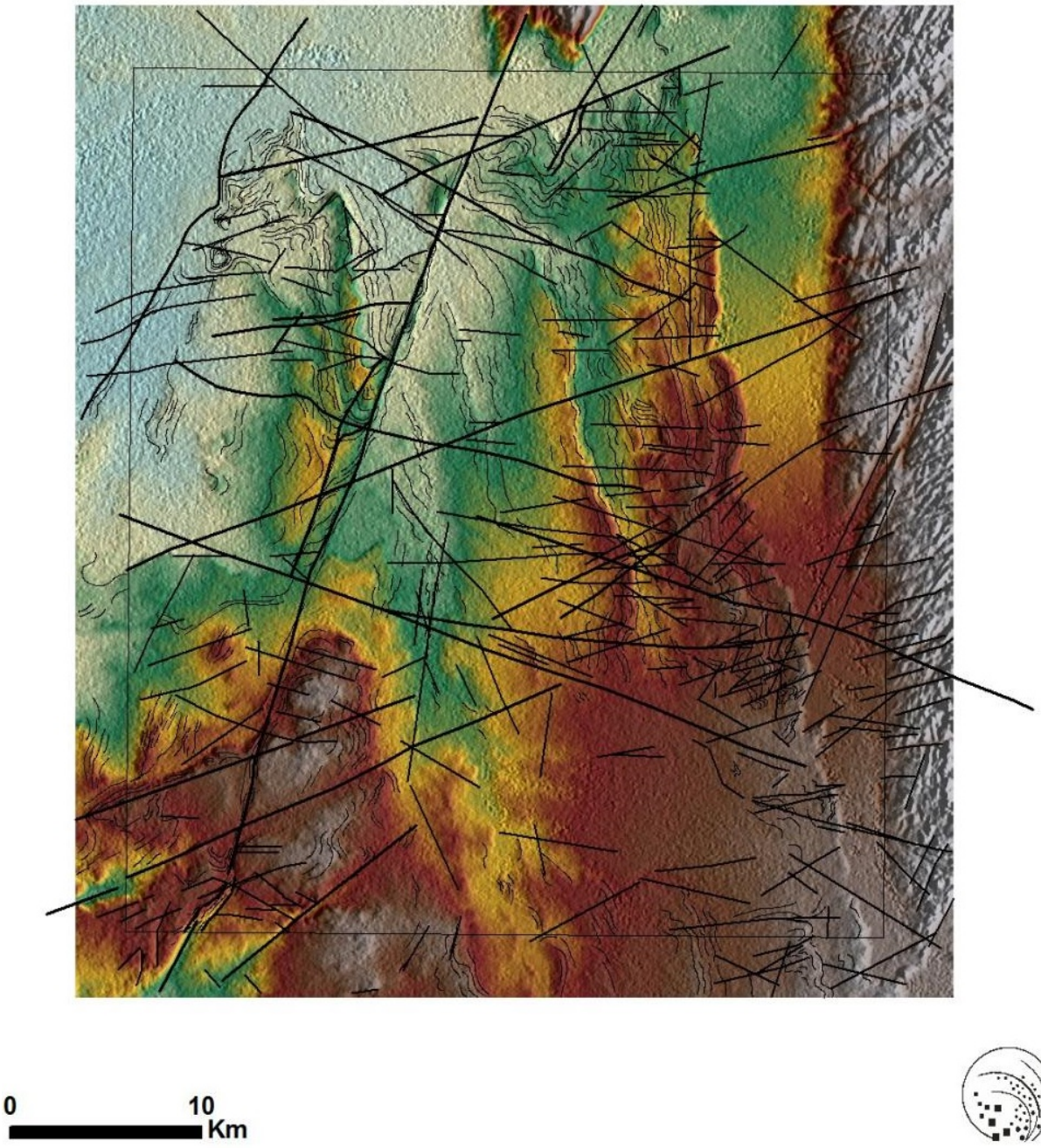


Figure 8-4: Structural analysis over SRTM Elevation image, was used to interpret major faults and fractures as well as bedding patterns.

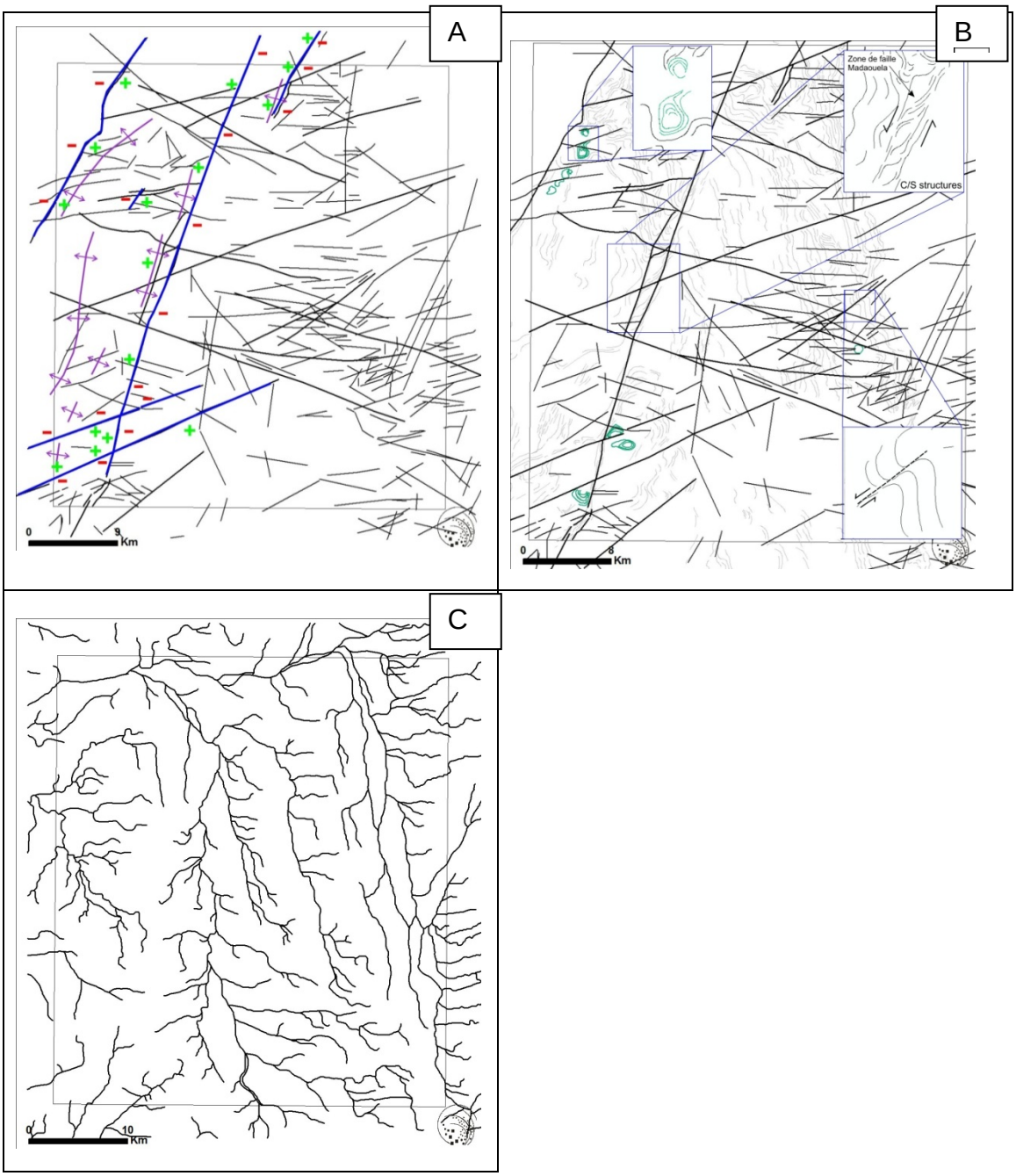


Figure 8-5: Interpretations of domes and also fault movements were interpreted from the structural interpretation. Hydrographical interpretation was also included from SRTM dataset

8.4.1 2009 to 2010

Mapping by GoviEx was carried out on the Marianne and Marilyn deposits. The mapping used preliminary photo interpretation, followed by field validation as access was restricted due to security concerns, associated with the military camp at the time.

From field observations a number proposals were made for the mineralisation event and confirmed the structural complexity proposed by MIR Teledetection.

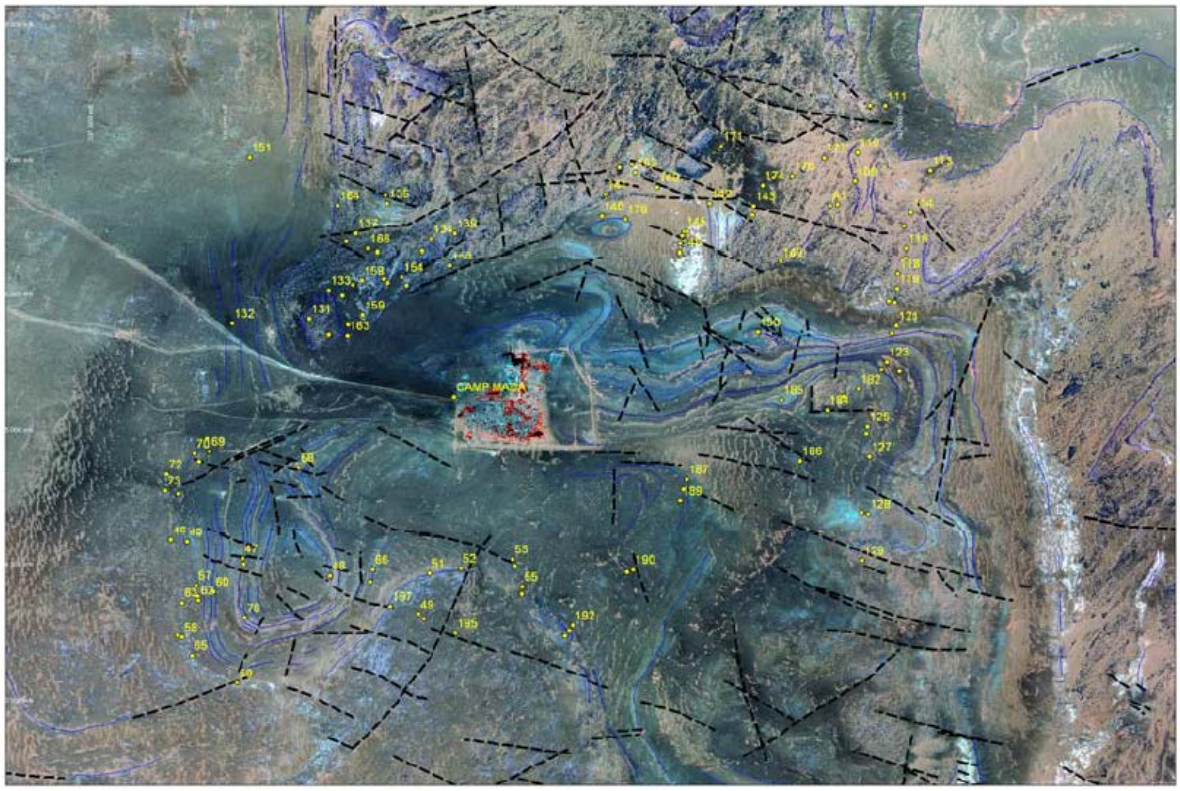


Figure 8-6: Quickbird image of the zone of study, with points of observation in yellow.

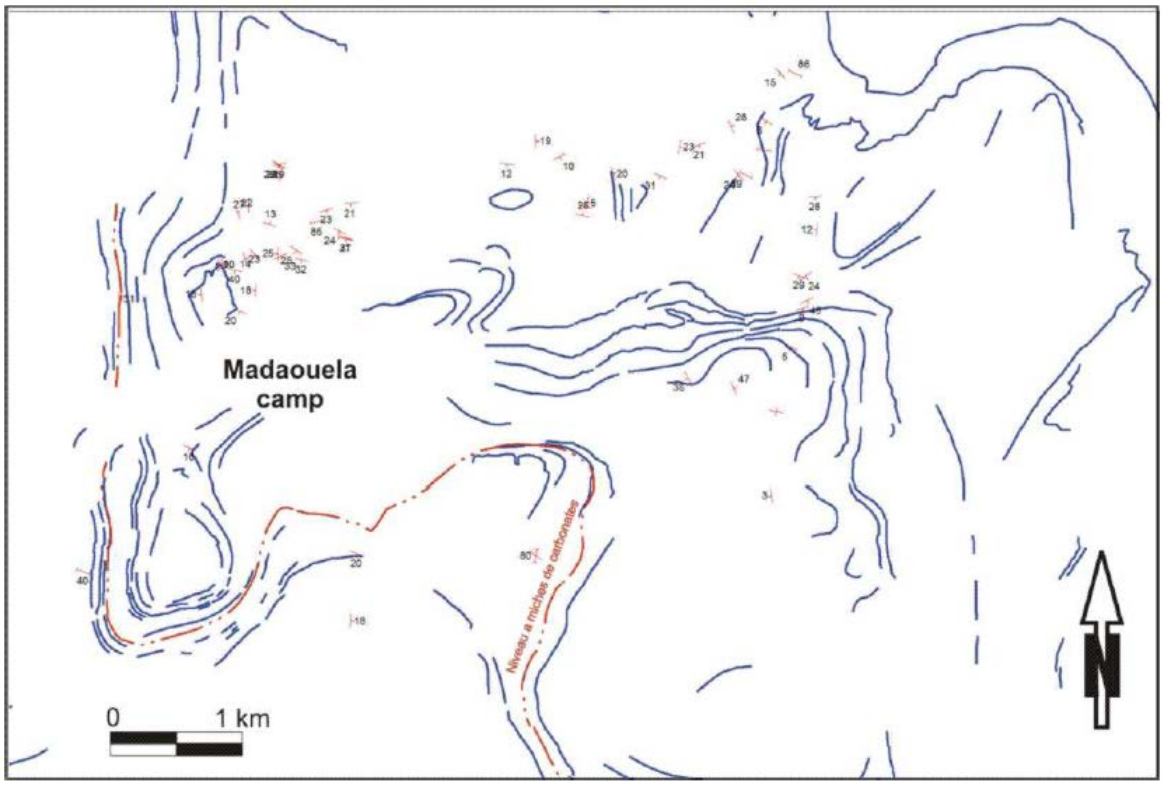


Figure 8-7: Stratification interpreted from Quickbird image and dip and strike from measurement in the field.

The presence of pyrite, organic matter, generally reduced sandstones within deeper “graben like structure bounded to one side by N80 fault created the ideal environment for uranium deposition. The observation of features such as evidence of fluid flows along fractures (Figure 8-8), and presence of pyrite and organic matter in outcrop led to those conclusions.

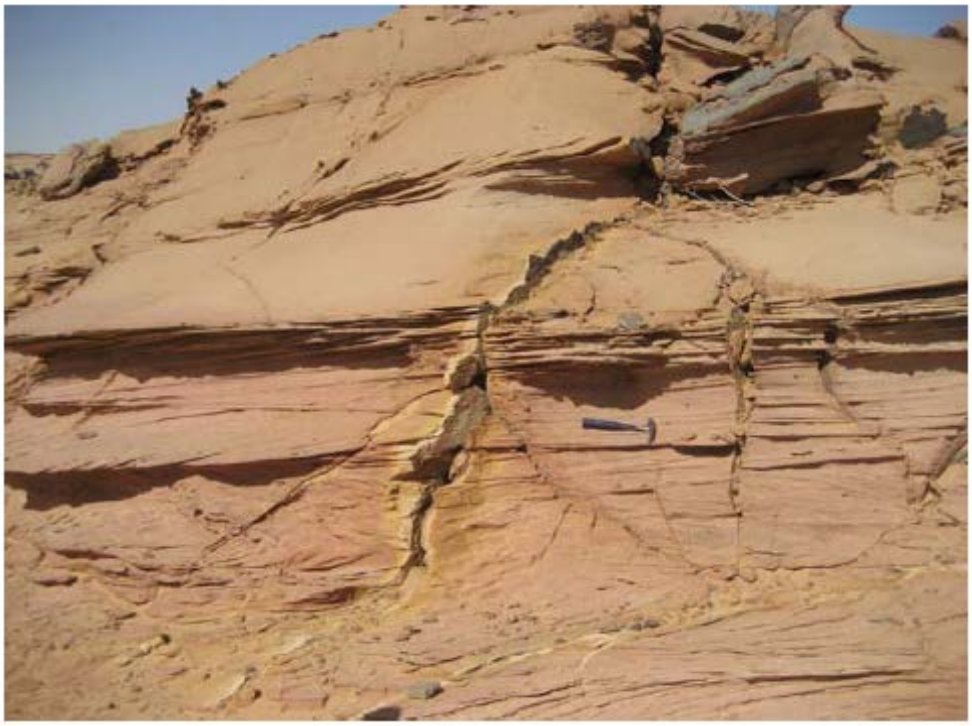


Figure 8-8: Large N80 joint showing evidence of strong hydrothermal alteration

8.4.2 2011-2012

Strip mapping along regular spaced lines was completed by GoviEx over MAD I, MAD II, MAD III and MAD IV. The main purpose of the exercise was to validate the regional geological map data, followed by measuring the direction of paleo-flow to determine channel development and help in defining drill sites. The reading of radioactivity using the SPP2 spectrometer help identified potential target horizons. The geologists primary task was the recording the lithology, stratigraphy, bedding orientation, presence of faults and fractures. Further to this once drilling was completed, it helped in the interpretation of drill sections.

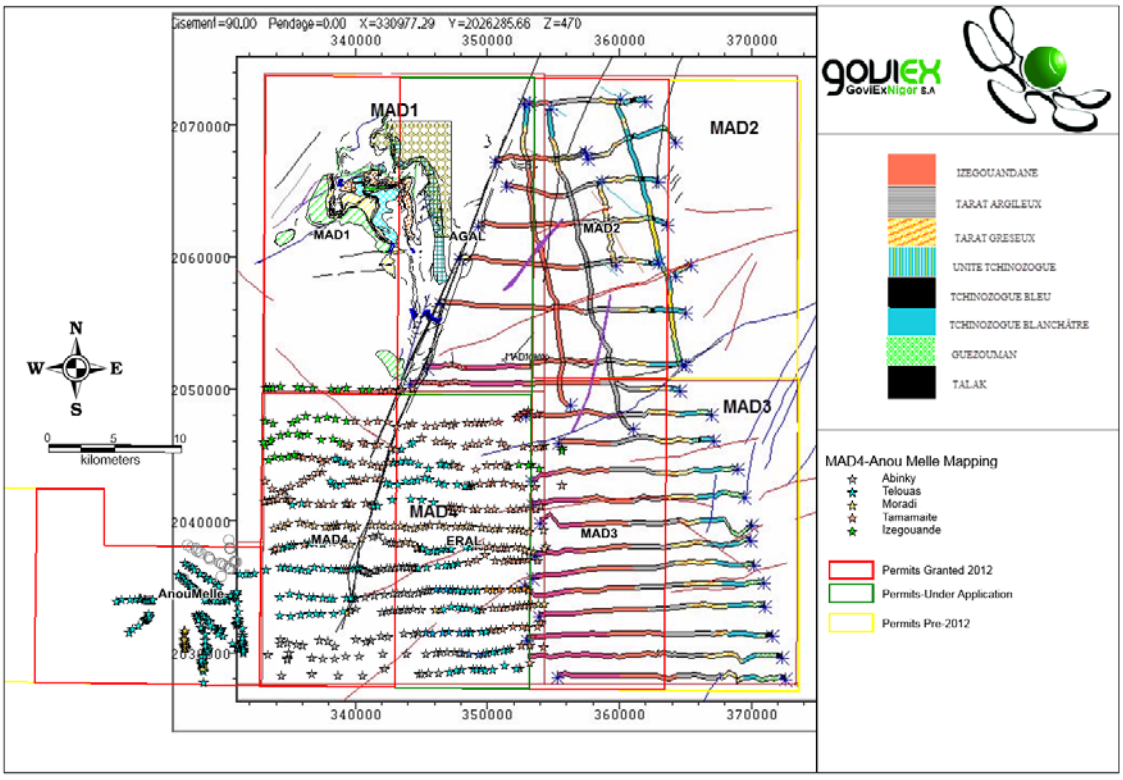


Figure 8-9: Strip Mapping over Madaoulea Uranium Project.

8.4.3 2014

In 2014, GoviEx completed field verifications on the Anou Melle licence. The main aim of the mapping programme was to confirm; the presence of faults interpreted by MIR Teledetection in 2009; previous work by CEA in the 60's; to confirm the stratigraphy and to verify a number of surface radiometric anomalies. This area is along strike from the Areva deposits (Akouta and Arlit to the north and Imouraren in the South). There was some initial drilling done in the area by the CEA, and also by GoviEx in 2010 that showed some anomalies in the target formations.

There were 10 sections identified which maximized the ground covered, focusing on the eastern part of the licence where there was a maximum of complexity, surface radiometrics and also downhole intercepts. (Figure 8-10)

There were an abundance of fractures on all 10 sections, most showing evidence of fluid flow, such as bleaching, siliceous or iron fill, and brecciation (Figure 8-11), as the area lies next to a major continental fracture like the Arlit Fault. The main directions identified were:

- NS parallel to the Arlit fault
- N70- such as the Capucines and Tagait Faults
- N40 parallel to the Izeretagen and Madaouela Faults
- N120 to 140, which is not as well recognized in the literature but may be significant.

The bedding was generally flat lying to very shallowly dipping to the west, except near some of the faults where steepening of the dips was evident.

The surface radiometric anomalies occur near faults and at the top of the Moradi and base of the Teloua formation within paleo channels of conglomerates and sandstone.

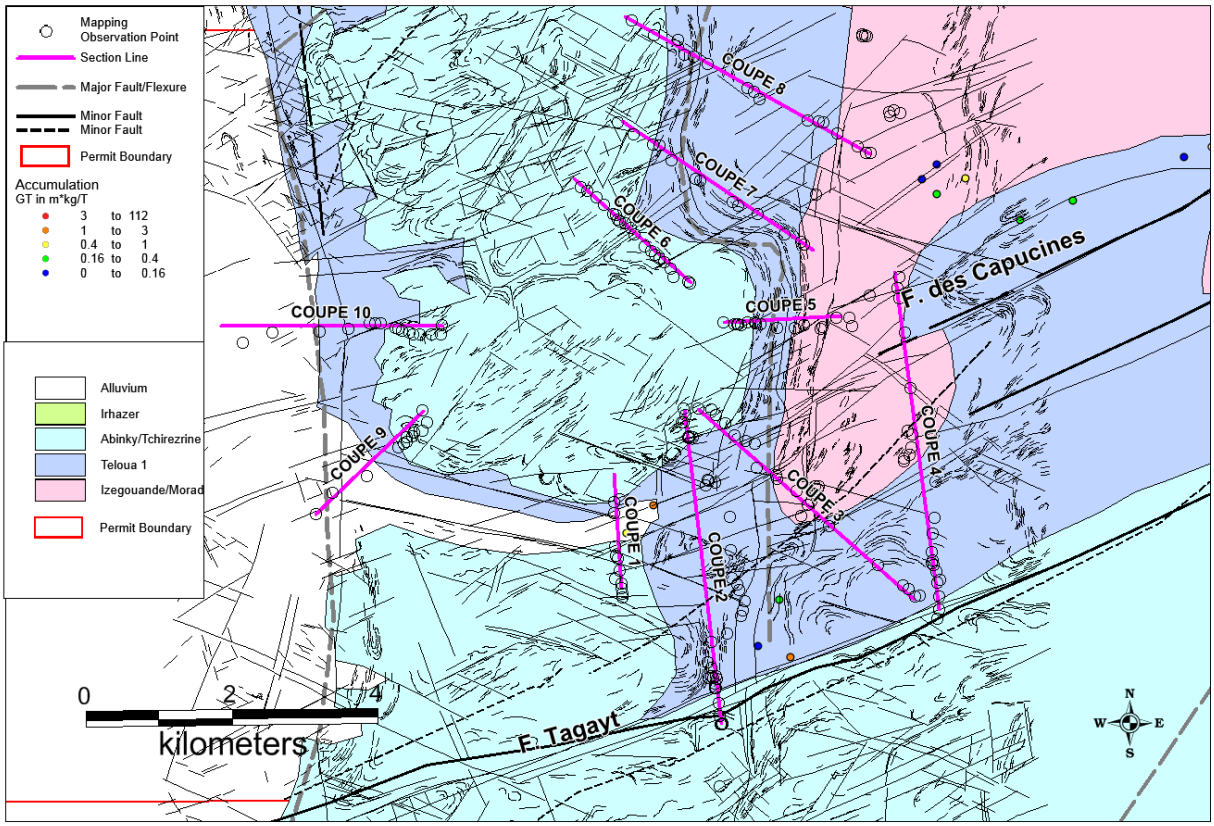


Figure 8-10: **Location map of the various sections mapped (pink lines), over the local geology map.**

Figure 8-11: **At point C01-P6, brecciated and silicifed rock, strikes N100 and dips steeply to the SSW. It seems to be the fault that has been interpreted from the remote sensing, which controls the edges of the creek. Evidence of fluid flow.**

The Trois Grace is a series of three distinct rocky mounds that line up along the central NS fault (The Trois Grace Fault along the middle of Coupe 3). It is heavily silicified, evidence of past fluid flows, but also potential to create physical barriers to fluid flow if the fault zone was sealed prior to major uranium remobilization event. The uranium would pond along these structures, hence making them a prime target. (Figure 8-12)

Figure 8-12: Three remnant outcrops of silicified material align NS along one of the interpreted faults. This is the feature known as Les Trois Graces (photo taken from the Northern side)

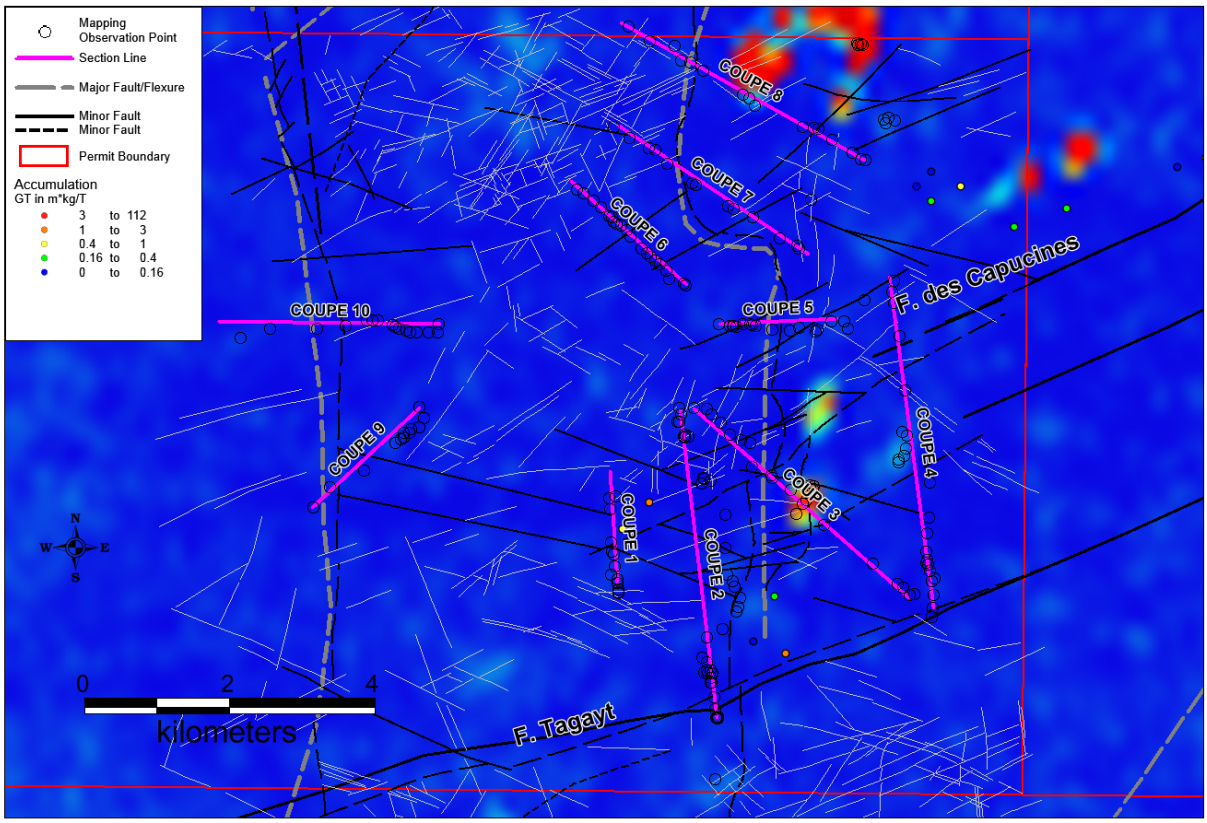


Figure 8-13: Mapping sections and locations over the U2/Th radiometric image, which shows the radiometric anomalies lying along faults.

8.5 Conclusions (SRK)

Extensive surface and sub-surface exploration has been conducted by GoviEx at Madouela using industry best practice for the style and extent of mineralisation which occurs here. The detailed and regular spaced drilling has allowed the deposits to be outlined with a high degree of confidence and, coupled with the field mapping, structural, hydrographic and remote sensing analysis, has enabled the identification of additional potential.

9 DRILLING (ITEM 10)

9.1 Introduction

Between 1963 and 1965, the CEA drilled the area extensively, leading to the discovery of the Madaouela deposits (Marilyn and Marianne). CEA conducted drilling operations using drilling grids of 800 m over large areas and then down to 100 m. The discovery of the Marilyn deposit was then drilled locally at 50 m and less spacing in the areas where a mining test was implemented with a view to sampling uranium mineralization and investigating the global rock quality underground.

From 2008 GoviEx started drilling and proceeded to drill Marianne and Marilyn to initially produce an Inferred resource, which led to further drilling which has finally led to the current Measured, Indicated and Inferred resources over a number of deposits.

Drilling was the main tool of exploration used by the CEA in the 1960's and GoviEx followed the same procedures by initially drilling on a large 800 to 1600m grid to obtain initial stratigraphic, redox and radiometric information. The use of historical results based on research of historical drilling data, from the Government archives mainly, proved to be partially successful even though only partial data could be found and some of the data content could not be verified. Historical results were extensively used to identify mineralized intersections, stratigraphic surfaces, and provided for orientation of the various drilling programs.

This led to re-drilling of old CEA holes either by reopening or fresh re-drilling and logging to confirm the results of the CEA, and furthermore the new results were considered the only reliable data collection that would be compliant with the new drilling data.

9.2 Organization

The drilling campaign has been monitored and supervised by GoviEx personnel. Geophysical logging has been conducted routinely with SEMM equipment operated by GoviEx personnel under the supervision of a geophysicist from SEMM. The SEMM geophysicist was in the first year responsible of the data acquisition and storage and conducted all QA/QC protocols, along with training of GoviEx staff. Geological survey, database entry and controls were implemented by GoviEx personnel. The raw data are stored in WellCad format. The working sequence (log circuit) is:

- Field Grid Implementation (pegging at various grid intervals dependent on purpose);
- Mud Pit on to be drilled sites;
- 30 minutes mud circulation upon completion of hole;
- Geophysical logging;
- Data control (WellCad, along Drillhole)
- Data transfer (WellCad to Utimine, and Coralis Data Base); 3D controls operated in UTIMINE prior to entry of new data in the GoviEx Coralis Database. This database has been tailored to our needs by CORALIS and is compatible with any commercial database;
- Geological survey of rock chips (1 m spacing of samples, lithology, stratigraphy, colour of grain coatings and matrix composition, definition of standard redox state);
- Reporting drill data into Well Cad, Utimine, Coralis database;
- Daily afternoon meeting;

- QA/QC control;
- Finalizing of logs, Log transformation and grade transfers;
- Updating accumulation tables; and
- Sectional controls in UTIMINE.

The GoviEx exploration program consisted of the following:

- A drilling contract was secured with the local drilling contractor ESAFOR. ESAFOR is based in Arlit and has been drilling in the area for over 20 years on all the main Areva deposits. They are highly specialized in drilling the local rock formations and have an excellent record;
- GoviEx recruited an exploration manager, Mr Tiemogo Mahaman, who has worked on the Tim Mersoi basin in exploration since the early CEA period and on all the development projects of Cogema/Areva (he was in charge of the training and supervision of the geological team at Imouraren for Areva);
- GoviEx recruited a qualified staff of geologists and prospectors having between four and ten years' experience in the geology and the exploration methods applied by Cogema/Areva, or its subsidiaries (Somaïr and Cominak). Some of the hired geologists had also worked with open pit and underground grade control. This unique panel of personal with local experience helped enter directly into an intensive drilling phase and to apply the tried-and-tested procedures and methods developed at the mines in the district. To prevent confusion and a steep learning curve, GoviEx has used similar software and geological criteria use in the district;
- GoviEx organized three (3) down-hole gamma logging units and personal to run them. SEMM Logging of France (with an office in Niamey) was contracted to supply and organize the installation of the logging vehicles and to supervise GoviEx logging technicians. The GoviEx logging technicians were recruited locally, each with over three years' experience, acquired from Areva projects in Niger. In addition, GoviEx purchased and mounted a logging unit identical to the systems used by Areva (Mount Sopris) with a DHT 27 gamma probe, which is the probe used locally by Areva and also as an international reference for in situ gamma measurements. This system was instrumental in determination of correlations between historical counts per second ("CPS") gamma measurements and the CPS recorded by the routine SEMM equipment. The DHT 27 unit is also used as a permanent QA/QC control of radiometric probe data for over 25 % of the holes drilled;
- GoviEx purchased a complete set of digital global positioning ("DGPS") surveying equipment (Rascal), and trained staff to use the equipment, in order to obtain rapid and accurate coordinates of drillholes; and
- GoviEx opened an office and base location in Arlit, then Akokan, which is fully equipped with accommodation and office equipment to manage the exploration program. The facilities today include a walled storage facility with a water well in Arlit and offices in a secure compound in Akokan equipped with two main workstations, a portable computer for each geologist, a central printing and scanning unit, a large format plotter, and some temporary housing, offices, and food service facilities. All project working documents are prepared and printed at the Akokan office.

Drilling was routinely performed using the rotary mud drilling technique, followed by geophysical logging (deviation, resistivity and radioactivity measured with scintillometer and Geiger Muller).

9.3 Work Programs

All exploration programs were executed and supervised by GoviEx staff, using local or external contractors including:

- ESAFOR, LEGENI and FORACO – Drilling;
- SEMM – Logging;
- ALS-Chemex – Assays of core for uranium and trace elements;
- MIR Technologies;
- ULC, with M Patrick Brunel, geophysicist, to validate the process of correlating SEMM gamma probes and the DHT27 probe; and
- Dr Amit Tripathi, independent consultant, for the training and QA/QC of Rascal DGPS surveying system.

9.4 Exploration and Development Drilling

9.4.1 MAD I (including Agaliouk)

Historically the Marilyn and Marianne deposits were drilled by CEA teams (1963-1965) using an average 100 m square grid. This spacing was considered in 1965 sufficient to define reasonably assured resources. A number of zones of interest were identified using a much larger grid (from 800 m to 400 m), with the exception of a small area in the southern part of Mad I. PNC drilling from 1980 to 1992, largely as in-fill definition drilling for the PNC feasibility level study of Marianne-Marilyn.

The GoviEx exploration program commenced in August 2008, following property acquisition in 2007. Between 2008 and 2010 almost all of the drilling undertaken was focused on the Mad I property, and was a combination of exploration and in-fill resource definition drilling. The main reasons for the focus on the Mad I property were;

- known resources at Marianne and Marilyn;
- in August 2008 GoviEx entered into a USD 28 M funding agreement with Cameco, which was conditional on a commitment to invest the majority of the funds in Mad I, and;
- during 2008 - 2009 security issues required GoviEx to operate close the the military camp on Mad I.

GoviEx's drilling program on the Madaoeula Uranium Project has been a combination of exploration and development and has been driven by a number of factors including commercial and security.

Table 9-1 summarises GoviEx's exploration drilling program for the period August 2008 to December 2013.

Table 9-1: Summary of GoviEx drilling program for the period August 2008 to December 2013

	Drill	2008		2009		2010		2011		2012		2013		TOTAL	
Deposits	**Type**	**Number**	**Total (m)**	**Number**	**Total (m)**	**Number**	**Total (m)**	**Number**	**Total (m)**	**Number**	**Total (m)**	**Number**	**Total (m)**	**Number**	**Total (m)**
MARTINE	Water well									0	0	0	0		
	RDH			0	0	27	1351	0	0	0	0	0	0	27	1351
	mixed RDH-DDH			0	0	0	0	0	0	0	0	0	0	0	0
	Reopening historical DH					4	201	0	0	0	0	0	0	4	201
MARIANNE	Water well	1	84							0	0	1	266	2	350
	RDH	560	57162	351	29740	97	11627	24	2824	1	266	499	44129	1532	145749
	mixed RDH-DDH	42	316	24	150	0	0	0	0	112	12270	13	569	191	13304
	Reopening historical DH	59	5086	42	3192	5	592	0	0	19	1704	0	0	125	10574
MARILYN	Water well									0	0	0	0	0	0
	RDH			643	37323	61	3140	59	2179	141	9238	114	6517	1018	58396
	mixed RDH-DDH			5	30	0	0	48	981	7	470	0	0	60	1480
	Reopening historical DH			134	9612	18	875	0	0	0	0	0	0	152	10487
MAD SOUTH	Water well			0	0	1	152			2	288	0	0	3	440
	RDH			165	23141	577	77458	49	6556	898	109632	202	19308	1891	236096
	mixed RDH-DDH			0	0	0	0	20	208	49	3374	38	2698	107	6280
	Reopening historical DH			24	2648	40	5157	40	4488	45	5049	0	0	149	17341
MAD 1 EXPLO	Water well			0	0	1	409			1	192	1	137	3	738
	RDH			0	0	8	3140	176	48077	95	22551	0	0	279	73768
	mixed RDH-DDH			0	0	0	0	0	0	3	796	0	0	3	796
	Reopening historical DH			0	0	2	767	0	0	0	0	0	0	2	767
MAD 1 CUMUL	Water well	1	84	0	0	2	561			4	745	2	403	9	1793
	RDH	560	57162	1159	90204	770	96717	308	59637	1246	153690	815	57186	4858	514596
	mixed RDH-DDH	42	316	29	179	0	0	68	1189	78	6344	51	3424	361	11452
	Reopening historical DH	59	5086	200	15452	69	7592	40	4488	45	5049	0	0	413	37667

Note:

- Mad I Cumul includes Madaouela I and Agaliouk

Since GoviEx began the majority of its drilling program it has completed on the Madaouela I license (now the Madaouela I and Agaliouk licenses). The program started in August 2008, after permission was granted to conduct drilling west of the Madaouela Army Base, and initially focussed on development drilling of Marianne on a 50 m grid with a total of 62,648 m drilled in 2008.

Drilling east of the army camp was authorized in April 2009, enabling drilling to commence the 50 m grid on Marilyn. Development drilling on Marianne and Marilyn has continued until March 2013 as the size of the deposit has increased and further in-fill definition drilling has been required. The development drilling completed on Marianne and Marilyn since September 2012 has now been included in the resource update.

In order to increase the global potential and define inferred resource in areas historically drilled on a large grid (800 m to locally 400 m), from 2009 drilling targets were identified in the area south of the Marianne deposit. The drilling targets were identified by similarity with the geological controls defined for the Marianne-Marilyn mineralization as previously discussed.

Known historical drill intercepts that were in the suspected redox boundary trend line were drilled on a 200 m grid basis.

One area was discovered in the north east (“MSNE”) and two smaller zones are currently being recognized further south (MSCE, which was in fact the “historical” main CEA target). Drilling since February 2010 in the southern part of the area has confirmed the newly discovered Miriam deposit, together with the discovery of a new area termed the MSEE (Mad South Extreme East) deposit, of similar thickness and grade to the MSNE deposit. In 2012 the Maryvonne deposit was discovered.

Between 2008 and the end of 2013 a total of 2550 holes were drilled totalling 20,415 m on the Marianne and Marilyn deposits. Over the same period a total of 514,596 m were drilled on the Mad I and Agal licenses with 4,858 holes. Included in this drilling was the discovery of the La Banane deposit, which is the first deposit to be discovered in the Madaouela sandstones.

Table 9-2: Summary of core drilling and sampling

Secteurs	Number Holes	Total Metres
Marianne	191	13,304
Marilyn	60	1,480
MAD South	107	6,280
MAD 1 Explo	3	796
Totals	**361**	**11,452**

Diamond core drilling was carried out initially on Marianne to obtain a correlation between assays and radiometric data, this was subsequently followed by drilling over the other known deposits for stratigraphic correlation, but also as further validation of the correlation factor used from the initial Marianne core. Cores were sampled and prepared on the basis of industry standards, and assayed by ALS-Chemex in Johannesburg, South Africa.

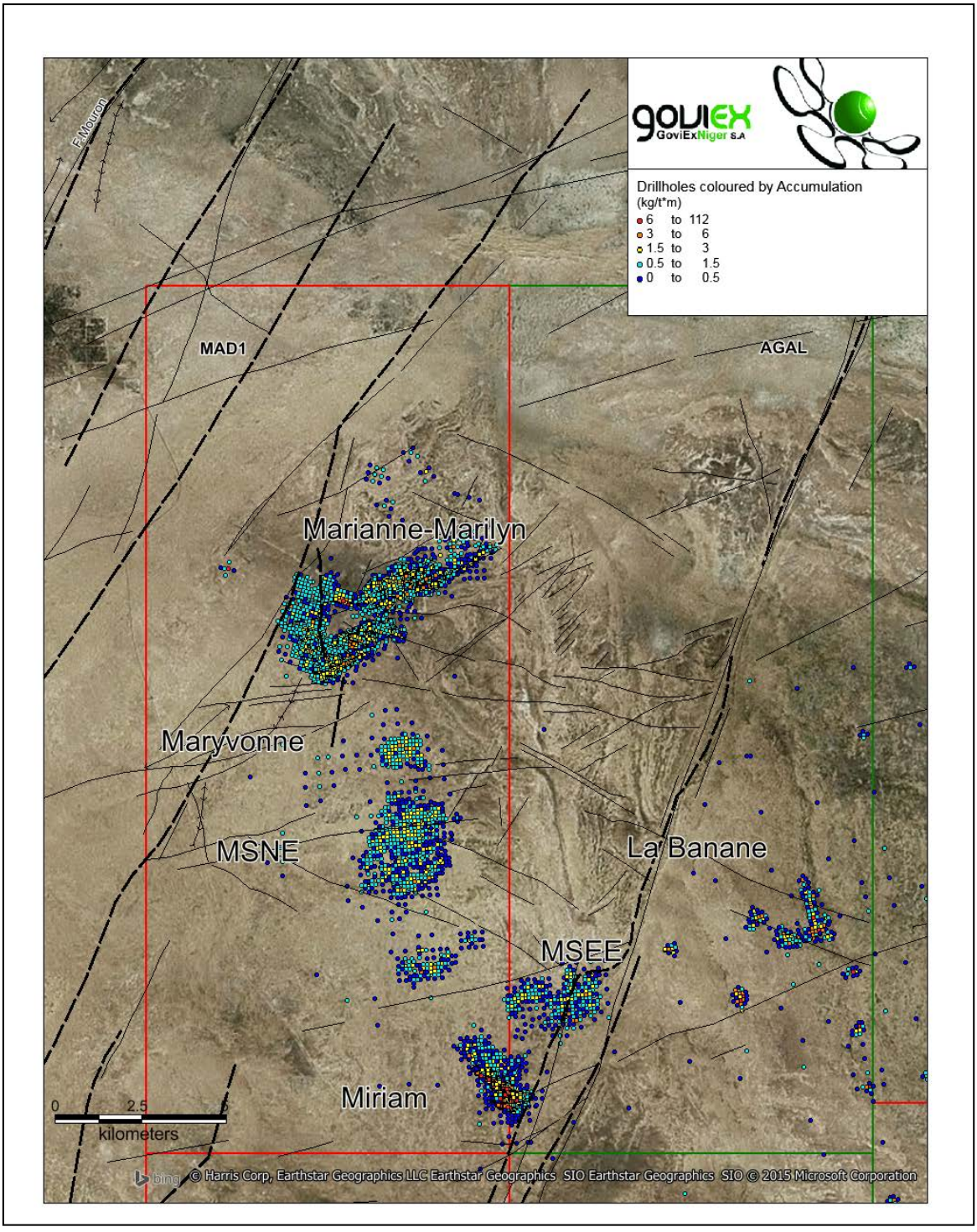


Figure 9-1: Drill hole positions and GT on Mad I and Agal licences

GoviEx has 100% ownership of Mad I, II, III and IV, and applications filed November 22, 2012 for Agal and Eral are pending. GoviEx has been advised by senior government officials that Agal and Eral can expect the applications to be approved.

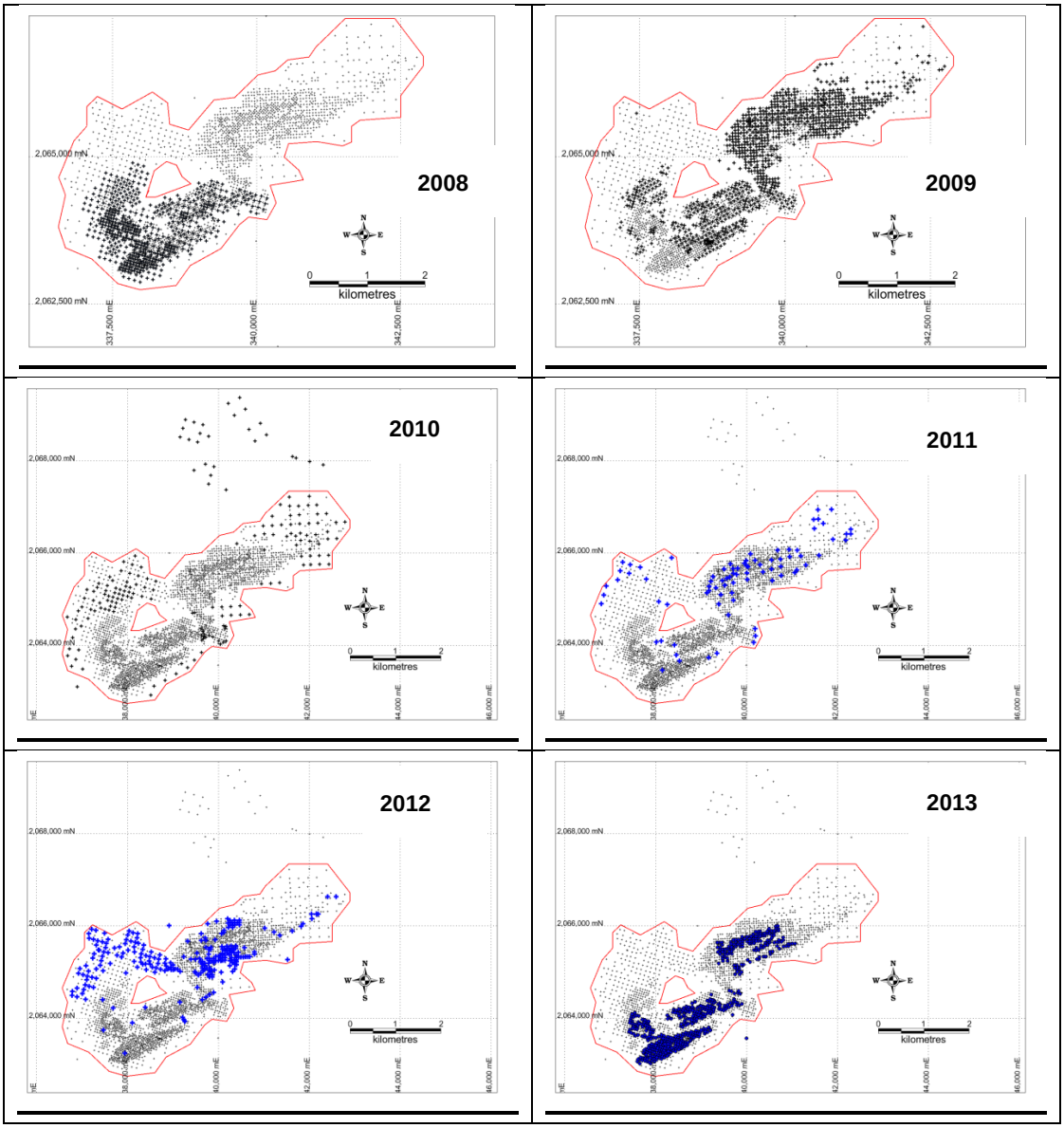


Figure 9-2: Drilling Progress at Marianne and Marilyn over time.

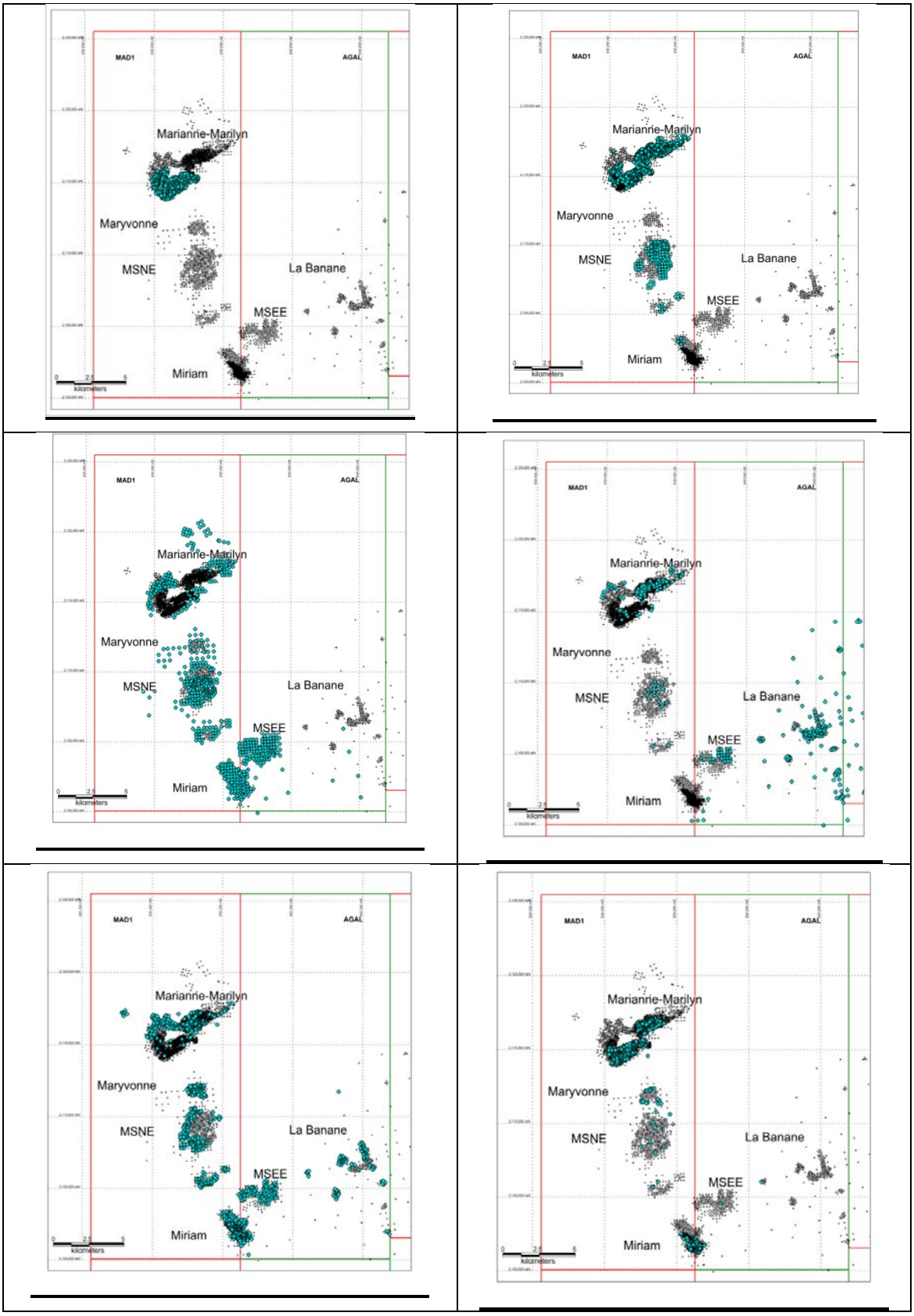


Figure 9-3: Drilling Progress at MAD I (incl Agaliouk) over time (2008 – 2013)

GoviEx has 100% ownership of Mad I, II, III and IV, and applications filed November 22, 2012 for Agal and Eral are pending. GoviEx has been advised by senior government officials that Agal and Eral can expect the applications to be approved.

Madaouela II, MAD III, IV, Eral and Anou Melle

Initially exploration on these properties started in April 2010 and continued until 2013 except at Anou Melle where it ended in July 2010. The distance to the drilling camp and the poor state of available drilling rigs at the time, resulted in an earlier stop to this program.

The subsequent exploration activity then was concentrated east of the Madaouela fault on Mad II, Mad III, Mad IV and Eral from August 2010 to January 2013, following the purchase of three new rotary drills by GoviEx. The exploration was conducted at 3,200 m grid on sub EW profiles, and following redox interpretation profiles at a 1,600 m grid were drilled on the northern part of Mad IV and Mad III.

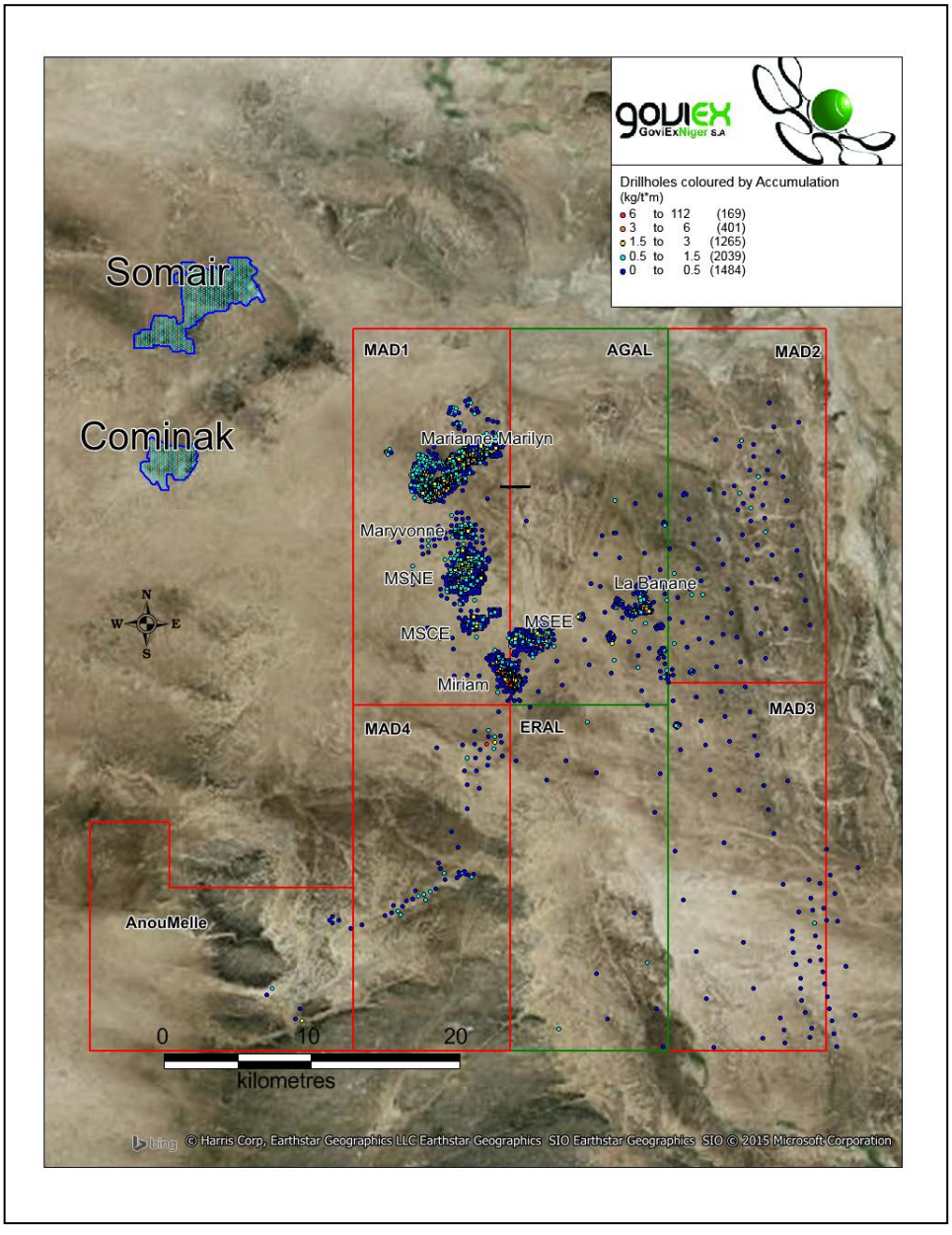


Figure 9-4: Position of GoviEx drill holes

GoviEx has 100% ownership of Mad I, II, III and IV, and applications filed November 22, 2012 for Agal and Eral are pending. GoviEx has been advised by senior government officials that Agal and Eral can expect the applications to be approved.

MAD II

As of 2010 onwards activities started on MAD II, including surface mapping along sections (Figure 9-5), debouchage/reopening of old CEA holes to check their readings and grid drilling at 1.6 to 3.2 km spaced lines. The results were encouraging with intercepts within the Madaouela, Tarat, Tchinezogue and the Guezouman Formations, see Table 9-4, Table 9-5 and Figure 9-6.

Table 9-3: Summary drilling activities- Debouchage means re-opening of old CEA holes, and re-logging with resistivity and gamma tools.

	Year	Number holes	Depth (m)
Debouchage	2010	2	746.52
	2011	10	1,687.21
Drilling	2010	3	570.85
	2011	48	9,624.99
	2012	0	0
	2013	7	11,75.24

Table 9-4: GoviEx drill intercepts at 400 ppm cut off within the Guezouman

Hole_ID	From (m)	To (m)	Th (m)	GTh	G (kg/t)	Formation
MAD20004	145.8	146.4	0.6	0.44	0.74	GUEZ
MAD20005	167.8	168.2	0.4	0.19	0.49	GUEZ
MAD20009	94.2	94.6	0.4	0.17	0.43	GUEZ
MAD20011	214.8	215.2	0.4	0.24	0.6	GUEZ
MAD20013	136.6	137.2	0.6	0.53	0.89	GUEZ
MAD20020	255.2	255.6	0.4	0.28	0.69	GUEZ
MAD20029	265	265.4	0.4	0.33	0.82	GUEZ
MAD20035	171	171.6	0.6	0.37	0.62	GUEZ
MAD20044	280	280.8	0.8	0.48	0.6	GUEZ
MAD20046	242.2	243	0.8	0.42	0.52	GUEZ
TALA197	143.8	144.6	0.8	0.78	0.97	GUEZ
TALA228	123.6	124	0.4	0.23	0.57	GUEZ
TALA229	122.4	122.8	0.4	0.29	0.73	GUEZ
TALA231	152.2	152.6	0.4	0.22	0.54	GUEZ
TALA234	155.2	155.8	0.6	0.54	0.89	GUEZ
TALA248	166	166.8	0.8	0.96	1.2	GUEZ

Table 9-5: GoviEx Drill Intercepts within other formations, including the Tarat, Madaouela and Tchinozogue (UTT)

Hole_ID	From (m)	To (m)	Th (m)	GTh	G (kg/t)	Formation
MAD20004	17.4	18.4	1	0.58	0.58	TARA
MAD20005	20.2	20.6	0.4	0.26	0.66	TARA
MAD20020	120.6	121.2	0.6	0.33	0.55	TCHI
MAD20036	119.2	120.2	1	1.04	1.04	TCHI
MAD20043	124	124.4	0.4	0.23	0.58	TCHI
MAD20047	96.2	96.8	0.6	0.32	0.53	MADA
MAD20047	130	131.4	1.4	0.81	0.58	TARA
MAD20048	95.2	96.4	1.2	0.77	0.64	MADA
MAD20048	122.6	123	0.4	0.24	0.59	MADA
MAD20049	93	93.4	0.4	0.2	0.49	MADA
MAD20050	118	119.2	1.2	0.96	0.8	MADA
TALA206	51	51.6	0.6	0.41	0.68	UTT
TALA231	27	27.4	0.4	0.21	0.53	TARA
TALA234	26.8	27.2	0.4	0.23	0.56	TARA

Figure 9-5: MAD II GoviEx drill hole locations showing holes that were re-opened and newly drilled holes, coloured by date drilled.

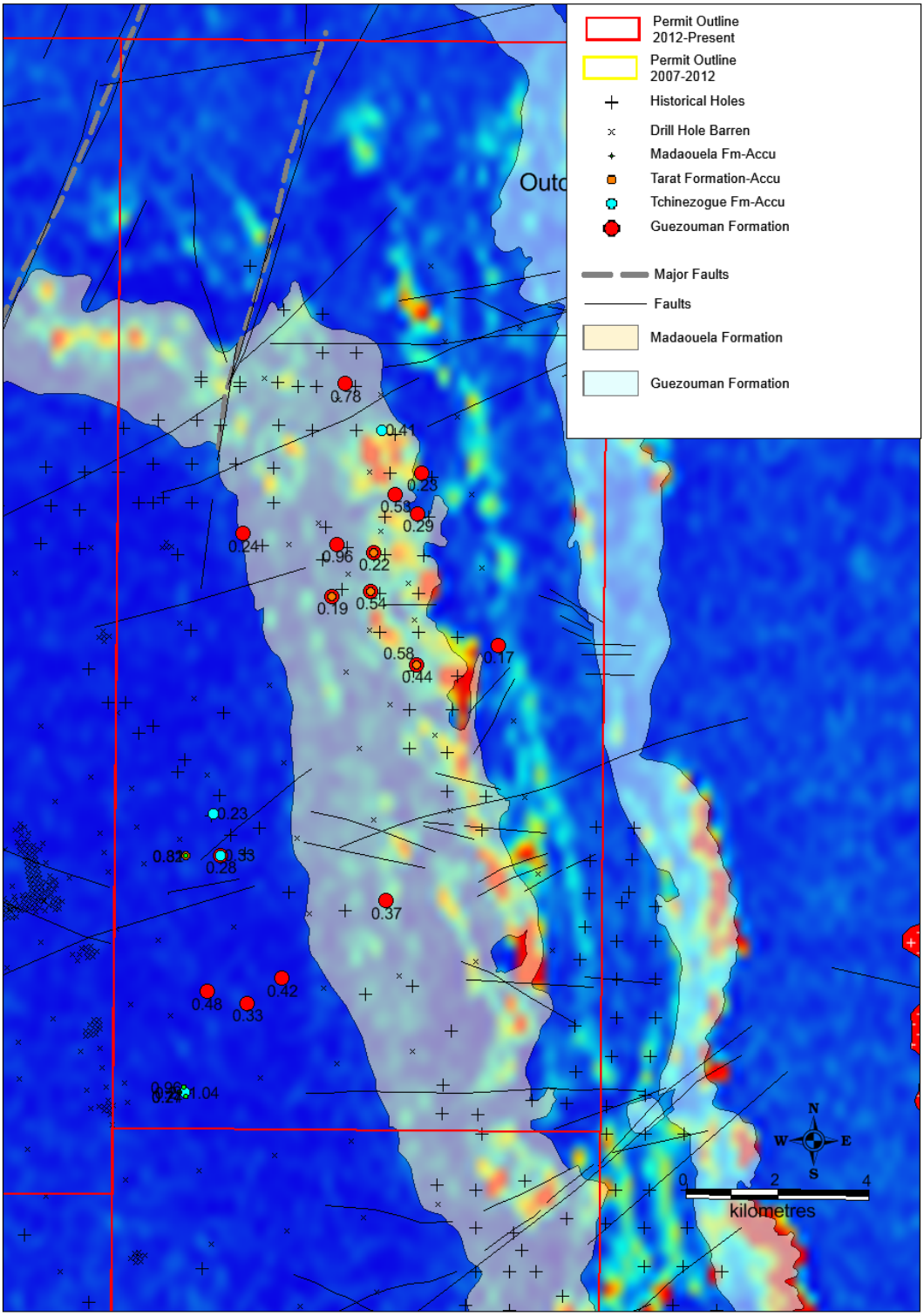


Figure 9-6: MAD II Significant intercepts by formation.

MAD III

From 2011 onwards activities started on MAD III, including surface mapping along sections, grid drilling at 800 m to 1600 m spaced lines (Figure 9-7). The results were encouraging with intercepts within the Madaouela, Tarat, Tchinezogue and the Guezouman. See Table 9-7.

Table 9-6: Summary drilling activities.

Year	Number holes	Total metres
2011	64	16,292
2013	1	424

Table 9-7: GoviEx drill intercepts at 400 ppm cut off

Hole_ID	Project	From (m)	To (m)	Th (m)	Gth	G (kg/t)	Formation
SMAD300021	MAD_3	288	288.4	0.4	0.18	0.46	GUEZ
SMAD300041	MAD_3	302	302.4	0.4	0.28	0.7	GUEZ
SMAD300421	MAD_3	285.4	286	0.6	0.44	0.73	GUEZ
SMAD300541	MAD_3	102	102.6	0.6	0.43	0.72	TARA
SMAD300541	MAD_3	198	198.4	0.4	0.36	0.9	GUEZ
SMAD300541	MAD_3	199.2	199.8	0.6	0.36	0.59	GUEZ
SMAD400031	MAD_3	160.6	161.6	1	0.91	0.91	TARA
SMAD400151	MAD_3	156.6	157	0.4	0.31	0.78	MADA
SMAD400161	MAD_3	163.2	163.6	0.4	0.36	0.9	MADA
SMAD400171	MAD_3	152.8	153.8	1	3.21	3.21	MADA
SMAD400281	MAD_3	151.2	151.8	0.6	0.92	1.54	MADA

Note: MAD IV prefixed holes were misnamed and is reported under MAD4 drilling

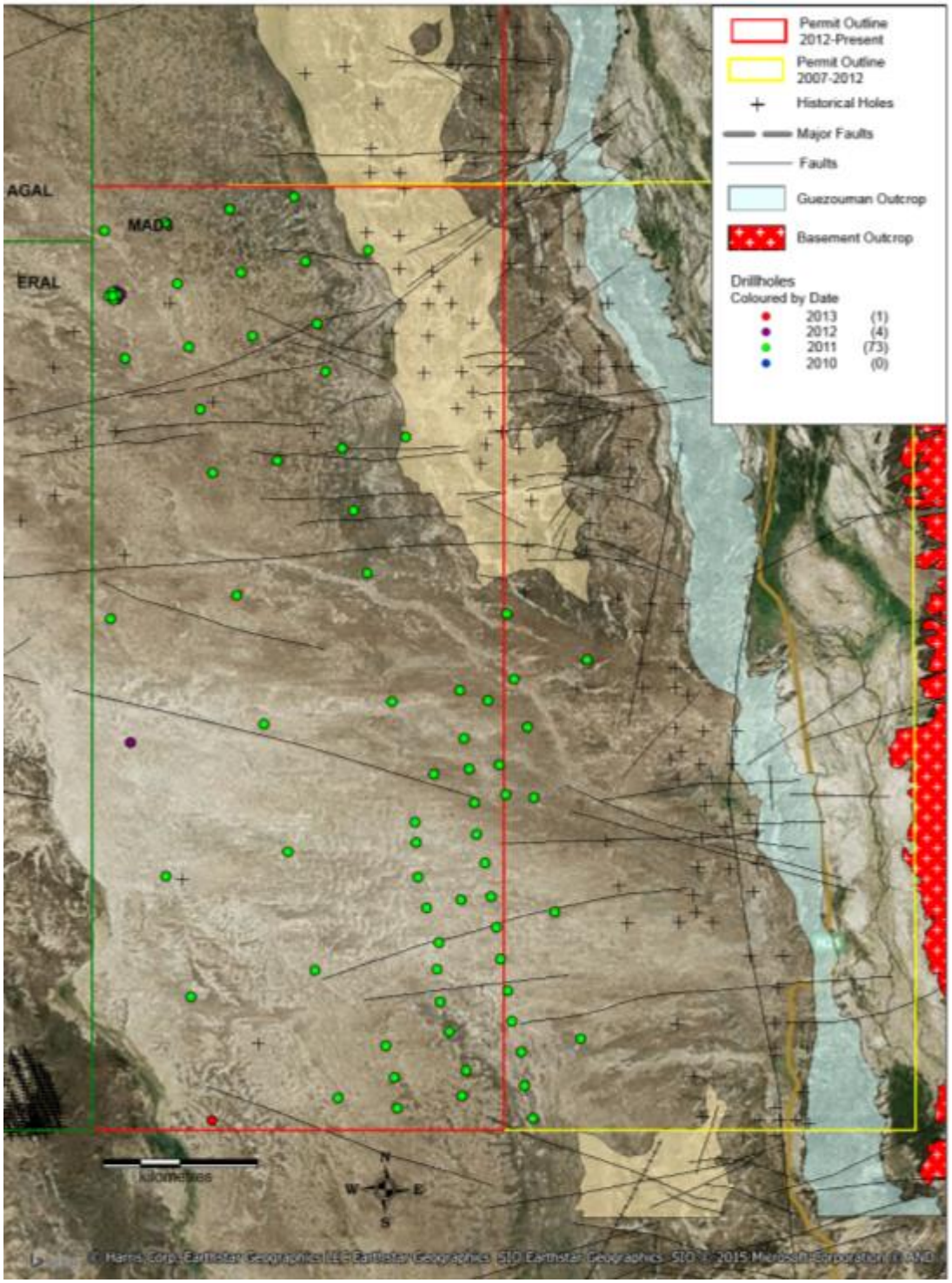


Figure 9-7: GoviEx drill hole locations, coloured by date drilled.

MAD IV (including Eralrar application area)

The drilling carried out over the MAD IV licence area (including Eralrar) spans the period 2010 to 2013. The 2010 drilling was focussed around known anomalies from CEA or PNC drilling results, from there follow up drilling in 2012 and 2013 around the same areas as well as a focussed drilling over an area of radiometric anomaly in 2013 (Figure 9-8). In 2011, most of the drilling consited of large spaced grid drilling for mainly stratigraphic and exploration purposes.

Table 9-8: Summary drilling activities.

Year	Number holes	Total metres
2011	20	7,959
2012	19	6,096
2013	37	11,217

Table 9-9: GoviEx accumulated drill intercepts at 400 ppm cut off

Hole_ID	From (m)	To (m)	Thickness (m)	GT	G (kg/T)
SMAD400051	189.2	190	0.8	0.72	0.9
SMAD400061	365	365.4	0.4	0.16	0.41
SMAD400081	408.8	409.2	0.4	0.25	0.62
SMAD400111	474.4	475.6	1.2	0.56	0.46
SMAD400211	446.4	470	1.2	1.24	1.04
SMAD400231	213.4	273.6	1.6	1.34	0.83
SMAD400321	224.8	225.2	0.4	0.21	0.53
SMAD400341	115.8	117.4	1.6	6.21	3.88
SMAD400411	116.8	117.4	0.6	0.65	1.08
SMAD400431	150	150.4	0.4	0.18	0.45
SMAD400481	287	288	1	0.95	0.95
SMAD400491	259.2	277	1.4	0.93	0.67
SMAD400501	276.2	276.8	0.6	0.86	1.43
SMAD400511	240.8	241.2	0.4	0.17	0.42
SMAD400521	4.2	4.8	0.6	0.44	0.73
SMAD400531	254.4	255.4	1	0.86	0.86
SMAD400541	370.6	371	0.4	0.26	0.66
SMAD400551	6.2	7.4	1.2	0.93	0.77
SMAD400571	282.4	282.8	0.4	0.21	0.52
SMAD400591	417.2	417.8	0.6	0.32	0.53
SMAD400601	10.2	12.2	1.4	0.76	0.54
SMAD400611	245.8	288	0.8	0.4	0.5
SMAD400621	283.4	284.2	0.8	1.11	1.39
SMAD400641	248	248.6	0.6	0.35	0.58
SMAD400651	268.2	268.6	0.4	0.25	0.63
SMAD400731	255.4	256	0.6	0.4	0.67
SMAD400751	305.6	306.4	0.8	0.53	0.67
SMAD400761	292.2	293.2	1	0.66	0.66
SMSNE44581	73.8	74.4	0.6	0.83	1.39
SMSNE45211	92	92.8	0.8	0.79	0.98
SMSNE45241	97.8	139.6	2.4	1.77	0.74

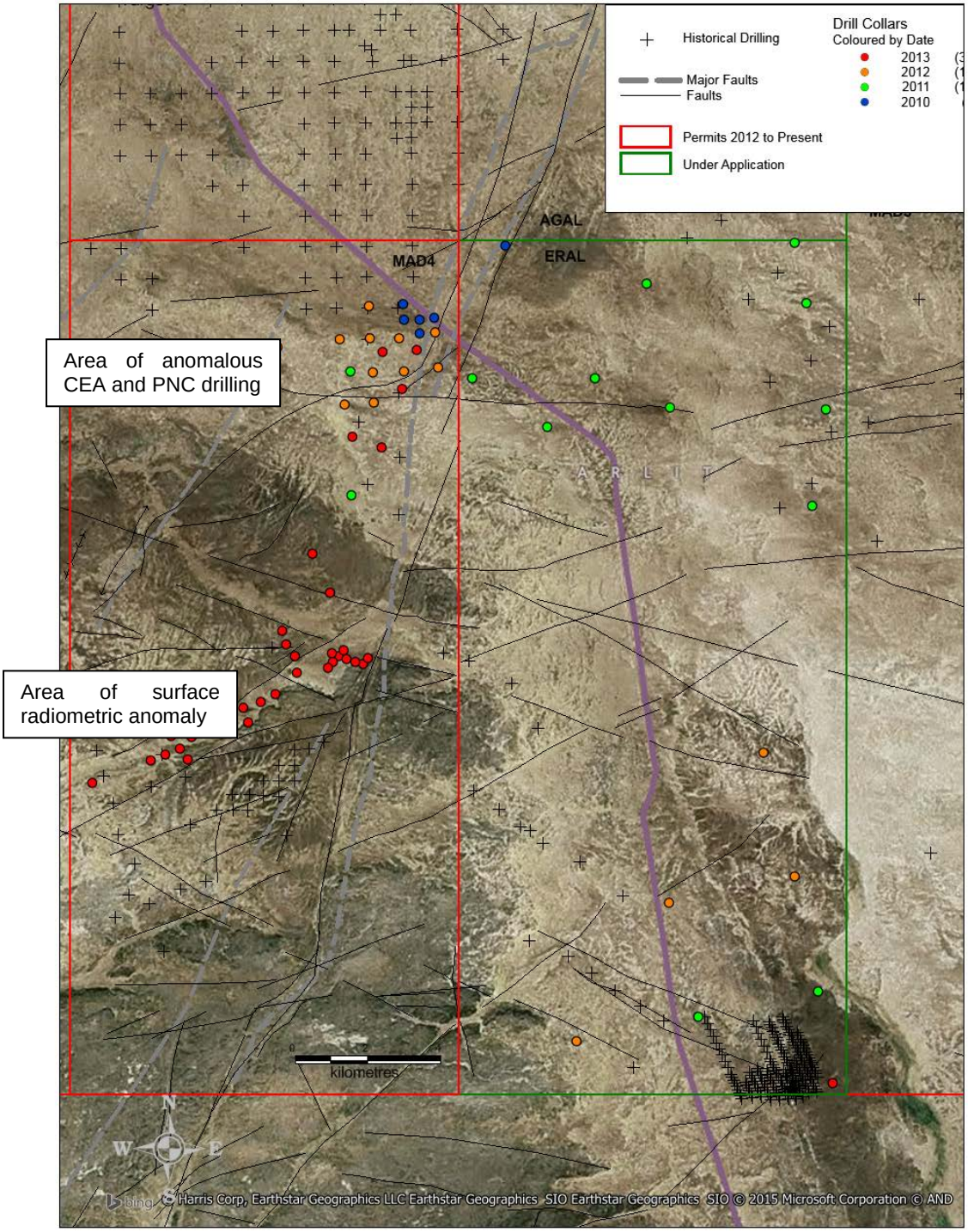


Figure 9-8: MAD IV including Eral drilling collar position, coloured by date.

GoviEx has 100% ownership of Mad I, II, III and IV, and applications filed November 22, 2012 for Agal and Eral are pending. GoviEx has been advised by senior government officials that Agal and Eral can expect the applications to be approved.

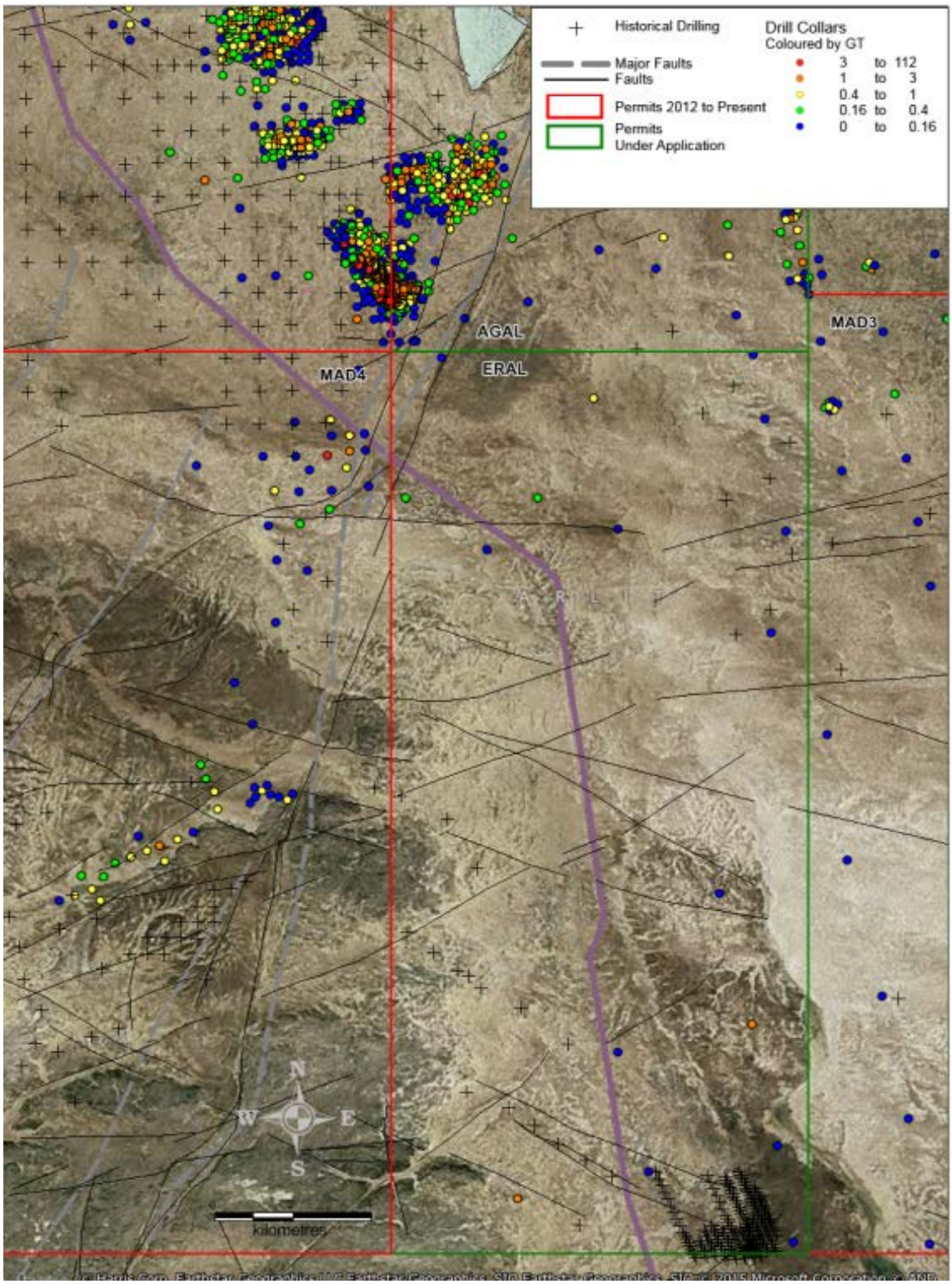


Figure 9-9: MAD IV including Eral drilling collar position, coloured by grade accumulations.

GoviEx has 100% ownership of Mad I, II, III and IV, and applications filed November 22, 2012 for Agal and Eral are pending. GoviEx has been advised by senior government officials that Agal and Eral can expect the applications to be approved.

Anou Melle

In 2010, GoviEx carried a preliminary drill program over the Anou Melle licence. As previously reported for logistical and security reasons and the poor quality of the drill rigs, the program was abandoned and no further follow up were carried out due in changes in priority. Mapping in 2014 reveals the prospectivity of the area.

Table 9-10: GoviEx drill intercepts at Anou Melle, using a 400 ppm cut off

Hole_ID	From (m)	To (m)	T (m)	GT	G (kg/T)	Formation
SMAD40056	281.6	282.2	0.6	0.24	0.4	Tarat
SANOU5001	329.2	329.6	0.4	0.24	0.61	Tchinozogue
SANOU5003	40.6	41.2	0.6	0.26	0.44	Moradi
SANOU5003	328.6	329	0.4	0.23	0.57	Guezouman
SANOU5005	404.6	405.2	0.6	0.49	0.82	Guezouman
SANOU5007	351.2	351.6	0.4	0.33	0.83	Guezouman
SANOU5008	20	20.8	0.8	0.49	0.62	Teloua
SANOU5008	307.8	308.2	0.4	0.28	0.7	Tchinozogue
SANOU5008	346.6	347	0.4	0.19	0.47	Guezouman
SANOU5008	348.6	349.2	0.6	0.5	0.84	Guezouman
SANOU5008	350.6	351.4	0.8	0.85	1.06	Guezouman
SANOU5009	299.4	301.4	2	1.11	0.56	Tarat
SANOU5009	400	400.4	0.4	0.26	0.65	Guezouman

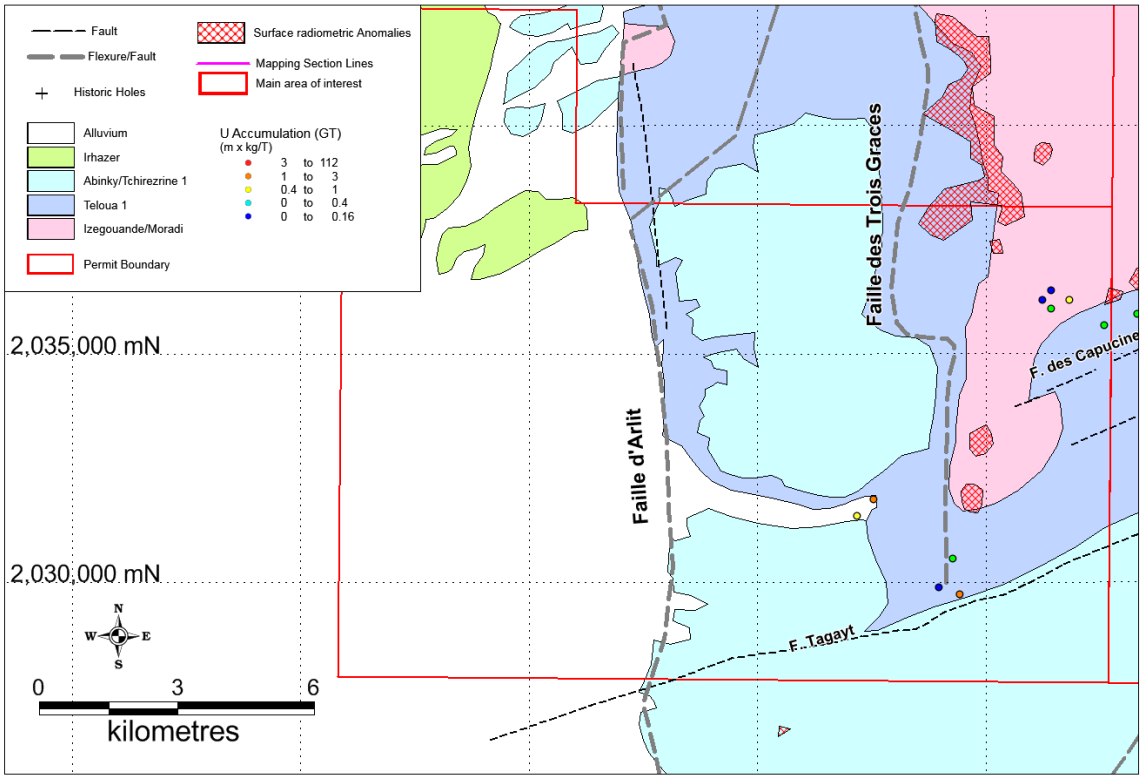


Figure 9-10: Anou Melle Drilling location, coloured by grade accumulation (GT). All holes were drilled in 2010.

9.5 Surveys and Investigations

9.5.1 Surveying

Surveying is done with precision, care and crosschecking in the field, using the DGPS equipment; thus providing collar locations to centimetre accuracy. Surveying uses a network of permanent survey monuments for base stations and is tied to real-world coordinates using WGS 84 as a format.

9.5.2 Survey Methodology

Allen Osborne Associates Rascal DGPS equipment was purchased and a crew trained on its utilization both in static and dynamic modes.

In the absence of a reliable certified geodesic benchmark (reference point), the establishment of a reference point with a VLBS (Very Long Base Line) was implemented in August 2008, by measuring from the roof of an office in Arlit. A second base point was then established on Marianne and used for surveying the drillholes in the field from August until January 2009. The measurements were conducted in the static mode, which required a post-treatment.

While pursuing the survey in a static mode, the elevation (Z) of the surveyed points, although very precise in relative terms, was at a variance with the IGNN (Institut Geographique National du Niger) grid by about 25 m. This discrepancy remained because the elevation was measured in reference to the mean sea level instead of the ellipsoid height as used by the IGNN.

In February 2009, the survey team obtained the coordinates of one geodesic point that was apparently used by COGEMA (AREVA) since 1968/1969. Those coordinates were later confirmed by IGNN. This point was located in the field and served to calibrate the survey grid. From this reference benchmark, a reference elevation was adopted and previous data corrected accordingly.

Thus, a new VLBS station was established on the roof of the current office in Akokan. At this location, a GPS base was established on February 12, 2009, and a continuous 28 hour GPS observation was made. Subsequent calibration of this observation was made using nine IGS stations located in Spain, Malta, Italy, Greece, Uganda, Cote d'Ivoire and Canary Islands. The GPS processing files were downloaded from these stations and used as reference to reprocess the 28 hour Akokan office base station data. The calibrated position of the reference station in UTM zone 32 WGS 84 is:

E 325,109.941m	N 2,069,339.230m	Elevation 442.756m above mean ellipsoid

All DGPS points have then been reprocessed with the new reference coordinates. Out of 800 surveyed points, 98 % had very high degree of confidence in relative positional fixes. 2 % of the points that did not meet the same confidence have since been resurveyed in the field and corrected in the database.

Three new base stations in the field have been established and calibrated using the new coordinates of the Akokan office base station.

Since February 2009, the survey team has been trained on the use of Graf Nav Kinematic software for the Rascal controller. Radio modems are used to transmit the error matrix from the DGPS reference station to the Rover, which provides a highly precise positional fix in real time. This method removes the need for post processing of the DGPS data. Table 9-11 lists the current GoviEx reference survey points.

Table 9-11: DGPS, listing of current GoviEx reference survey points

Reference (Topo 23 07 09)			
Survey Points	**E**	**N**	**H**
Office base AKOKAN	325,109.925	2,069,339.225	418.813
Former offices in ARLIT (Mahmud's villa)	331,312.427	2,071,357.210	430.487
BASE CHANT IER 2	339,139.389	2,063,580.003	440.605
BASE CHANT IER 3	339,187.784	2,063,634.480	440.745
BASE CHANT IER 4	338,800.109	2,065,718.820	435.290
IGN(D)(Akokan)	328,750.028	2,071,149.985	420.546
BASE 5	340,491.971	2,059,202.602	440.885
BASE 6	341,953.446	2,065,716.220	440.305

The DGPS team is under the supervision of the chief geologist and comprises of one geologist and one qualified prospector (working alternatively) accompanied with two field assistants (4 persons in total). Surveys of drill collars are transferred to the database daily; therefore, geologists are provided coordinates of drillhole positions for interpreting and validating exploration results in real time. In the event that geological interpretations reveal that some holes are not correctly reported, the collars are resurveyed, and the resurveying includes often neighbouring points for homogeneity controls.

9.5.3 Drilling

Drilling is done with mud-rotary drill rigs that drill approximately 120 mm diameter holes. Historical drillholes have been located where possible in the field, and reamed with this larger diameter drill bit, and re-logged. GoviEx is therefore relying of GoviEx data from historical drillholes, not historical data. The few historical holes that could not be found to re-enter (re-drill) were offset and a new hole was drilled (approximately 15 such holes to date). Upon completion of drilling to 8 to 9 m below the mineralized horizon, the hole is cleaned and circulated with drilling mud for 30 minutes, to allow for fluid-filled holes for electrical logs as well as gamma logs, and avoid radon contamination.

All mineralized holes (intercepts over 100 c/s (scintillometer (PM) and Geiger probe (GM)) are relogged with the Mount Sopris GoviEx logging unit equipped with DHT27 Areva standard probe, and by ULC using Geovista probes (UMC logging include caliper, deviation and resistivity as well as PM and GM)

Core drilling is done selectively, with:

- Rotary drilling up to a defined depth
- Core drilling with specific core drilling rig (Chinese copy of diamond Boart core drilling machines to collect core from the mineralized horizons. Core samples were assayed to define the 'counts/second' eU correlation line The 42 core hole intersections in the Marianne deposit for approximately 320 individual core samples were used initially. More samples were collected on Marilyn, MSNE and Miriam, but the correlation line of Marianne was kept for all deposits, as it proved to be acceptable. It is a conservative

option, as it may be locally minoring grades from several %, but it is preferred at this stage to use one single correlation line.

9.5.4 Logging

Logging is done using; three SEMM designed logging units, with probes modified for GoviEx. Internal QA/QC of intersections greater than 100 raw c/s (GM probe) is conducted using one Mount Sopris logging unit (GNSA owned) equipped with DHT27 Areva probe. For each drillhole logging unit, two probes are used;

- a resistivity and natural gamma (scintillation(PM)) probe,
- and a probe containing natural gamma (by Geiger tube GM) and by scintillation counter (PM)) and deviation (magnetic/inclinometer) instrumentation. GM logs are used to define in-situ uranium grades for the drillhole database.

9.6 Interpretation (SRK)

SRK's interpretation of the GoviEx exploration program is that of a well-planned and executed exploration and resource definition-drilling program. Contractors and GoviEx personnel are knowledgeable in drilling and logging equipment and procedures. And GoviEx has installed procedures for data checks and verifications to ensure data accuracy and consistency. The GoviEx exploration team in place are a competent crew of geologists that work very well together, and have been sufficiently trained and instilled with the need for data verification that allows for minimal errors in the drillhole database. SRK found that the procedures in place for exploration drilling, data gathering, and data verifications are at a high level, and meet or exceed industry norms for uranium exploration methods.

The exploration programs are appropriate for the style of uranium mineralization. SRK found no deficiencies in the exploration methods. The field procedures are appropriate and adequate for developing a drillhole database sufficient for resource estimation.

Significant up-side exploration potential exists on GoviEx concessions at the Madaouela Uranium Project.

Source: GoviEx, 2009

Figure 9-11: Digital GPS surveying in the field; reference and rover

Source; SRK, 2010

Figure 9-12: Drill Rig at Madaouela south

9.7 Results and Conclusions (by SRK)

SRK concludes the drilling methods used by GoviEx are industry standard methods, and are appropriate for the style of uranium mineralization at the Madaouela Uranium Project. Following SRK recommendations, GoviEx has conducted a large amount of core drilling, originally with ESAFOR, and ultimately with Geodrill, within the frame of the contract signed to execute the geotechnical core drilling.

10 SAMPLE PREPARATION, ANALYSES AND SECURITY (ITEM 11)

10.1 Sampling Method and Approach

The industry standard methods of determining in-situ uranium grades are by assay methods on core and by equivalent uranium grade determination from down-hole gamma logging equipment. The gamma logging probe uses either a scintillometer or a Geiger-Muller tube to record gamma radiation as counts per second ("CPS"). The gamma probe is calibrated against a known source and correlated with core assays to derive conversions of CPS to eU_3O_8 grades.

GoviEx has used the Geiger-Muller gamma probe data as the primary sampling tool in determining uranium mineralization thickness and grade. Radiometric probing (gamma logs) and the conversion to eU_3O_8 data have been industry-standard practices used for in-situ uranium determinations since the 1960's.

10.1.1 Sampling Methods

10.1.1.1 Rotary Cuttings

Rotary drill cuttings are collected on 1 m intervals and laid out at the drill site, for geological logging only, cuttings are not retained for future use. Rotary mud drilling provides samples that by the nature of the drilling medium are not sufficient for accurate chemical analysis of in-situ uranium grades. GoviEx has not analyzed rotary drill cuttings.

Nevertheless, rotary drill cuttings are an important geological sample as they are compared with geophysical logs (electrical logs and gamma by scintillation) to determine lithological contacts, redox state, dominant granulometry which are important to stratigraphic correlation and therefore mineralization correlations hole-to-hole.

10.1.1.2 Core Sampling

Drill core PQ size (93 mm) was sampled, as whole core, at 30 cm intervals for establishing the Ra/Grade relationship.

GoviEx sampling procedures for metallurgical testing includes core size of 93 mm, assaying ¼ core on 30 cm samples, retaining ½ core samples for metallurgy and/or a thick (5 to 10 mm) slice of core (length-wise along the core axis) is glued to a wood strip to preserve a sample of for lithology/mineralogy/alteration reference.

GoviEx also uses core samples for specific gravity determinations as a means to derive bulk density measurements. Historical determinations of density resulted in an average dry density of 2.3 g/cc (or t/m^3) for Guezouman sandstone, which is verified by GoviEx data and consistent with that expected from sandstone. All mineralization is in Guezouman sandstone, and there is limited or no expectation of density variance within the sandstone.

10.1.1.3 Radiometric Sampling

The standard method used by GoviEx is the gamma probe as further described below. GoviEx uses two types of probes to log each drillhole:

1. A 42 mm diameter probe, called BDGG. It records two natural gamma reading using.
    a. One photomultiplier tube ("PMT"), equipped with a 2.5 cm NaITh crystal.

b. One scaling unit attached to two Geiger Muller tubes: ZP 1200 (gamma CPS used in grade estimation).
c. Borehole deviation recording (tilt and azimuth) using a three axis magnetometer and inclinometer.

2. The second probe called GUIP (38 mm diameter) records natural gamma (PMT with 2.5 cm Nail crystal), resistivity and conductivity.

Only the BDGG gamma probe records are used for estimation purposes and for converting radioactivity into the equivalent uranium grade.

Gamma logging speeds are set at 1 m per minute in mineralization, and 5 m per minute in non-mineralized rock. The resistivity (GUIP) probe is logged at 5 m per minute. Typical logging procedures are to monitor the probe data while the probe is lowered to maximum drillhole depth, to identify the mineralization, then set the logging speed to 1 m/min and log the mineralization as the probe is winched out of the hole; changing to the faster logging speed above mineralization.

CPS data are gathered as composited 10 cm data, for conversion to eU data in Utimine software. Each gamma probe is calibrated at an onsite calibration pad, using a caesium source for 20 minutes. This calibration is to verify each probes functionality and lack of drift. Probes are compared to a reference probe ensuring that all probes are calibrated correctly.

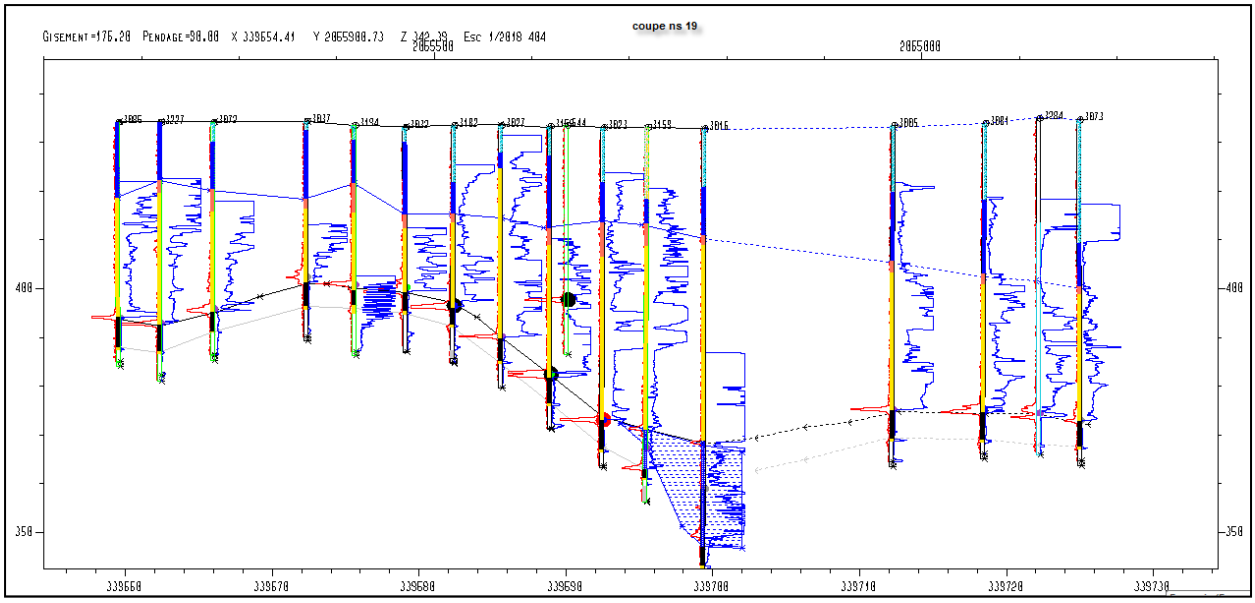
10.1.2 Factors Impacting Accuracy of Results

The most important factors affecting the accuracy of gamma probe derived eU grade data are the calibration and correlation of the gamma probes. The probes have been calibrated and a K factor determined at external logging facilities (Canada and France). The QA/QC procedures are summarized in Section 11 (Data Verification).

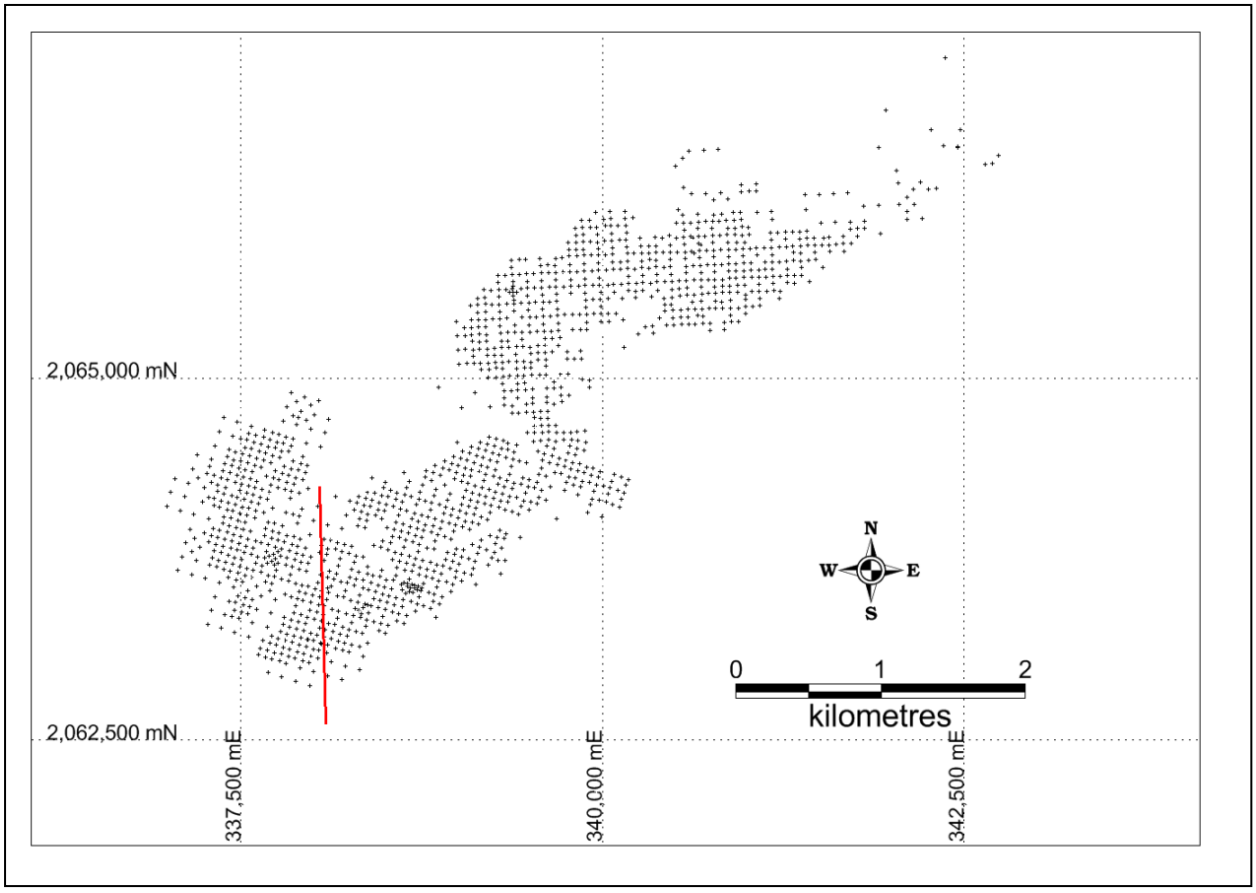
SRK considers the sampling method and approach to be appropriate for the mineralization, and adequate for generating a reliable database to be used in resource estimation. The methods are standard industry practice for sampling uranium deposits.

Source; SRK, 2010

Figure 10-1: Down-hole gamma logging at Madaouela



Note: Vertical exaggeration in the vertical scale; 50 m drillhole spacing.
Source: GoviEx, 2009 (line of section below)



Source: GoviEx, 2009

Figure 10-2: North-south section in Marianne, showing down-hole logs and correlations

10.2 Sample Preparation and Assaying Methods

10.2.1 Chemical Analyses

A total of 165 cored drill holes were completed over the various deposits and prospects. Core from 65 drillholes in Marianne-Marilyn were sampled; 400 samples. Whole core was sent to ALS for sample preparation and analysis. Chemical analyses of core for uranium (U) are used to compare against radiometrically determined uranium (expressed as gamma log CPS data) for the purpose of establishing a correlation curve to apply to determine the equivalent uranium grades (eU) that forms the basis for the drillhole database. Other assays were used to verify that the factor used derived from Marianne data could be used at the other deposits. It was concluded that the Marianne factor was conservative, and could be safely used.

Table 10-1: Totals of core samples actually sent for assays.

Secteur	Holes	No Samples	Length
MARIANNE	65	400	118.2
MARYLIN	53	394	116.5
MIRIAM	23	1,157	346.7
MSCE	2	27	5.4
MSSE	5	81	28.5
MSNE	17	116	34.5
Totals	**165**	**2,175**	**649.8**

GoviEx uses ALS Chemex in Johannesburg, South Africa, for analytical work on core samples. Sample preparation includes crushing and pulverizing to 75 µm the entire 30 cm length sample interval, without keeping representative samples on site. This was done for the purpose of analyzing the entire volume of core, as GoviEx's intent was to include all mineralization to provide the most representative sample possible. Subsequent to initial core drilling and sampling for determination of the radiometric/grade correlation, some core holes, or representative sections are being retained on site.

Analyses for uranium and other elements were done by ALS Chemex by standard 4-acid digestion on pulps samples, and ICP-MS analytical methods. ISO registration and accreditation provides independent verification that analytical labs have a Quality Management System in place. Most ALS Chemex laboratories are registered or are pending registration to ISO 9001:2000, and a number of analytical facilities have received ISO 17025 accreditations for specific laboratory procedures.

Uranium assay results were composited to establish the correlation line with radiometric data. Uranium assays are not used in resource estimation.

Bi-variate statistical analyses of assays were carried out and possible geochemical correlations between elements were investigated. Similarly, statistics were done on "composites" at 400 ppm eU cut-off along drillholes for the core samples analyzed. A summary of 42 mineralized intersections are summarized in Table 10-2.

- The average Mo, V and CO_2 grades show that Madaouela mineralization is very similar to the Cominak ore. The current core sampling may not be fully representative of the entire Marianne-Marilyn deposit, but it is considered to be unbiased;

- Mo, V and CO_2 are poorly correlated to U. The relative abundance of such metals in the mineralization is representative of the Redox conditions of the sediments at the time of uranium deposition; and
- The average uranium grade of this sampling batch is matching the current estimated mean grade of Marianne, which is a demonstration of that it is representative.

Table 10-2: Average geochemical content of Marianne-Marilyn mineralization based on drill core analysis

Average Grade (ppm)			Average Grade (%)
U	Mo	V	CO_2
1,611	502	157	2.9

Source GoviEx, 2009

Molybdenum grades from underground sampling shows greater variability in the deposit with concentrations up to 4200 ppm reported (GoviEx, 2013). Bulk metallurgical samples taken from Marianne-Marilyn and analysed at Mintek and SGS showed an average for molybdenum of 1067 ppm (see section 12).

Correlation X-Y plots of the elemental relationships of Mo, V, and CO_2 with U indicate the following:

- Mo, V and CO_2 are not correlative or only very weakly correlative with U;
- There is a general spatial correlation between anomalous values of Mo, V, and CO_2 with the uranium deposit; however
- High U/Mo does not directly compare to a high U/V ratio, and high CO_2 values do not correspond to high U grades.

10.2.2 Radiometric Determinations

The basic analysis that supports the uranium grade reported in the Madaouela database of uranium grades and thickness of drill intercepts is the down-hole gamma log created by the down-hole radiometric probe. That data is gathered as digital data and composited to 10 cm data as the radiometric probe is extracted from a drillhole.

The down-hole radiometric probe measures total gamma radiation from all natural sources, including potassium (K) and thorium (Th) in addition to uranium-bearing minerals. In most uranium deposits, K and Th provide a minimal component to the total radioactivity, measured by the instrument as CPS. At the Madaouela Uranium Project, the uranium content is high enough that the component of natural radiation that is contributed by K from feldspars in sandstone, and minor Th minerals is expected to be negligible. The conversion of CPS to equivalent uranium concentrations is therefore considered a reasonable representation of the in-situ uranium grade. Thus, determined equivalent uranium analyses are typically expressed as ppm eU_3O_8 ("e" for equivalent) and should not be confused with U_3O_8 determination by standard XRF or ICP analytical procedures. The conversion process can involve one or more data corrections; therefore, the process used for Madaouela is described here.

The gamma probes are 42 mm in diameter and about 1.5 m in length. The probe has a standard sodium iodide (NaI) crystal that is common to both hand-held and down-hole gamma scintillation counters. GoviEx constructed probes include the scintillation counter and the Geiger-Muller tube, both of which function similarly to count natural radiometric emanation from uranium and its daughter products (the uranium decay series). GoviEx is using the Geiger tube readings for uranium grade determinations and the scintillation counter for depth

correlations and for lithology picks. The logging system consists of the winch mechanism (which controls the movement of the probe in and out of the hole) and the digital data collection device (which interfaces with a portable computer and collects the radiometric data as CPS at defined intervals in the hole). Radiometric readings are collected digitally into WellCad software for correlation with geology and resistivity. Subsequently, data is transferred to Utimine software for conversion to eU grade data (G), along with thickness (T), and accumulation (GT; Grade-thickness product).

Raw data can be viewed and plotted graphically from WellCad software, to provide a graphic down-hole plot of CPS. The CPS radiometric data may need corrections prior to conversion to eU or eU_3O_8 data. Those corrections include: accounting for water in the hole (water factor) which depresses the gamma response, hole diameter variations, the instrumentation lag time in counting (dead time factor), and corrections for reduced signatures when the readings are taken inside casing (casing factor). The water factor and casing factor account for the reduction in CPS that the probe reads while in water or inside casing, as the probes are typically calibrated for use in air-filled drillholes without casing. Water factor and dead time factor corrections are made to the data at Madaouela; there are no instances of mineralization inside casing at Madaouela.

Conversion of CPS to eU or eU_3O_8 is done by determining the relationship of core to radiometric data for a set of core-hole sample intercepts, and developing a correlation curve.

The procedure used by GoviEx at Madaouela is to convert CPS per anomalous interval by means of a correlation curve developed by comparing core intervals with gamma-log intervals for the 46 core hole intervals drilled at Marianne. The process involves re-positioning the core pieces for the whole-core interval of mineralization, and determining the contacts and peak radiometric reading with a hand-held scintillometer on the core. This is then matched with the radiometric curve developed from a down-hole plot of CPS. The core is cut and analyzed for uranium content for the same interval as the radiometric indicate. A best fit line defines the relationship of GT as follows:

$$GT_{core} = U_{core} \times T_{core} = (Factor \times CPS \times T_{probe}) = GT_{probe}$$

The same can be done on composited grade (U%) versus (CPS) at a given composite interval for each; the relationships have been found to be similar to that for GT. The factor is then used to convert CPS to eU grade.

GoviEx has found that the coefficient of correlation between GT_{core} and GT_{probe} is 0.968, with a 2 sigma precision on the mean of 8.1 %; a relatively close clustering of data along a linear relationship.

In uranium exploration, the usual technique consists in:

- Measuring the radioactivity in the drillholes using dedicated probes; and
- Transforming the radioactivity into uranium grade (equivalent uranium grade (eU or eU_3O_8).

In mineral deposit of recent age or close to surface alteration, where the uranium is not in equilibrium with its daughter products, the measurement of the gamma radioactivity to determine the uranium grade is imprecise and inaccurate. In such cases, the use of radioactivity estimations is completely incorrect.

In deposits such as Marianne-Marilyn at Madaouela, and all known mineralization in the carboniferous in Niger, the disequilibrium problem has never been encountered. These

uranium deposits are characterized by a secular equilibrium of uranium with its daughter products (uranium/radium). In specific locations within a deposit, the ratio U/equivalent Radioactivity can vary (from 0.8 to 1.2), but the weighted mean value remains close to 1. This fact is well established through all analyses carried out on cores and mine face samples, as well as through the numerous reconciliations of mining and development or grade control estimation in the district.

On the Madaouela Uranium Project, this particular aspect was carefully examined historically by CEA; unfortunately none of the relevant historical technical data was available to GoviEx. GoviEx did require ALS to carryout gamma spectrometry on 100 core sample pulps already assayed for G/Ra correlation. These measures were outsourced by ALS laboratory in RSA to Becquerel Laboratories Inc., Mississauga, Canada.

The following isotopes were to be measured (measurement method in bracket, results in Bq/g): [Bi214, Pb-210, Pb-212, Ra-226, Ra-228, Th-228, Th-230, Th-234, U235](GAMMA), [U-238, Th-232](NAA). Note that results relate only to the samples analysed and only to the items tested. Ra-228 was estimated from Ac-228 and Th-228 from Pb-212.

The lab delivered only the value of disequilibrium with the ratio U-238/Ra-226. GoviEx did perform grade re-composition with the help of specialists from CEA lab. Results are summarized in Table 10-3.

Table 10-3: Disequilibrium and reconstructed U grades from isotopic spectrometry (Becquerel data, ALS chem, GoviEx recalc)[1]

	U235	U238	Ra226
Période (secondes)	2.22E+16	1.41E+17	5.05E+10
Masse molaire (g/mol)	235.044	238.051	226.026
Avogadro (atomes/mol)	6.02E+23	6.02E+23	6.02E+23
Activité massique (Bq/g)	7.996E+04	1.244E+04	3.658E+10

Hypothèse = valeurs en Bq/g de roche

ME-MS61	SPECTROMETRIE (GAMMA, NAA) données en Bq/g par le labo								Calculs des teneurs massiques (en g / g de roche)			
V	SAMPLE	res en spectro gamma pour Ra et U235, en NAA pour U238)										
ppm	DESCRIPTION	U-238/Ra-226	Ra226	U235	U238	235/u238(%)		U‰ (recalculé d'apres les mesures spectro ou pluto NAA)	U235	U238	Ra226	Teneur massique en U ‰ total
217	E004/2200	**1.06**	16.96	0.8	17	5	1.3	1.298211538	1.001E-05	1.367E-03	4.637E-10	1.377
240	E003/2200	**1.09**	11.99	0.5	12	4	0.9	0.916384615	6.253E-06	9.649E-04	3.278E-10	0.971
381	E002/2200	**1.36**	45	2.3	61	4	4.7	4.658288462	2.876E-05	4.905E-03	1.230E-09	4.934
227	E0010/2202	**1.13**	16.95	0.7	17	4	1.3	1.298211538	8.754E-06	1.367E-03	4.634E-10	1.376
277	E008/2202	**1.18**	103.84	4.4	104	4	7.9	7.942	5.503E-05	8.363E-03	2.839E-09	8.418
431	E0017/2203	**1.22**	71.98	3	72	4	5.5	5.498307692	3.752E-05	5.790E-03	1.968E-09	5.827
208	E0015/2203	**1**	15	0.6	15	4	1.1	1.145480769	7.504E-06	1.206E-03	4.101E-10	1.214
338	E0289/2204	**1.05**	44.1	1.9	44	4	3.4	3.360076923	2.376E-05	3.538E-03	1.206E-09	3.562
177	E0037/2207	**1.04**	26	1.2	26	5	2.0	1.9855	1.501E-05	2.091E-03	7.108E-10	2.106
186	E0036/2207	**1**	13	0.6	13	5	1.0	0.99275	7.504E-06	1.045E-03	3.554E-10	1.053
323	E0045/2208	**0.76**	15.96	0.7	16	4	1.2	1.221846154	8.754E-06	1.287E-03	4.363E-10	1.295
113	E0044/2208	**1.15**	29.9	1.2	30	4	2.3	2.290961538	1.501E-05	2.412E-03	8.175E-10	2.427
127	E0055/2209	**1.54**	20.02	0.7	20	4	1.5	1.527307692	8.754E-06	1.608E-03	5.473E-10	1.617
102	E0062/2211	**1.13**	25.99	1.2	26	5	2.0	1.9855	1.501E-05	2.091E-03	7.106E-10	2.106
64	E0098/2212	**0.93**	26.97	1.1	27	4	2.1	2.061865385	1.376E-05	2.171E-03	7.374E-10	2.185
189	E0097/2212	**1.14**	72.96	3.4	73	5	5.6	5.574673077	4.252E-05	5.870E-03	1.995E-09	5.913
71	E0087/2213	**1**	15	0.6	15	4	1.1	1.145480769	7.504E-06	1.206E-03	4.101E-10	1.214
260	E0086/2213	**1**	16	0.8	16	5	1.2	1.221846154	1.001E-05	1.287E-03	4.374E-10	1.297
286	E0085/2213	**1**	22	0.9	22	4	1.7	1.680038462	1.126E-05	1.769E-03	6.015E-10	1.780
101	E0103/2214	**0.93**	25.11	1.1	25	4	1.9	1.909134615	1.376E-05	2.010E-03	6.865E-10	2.024
166	E0102/2214	**1.18**	47.2	1.8	47	4	3.6	3.589173077	2.251E-05	3.779E-03	1.290E-09	3.802
228	E0120/2215	**1**	14	0.6	14	4	1.1	1.069115385	7.504E-06	1.126E-03	3.828E-10	1.133
226	E0122/2217	**0.88**	51.04	2.2	51	4	3.9	3.894634615	2.751E-05	4.101E-03	1.395E-09	4.129
247	E0127/2218	**0.87**	13.05	0.6	13	5	1.0	0.99275	7.504E-06	1.045E-03	3.568E-10	1.053
283	E0126/2218	**0.88**	21.12	1	21	5	1.6	1.603673077	1.251E-05	1.689E-03	5.774E-10	1.701
49	E0133/2219	**0.93**	13.95	0.5	14	4	1.1	1.069115385	6.253E-06	1.126E-03	3.814E-10	1.132
130	E0132/2219	**1.13**	33.9	1.4	34	4	2.6	2.596423077	1.751E-05	2.734E-03	9.268E-10	2.752
275	E0131/2219	**0.87**	13.05	0.5	13	4	1.0	0.99275	6.253E-06	1.045E-03	3.568E-10	1.052
153	E0137/2220	**0.93**	26.04	1.1	26	4	2.0	1.9855	1.376E-05	2.091E-03	7.119E-10	2.104
465	E0136/2220	**1.08**	82.08	3	82	4	6.3	6.261961538	3.752E-05	6.594E-03	2.244E-09	6.631
197	E0154/2222	**1**	38	1.5	38	4	2.9	2.901884615	1.876E-05	3.056E-03	1.039E-09	3.074
230	E0153/2222	**1.06**	36.04	1.4	36	4	2.7	2.749153846	1.751E-05	2.895E-03	9.853E-10	2.912
63	E0160/2223	**0.85**	22.95	1	23	4	1.8	1.756403846	1.251E-05	1.849E-03	6.275E-10	1.862
309	E0161/2223	**0.93**	25.11	1	25	4	1.9	1.909134615	1.251E-05	2.010E-03	6.865E-10	2.023
242	E0165/2224	**1.21**	16.94	0.6	17	4	1.3	1.298211538	7.504E-06	1.367E-03	4.631E-10	1.375
299	E0164/2224	**0.96**	66.24	2.6	66	4	5.0	5.040115385	3.252E-05	5.307E-03	1.811E-09	5.340
311	E0170/2225	**1.05**	46.2	2	46	4	3.5	3.512807692	2.501E-05	3.699E-03	1.263E-09	3.724
118	E0178/2226	**0.94**	31.96	1.3	32	4	2.4	2.443692308	1.626E-05	2.573E-03	8.738E-10	2.589
285	E0177/2226	**1.1**	147.4	4.9	147	3	11.2	11.22571154	6.128E-05	1.182E-02	4.030E-09	11.882
330	E0176/2226	**0.85**	28.05	1.2	28	4	2.1	2.138230769	1.501E-05	2.252E-03	7.669E-10	2.267
43	E0187/2227	**1.07**	16.05	0.6	16	4	1.2	1.221846154	7.504E-06	1.287E-03	4.388E-10	1.294
105	E0186/2227	**0.88**	43.12	1.7	43	4	3.3	3.283711538	2.126E-05	3.458E-03	1.179E-09	3.479
97	E0193/2228	**1.1**	22	0.9	22	4	1.7	1.680038462	1.126E-05	1.769E-03	6.015E-10	1.780
168	E0192/2228	**1.09**	25.07	1.2	25	5	1.9	1.909134615	1.501E-05	2.010E-03	6.854E-10	2.025
311	E0191/2228	**1**	34	1.6	34	5	2.6	2.596423077	2.001E-05	2.734E-03	9.296E-10	2.754
286	E0199/2229	**1.13**	16.95	0.8	17	5	1.3	1.298211538	1.001E-05	1.367E-03	4.634E-10	1.377
335	E0204/2230	**1.19**	74.97	3.1	75	4	5.7	5.727403846	3.877E-05	6.031E-03	2.050E-09	6.070
417	E0203/2230	**0.22**	2.42	0.08	2.4	3	0.2	0.183276923	1.001E-06	1.930E-04	6.616E-11	0.194
323	E0208/2231	**1.09**	23.98	1.2	24	5	1.8	1.832769231	1.501E-05	1.930E-03	6.556E-10	1.945
141	E0215/2232	**1.08**	12.96	0.5	13	4	1.0	0.99275	6.253E-06	1.045E-03	3.543E-10	1.052
285	E0214/2232	**1.14**	15.96	0.7	16	4	1.2	1.221846154	8.754E-06	1.287E-03	4.363E-10	1.295
45	E0222/2233	**1.12**	29.12	1.3	29	4	2.2	2.214596154	1.626E-05	2.332E-03	7.961E-10	2.348
201	E0221/2233	**1.06**	36.04	1.5	36	4	2.7	2.749153846	1.876E-05	2.895E-03	9.853E-10	2.914
14	E0231/2234	**1.07**	14.98	0.7	15	5	1.1	1.145480769	8.754E-06	1.206E-03	4.096E-10	1.215
229	E0228/2234	**1.24**	26.04	1.2	26	5	2.0	1.9855	1.501E-05	2.091E-03	7.119E-10	2.106
37	E0237/2235	**1.27**	13.97	0.7	14	5	1.1	1.069115385	8.754E-06	1.126E-03	3.819E-10	1.135
260	E0236/2235	**0.96**	24	1.1	24	5	1.8	1.832769231	1.376E-05	1.930E-03	6.562E-10	1.944
365	E0241/2236	**1.03**	30.9	1.4	31	5	2.4	2.367326923	1.751E-05	2.493E-03	8.448E-10	2.510
209	E0249/2237	**1.05**	23.1	1.2	23	5	1.8	1.756403846	1.501E-05	1.849E-03	6.316E-10	1.864
89	E0256/2238	**0.94**	15.04	0.7	15	5	1.1	1.145480769	8.754E-06	1.206E-03	4.112E-10	1.215
136	E0255/2238	**1.41**	38.07	1.6	38	4	2.9	2.901884615	2.001E-05	3.056E-03	1.041E-09	3.076
32	E0263/2239	**1.08**	7.668	0.31	7.7	4	0.6	0.588013462	3.877E-06	6.192E-04	2.096E-10	0.623
60	E0262/2239	**1.13**	18.08	0.84	18	5	1.4	1.374576923	1.051E-05	1.447E-03	4.943E-10	1.458
113	E0260/2239	**1.12**	28	1.2	28	4	2.1	2.138230769	1.501E-05	2.252E-03	7.655E-10	2.267
75	E0266/2241	**1.06**	34.98	1.5	35	4	2.7	2.672788462	1.876E-05	2.814E-03	9.564E-10	2.833
77	E0265/2241	**1.08**	28.08	1.3	28	5	2.1	2.138230769	1.626E-05	2.252E-03	7.677E-10	2.268
320	E0269/2242	**0.97**	33.95	1.3	34	4	2.6	2.596423077	1.626E-05	2.734E-03	9.282E-10	2.750
35	E0278/2243	**1.05**	39.9	1.5	40	4	3.1	3.054615385	1.876E-05	3.216E-03	1.091E-09	3.235
94	E0275/2243	**0.88**	14.96	0.5	15	3	1.1	1.145480769	6.253E-06	1.206E-03	4.090E-10	1.212
371	E0274/2243	**1.15**	63.25	2	63	3	4.8	4.811019231	2.501E-05	5.066E-03	1.729E-09	5.091
165	E0281/2244	**1.15**	14.95	0.6	15	4	1.1	1.145480769	7.504E-06	1.206E-03	4.087E-10	1.214
52.1	E0282/2245	**0.95**	18.05	0.9	18	5	1.4	1.374576923	1.126E-05	1.447E-03	4.935E-10	1.459
34.9	E0285/2247	**0.91**	20.93	0.9	21	4	1.6	1.603673077	1.126E-05	1.689E-03	5.722E-10	1.700

[1] Note V grade indicated as an indication of visual redox context

SRK notes that the mineralization at Madaouela is reduced, has not been exposed to secondary oxidation/mobilization by groundwater, and is therefore not likely to have any disequilibrium issues on a global deposit basis. Rare presence of yellow products in drilling chips was observed northwest of Marilyn (Martine sector) associated with very shallow mineralization (mineralization that has been oxidized to U (VI) minerals). No mineralization of this area is included in the resource.

GoviEx estimations are based on Grade/Radioactivity conversion built from dedicated core sampling carried out initially on Marianne. The relationship has been controlled now for Marilyn, the group MSNE, MSCE and recently Miriam. As mentioned above the Marianne relationship appeared relatively conservative, and has consequently been kept for all deposits.

10.3 Quality Controls and Quality Assurance ("QA/QC")

10.3.1 Radon Survey

All mineralised holes are relogged within 3 days after completion for Radon control. This procedure, derived from CEA Instructions Techniques is very important, especially:

- in drilling phases of high drilling rate;
- in the case of well mineralized holes.

10.3.2 Chemical Assays

GoviEx inserted a small number blanks at random interval within their sampling programme, and relied solely on the internal lab QA/QC of ALS-Chemex for the quality of their data. Since chemical assay results were not used directly as part of the resource model, GoviEx did not need to implement a detailed QA/QC programme of the assay results with standard, blanks and duplicates. The quality of the radiometric data was backed up by other means as described elsewhere in this chapter.

10.3.3 Radiometric Determinations

GoviEx has implemented a significant program of data checks and calibrations to insure accuracy and comparative determination of eU from the various probes in use at Madaouela.

The quality assurance and quality control on the logging equipment and on the recorded logs is continuous and involves two sets of control:

1. Equipment control: the counting rate of probes is regularly checked for evidence any bias or counting loss. The controls are of two types.
    a. Static mode: control checks daily against calibrated sources. Once daily a probe is checked at a dedicated calibration pad on-site, where a cesium source is places along-side the probe for a 20 minute period to verify stability and daily repeatability of the gamma readings.
    b. Dynamic mode: All probes are regularly checked by comparison with the log from a Mount Sopris gamma probe (DTH 27 reference probe); which has been done for approximately 25 % of the drillhole data to-date. GoviEx schedules a minimum of one in every 20 holes for re-logging with the reference probe.

2. Record control: The controls consists of:

    a. Re-logging holes with a reference probe and/or with another SEMM logging unit (identical to item “1.b” above). Data logs are compared for accuracy of depth, grade, thickness, and GT of mineralization.
    b. Re-logging holes after three days (Radon half-life period). This is done to determine if radon is an issue in the radiometric determinations; which has been determined thus far to be on minimal consequence.

All data is reviewed and plotted by GoviEx to determine if any radiometric probe is out of calibration or faulty. Any holes in question are re-logged. Any probes in question result in the hole being re-logged by another probe for all holes logged that day. Figure 10-3 is a sample plot of the calibration check for one gamma probe demonstrating minimal (± 2 %) daily variance in CPS counting.

A Tubular SEMM reference source is used for probe calibration checks on a daily basis. The source was built with stabilized crushed ore from a French mining site. The ore grade is not precisely given, but the activity of the source is moderate to high. The counting rates of this source for the various probes in use at Madaouela are shown in Table 10-4.

Table 10-4: Reference standard counting rate on SEMM source

	SEMM Source 15O ms Sampling	
Probe	**Average Geiger**	**Average Scintillometer**
BDGG 8101	32.42	4132.44
BDGG 8102	30.40	4001.72
BDGG 8103	31.63	4120.11
BDGG 828	32.35	4039.54
BDGG 829	32.05	4012.32
BDGG 830	31.99	4179.17
GUIP 8111	na	3951.89
GUIP 8112	na	3798.34
GUIP 8113	na	3847.11
GUIP 8114	na	3971.14
GUIP 842	na	3804.79

Additional static radioactive control sources were purchased from CERCA LEA, France. The type and certificate references of each source are summarized in Table 10-5.

Table 10-5: Caesium reference sources

Elem	Reference	N° id	Ext dia.	Sealed Sources Classification	Activity kBq	Ext. Relative Uncertainty (%k=2)
^{137}Cs	CS137EGSB15	50547	38mm	C22212	38	5
^{137}Cs	CS137EGSB15	50548	38mm	C22212	38.4	5
^{137}Cs	CS137EGSB15	50549	38mm	C22212	36.6	5
^{137}Cs	CS137EGSB15	50549	38mm	C22212	39	5
^{137}Cs	CS137EGSB20	50551	38mm	C22212	401	5
^{137}Cs	CS137EGSB20	50552	38mm	C22212	397	5
^{137}Cs	CS137EGSB20	50553	38mm	C22212	400	5
^{137}Cs	CS137EGSB20	50554	38mm	C22212	400	5

Source: CERCA LEA, 2008

The radioactive caesium sources are fitted into a special housing and the probe is laid against it for measurement at a dedicated calibrations pad. All radioactive sources are stored at the Project logging/calibration site inside a locked and secure facility.

10.3.4 Independent Verification

In order to have an independent validation of GoviEx geophysical logging results, ULC, an independent geophysical logging company was appointed to initially log about 60 drill holes and to repeat the log measurements and compare the results with those used for the resource estimate. This initial work was completed between October and December 2009, using their equipment (Geovista) and personnel. This verification work showed the reliability of GoviEx logging teams and units, and confirmed the repeatability of the logs. ULC continued to supply logging services up to the closure of work sites in 2013. A total of 343 holes were logged, with excellent comparisons with GoviEx results.

Table 10-6: Summary of ULC logging

Number	Sector	Year
13	MAD1	2011-2012
2	MAD4	2011-2012
171	MSNE	2011-2012-2013
45	MARI	2008-2009-2011-2012-2013
57	MARL	2009-2011-2012-2013
55	BSRK	2012-2013

10.4 Interpretation (SRK)

SRK concludes that GoviEx's sample preparation, methods of analysis, and sample and data gathering are being implemented with a high degree of care in data collection, data transfer, data conversion, and gamma probe QA/QC. The methods are acceptable by industry-standard procedures and are applicable to the uranium deposits at the Madaouela Uranium Project.

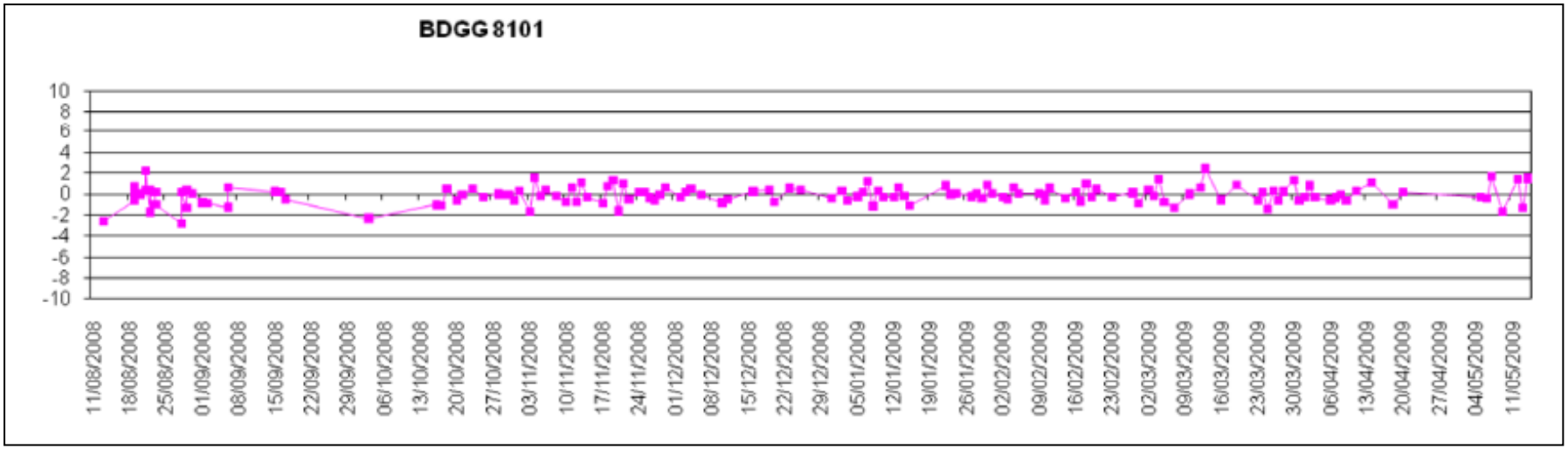


Figure 10-3: Example daily plot of % variance in CPS for probe BDGG 8101

11 DATA VERIFICATION (ITEM 12)

11.1 SRK Data Verification

SRK data verification has involved the following:

- Database verification;
    - Spot checks of a number of drillhole gamma logs for grade determination, thickness, and depth of mineralization;
    - Examination of the process of data acquisition, data transfer, and data conversion;
    - Verification of drillhole locations as plots; and
    - Verifications of uranium deposit shapes, average grades, and thickness of mineralization as previously determined by GoviEx.
- Visual data verification;
    - Verification of anomalous radioactivity in drill cuttings corresponding to mineralized intervals, and mineralized stockpiles from former CEA underground;
    - Visual confirmation of lithological designations;
    - Visual confirmation of surface geology and controls to mineralization; and
    - Visual verification of historical and current drillhole collar locations and documentation; confirmed as identified on maps.

SRK did not determine any variance from the GoviEx determinations of mineralization, or geology. Therefore, SRK did not verify the grade and thickness determinations in all drill holes in the database.

SRK did not independently collect samples for assay determinations of uranium content, as radiometric confirmation minimized the need to verify presence of uranium mineralization by chemical assays.

11.2 GoviEx Data Verification

11.2.1 Database Structure

The drilling database is hosted in CORALIS database structure was designed by Coralis to GoviEx specifications.

Working databases continue to be active in their hosting software, they are:

WellCad software is comprised of WellCad and Excel files. WellCad data is transferred into Utimine during drilling.

Utimine is software that was developed for GoviEx to perform geological interpretations and correlations. Utimine calculates all mineralization data (accumulations) and is user-friendly software to interpret logs, sections, and maps. The software offers several functions that are needed in uranium exploration and development. Management control of daily progression of drilling is performed through Utimine with sectional interpretations. Utimine contains the databases and all geostatistical modeling tools necessary to estimate resources.

Mapping data are in Utimine and in MapInfo. MapInfo hosts all map data from external sources as well as updates of practical maps to follow every day development and drilling program.

Export/Import from/into Utimine can take place in pre-formatted text files include the following: Excel (csv, pin), Surpac, Gemcom, Vulcan, GoCad, Isatis.

SRK noted that the entire database has multi-person checks during the WellCad and the Utimine data process stages. Multiple geologists will have input to the logging, data entry and resulting sectional interpretations.

- While each geologist is responsible for his own daily geological logs, the drillholes will need to correlate lithologically into the database of existing drillholes;
- Geologist import their geological logs into excel and WellCad on a daily basis, and the data gets validated by the geologist in charge of stratigraphic data;
- Drillhole data is transferred from WellCad to Utimine by geologists dedicated to data entry, and the inputs are verified by a second geologist. The data is verified in cross-section to validate the lithology picks. The Chief geologist also examines the data inputs for visual errors;
- The project geologist and Chief geologist also verify geological determinations as a final check daily.

11.2.2 Collar Co-ordinates

The survey database integrity and homogeneity is verified through the following procedure:

- Drill collar elevations are plotted on a topographic map interpreted from all data. All data points that appear as surface spikes are re-surveyed. Re-surveying is carried out on each abnormal point and all neighbouring points;
- Trace of the surface is also visualized in sections, and any particular surface feature that seems abnormal is re-checked in the field; and
- A certain amount of historical holes on Mad I are still plotted after approximate reconversion of their coordinates, which are derived from map documents. None of those holes are included within the resource areas, unless they have been re-drilled and surveyed.

11.2.3 Drillhole Deviation

Drillholes are all vertical by design. The angle deviation that may occur during drilling is measured with initial drillhole logging. Verification of the deviation measuring equipment reliability was conducted as follows:

- In a limited number of drillholes, cross checks were carried out with the deviation device attached to the DHT27 reference probe;
- If suspicious probe deviation records are indicated, the equipment is checked on surface by pulling the probe on the ground along a marked line to verify the measurement are correct; and
- Every 20 to 50 holes, randomly selected holes are tested again for deflection using one of the other probes.

11.2.4 Geological Logs

Geological logs are subjected to a series of verifications:

Depth Errors

Logging depth is corrected for resistivity and scintillometer records by the probe manufacturer. Strict zero collar point in resistivity and gamma logging results in marginal differences

between the geological depth and the gamma depth. For estimation purposes, the depth as measured by the gamma probe is used as the depth in the database.

Some visible errors occur in total depth lithology picks, as geologists often note the depth rounded to the number of chip sample intervals below the RT geological marker. A final correction is done to the correct depth from gamma probe data.

Coding Errors

The applied field procedures are as follows:

- Compare geological with geophysical logs insuring depth reporting of lithology is logical;
- Rechecking in office of field data and control of main stratigraphic breaks. Any subsisting doubts are verified in the field the following day with the chief geologist;
- Re-control of depths by entering data into Excel and WellCad; and
- Converted WellCad logs with deviated trace are plotted in cross sections and lateral homogeneity is verified with neighbouring holes.

Additional validation occurs when interpreting sections and maps. Prior to a revision of the coding, in most cases the geologist revisits the chips on the drill site, as due to the dry climatic conditions, chips are preserved sometimes for years at the drill site. The coding revision is then defined with respect to neighbouring drillhole data.

11.2.5 Gamma Logs

Gamma log verifications are discussed in Section 10 (Radiometric Determinations); a discussion of QA/QC procedures for gamma logging. Those QA/QC procedures include:

- Duplicate determinations (re-logging with the same probe, if necessary);
- Replicate determinations (re-logging with a second probe or the reference probe); and
- Third Party determinations, equivalent to third party lab analyses (re-logging of a percentage of the drillholes by and independent contractor) has been done as planned, reporting is still outstanding.

All Data verification and QA/QC procedures have been compiled into a document by GoviEx staff.

11.3 Conclusions (SRK)

SRK reviewed the field procedures which are essentially all drilling related, and include the following:

- Rotary mud drilling;
- Selected Core drilling of mineralized horizon;
- Drillhole gamma-logging and resistivity-logging procedures including local gamma probe calibration measurements;
- Geological logging of cuttings;
- Core logging and hand-held radiometric procedures;
- DGPS drillhole collar surveys; and
- Hole plugging, capping and documentation.

SRK concludes that the field procedures are appropriate and adequate for developing a drillhole database sufficient for resource estimation. Furthermore, the process includes many data checks and verifications to assure the data is accurate. Multiple QA/QC procedures are

in place to validate the data as it is acquired. The geological staff are sufficiently trained and instilled with the need for data verification, and multiple geologists examine the data inputs into the database. Therefore, the procedures in place allow for proper data verification with minimal database errors.

Of particular note are the following QA/QC procedures noted by SRK that contribute to the overall integrity of the database:

- Individual gamma probes are calibrated against the DHT 27 reference probe to insure all probes are reading the same;
- Drillholes with greater than 1.5 GT are re-logged after three days to check for radon build-up (which has been determined to be negligible);
- Approximately ¼ of the mineralized holes are re-logged with the DHT 27 reference probe as a check;
- Two probes are run for each hole, one with gamma by Geiger tubes, and one with gamma by scintillation counter, which provides a check;
- Scintillation logs are used for depth correlation and for lithology picks as a check against geological picks from cuttings;
- Each probe in calibration checked against a standard (Caesium source) for 20 minutes at an above ground calibration station each day to verify accuracy;
- Geology of the drill chips, collected on 1.0 m intervals, is logged with the assistance of the resistivity logs as a check;
- No historical drillhole intercept data is used – all historical holes use have been re-drilled and re-logged; therefore, all the data in the database is GoviEx collected gamma log data and internally consistent;
- Geology logs in the field are entered into Excel spread sheets and the Geologist imports the data into WellCad with the geophysical logs on a daily basis, and can make visual QA/QC checks at that time for any errors;
- The conversion of log data by the geologist is then checked by another geologist for errors and for correlations with adjacent drillholes in cross-section in WellCad prior to conversion from CPS to eU% which is done in Utimine software;
- A final check of the drillhole data is done by Chief geologist as new drillholes are entered into the database; and
- Drill core correlations with gamma logs are carefully done at the core logging site in the field. Core is photographed and radiometrically scanned with a hand-held counter to verify and compare mineralized core intervals with the gamma logs, prior to cutting and quartering for assay on 30 cm intervals.

SRK therefore concludes that GoviEx has in place sufficient QA/QC and database verification procedures to render the drillhole database consistent, verifiable, and appropriate for use in resource estimation.

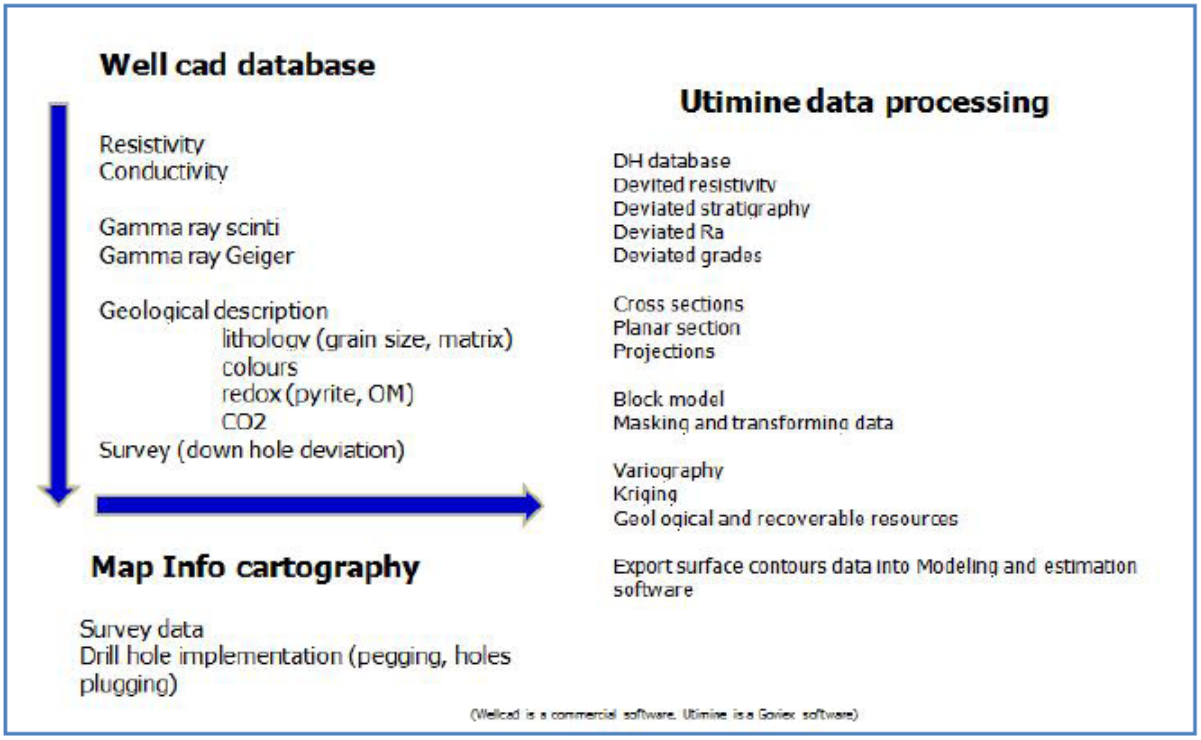


Figure 11-1: GoviEx Madaouela database structure and organization

12 MINERAL PROCESSING AND METALLURGICAL TESTWORK (ITEM 13)

12.1 Historical Works

As part of the CEA evaluation in 1965 a sampling station was set up underground in the Marianne deposit and 2,000 t of mineralized material was extracted and stockpiled into two piles (high and low grade) on site (currently situated in the military camp). The main purpose of the sampling station was to evaluate the mineralization grade from underground to compare it with the results of drillhole samples and gamma-log measurements. Their intention was certainly to perform some mineral processing tests, but no records of this work could be found in the archives.

Near-by mines have been extracting uranium using an acid leach mill recovery process since the 1970's, and Madaouela mineralization is similar mineralogically; although, Madaouela mineralization is potentially higher in carbonates and lower in average grade.

A specimen of mineralization, taken from the CEA stockpiles of underground extracted mineralization, was subjected to preliminary tests.

12.2 GoviEx Studies

GoviEx completed preliminary mineralogical studies at the School of Mines, St Etienne, France on core samples in selected parts of the deposit where potentially economic extraction may be possible, a summary of the results can be found in the previous SRK Preliminary Economic Assessment ("PEA") report (SRK Consulting, 2011).

In addition testwork was undertaken during the scoping study on core samples at SGS Lakefield (SGS Minerals Services, 2010) and (SRK Consulting, 2011), and included in the summaries below.

12.3 Methodology

An 8 tonne metallurgical sample was collected in 2011 from existing high grade ("HG") (4 t), low grade ("LG") (2 t) and waste rock material ("WR") (2 t). Concern over representativity led to detailed evaluation of this ore material as being representative of the Madaouela ore. Comparison in mineralogy, metallurgy and uranium deportment showed consistency with previous metallurgical studies on Madaouela drill core (SRK, 2012). As such the sample is considered suitable for evaluation. This bulk sample is considered acceptable to simulate the potential feed from underground operations and overall grade was determined to be approximately 0.15 - 0.16 % U_3O_8. GoviEx staff collected the sample on site.

Expanding on the previous testwork undertaken as part of the feasability studies the following activities were carried out to further develop the current understanding and to help establish the mineral processing requirements for the Madaouela Uranium Project:

- Mineralogical studies for further characterisation of high grade GoviEx Ore samples;
- Mineralogical studies for characterisation of Miriam ore and leach residue samples;
- Acid pugging testwork on high grade ("HG") ore samples and core samples (conducted by SGS minerals) to determine whether improved uranium extractions can be achieved by curing the ore with high strength sulfuric acid and an oxidant(s), before a weak acid leach;
- Diagnostic acid leach on HG ore samples to establish the susceptibility of uranium within

the ore samples to further extraction by acid leaching;

- Bottle roll tests to provide an estimate of possible uranium extraction, acid consumption and uranium extraction kinetics;
- Column leach testwork to provide an understanding of the potential performance of a heap leach processing circuit for uranium extraction;
- Beneficiation: pre-assessment steps can be incorporated upfront of any leaching circuit (heap leach, tank leach etc.) to increase the uranium content in the head feed and reduce the level of acid consuming gangue minerals in the plant feed. The beneficiation process reduces the throughput requirement in the leaching circuit for a given uranium mass recovery and can reduce operating costs by reducing the amount of acid required during leaching. Four methods were assessed:
    - Radiometric Ore Sorting ("ROS") that relies upon uranium detection (discarding low uranium concentrations below a set cut-off point);
    - Scrubber testwork (based on particle size pre-concentration) was conducted on the GoviEx ore;
    - Desliming and flotation of a milled product to selectively remove sterile gangue carbonate minerals;
    - Ablation separates the non-uranium bearing host rock from the ore and reduces the throughput requirement of the leaching process.
- Downstream pregnant leach solution ("PLS") processing: - several methods were assessed to determine the most appropriate downstream PLS processing option for uranium recovery:
    - Ion exchange;
    - Solvent extraction, including:
        - conventional commercially available reagents;
        - an extraction reagent capable of extracting Mo in addition to U, enabling the recovery of a Mo product and lower impurity U product.

By assessing the performance of the above activities conceptual flow diagrams can then be developed and optimised using the best practicable combination of options in terms of uranium extraction/recovery. This conceptual flowsheet can then be further refined taking cognisance of: financial considerations (capital and operating costs); regulatory requirements; physical location constraints; climatic conditions; tailings deposition requirements, water consumption etc.

12.4 Mineralogy

Section 12.4.1 provides a summary of two phases of mineral characterisation carried out by Mintek on grab samples of waste rock and ore samples collected from the high grade and low grade ore stockpiles located in the former mine camp. Section 12.4.3 outlines the mineral characterisation of diagnostic leach residues conducted on GoviEx ore samples (Mintek, 2012).

12.4.1 Primary Ore Mineralogy (Mintek)

Phase one of the Mintek testwork used petrography and qualitative automated scanning electron microscopy (AutoSEM) to identify the mineralogy of three bulk samples of the low grade ore, high grade ore and waste rock. As these samples were obtained from stockpiles that have been exposed to the environment for approximately 50 years the amount of

weathering and oxidation of the samples was also determined.

Phase two used quantitative X-ray diffraction ("XRD") and AutoSEM utilizing electron microprobe analysis to determine uranium-bearing minerals for the high grade ore sample in terms of speciation, grain size distribution and mineral associations.

Low Grade Ore

The low grade ore sample examined in phase one of the (Mintek) study was determined to consist of two generations in which sediment was deposited. The earlier formed spherical "concretions" comprising impure sandstone in a darker dolomitic carbonate matrix, with goethite and limonite cementing the clasts and forming a thin reddish brown rim around the outer surface. This limonitic goethitic layer is coated with a layer of gypsum. The younger sediment is quartzitic with a chert matrix. Limonite and goethite form in thin oxidised layers between some of the bedding planes and also occur on the outer surfaces.

Pyrite is present in fine grains < 1 % by volume. The LG ore sample contains: brannerite; uraninite; a U-clay mineral; and U-leucoxene forming fine grains or wisps within the matrix of the sandstone, it should be noted however that the composition was determined by AutoSEM. Figure 12-1 illustrates the earlier centres of sandstone spheres where dolomite forms the matrix (left), and is rimmed by a zone of gypsum (middle) and the younger sandstone is cemented in a chlorite matrix (right). A rim of gypsum (green) surrounds the sphere of earlier formed sandstone and this is where the uranium-bearing mineral phases are mainly situated.

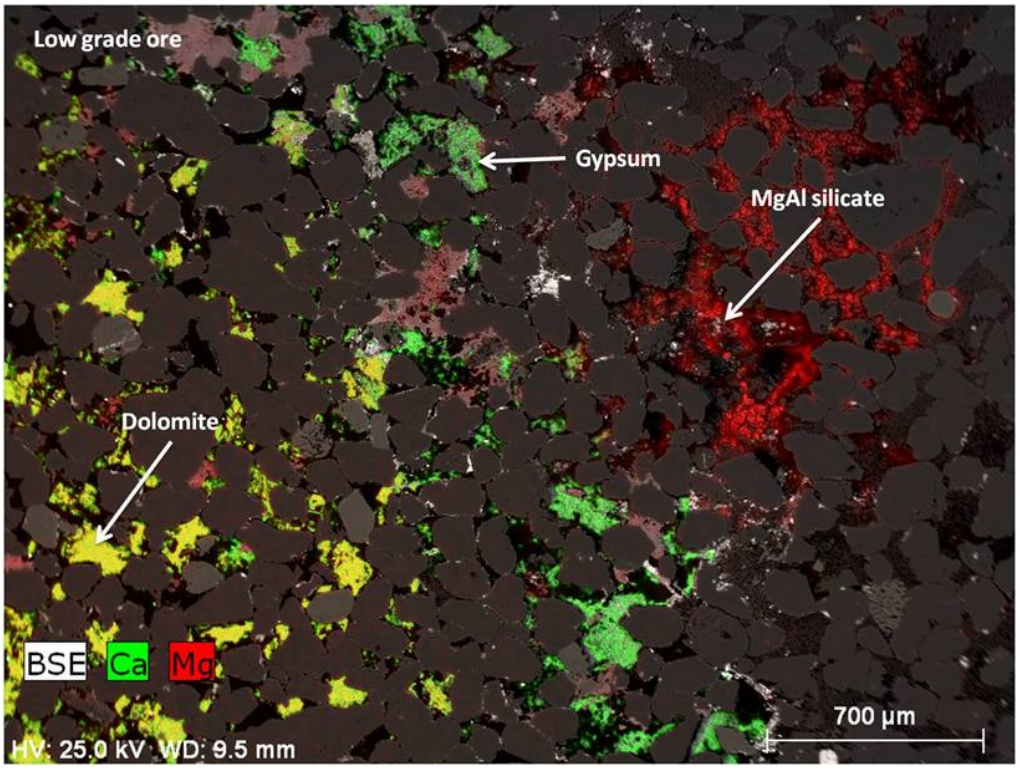


Figure 12-1: An element map of the low-grade sample. Scale bar = 700 µm

High Grade Ore

Phase one of the Mintek study determined that the high grade ("HG") ore utilizing AutoSEM analysis showed that the sample contained U-clay mineral, uranophane and U-leucoxene with uranophane contributing 50 % to the total uranium budget. Phase one indicated that

brannerite type minerals had been confirmed, however the more extensive phase two testwork concluded that these occurrences are in fact both fine intergrowths of pitchblende/silica mixed with TiO_2 or coffinite mixed with TiO_2, with very minor amounts of true brannerite, as orthobrannerite.

The most abundant uranium-bearing minerals in the sample are "pitchblende/silica-mix-TiO_2", "coffinite-mix-TiO_2" and coffinite. Uranite and autunite are less abundant. The "pitchblende/silica-mix-TiO_2" and "coffinite-mix-TiO_2" phase contribute 65 % of the total uranium. The uranium-bearing minerals in the high grade ore mostly report to the 10-25 µm size fraction. Phase two of the testwork concluded that the majority of uranium bearing minerals exists as U (IV), the unoxidised form of the element.

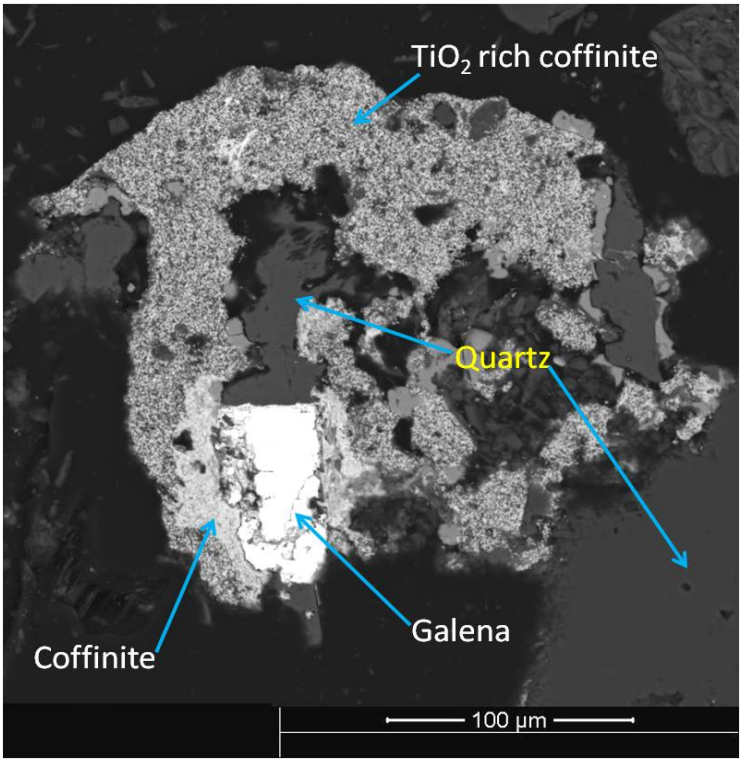


Figure 12-2: BSE image of the fine grained intergrowths of coffinite with TiO_2.

The bulk modal mineralogy analysis was carried out by Mintek and an external lab to provide a QA/QC check on the results obtained.

The sample consists of predominately quartz followed by K-feldspar (in the form of microcline, 3.7 mass %). Plagioclase was not detected in analysis one. The quantity of kaolinite is similar to K-feldspar (3.5 - 3.7 mass % and 5.2 - 5.5 mass % for analysis one and two respectively). Chlorite and muscovite were not detected in analysis one. Small quantities of Ca bearing phases (calcite, dolomite and gypsum) were noted together with plagioclase, muscovite, chlorite and hematite. The mineral illite, has a crystal structure very similar to muscovite and as very little muscovite is identified in the form of plates in the petrographic analysis, it is assumed that most of the muscovite identified by XRD is the clay mineral illite.

The different quantities of kaolinite and muscovite (illite) in the XRD spectra, indicate that these clay minerals are not ubiquitous, and occur sporadically as alteration products, within microcline and plagioclase clasts in the sandstone in <5 mass %. The clay minerals kaolinite and illite are therefore not expected to have a major effect on the percolation of the lixiviant through the uraniferous sandstone particles, owing to their heterogeneous dispersal in the

sample.

The petrographic analysis shows that the matrix of the high-grade sample is predominantly carbonate which is likely to consume high amounts of acid.

Waste Rock

The waste rock sample contains U-clay mineral, U-leucoxene, brannerite and uranophane. Uraninite is not present in the waste rock sample. This sample shows very little oxidation. The occurrence of clay minerals /chlorite can influence the leaching process by preventing the flow of the lixiviant through the sample. The presence of known acid consumers carbonate and gypsum was also noted in the samples.

Field Observations (SRK)

Surface exposure of the Guezouman sandstone reveals high permeability along the fault zones and bedding planes is relatively high as evidenced by the strong bleaching and hematization associated with faults and bedding planes. When fresh surfaces are viewed however the frequent carbonate-dominated cement indicates a low effective porosity in the uranium-bearing horizons. This was tested in the field by pouring water on a freshly exposed broken face. There was minimal infiltration indicating a low effective porosity.

Further evidence for low porosity was observed in the field with respect to the depth of the redox front. At a drillhole location (MARL-34-01) the redox front is at approximately 22 m depth. Given that this hole was less than 200 m from the start of the outcrop of the same horizon in the Guezouman sandstone this indicates that connected porosity in the sandstone is limited. This is most likely a reflection of the carbonate cement in the sandstone that reduces effective porosity and limits migration of fluids through the unit.

Observations on the Historic Mineralization Stockpile (SRK)

Two historic stockpiles exist, situated in the former mine camp, currently an army base at Madaouela. The lower grade material contains approximately 1,200 t, while the higher grade pile contains approximately 800 t.

Both stockpiles reveal the reduced “green colored” nature of the uranium-bearing horizon. Relic wood fragments and organic carbon are evidenced in both along with sulfide minerals as well as siderite-hematite nodules after marcasite. In the partially to wholly oxidized samples iron staining is common and the dolomite cement is more apparent. As with the core low effective porosity was indicated from the abundant carbonate cement in the material and by the lack of depth to surface oxidation on mineralization pieces. Despite 50 years of weathering, the only iron alteration to hematite or formation of uranium (VI) minerals was on fracture surfaces or exposed weathered faces on the stockpile and even here black uranium U (IV) minerals and sulfides were still in evidence. Consequently, the material appears acceptable as an analogue for the untapped mineralization for purposes of metallurgical testing of the deposit.

12.4.2 Geometallurgy (SRK)

In terms of processing the important aspects that have been realized from the field examinations, initial and subsequent mineralogy, and metallurgy testwork undertaken, are:

- Uranium is present as coffinite (60 %) and uraninite (40 %) with negligible other phases such as uranium (VI) weathered surface coatings, and so on. There is no evidence of brannerite-group minerals occurring in the deposits currently sampled for the proposed underground operations;
- Uranium minerals are present as fine grained, typically less than 30 µm with occasional grains up to 100 µm in size, minor to brannerite;
- The mineralogy of the deposits changes across the license area (Section 12.4);
- Uranium minerals dominantly occur as fine grained constituents of coffinite-uraninite-siderite-sulfide replacement assemblage of clay minerals and dolomite in the cement of the quartz-feldspar sandstone lenses;
- Molybdenum largely occurs as a trace element in pyrite or coffinite with less than 20% occurring as molybdenite (MoS_2) or powellite ($CaMoO_4$);
- The majority of samples have carbonate contents dominated by dolomite (up to 3 %) with minor siderite and possible ankerite, none of which are efficient acid consumers;
- Minor organic carbon as wood fragments, often replaced by sulfide minerals and/or oxidized;
- Process needs to focus on separation with minimal over-grinding in order to avoid high viscosity in agitated leaching and slow solid/liquid separation rates;
- Evaluate the potential processing advantages of RADOS/ROS beneficiation, ablation and Mo solvent extraction.

The mineralogical studies carried out on the testwork leach residues show clearly that they are mainly composed of quartz and clays minerals, the Miriam residues contained U-Ti minerals. Low U recovery during the classical leaching tests seems to result from:

- grains of pyrite or Fe-Ti-oxides with isolated uranium concentrations that are associated with phosphorus phases and are presumably not accessible for leaching;
- uranium with clay minerals, indicated by high aluminum concentrations in the EDX spectra, locking uranium in the matrix between quartz grains; and
- trace brannerite minerals present in some of the Miriam ore samples.

12.4.3 Residue Mineralogy

Residue mineralogy was carried out on four residue samples resulting from four diagnostic leach tests carried out on the high grade ore stockpile from Madaouela. The test conditions are detailed in Table 12-1 and the full test methodology and results are contained in the Mintek report, report number 6267 (Mintek, 2012).

Table 12-1: Samples received and test parameters

Sample label	Sample grind size p80 (µm)	Temperature leached (°C)	U_3O_8 extraction	MNL traceability number	Report short name
Leach residue test 4	500	80	71 %	IMO/12/211	Residue T4
Leach residue test 6	500	60	74 %	IMO/12/212	Residue T6
Leach residue test 7	60	60	72 %	IMO/12/213	Residue T7
Leach residue test 8	60	60	27 %	IMO/12/214	Residue T8

The majority of the uranium minerals present in the residues for tests seven and eight were < 10 µm with the U-rich titanate exhibiting the largest size distribution. The uranium minerals

for tests four and six (larger sample grind size (500 μm) had a larger size distribution, with U-rich titanate in particular having > 45 % of uranium minerals greater than 10 μm for both tests.

For all the residues the majority (> 65 %) of Ti-rich coffinite minerals were locked (i.e. 0 % of the uranium mineral grain shows an exposed perimeter) and the majority (> 60 %) associated with quartz. For the majority of the residues coffinite was partially exposed (> 60 %) the exception was Test 7 where only 42 % was partially exposed and the remaining 58 % was locked. Residues from the testwork seven and eight exhibited high percentages of liberated U-rich titanate indicating that this mineral could be refractory under these leach test conditions.

12.4.4 Miriam Mineralogy

As part of the assessment of ore mineralogy for metallurgical and processing implications for the Madaouela project, seven samples of the ore from the Miriam prospect were submitted for mineralogical assessment completed by SRK. These were selected to assess the speciation and textures of the uranium-bearing minerals, and in particular what implications these have for potential process plant design. This assessment forms part of the wider metallurgical and process plant study at Madaouela. Three U bearing samples were submitted to SGS Lakefield, two residues that were generated during the SGS bottle roll testwork and a *Miriam 2013 Met Comp* sample. The *Miriam 2013 Met Comp* composed of a blend of two drums of half core material received from the Miriam deposit and blended as per SRK instructions.

The lithologies that were submitted consisted of arenites with high percentages of quartz and only minor to trace concentrations of other minerals such as interstitial kaolinite, calcite and occasionally pyrite. Calcite was a consistent minor mineral (1 – 10 %) for each sample apart from one, where it was a major constituent (> 10 %). The presence of this calcite will have implications for conventional acid leaching process as it will consume some of that acid reducing the efficiency of uranium extraction.

The main uranium minerals observed were coffinite and becquerlite with the occasionally observation of silica-mix-TiO_2 minerals, uraninite and uraniferous clay. Generally the uranium minerals were fine to medium grained but also arranged as composites that were coarse-grained. Texturally, they were located in the interstitial space between the quartz grains and closely associated with pyrite and/or calcite. Although some uranium particles were very fine-grained there was no observed encapsulation within quartz grains that is a favourable observation for conventional extraction processes.

In summary, the deportment of uranium-bearing minerals primarily as medium to coarse-grained composites in interstitial textures and as generally readily leachable minerals is favourable to conventional leaching processes. However, there are some factors that may present a concern for uranium extraction. These are:

- the presence of calcite which will increase acid consumption;
- the trace deportment of some of the uranium as orthobrannerite which is a refractory grain resistant to leaching;
- the minor presence of uranium as solid solution within clay-type particles which will likely be very difficult to recover; and
- the possible poor liberation of the small amount of very fine-grained uranium particles which do not form part of a larger medium or coarse grained uranium-bearing composites. The summary of minerals detected in the samples can be found in Table 12-2.

Table 12-2: Summary of minerals found in thin section and polished block

		Sample ID	SRK3601	SRK3602	SRK3603	SRK3604	SRK3605	E124 4818 Low	E124 4818 High
		Lithology →	Arenite	Arenite	Arenite	Arenite	Arenite	Arenite	Calcareous Arenite
		Ideal chemistry ↓							
Silicates	Quartz	SiO_2	XXX	XXX	XXX	XXX	XXX	XXX	XXX
	Zircon	$ZrSiO_4$				X		X	X
Uranium-Bearing Minerals	**Orthobrannerite**	$U^{4+}U^{6+}Ti_4O_{12}(OH)_2$		X					
	Becquerlite	$Ca(UO_2)_6O_4(OH)_6.8H_2O$	X		X	X	X		X
	Coffinite	$(U^{4+},Th)(SiO_4)_{1-x}(OH)_{4x}$	X	X	X		X	X	X
	Uraninite	UO_2				X			
Oxides	Rutile	TiO_2		X					
Feldspar Grains	Albite - Andesine	$NaAlSi_3O_8$ - $(Ca,Na)[Al(Al,Si)Si_2O_8]$	X	X	X	X	X	X	X
	Microcline	$KAlSi_3O_8$	X	X	X	X	X		X
Phosphates and Sulfates	Fluorapatite	$Ca_5(PO_4)_3F$	X						
	Baryte	$BaSO_4$		X					
Clay Minerals	Phlogopite - Annite	$K(Mg,Fe)_3(AlSi_3O_{10})(OH,F)_2$		X				X	
	Clinochlore - Chamosite	$(Mg,Fe^{2+})_5Al(AlSi_3O_{10})(OH)_8$						X	
	Illite - Muscovite	$K_{0.65}Al_{2.0}[\,][Al_{0.65}Si_{3.35}O_{10}](OH)_2$							
	Kaolinite	$Al_2(Si_2O_5)(OH)_4$	XX	XX	XX	XX	XX	XX	XX
Sulfides	Pyrite	FeS_2	XX	X	X	X	XX	X	XX
	Galena	PbS	X	X		X		X	
Carbonates	Calcite	$CaCO_3$	XX	X	XX	XX	XX	XX	XXX
X	Trace Minerals (<1% by area)								
XX	Minor Minerals (1-10% by area)								
XXX	Major Minerals (> 10% by area)								

SGS Miriam 2013 Met Comp sample

The XRD analysis for the *Miriam 2013 Met Comp* sample consists of major amounts of quartz, minor K-feldspar and chlorite with trace amounts of calcite. The QEMSCAN results correlate with the XRD analyses for the *Miriam 2013 Met Comp* sample consisting of quartz (82.3 %) and K-feldspar (6.3 %), with minor clays (3.8 %), chlorite (2.5 %) along with trace amounts (0.6 %) of pyrite and other minerals (the full modal abundance is detailed in Table 12-12). The uranium bearing minerals account for 0.7 % and include uraninite (0.03 %), coffinite (0.4 %) and brannerite (0.3 %). The D_{50} for uranium minerals is 26 µm; silicates 229 µm; carbonates 114 µm and overall particles 229 µm. The uranium is dominantly hosted by coffinite (71.6 %), U-Ti (brannerite) 21.3 % and uraninite (7.0 %). At -10 mesh the majority of the uranium minerals are 0-30 % grain area exposed and only 12 % are free and liberated.

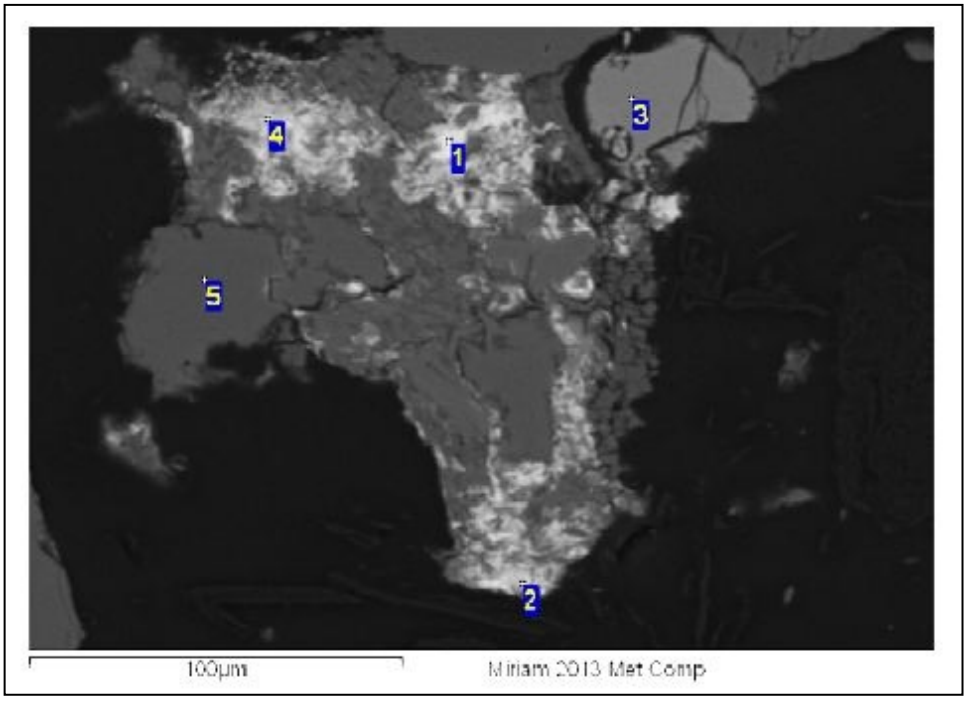


Figure 12-3: SEM – BSE image of the *Miriam 2013 Met Comp* sample

Residue B10 and Residue B12

The residues from the bottle roll leach tests operated at 1.7 mm crush and pH 2 (B10) and pH 1.2 (B12) were subjected to mineralogical analyses. For both samples the SEM-EDS yielded predominantly U-Ti minerals (e.g. brannerite) and the uranium minerals occur as fine-grained locks in quartz (3 < 100 µm). A few grains of uraninite and coffinite were also detected.

12.5 Quality Assurance/Control

A second laboratory was used to analyse the samples from the acid pugging testwork and a statistical comparison of results was undertaken, the analyses are displayed below in Figure 12-4 and show a good correlation between the analyses from the two laboratories.

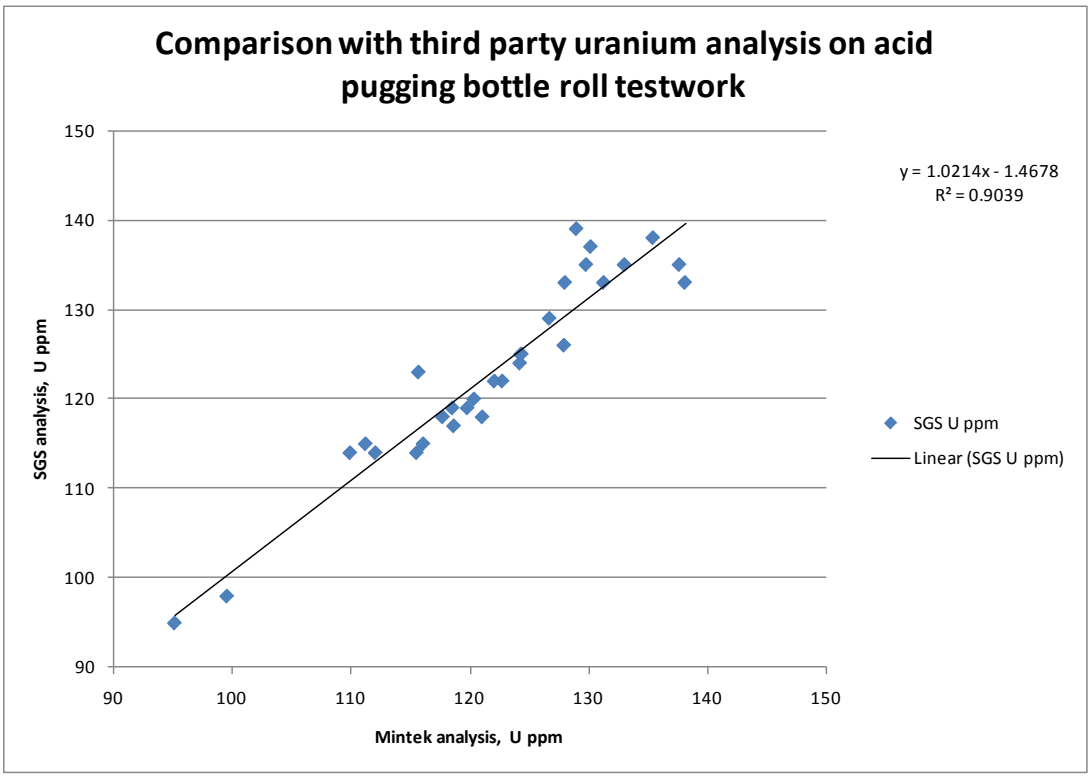


Figure 12-4: Comparison of acid pugging results from Mintek and SGS

12.6 Comminution

Bond ball work index ("BBWI") tests were carried out for the HG, LG and WR samples. The tests were conducted in a standard Bond tumbling ball mill with dimensions 305 mm x 305 mm with no lifters. Based on the typical classification table for BBWI the HG ore sample contained hard and very hard components, the LG ore sample contained medium hard and hard components and the waste rock sample contained hard components. The breakage characteristics of the HG ore and waste rock sample are approximately the same at 300 & 212 μm and 212 & 150 μm respectively. A summary of the BBWI results can be found in Table 12-3 below.

Table 12-3: Bond ball work index results

Limiting Screen Size (μm)	BBWI (kWh/t)		
	HG ore	LG ore	Waste rock
300	18.15	12.44	17.48
212	18.39	14.79	19.14
150	23.12	15.56	19.35
106	25.96	16.28	20.29

12.7 Beneficiation

A composite sample (100 % passing 30 mm) comprising low grade (30 %), high grade (35 %) and waste rock (35 %) was prepared for the beneficiation testwork. In addition to the RADOS testwork (equivalent to radiometric sorting detailed in Section 12.7.1), Mintek also conducted scrubber testwork to assess the effectiveness of this process in producing a more concentrated uranium feed.

12.7.1 Radiometric Sorting

Introduction

Radiometric ore sorting is an operating plant method of pre concentration that can separate a higher grade portion from a lower grade portion according to their respective radioactivity.

Current ROS technology is applicable on mineralization of a size between 15 mm and 150 mm. It will therefore be applied to the Run of Mine ("ROM") after screening above 15 mm. The resulting fines (less than 15 mm) will certainly contain the majority of the uranium and will then have to be treated separately.

Madaouela RADOS Testwork (Mintek)

A composite sample (100 % passing 30 mm) comprising low grade (36 %), high grade (32 %) and waste rock (32 %) was prepared for the beneficiation testwork. It should be noted that only 13 % of the available -50 mm material was recovered to the +30 mm fraction. The 750 kg composite sample (bold column in Table 12-4) was used for RADOS testwork.

Table 12-4: Masses of screened fractions

Sample	Mass of Sample (kg)		
	-50 + 30 mm	**- 30 mm**	**Total**
HG	**240**	1,733	1,973
LG	**267**	1,699	1,966
Waste Rock	**244**	1,476	1,720
Total	**751**	4,908	5,659

The beneficiation composite sample was screened to -50 +30 mm and was subjected to a RADOS XRF ore sorter at a uranium cut-off grade of 600 ppm. Figure 12-5 demonstrates the potential to pre-concentrate the uranium concentration in the head feed to the processing circuit.

The RADOS testwork showed that the uranium content of the composite sample was upgraded from 1,080 ppm to 1,619 ppm while recovering 89.6 % of the uranium and ejecting 35.7 % of the feed mass.

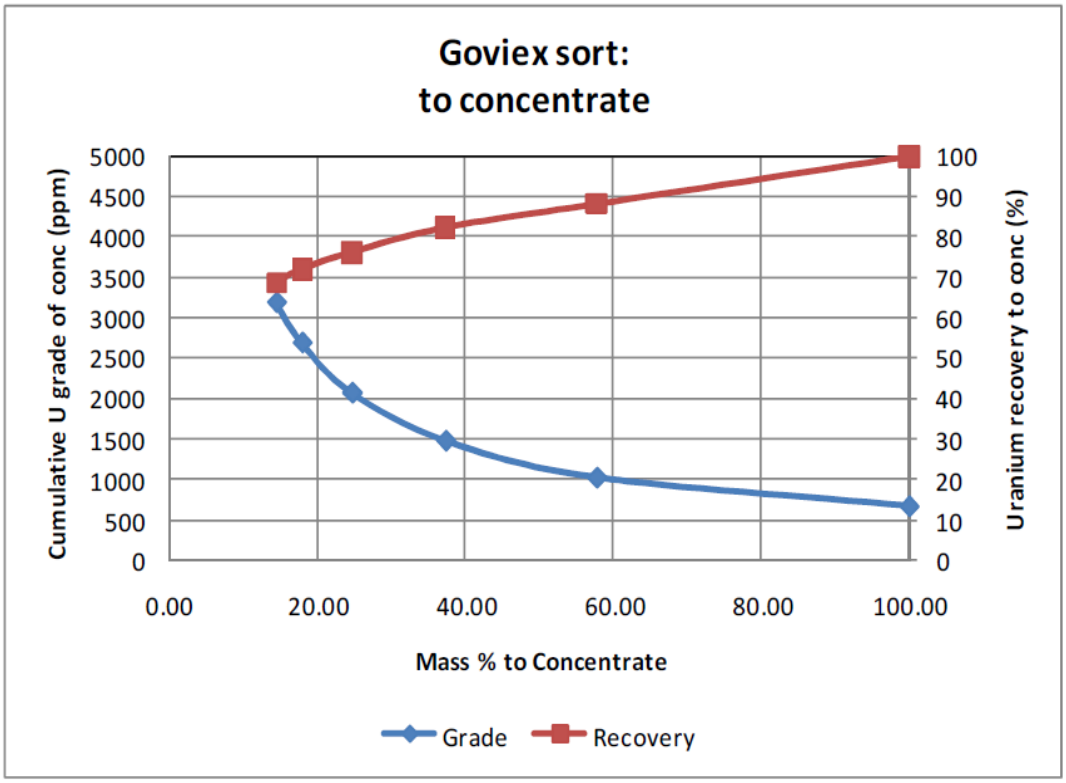


Figure 12-5: RADOS testwork – uranium recovery

KRIPTO ROS Assessment

As an extension of this assessment discussions were held with an active manufacturer of radiometric sorters, EC KRIPTO in Ukraine. This company has designed manufactured and commissioned ROS equipment since 1981. Their equipment has been piloted extensively throughout the former Soviet Union countries with operational equipment utilized in several operations (Table 12-5).

Table 12-5: Summary of operations utilizing Kripto-ROS equipment

Operation	Country	Started	Finished	Geology
Martelskoye	Ukraine	1981	operating	Sedimentary hosted
Ingelskoye	Ukraine	1983	1999	Sedimentary hosted
Smolinskoye	Ukraine	1986	operating	Granite hosted
Strekopskoye	Kazakhstan	1990	1995	Sedimentary hosted
Priaguanskoye	Russia	1997	operating	Sedimentary hosted

Summary of Operation

The first operation at Ingelskoye operated a small ROS at 15 tph and was able to recover more than 95 % of the uranium into a mass pull of 50 % of mined material. Two types of separators have been developed;

- Cone separators; Not as sensitive but useful for higher grade ore and coarse material (typically 100-300 mm) and higher throughput (130-140 tph). Generally more expensive to operate (more than USD 2 /t);
- Belt separators; slower, limited in terms of throughput (upper limit of 90 tph quoted) but more sensitive and can utilize finer grained material down to 20 mm. Operating cost is defined as USD 0.50 /t for the ROS unit.

In addition to these the company also produces Electromagnetic separators that they have had success in applying to some uranium ores and heavy mineral concentrates. The benefit of these is that they can handle material down to 0.1 mm but are limited in capacity (20 tph).

The system does require water on the order of 34 m^3 in circulation with typically 20 % loss of water over 24 hours for 100 tph belt separator.

Typical cost of 100 tph belt separator is USD 12 M and this covers two track belts with one belt working on material in the size range 90-300 mm, one operating at 20 - 90 mm (although may be limited to 40 - 90 mm depending on material). The less than 20 mm size fraction would be recovered and fed to the ore bin as there is a lot of uranium minerals generally associated with this fraction but this can also be confirmed by use of an overhead ROS as well. The sensitivity of the ROS can be controlled and Kripto have operated them from 100 - 800 ppm.

The typical layout of a ROS utilizes a hopper that feeds material onto a belt where it is size sorted for separation into;

- +300 mm (return fraction to crusher);
- 90 - 300 mm belt separator;
- 20 - 90 mm belt separator;
- -20 mm fines.

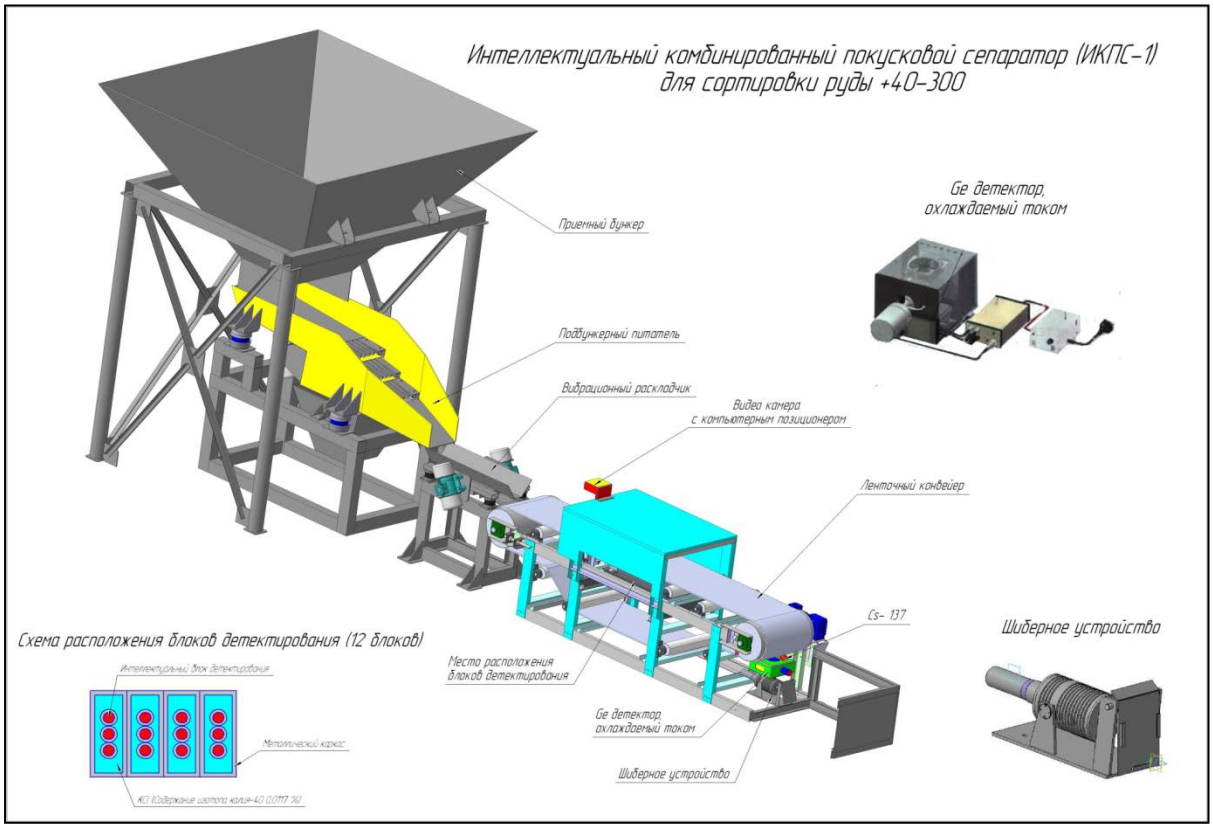


Figure 12-6: Kripto ROS schematic

Final design would be based on material characteristics (grade, mineralogy, size distribution).

Performance of Kripto ROS

Performance data was provided based on averages for 5 years at Smolinskoye (Table 12-6).

Table 12-6: Summary of performance, 2007-2012 Smolinskoye

Index	Total t	U_3O_8, %	Uranium t
Processing Volume, 2007-2012.	4,784,200	0.0181	86,594
Primary separation	913,193	0.0458	41,824
Secondary separation	993,451	0.0396	39,341
Final tail	2,877,556	0.0019	5,429

The uranium is contained in two fractions. The initial belt and cone separation is in the process plant and secondary separation by belt filters of the primary tails occurs in the waste rock area. The_primary separate is shipped by train to the hydrometallurgical plant at Zhalty vater and the secondary separate is processed on site using heap leach. The final waste containing less than 20 ppm U_3O_8 that comprises the majority of the mined rock from underground is used for road aggregate and building materials.

Thus the Kripto ROS separators concentrate more than 93 % of the uranium into a 40 % mass pull. This compares to the RADOS operation on Madaouela ore that was able to concentrate approximately 90 % into a 60 % mass pull with rejection of 40 % of the material. It should be noted that for the operations that used Kripto ROS in Russia and Kazakhstan, RADOS was also tested and gave similar results but was not selected due to complexity of operation. Given that Smolinskoye has similar uranium mineralogy (coffinite and uraninite) and a similar deportment of uranium (50 % of uranium grains less than 50 µm) it is proposed that despite the ore being granitic it will behave similar in ROS to Madaouela and so the results are applicable.

Based on the information provided by Kripto and performance of RADOS at Mintek on a large sample of Madaouela ore it is proposed that the following performance values be considered for Madaouela ore sorted through a Kripto ROS.

- 93 % separation of uranium into 40 % mass pull where there is resuing and 93 % separation into 55 % mass pull where there is no resuing;
- Capital cost of USD 17 M assuming two streams of maximum capacity 2,200 tpd belt separators;
- Operating cost of USD 0.50 /t for ROS.

It should be noted that the evaluation of the performance of ROS applied for the Madaouela ROM is conservative allowing for more mass in the accepted sorted material and lower uranium recovery than reported in full scale operations. ROS in the process design will be to target the removal of hanging wall waste that contains very little to no uranium.

12.7.2 Scrubber Testwork

The objective of the scrubber testwork was to recover the uranium that was indicated to occur with the fine interstitial cement and separate this from the coarser quartz particles in the sandstone. The aim is to liberate and concentrate finer uranium bearing particles without concentrating gangue so as to reduce the mass of ore that requires leaching.

Due to the hardness of the sample (as indicated in the BBWI testwork) the U_3O_8 could not be efficiently liberated from the GoviEx composite sample.

12.7.3 Flotation Testwork

Six flotation tests were conducted using various test setups, various conditioning reagents and at various pH values.

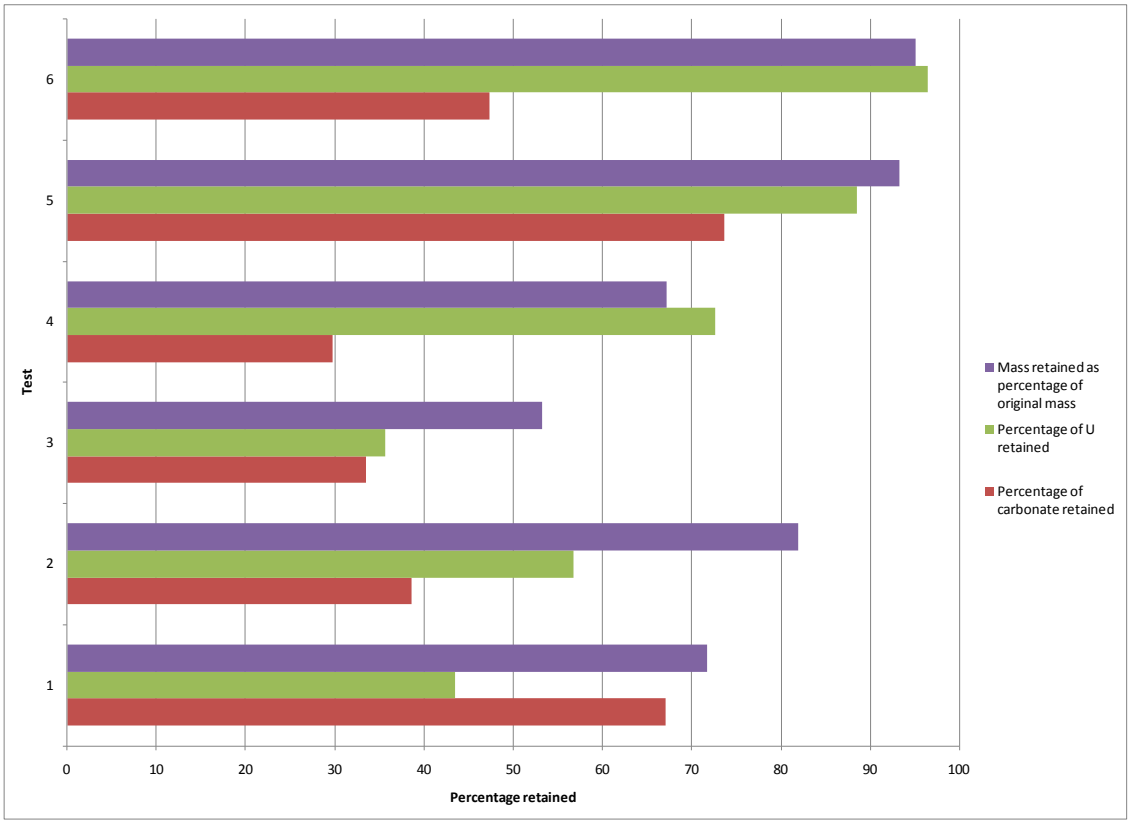


Figure 12-7: Flotation test results

The optimum test conditions (Test 6 in Figure 12-7) were reverse U flotation on a deslimed sample with an ore pH of 7.5, using: yellow dextrin; and fuel oil as conditioning reagents at P_{80} of 212 µm. The float resulted in a 96 % recovery of U, 53 % rejection of the carbonate material and 5 % reduction in leach feed mass.

12.7.4 Ablation

Ablation works by combining the ore with water to form slurry and impacting the slurries together to form a high energy impact zone that separates heavier uranium minerals from the host rock, resulting in a reduced throughput requirement (Figure 12-8) on the downstream leaching circuit.

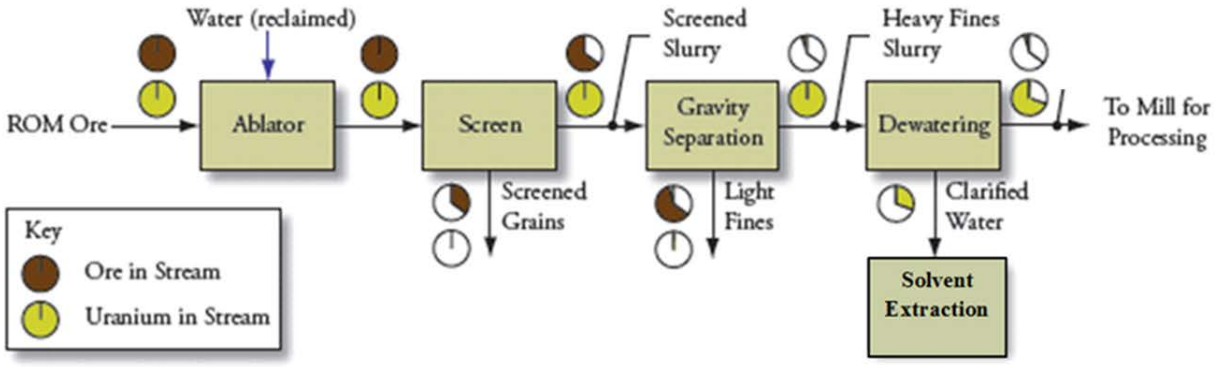


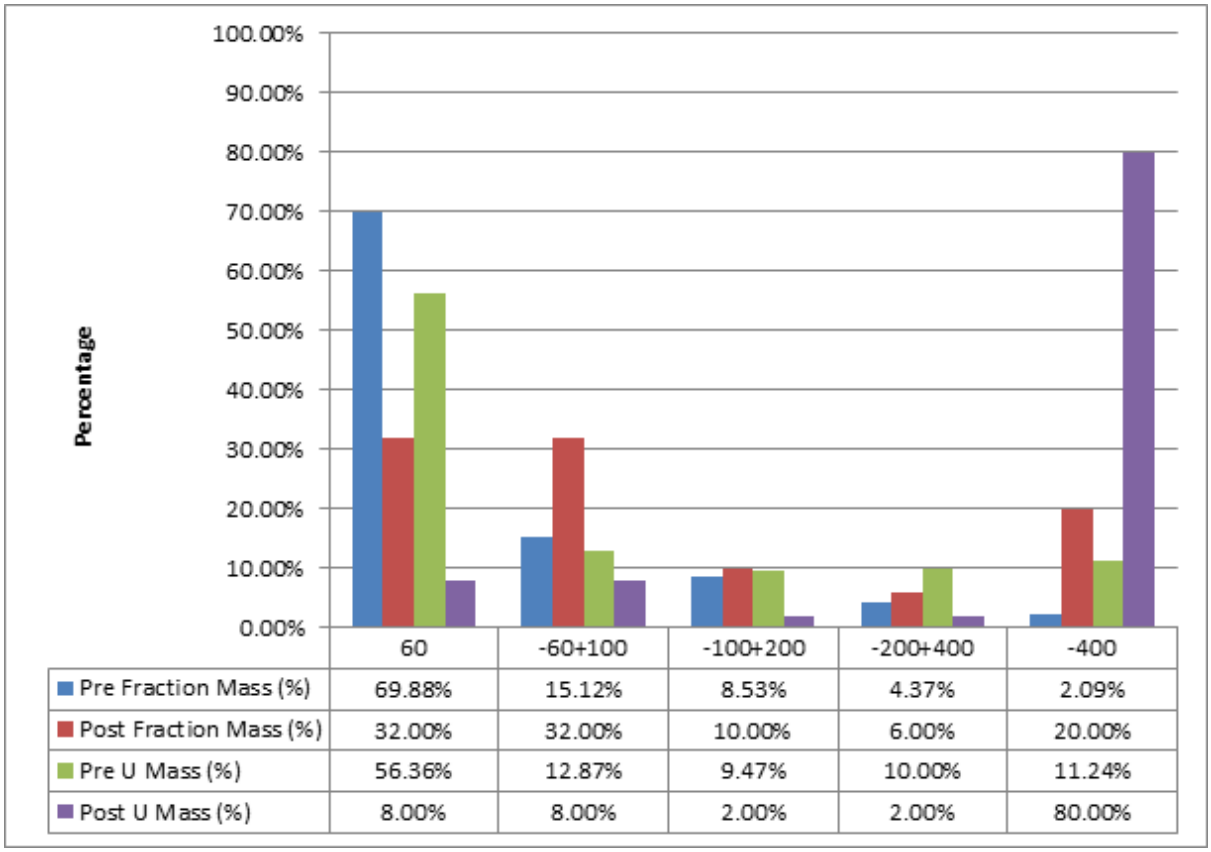
Figure 12-8: Typical ablation process flow diagram (source: Ablation Technologies)

Uranium is recovered from two streams, soluble uranium is recovered through dewatering and this is fed to the solvent extraction and is also recovered from the leach circuit in the mill within the sulfuric acid stream.

Two tests were conducted using different slurry total suspended solids concentrations, slurry velocities, and ablation time. Table 12-7 details the elemental deportment by PSD, the small scale tests initially indicated that 84% of the uranium can be isolated in the fines fraction (<149µm) and contained within the slurry. The recovery of uranium in a large scale pilot and the full scale plant will be significantly greater as the throughput and velocities increase the energy in the high impact zone resulting in greater ablation of the cemented particles, furtherfurther teswork has shown the recovery to be in the order of 99%. 72% of the molybdenum contained within the ore also reports to the fines fraction (<149µm). The amount of material requiring to be leached is reduced by 64%.

Table 12-7: Initial testwork - Post ablation particle size elemental deportment

Fraction size (mesh)	+50	-50+100	-100+200	-200+400	-400	total
sample mass (g)	6747.54	6781.51	1997.55	1337.60	4092.35	20956.55
mass fraction %	32%	32%	10%	6%	20%	100%
U (ppm)	787	803	742	919	12900	
U mass (g)	5.31	5.45	1.48	1.23	52.79	66.26
U mass (%)	8%	8%	2%	2%	80%	100%
Mo (ppm)	71.7	80.7	71.7	68.7	557.5	
Mo mass (g)	0.48	0.55	0.14	0.092	2.28	3.55
Mo mass (%)	14%	15%	4%	3%	64%	100%
V (ppm)	11.3	35	56	125.7	131.3	
V mass (g)	0.076	0.24	0.12	0.17	0.54	1.13
V mass (%)	7%	21%	10%	15%	48%	100%
Fe (ppm)	1192.7	1250.3	2221.7	4561	18733	
Fe mass (g)	8.05	8.48	4.44	6.10	76.66	103.73
Fe mass (%)	8%	8%	4%	6%	74%	100%
Ti (ppm)	284	404.7	537.3	1129.7	5158.7	
Ti mass (g)	1.92	2.74	1.07	1.51	21.11	28.36
Ti mass (%)	7%	10%	4%	5%	74%	100%
Ca (ppm)	6274	2098.7	6907	14233.3	32800	
Ca mass (g)	42.33	14.23	13.80	19.04	134.23	223.63
Ca mass (%)	19%	6%	6%	9%	60%	100%



	60	-60+100	-100+200	-200+400	-400
Pre Fraction Mass (%)	69.88%	15.12%	8.53%	4.37%	2.09%
Post Fraction Mass (%)	32.00%	32.00%	10.00%	6.00%	20.00%
Pre U Mass (%)	56.36%	12.87%	9.47%	10.00%	11.24%
Post U Mass (%)	8.00%	8.00%	2.00%	2.00%	80.00%

Figure 12-9: Initial Testwork - Pre and Post ablation elemental deportment

Figure 12-9 indicates the increased concentration of elements in the fine fractions (in particular <400 Mesh, equivalent to <37µm) and the overall increase of particle mass to the finer end of the PSD due to the ablation process. Further testwork is required to establish the optimum particle size cut off and process parameters. Commercial scale units with higher pump velocities and throughput capacities are likely to result in improved PSD elemental deportment and hence uranium/molybdenum concentrations. Initial methods have already been tested by Ablation that show improved uranium recovery.

One 20 kg sample was run in late 2013 through the Wyoming pilot plant with SRK present, sub-splits were taken for analysis in the UK. Chemical analysis was by XRF and calcite mass by XRD. Table 1 details the elemental deportment by PSD, the small scale tests to date show that 90% of the uranium can be isolated in the fines fraction (<149µm) and contained within the slurry. Approximately 69% of the molybdenum is returned in the same material along with 33% of the calcite present. Approximately 39% mass of the total sample is recovered for the leaching circuit. Results show a slight decrease in molybdenum and calcite recovered and an increase in uranium and the mass recovered.

The current costs although conservative will be maintained as it is considered we have insufficient data for any refinement. However, due to the recirculation in the system and greater pressure from the nozzles the separation of uranium from carbonates has been improved.

Table 12-8: Results of 20 kg Ablation test, Wyoming

Fraction size (mesh)	+50	-50+100	-100+200	-200+400	-400	Total
sample mass (g)	5778	5543	1098	3680	4325	20424
mass fraction %	28%	31%	10%	18%	21%	100%
U (ppm)	389	655	770	1415	12560	15789
U (g)	2.248	3.63	0.846	5.207	54.322	66.253
U mass (%)	3.4%	5.5%	1.3%	7.9%	82.0%	100%
Mo (ppm)	88.9	120	342	89.6	711	1351.5
Mo (g)	0.514	0.665	0.376	0.33	3.075	4.959
Mo mass (%)	10%	13%	8%	7%	62%	100%
Calcite (%)	19%	5%	43%	11%	22%	100%

Recycle of the residue through a second ablation circuit gave the same results for uranium and molybdenum recovery although no appreciable further calcite was rejected (Table 12-9).

Table 12-9: Results of Second Cycle Ablation test on +200 µm fraction

Fraction size (mesh)	50	50	100	200	-400	Total
sample mass (g)	2766.7	3450	1239	2479.2	2484.1	12419
mass fraction %	22%	28%	10%	20%	20%	100%
U (ppm)	93.054	59.866	22.494	152.37	1486.21	1814
U (g)	0.345	0.242	0.0824	0.565	5.49	6.724
U mass (%)	5.1%	3.3%	1.2%	8.4%	81.9%	100%
Mo (ppm)	61.315	65.557	42.968	40.051	341.02	550.9
Mo (g)	0.173	0.184	0.121	0.113	0.963	1.554
Mo mass (%)	11%	12%	8%	7%	62%	100%
Calcite (%)	18%	5%	51%	15%	11%	100%

Thus overall 99% of uranium and 90% of molybdenum is recovered to the leach feed material along with 50% of the calcite gangue. Consequently the gangue acid consumption has been reduced with calcite concentration in the leach material reduced to the equivalence of 13 kg H2SO4/t assuming one mole of acid is consumed by one mole of calcite in addition to the acid requirement for uranium leaching.

12.8 Alkaline Leach

A 2 kg 80 % passing 250 µm HG ore sample was pulped to 30 % solids (m/m) using Rand Water Board water. The slurry was heated to 90°C in a 5 L glass reactor on a hot plate connected to a temperature controller. Sodium carbonate was added to a concentration of 30 g/L carbonates and sodium bicarbonate was added to a concentration of 10 g/L bicarbonate, and the concentrations were checked via titration and adjusted accordingly. The test was run for 48 hours.

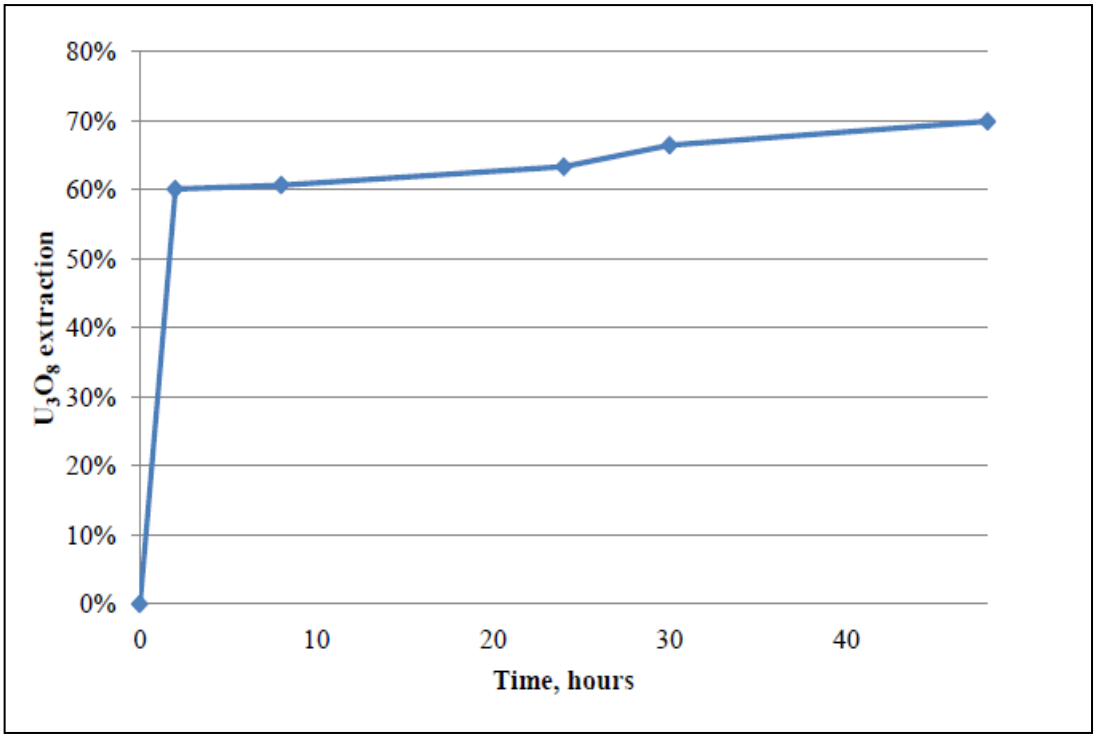


Figure 12-10: Alkaline leach kinetics

Overall (after 48 hours) the alkaline leach test resulted in a 70 % extraction efficiency for U_3O_8, however after 2 hours the extraction efficiency was already 60 %. The kinetics of the alkaline leach are somewhat comparable to the acid leach kinetics however the final extraction efficiencies are significantly lower (particularly when compared to the acid pugging leach extraction efficiencies).

12.9 Diagnostic Acid Leach

A diagnostic acid leach study was carried out by Mintek (Mintek, 2011) on the HG ore. The extent and kinetics of uranium dissolution were determined as a function of grind size, acidity, pre-treatment, temperature and re-grinding. The results from these testworks are summarised in this section.

There was no significant difference in the uranium extractions after 24 hours in relation to differing grind sizes (60 μm, 100 μm, 300 μm and 500 μm), however the rate of leaching increased with decreasing particle size.

The effect of altering the pH and residual sulfuric acid concentrations (pH 1.5, pH 1.85 and 25 g/l) during the acid leach showed no significant difference in the uranium extraction for the tests after 24 hours. It should be noted that the pH variation tests were carried out at a coarse grind (500 μm) and 60 ^{o}C. Acid curing also did not result in any significant effect on uranium extraction efficiency after 24 hours.

The highest uranium extraction efficiency was achieved by re-grinding the residue from a coarsely ground leached sample and re-leaching it. However this required 130 kg/t of acid, a 48 hour leach and resulted in an 80 % extraction efficiency.

An increase in leach temperature (from 60 to 80 ^{o}C) increased leach kinetics and acid consumption (70 to 89 kg/t respectively) with no significant improvement in uranium extraction efficiency.

Overall, the diagnostic acid leach tests resulted in approximately 75 % uranium extraction efficiency after 24 hours.

12.9.1 Two Stage Acid Leach

A series of tests was conducted by Mintek utilising a multi-stage leach approach using the filtrate from one leach to act as the lixiviant in the next leach. This was repeated for 3 stages (i.e. 3 two stage leaches). Approximately 85 kg/t sulfuric acid was consumed in the leach cycle overall.

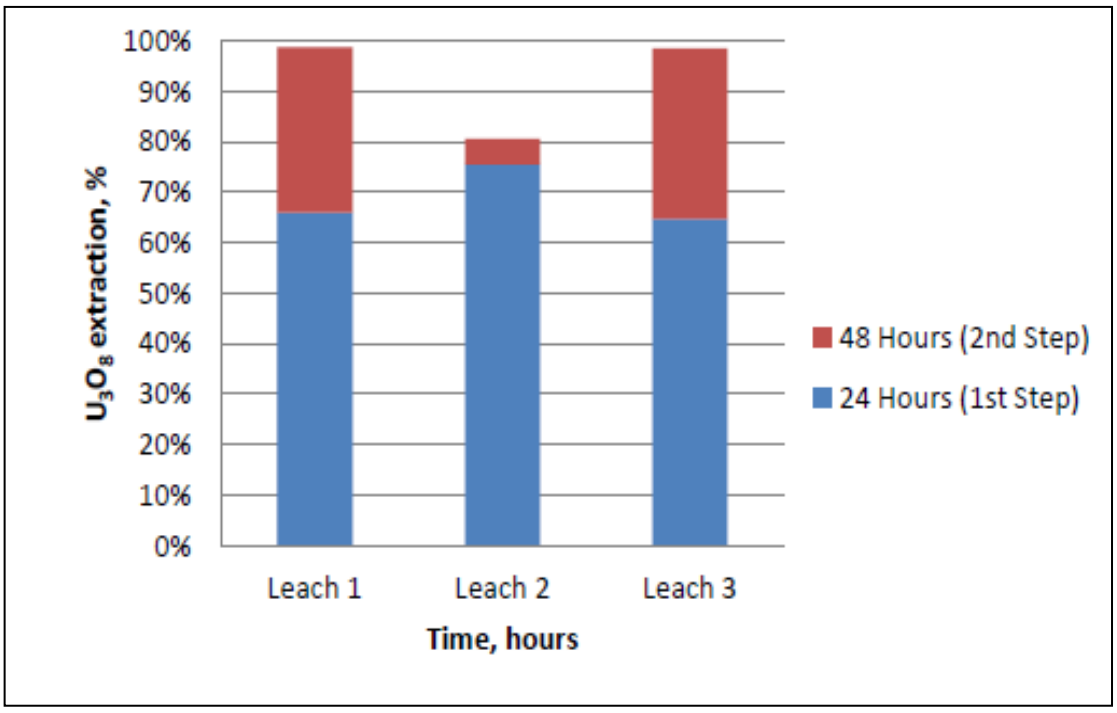


Figure 12-11: Uranium extraction for the two stage leach testwork

The extraction efficiencies for the two stage multiple leaches was very high (around 98 %) for leach 1 and 3 of the trials and 81 % for the leach 2 trial (the lower extraction most likely due to a larger dissolution volume used for stage 2 that caused dilution in the testwork).

The reusing of the filtrate from stage to stage allows any residual acid to be used for the next stage leach reducing opex and does not seem to have any negative impact on extraction efficiency as can be seen from the efficiency in leach 3 (Figure 12-11). The net acid addition required will be on the order of 20 kg/t based on calculations.

12.9.2 Acid Pugging (Ambient Temperature)

As part of the previous study carried out by SGS (SGS Minerals Services, 2010) on core samples obtained from the Madaouela deposit, testwork was conducted to measure the efficiency of uranium extraction under various leaching conditions. Part of the study investigated uranium extraction utilising a strong acid pugging and curing process (similar to the processing method used at the nearby Somair and Cominak mines), the extraction efficiencies were greater than 84 % and dependant on the test conditions some tests achieved 94 % efficiency. Due to the very favourable extraction efficiencies achieved in the previous study similar testwork has been undertaken on the HG ore samples obtained for this study.

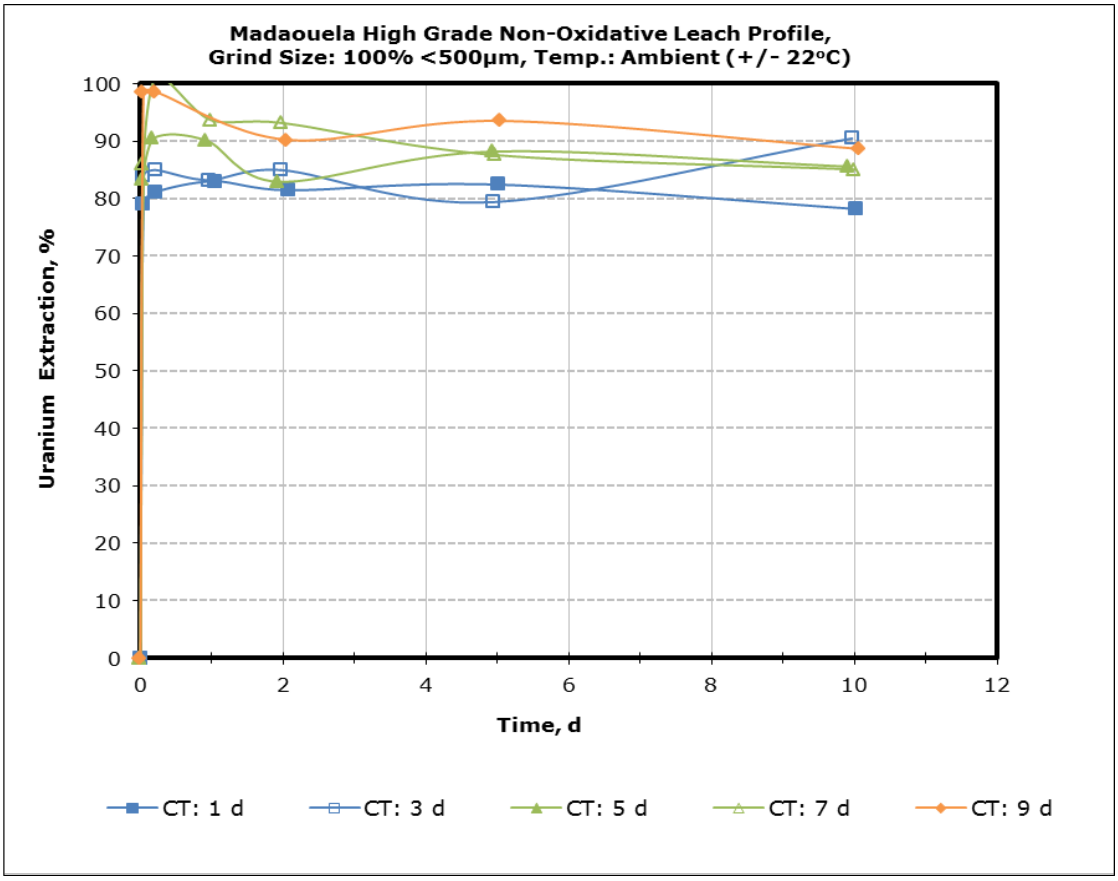


Figure 12-12: High grade ore acid pugging tests (non-oxidative) – U extraction over time

The extraction efficiencies obtained during this testwork are consistent with the previous testwork results and indicate that approximately 95 % is achievable using the acid pugging process. The extraction efficiency was proportional to duration of the acid curing, with the highest achieved after 9 days (96 %), however >90 % could be achieved with one day curing.

The extraction efficiency was independent of the oxidative state of the testwork. The testwork was only carried out on one grind size (500 µm). It should be noted that the high extraction efficiencies obtained during the pugging testwork requires a high acid consumption (120 kg acid/t dry ore compared to around 80 kg/t for the diagnostic leach resulting in 78 % extraction efficiency) and noxious fumes are generated during the pugging process, these will have to be collected and scrubbed.

12.9.3 Acid Pugging (High Temperature)

It was highlighted that an increase in temperature could be expected during acid pugging resulting from the dilution effect of the acid on the moist feed and the exothermic reactions between the acid and carbonates and other reactive minerals. However in the small scale tests conducted it was expected that the temperature increase would not replicate that which would be achieved in plant scale operations. It was also expected that the increased temperature achieved could reduce the required curing time prior to leaching.

In an attempt to reduce the residence time of the curing stage of the acid leach the previous testwork (Section 12.9.2) a high temperature cure phase was explored over 6 tests at varying acid additions (80-150 kg/t) and temperatures (at 100 °C and 160 °C). The curing phase was followed by an agitated tank leaching at 60 °C and pH 1.5.

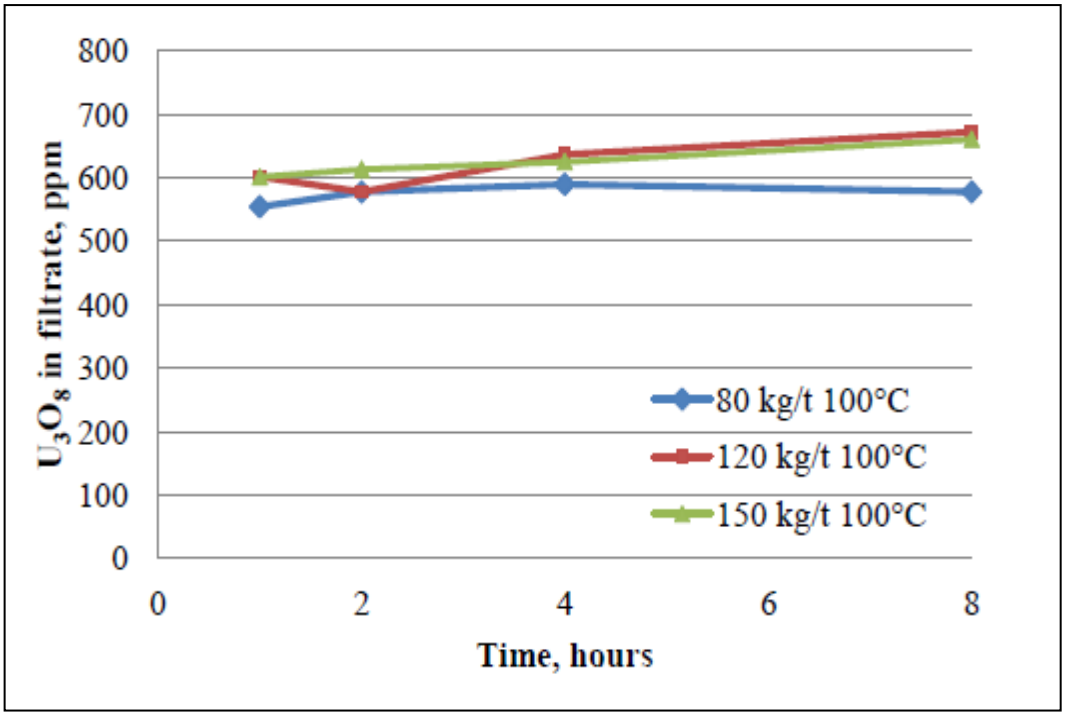


Figure 12-13: Leach kinetics at various acid consumptions

The optimum acid dose was 120 kg/t resulting in a U_3O_8 extraction efficiency of 96 % at 100 °C while the increased temperature tests (160 °C) did not result in higher extraction efficiencies. The optimum acid dose test was repeated at different grind sizes (250 and 500 µm), the results detailed in Table 12-10 indicate that the coarse grind material (500 µm) had the highest extraction efficiency (although it was higher in the previous test utilising the same test conditions).

Table 12-10: Repeat pugging and curing leach at optimum acid consumption

	U_3O_8 in residue (mg/kg)	U_3O_8 in filtrate (mg/L)	U_3O_8 extraction based on solid assay (%)	Accountability (%)
120 kg/t Repeat (-500 µm) 100°C	138	720	92	96
120 kg/t 80 % -250 µm 100°C	240	502	86	92
120 kg/t 80 % -250 µm 160°C	222	598	87	100

12.10 Bottle Roll Leach Test

12.10.1 Bottle Roll Leach Tests (Mintek)

Bottle roll tests were carried out on the low grade ore and RADOS concentrate samples, the amount of U_3O_8 dissolution for the LG Ore at the end of the bottle roll testwork was approximately 73 – 79 % irrespective of the pH of the leach medium or addition of a ferric oxidant.

The U_3O_8 extraction efficiency of around 78 % for the LG ore with ferric added at a pH of 1.7 only required 45-49 kg/tonne of sulfuric acid. It was inconclusive whether the addition of an oxidant increased the reaction kinetics because one of the duplicate tests without oxidant added produced the same percentage U_3O_8 extraction at day one as the oxidant added trials. All of the tests achieved > 85 % of maximal extraction efficiencies after two days, as can be seen from Figure 12-3.

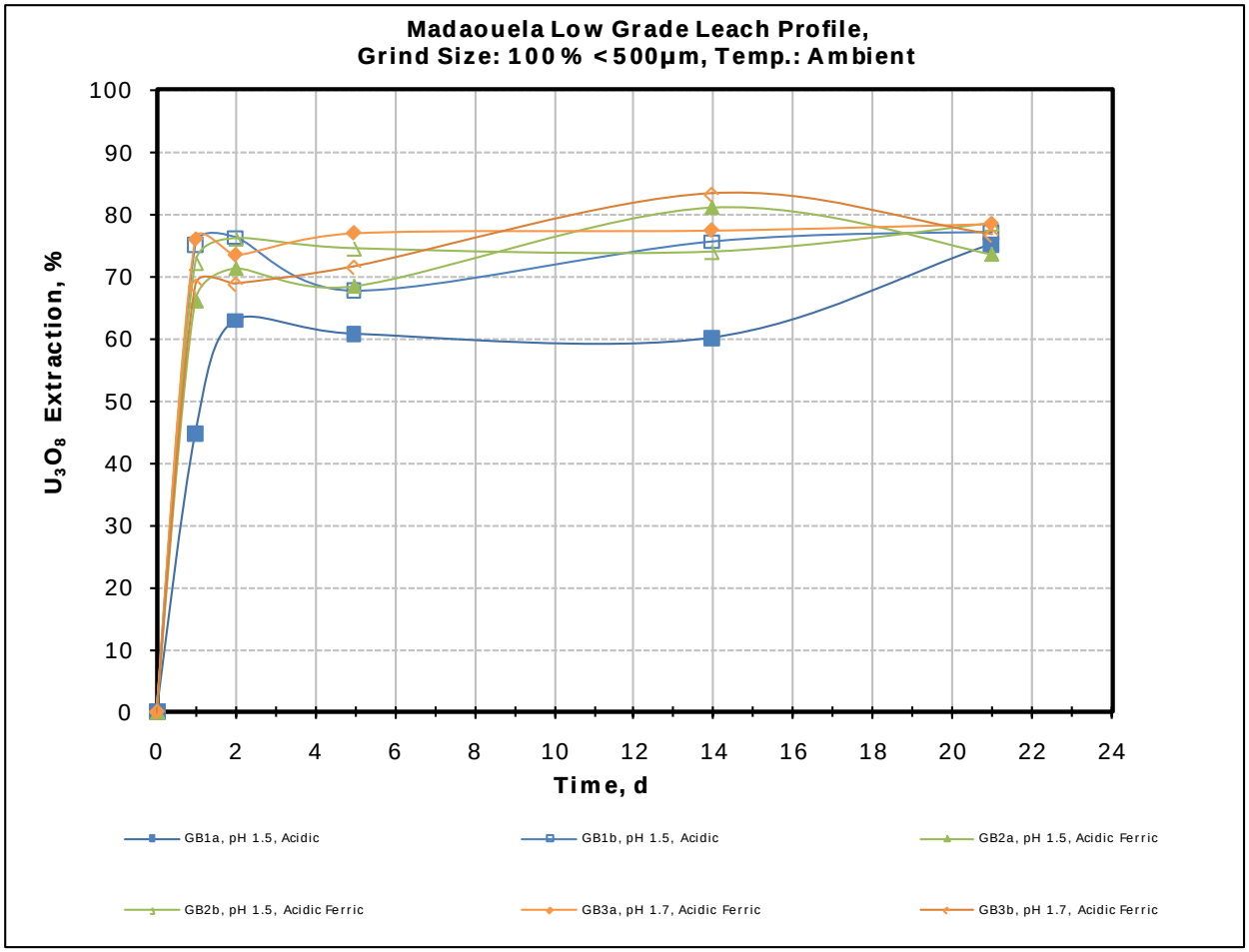


Figure 12-14: Low grade ore bottle leach testwork – uranium extraction over time

The acid consumed for the ferric addition tests operated at pH 1.7 used on average 26 % less acid than the pH 1.5 runs (84 kg/t versus 114 kg/t), however they provided nearly identical extraction efficiencies (77 % versus 75 % respectively).

The acid consumption (kg/t) increased rapidly up to day one, gradually up to day seven and then remained fairly constant until the end of the trial for the low grade ore and the RADOS concentrate.

The reaction kinetics and overall percentage of U_3O_8 dissolution for the RADOS concentrate samples increased with a ferric oxidant addition (circa 75 % compared to circa 80 % respectively) as can be clearly seen in Figure 12-15, however was independent of the leach medium pH for the oxidant runs.

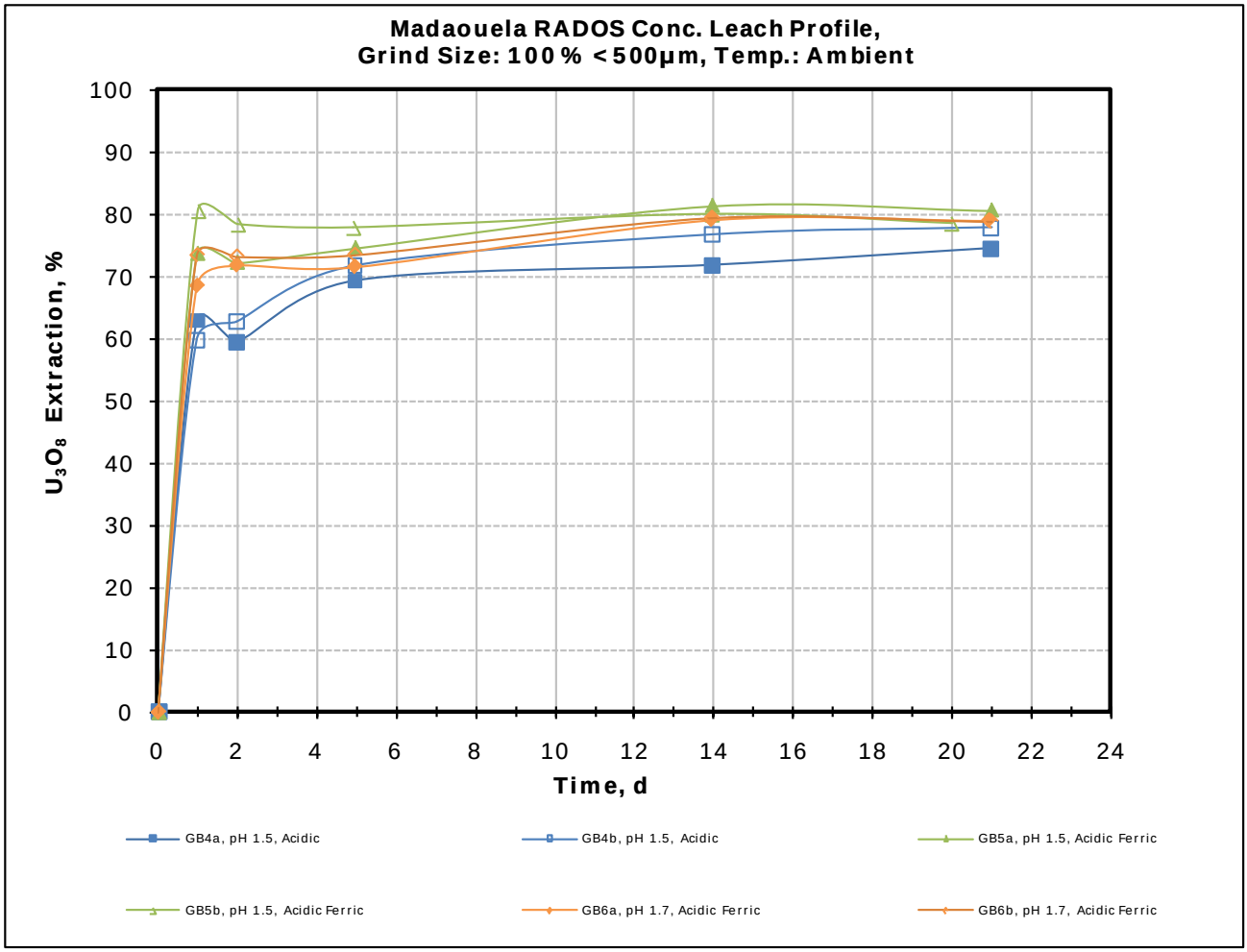


Figure 12-15: RADOS concentrate bottle leach testwork – uranium extraction over time

12.10.2 Bottle Roll Leach Tests on Miriam Sample (SGS)

A series of bottle roll tests were performed on the *Miriam 2013 Met Comp* sample and assay information is in Table 12-11.

Table 12-11: Head assay results

Sample ID	2013 Met Comp	Sample ID	2013 Met Comp
U %	0.26	Th %	0.11
S %	0.20	CO_2 %	0.74
SiO_2 %	87.4	TiO_2 %	0.24
Al_2O_3 %	4.72	P_2O_5 %	0.04
Fe_2O_3 %	1.36	MnO %	0.02
MgO %	0.55	Cr_2O_3 %	0.02
CaO%	1.77	V_2O_5 %	< 0.01
Na_2O %	0.20	LOI %	3.34
K_2O %	0.96	Sum %	100.6
Ag g/t	< 2	Mo g/t	159
Al g/t	24,500	Na g/t	942
As g/t	< 30	Ni g/t	< 20
Ba g/t	255	P g/t	168
Be g/t	0.74	Pb g/t	178
Bi g/t	< 20	Sb g/t	< 10
Ca g/t	10,200	Se g/t	< 30
Cd g/t	< 2	Sn g/t	< 20
Co g/t	< 7	Sr g/t	50.6
Cu g/t	14.5	Ti g/t	1,490
Fe g/t	8,760	Tl g/t	< 30
K g/t	7,950	V g/t	30
Li g/t	< 20	Y g/t	10.0
Mg g/t	1,690	Zn g/t	134
Mn g/t	319		

Table 12-12: Modal abundance (wt %) of head sample

Survey	**GoviEx Niger Holdings**	
Project	**Miriam 2013 Met Comp / MI5025-Jan 2013**	
Sample	**Miriam 2013 Met Comp**	
Fraction	**Combined**	
Mass Size Distribution (%)	**100.0**	
Calculated ESD Particle Size	**169**	
	Sample	
Mineral Mass (%)	Coffinite	0.4
	U-Ti Minerals	0.3
	Uraninite / Uranophane	0.0
	Quartz	82.3
	K-feldspar	6.3
	Plagioclase	0.4
	Clays	3.8
	Micas	0.0
	Chlorite	2.5
	Other-Silicates	0.1
	Zircon	0.1
	Fe-Oxides	0.0
	Ti-Oxides	0.1
	Calcite	2.9
	Other-Carbonates	0.0
	Pyrite	0.6
	Other-Sulfides	0.1
	Apatite	0.1
	Gypsum	0.0
	Other	0.0
	Total	**100.0**

The XRD analysis for the *Miriam 2013 Met Comp* sample consists of major amounts of quartz, minor K-feldspar and chlorite with trace amounts of calcite. The QEMSCAN results correlate with the XRD analyses for the *Miriam 2013 Met Comp* sample consisting of quartz (82.3 %) and K-feldspar (6.3 %), with minor clays (3.8 %), chlorite (2.5 %) along with trace amounts (0.6 %) of pyrite and other minerals. The full modal abundance is detailed in Table 12-12.

A series of twelve 20-day bottle roll tests were conducted using various crush sizes (19, 12.7, 6.4 and 1.7 mm) of the overall composite material and at different operational pH values (1.2, 1.6 and 2.0 pH units). Ferric iron was added to each test at 1 g/L Fe^{3+} and the ORP was maintained above 500 mv (vs. Ag|AgCl, saturated KCl) using a 200 g/L sodium chlorate solution.

Table 12-13: Bottle roll test conditions

Test No.	pH	ORP	Fe^{3+} Addition, g/L	Crush, mm	Temp., °C	Sample
B1	2.0	500	1	19	Ambient	¾ ” Overall Comp
B2	1.6	500	1	19	Ambient	¾ ” Overall Comp
B3	1.2	500	1	19	Ambient	¾ ” Overall Comp
B4	2.0	500	1	12.7	Ambient	½ ” Overall Comp
B5	1.6	500	1	12.7	Ambient	½ ” Overall Comp
B6	1.2	500	1	12.7	Ambient	½ ” Overall Comp
B7	2.0	500	1	6.4	Ambient	¼ ” Overall Comp
B8	1.6	500	1	6.4	Ambient	¼ ” Overall Comp
B9	1.2	500	1	6.4	Ambient	¼ ” Overall Comp
B10	2.0	500	1	1.7	Ambient	-10 mesh Overall Comp
B11	1.6	500	1	1.7	Ambient	-10 mesh Overall Comp
B12	1.2	500	1	1.7	Ambient	-10 mesh Overall Comp

Based on the testwork results, 77 % uranium extraction can be achieved while rejecting 99.8 % of the silicon found in the feed while maintaining moderately low pH levels over a period of 20 days. Table 12-14 graphically presents the uranium extraction over the 20 day test duration.

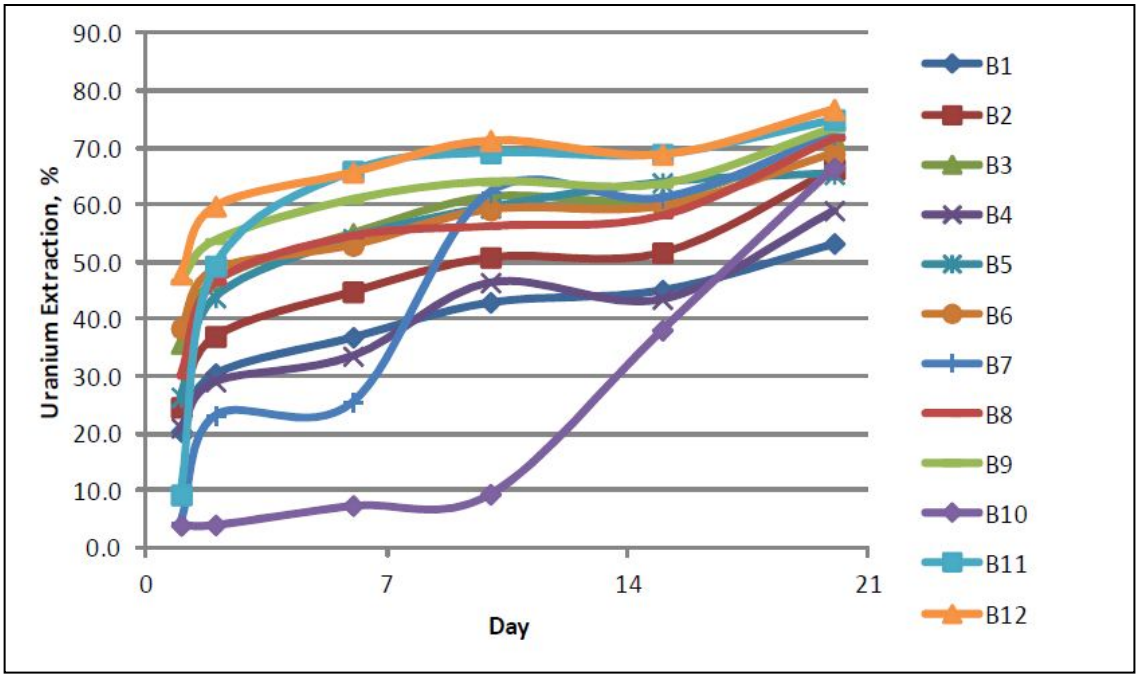


Figure 12-16: Bottle roll leach tests (SGS) - uranium extraction over time

The finer crush size and to a lesser extent pH level increase overall U extraction efficiencies and extraction kinetics.

Molybdenum co-extraction varied between 1 and 43 % broadly inversely proportional to the pH value (i.e. the highest Mo co-extractions occurred at pH 1.2). The maximum final PLS uranium tenor was at approximately 800 mg/L while molybdenum was found in solution at approximately 30 mg/L.

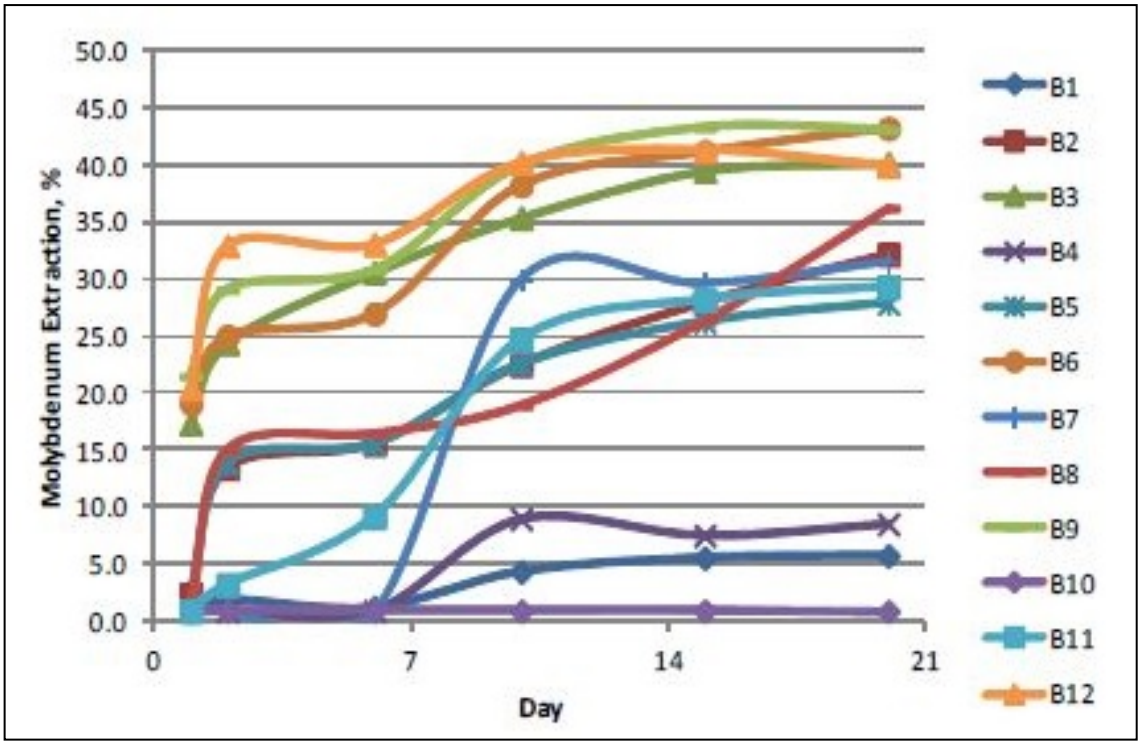


Figure 12-17: Bottle roll leach tests (SGS) - molybdenum extraction kinetics over the test period

Silicon was found in solution at a concentration of approximately 100 – 400 mg/L. It was found that crush size has a measurable effect on uranium extraction while acid dosage has a significant effect on silicon extraction.

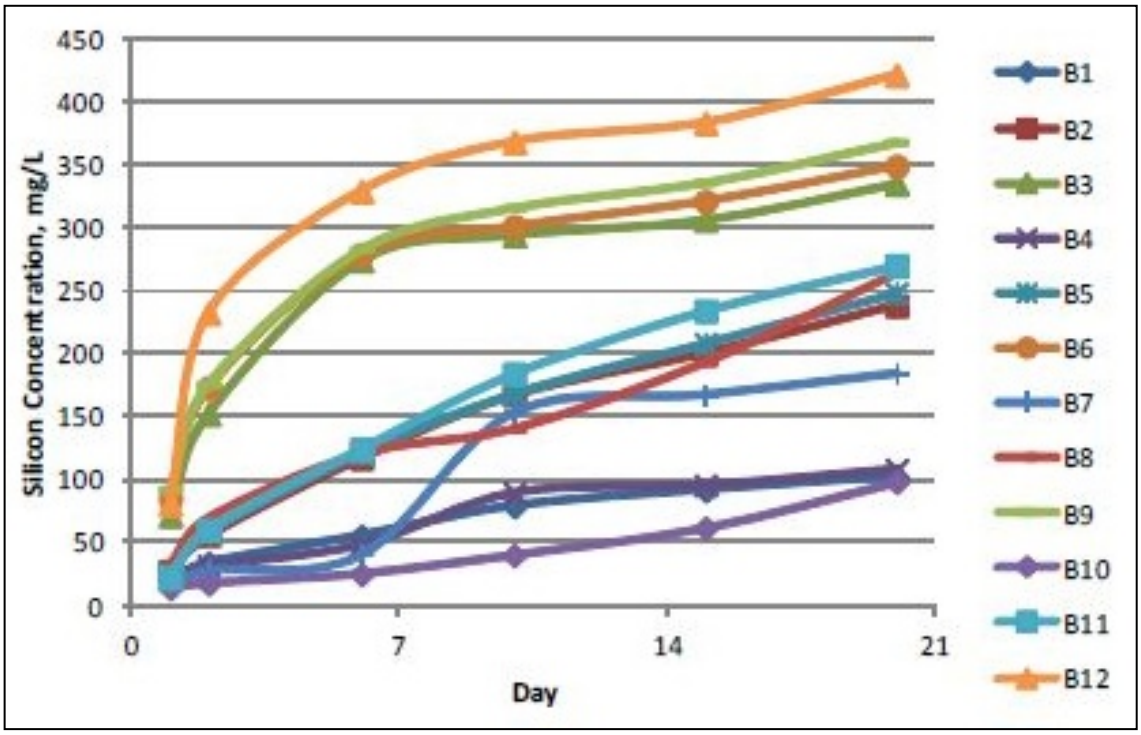


Figure 12-18: Bottle roll leach tests (SGS) - Silicon extraction kinetics over the test period

Figure 12-18 indicates that careful pH control/acid addition must be maintained during the leach process to ensure that the optimum U extraction is achieved whilst minimising Si co-extraction.

The residual uranium in the leach residues was found to be predominantly fine-grained brannerite locked in quartz particles. The uranium extraction efficiency varied between 53 – 78 %, sulfuric acid addition for the tests ranged from 16 – 85 kg/t and sodium chlorate ranged from 0.6 – 3.3 kg/t.

12.11 Column Leach Testwork

Twelve tests were carried out on the GoviEx samples, two test conditions in duplicate for the three GoviEx samples (HG ore, LG ore and RADOS concentrate).

The two tests had standard parameters for:

- Duration – 100 days;
- Column height – 1 m;
- Size (100 % passing) – 20 mm;
- Temperature – 25 ^{o}C;
- Irrigation rate – 6.0 L/m^2/hr;
- Feed acid – 3.0 g/L;
- Feed pH – 1.54 pH units;
- Agglomeration acid:
    - HG ore: 38 – 40 kg/t;
    - LG ore: 37 – 38 kg/t;
    - RADOS conc.: 55 – 57 kg/t.

The variable parameter was the oxidative state of the leaching medium, with each sample type running in oxidative and non-oxidative states. To allow comparisons of the column leach testwork results with the bottle roll tests the feed acidity of each of the six duplicate tests was increased on day 33 to 7 g/L for the HG and LG ores and 13 g/L for the RADOS concentrate.

The HG Ore U_3O_8 dissolution on days 6-7 was around 64-69 % (see Figure 12-19) irrespective of oxidative/non-oxidative state. Increasing acid feed concentration from 3 to 7 g/L resulted in an increased rate of U_3O_8 dissolution and the dissolution after 95 days for the increased acid dose was greater than the constant acid feed rates (82 % and 75-78 % respectively). The U_3O_8 dissolution for the oxidative steady acid feed test was greater than the non-oxidative steady acid feed; however there was no difference between the increased acid feed test and steady acid feed irrespective of the oxidative state.

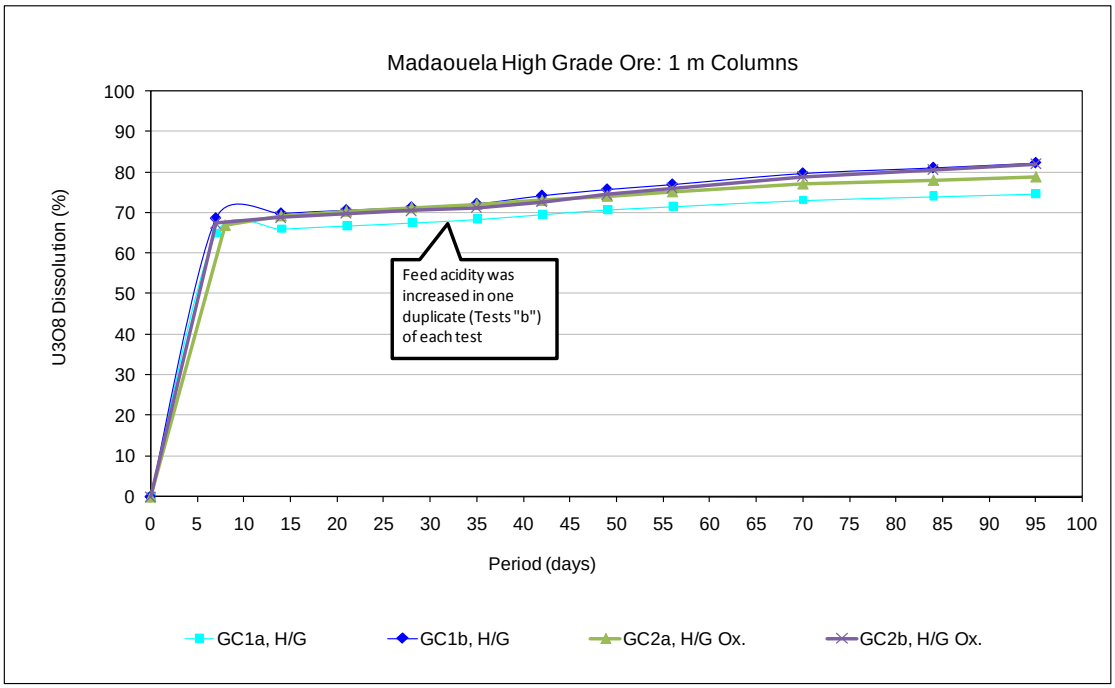


Figure 12-19: High grade ore column leach testwork - U_3O_8 dissolution over time

The LG Ore U_3O_8 dissolution on days 6-7 was around 60 % for the non-oxidative tests and one of the oxidative tests with the duplicate oxidative test having 66 % U_3O_8 dissolution. Increasing acid feed concentration from 3 to 7 g/L resulted in an increase rate of U_3O_8 dissolution. The dissolution after 95 days for the oxidative tests and the increased acid tests was approximately the same (80-82 %). However the non-oxidative constant acid feed U_3O_8 test dissolution was significantly lower (approx. 70 %).

The RADOS concentrate U_3O_8 dissolution efficiencies were widely varied compared to the LG and HG ore tests. On day 6 the U_3O_8 dissolution was between 56 % and 70 % with no clear correlation between the performance of the duplicates of the oxidative and non-oxidative tests. Increasing acid feed concentration from 3 to 13 g/l on day 33 resulted in an increased rate of U_3O_8 dissolution. The increased feed acidity tests had the highest U_3O_8 dissolution after 70 days and this continued to 95 days. The lowest U_3O_8 dissolution was 65 % after 95 days and corresponded to the non-oxidised steady acid test.

The pH of the column drainage significantly increased during the early phase of the trial up to days 7 – 10 and then gradually decreased up to the end of the trial. Each of the columns with the increased feed acidity concentration produced column drainage pH at or below the pH value of the feed after 65 days until the end of the trial.

An increased feed acidity concentration earlier in the trial (before day 33) is likely to have resulted in an increased U_3O_8 dissolution rate that potentially could have reached or exceeded the maximal dissolution percentage earlier in the trial. The increased feed acidity concentration will however have corresponded to an increased total acid consumption rate.

For all of the sample types the oxidative steady and increased acid tests outperformed the non-oxidative steady acid test. The optimum conditions for maximum U_3O_8 dissolution for all the samples were oxidative state and increased acid feed. The extraction efficiency of 82 – 85 % is encouraging as the material only needs to be ground to <20 mm, however the efficiency of this process is dependent on the clay content of the ore which could vary with different locations and other deposits.

12.11.1 Head and Residue Uranium Deportment

Particle size distribution ("PSD") analyses were carried out on the head and residue samples from the column test (crushed to 100 % < 20 mm for the column testwork). Each of the size fractions was then analysed for uranium concentration to determine the uranium deportment throughout the head and residue.

The HG ore and LG ore had similar head PSD analyses with the majority of particles smaller than 1.18 mm (64 and 76 % respectively) (as can be seen from Figure 12-20 and Figure 12-21). The residue PSD analyses were similar to the head analyses for the LG ore and HG ore, however the residue PSD of the RADOS concentrate was significantly smaller than the head PSD.

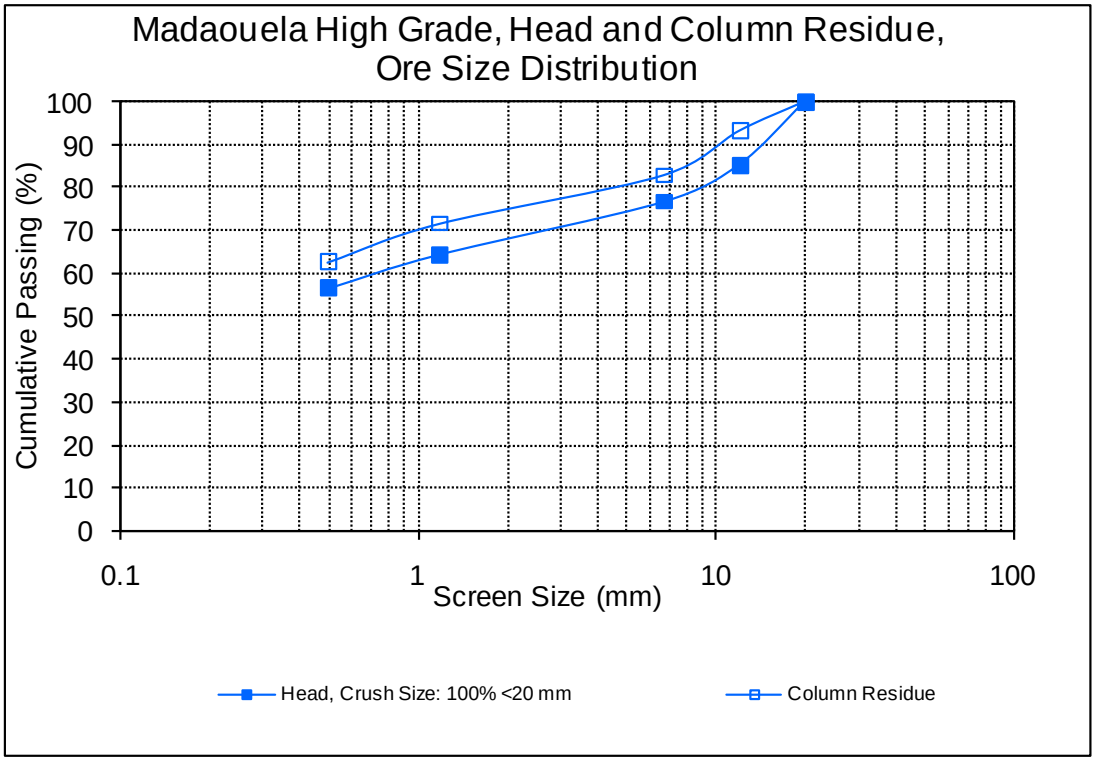


Figure 12-20: High grade ore column head and residue PSD

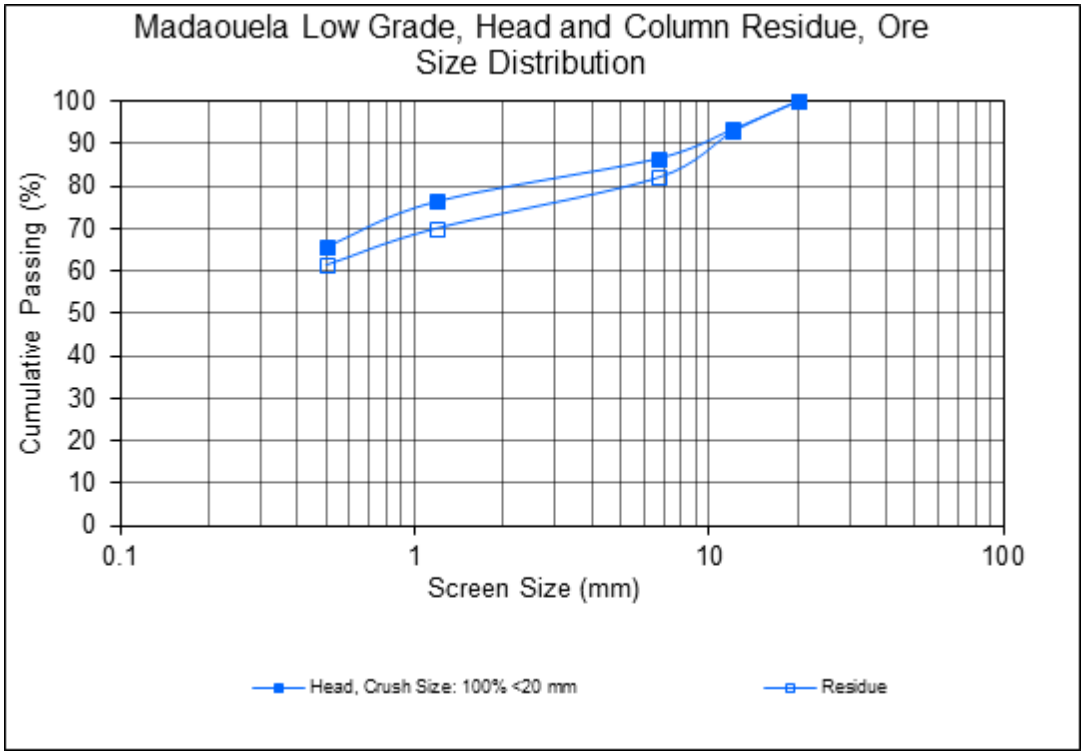


Figure 12-21: Low grade ore column head and residue PSD

The RADOS concentrate head was much coarser with only 31 % of particles finer than 1.18 mm (see Figure 12-22).

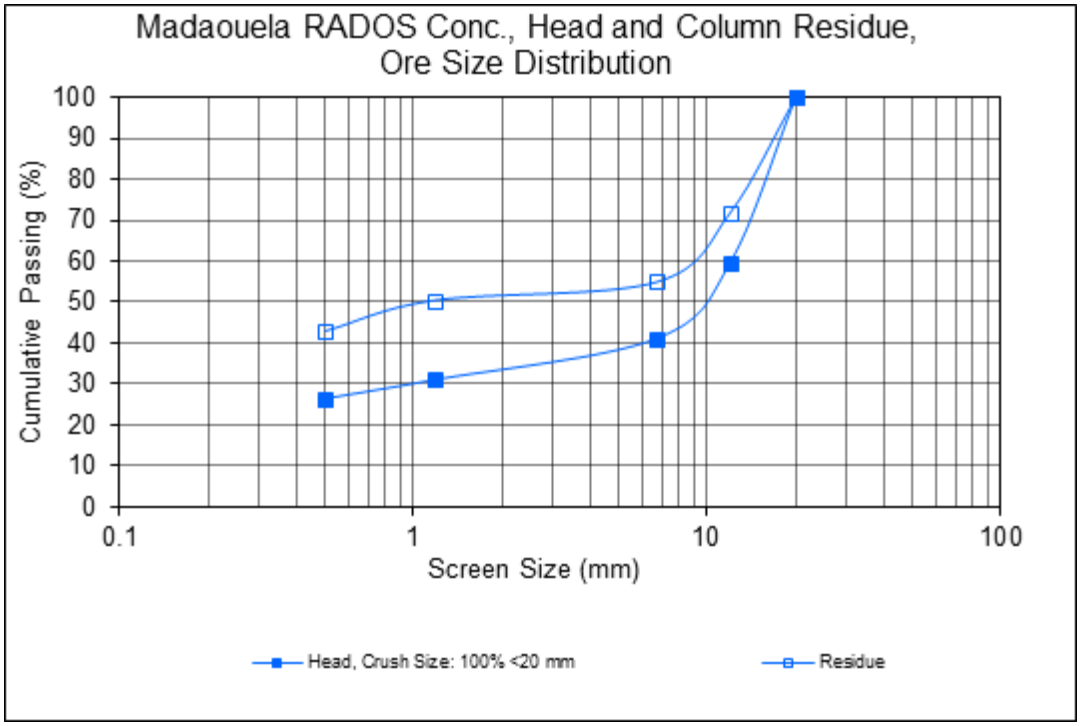


Figure 12-22: RADOS concentrate column head and residue PSD

The uranium distribution for the HG and LG ore head and residue samples across the PSD was very similar. The RADOS concentrate head uranium deportment mirrored the head PSD with a larger proportion of uranium reporting to the coarse fraction when compared to the LG

and HG ore deportments.

The uranium deportment in the residues after all the leach tests was skewed towards the coarse end of the PSD (with 62, 60 and 20 % cumulative uranium concentrations < 6.7 mm for the HG, LG and RADOS residues respectively compared to 79, 82 and 38 % for the head samples, a reduction of 17 - 22 %) indicating that the finer uranium particles were more susceptible to leaching.

12.11.2 Column Leach Residue U-Mineral Characterisation

Six leach residues were analysed to ascertain the quantitative mineralogical characterisation. The U-bearing minerals that are present in the leach residue samples include Ti-rich coffinite, U-rich titanite, U-rich coffinite, with minor amounts of uraninite and autunite.

Autunite makes up the least percentage of the leach residues indicating that is readily leachable under the leach conditions of the tests followed by uraninite for 4 of the 6 tests. For the majority of the leach residues U-rich titanite; Ti-rich coffinite; and coffinite contribute > 80 % of the U in the U minerals in the residue. There is some variation in the proportion of each of the minerals across the six residues.

In all of the residues the U-bearing minerals that were observed in the samples have grains sizes of mostly -38 µm. Most of the U-bearing minerals were found to be partially exposed and liberated. Only minor amounts of U-bearing minerals were found to be locked. The U-bearing minerals in the sample were found to be mostly associated with quartz.

12.12 Uranium Recovery from Pregnant Leach Solution

After leaching a bulk sample under optimised conditions specified by SRK, several uranium recovery options for the clarified pregnant leach solution (“PLS”) were investigated by Mintek.

12.12.1 Ion Exchange - National Institute for Metallurgy Continuous Ion Exchange (“NIMCIX”) Circuit

The bulk leach solution analysis used for the ion exchange testwork is presented in the Table 12-14.

Table 12-14: Bulk leach solution composition

Elements		Bulk leach 1	Bulk leach 2	Elements		Bulk leach 1	Bulk leach 2
Mg	mg/L	4,020	4,900	Co	mg/L	<2	5.6
Al	mg/L	3,890	4,470	Ni	mg/L	10	<2
Si	mg/L	707	714	Cu	mg/L	27	40
Ca	mg/L	608	667	Zn	mg/L	285	286
Ti	mg/L	<2	<2	Mo	mg/L	308	311
V	mg/L	33	63	Pb	mg/L	<2	<2
Fe	mg/L	8,700	10,100	U	mg/l	1,027	1,123

The concentration of molybdenum in the feed (308 - 311 mg/l) affected the efficiency of U_3O_8 recovery in the NIMCIX circuit.

Silica is also present in the PLS above 700 mg/L Si. At this level the silica will have a similar effect on the resin as Mo does. Silica is generally found in acidic solutions at pH around 2 as monosilicis acid, $Si(OH)_4$. When resin contact is made with a siliceous solution, the silica penetrates into the resin beads to various extents depending on the concentration of silica

present. Silica fouling presents a problem in the process of uranium recovery because it affects the kinetics of both loading and elution of the anion exchange resin.

A portion of the Mo and Si which fouls the resin will be eluted during the acid elution stage for U_3O_8 product; however the portion of Mo and Si that remains on the resin will build up over time, as discussed earlier, which ultimately impacts on the efficiency of the U_3O_8 recovery process. Molybdenum polymerised within the resin bead, is not completely removed by acid elution and as a result will build up over time resulting in a reduced uranium recovery. A caustic regeneration step can be employed however this can increase the resin degradation due to the large change in operating pH. Due to these operational issues activated carbon adsorption for molybdenum prior to the NIMCIX circuit was investigated (the performance of the various carbon sizes are displayed in Figure 12-23).

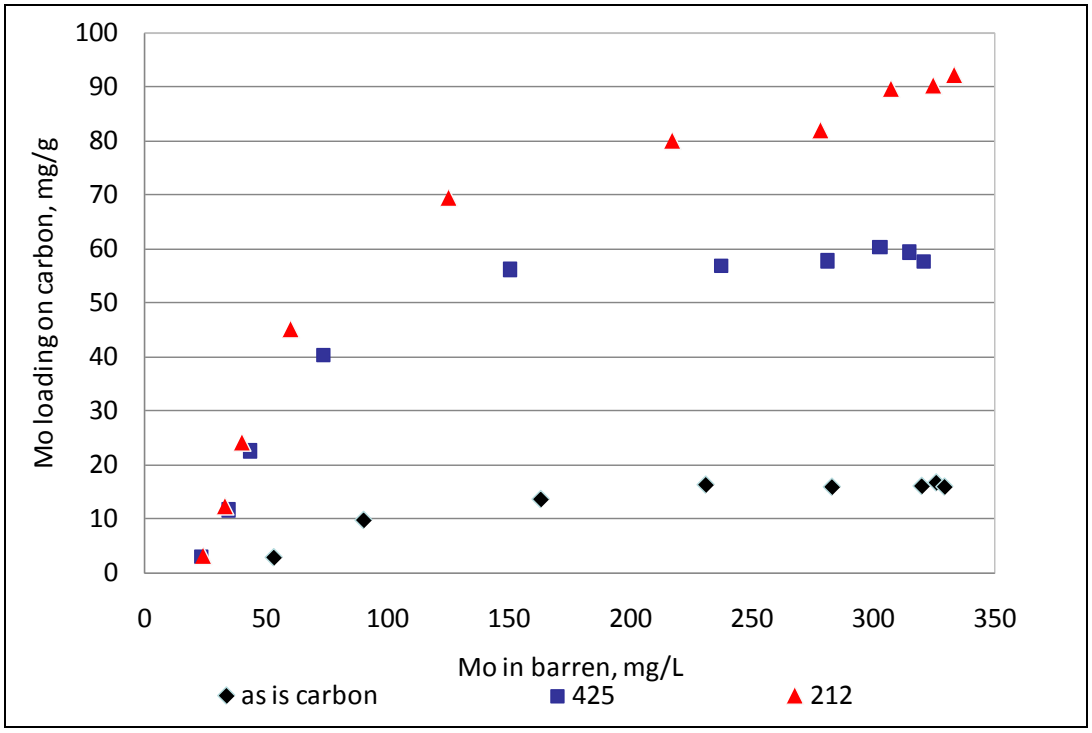


Figure 12-23: Molybdenum removal by activated carbon (@ 1.4 mm, 425 µm and 212 µm activated carbon particle size)

The activated carbon particle size required for efficient molybdenum loading would result in a significant pressure drop through the column resulting in precipitation of dissolved constituents in the column. A filtration stage after the activated carbon column and prior to the NIMCIX circuit may therefore be required to ensure the feed to the NIMCIX circuit is low in total suspended solids (“TSS”). The rate of Mo uptake onto the as received (1,400 µm) that mitigates the pressure loss observed with the finer carbon, the uptake at 20 hours does not correlate with the equilibrium adsorption testwork and has therefore been omitted by the laboratory, taking this into account the optimum residence time for optimal uptake of Mo (45 %) is 12 hours. The maximum Mo loading onto the carbon was 16.5 g/kg in equilibrium with 300 mg/L Mo in solution.

12.12.2 Solvent Extraction ("SX")

The solvent extraction reagents that were used for the test work are commercially available products, Alamine 336 (supplied by Cognis). Shellsol D70 (supplied by Shell Chemicals) was used as the diluent in the preparation of all organic phases and was selected to ensure optimum solubility of organic complexes under the conditions tested. The extractions were conducted at eight different organic to aqueous phase ratios. The PLS composition notably contained Mo (300 mg/L) and Si (700 mg/L).

Observations of solid build up, colour of PLS and the extraction equilibrium isotherm indicated that molybdenum was also extracted at the same time as uranium. Selective extraction was not possible. Further testwork would be required to investigate selective stripping of molybdenum and uranium.

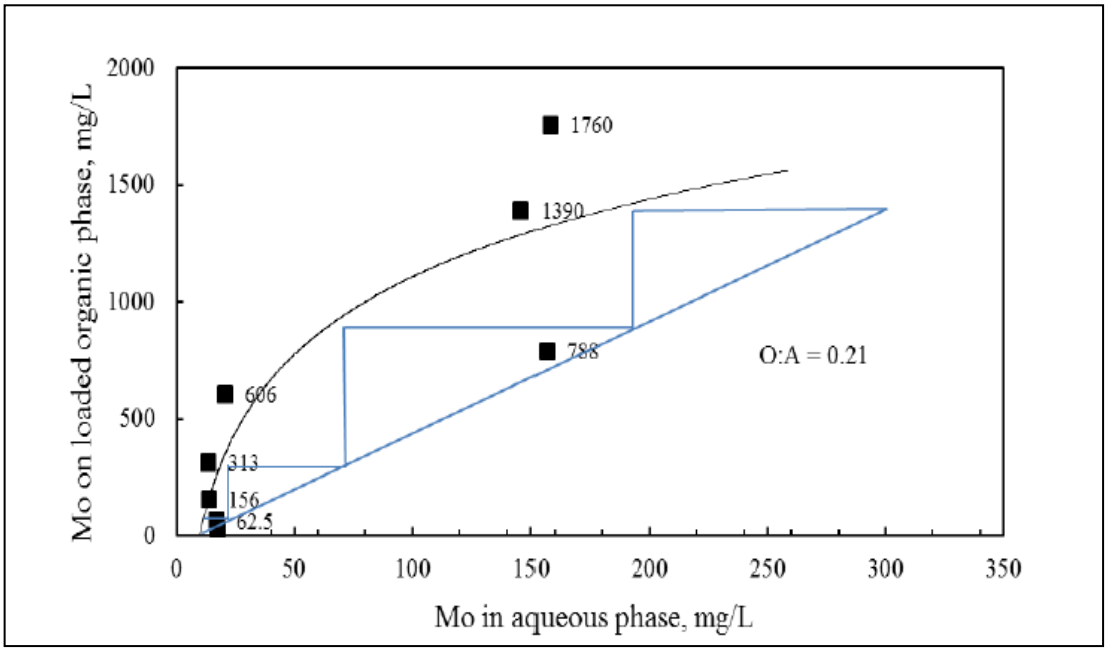


Figure 12-24: Mo extraction equilibrium isotherm using 4(v/v) % Alamine 336, 2 (v/v) % Isodecanol in diluent Shellsol D70

Figure 12-24 illustrates the McCabe-Thiele construction for Mo extraction. At an O:A phase ratio of 0.21, 4 stages are required for Mo removal.

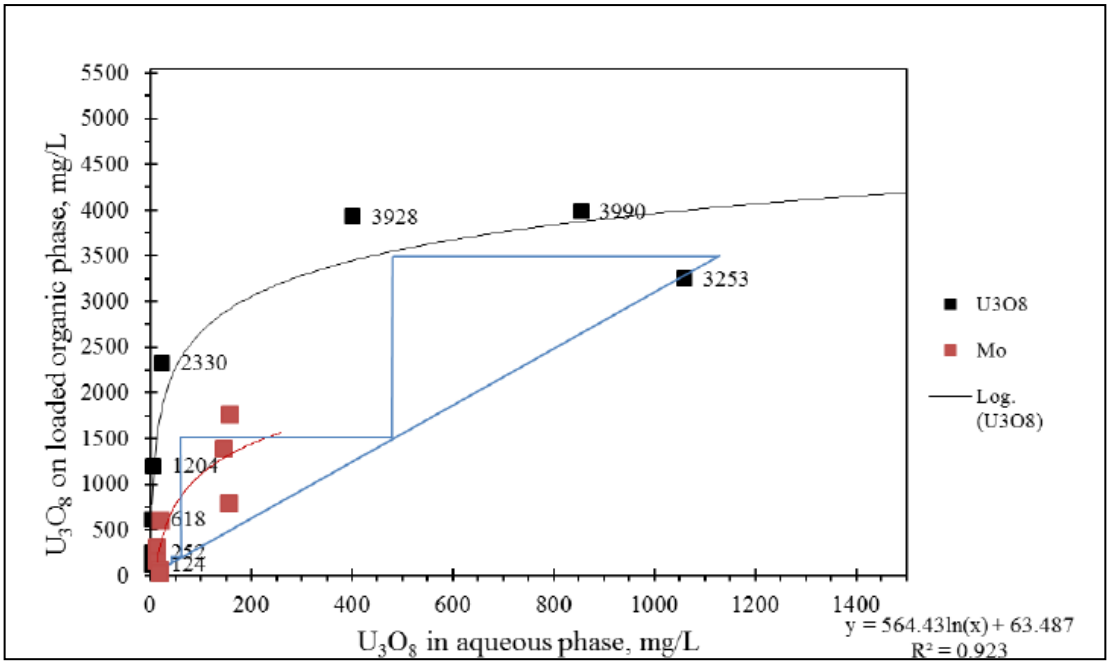


Figure 12-25: U extraction equilibrium isotherm using 4 (v/v) % Alamine 336, 2 (v/v) % Isodecanol in diluent Shellsol D70

Figure 12-25 illustrates that at an O:A phase ratio of 0.36, approx. three stages are required for U_3O_8 removal. Figure 12-25 also includes the Mo extraction equilibrium and indicates that the upgrade in concentration in the organic phase for Mo is almost 2 times greater than the upgrade of U_3O_8. Literature has indicated that the extraction coefficient for Mo (VI) (> 1,000) is much greater than that of U (IV) (< 1) for tertiary amines (Mintek, 2012).

Thus it could be possible to separate Mo and U_3O_8 by selectively extracting Mo prior to U_3O_8 extraction.

12.12.3 Solvent Extraction utilising Cyanex 600

A further solvent extraction approach has been tested. Cytec Industries have developed a reagent, primarily for the copper industry, which is capable of recovering both molybdenum and uranium from high acidity (low pH) PLS (Figure 12-26) with low impurity co-removal. Then preferential stripping can be undertaken to separate these two metals into commercial products with little carry over between the two allowing production of uranium and by-product molybdenum.

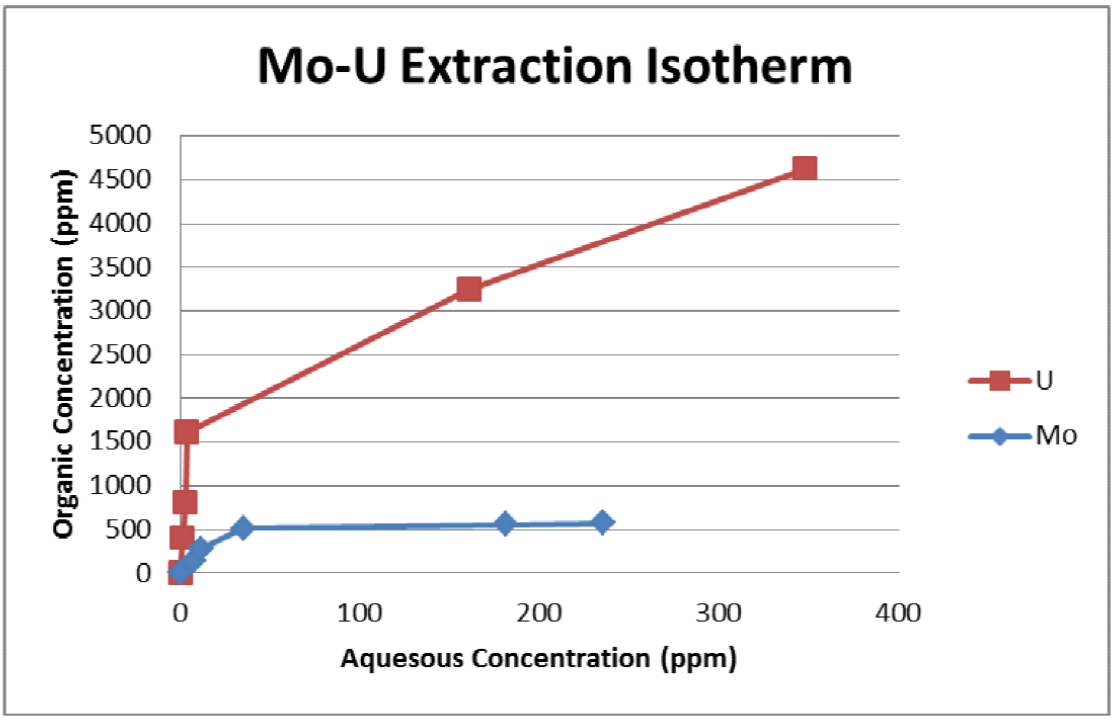


Figure 12-26: Molybdenum and uranium extraction isotherm

Initial results indicate that during a single solvent extraction step 99% of uranium and molybdenum can be recovered with minimial impurities co-extracted, however iron will be extracted to the organic phase at significant levels at aqueous concentrations above 5000 mg/L (Figure 12-27). There is a potential requirement for silica removal and likely requirement for an iron strip (dependent on PLS Fe and Si concentrations, during the testwork the PLS Fe concentration was 5579 ppm) upfront of the product recovery stages.

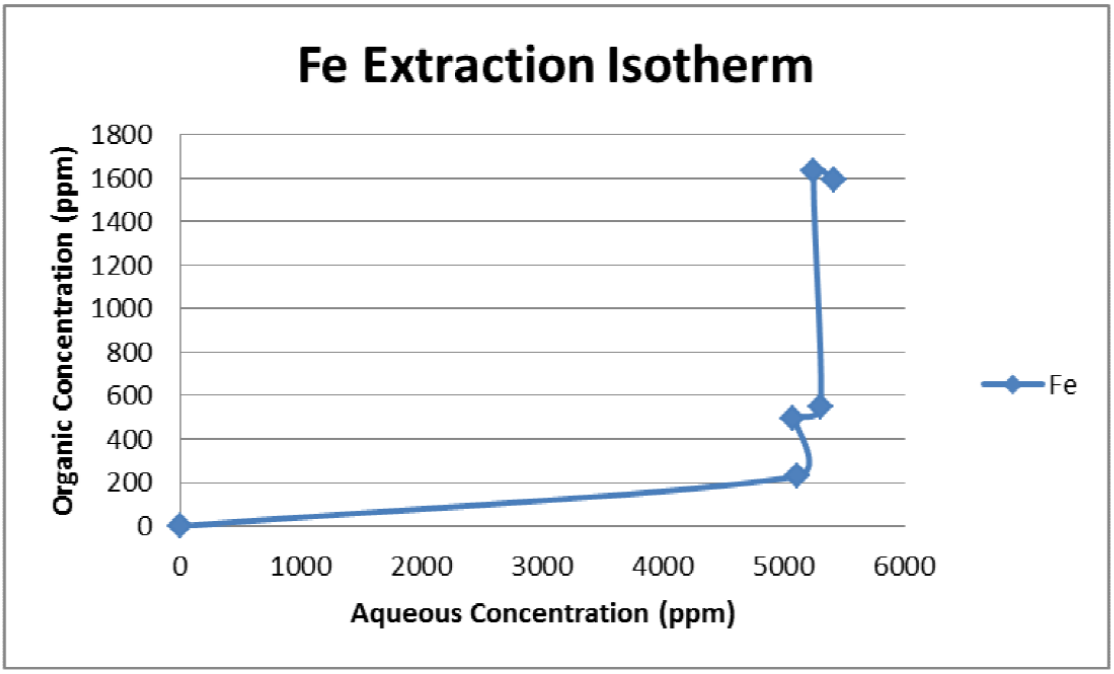


Figure 12-27: Iron extraction isotherm

The molybdenum and uranium extracted can be 2-stage stripped, to produce separate molybdenum and uranium saleable products (Figure 12-28 and Figure 12-29). The initial results indicated greater than 94% recovery is achievable for uranium and 98% is possible for molybdenum.Further testwork and process optimisation has improved the recovery efficiencies to approximately 99% for U and Mo.

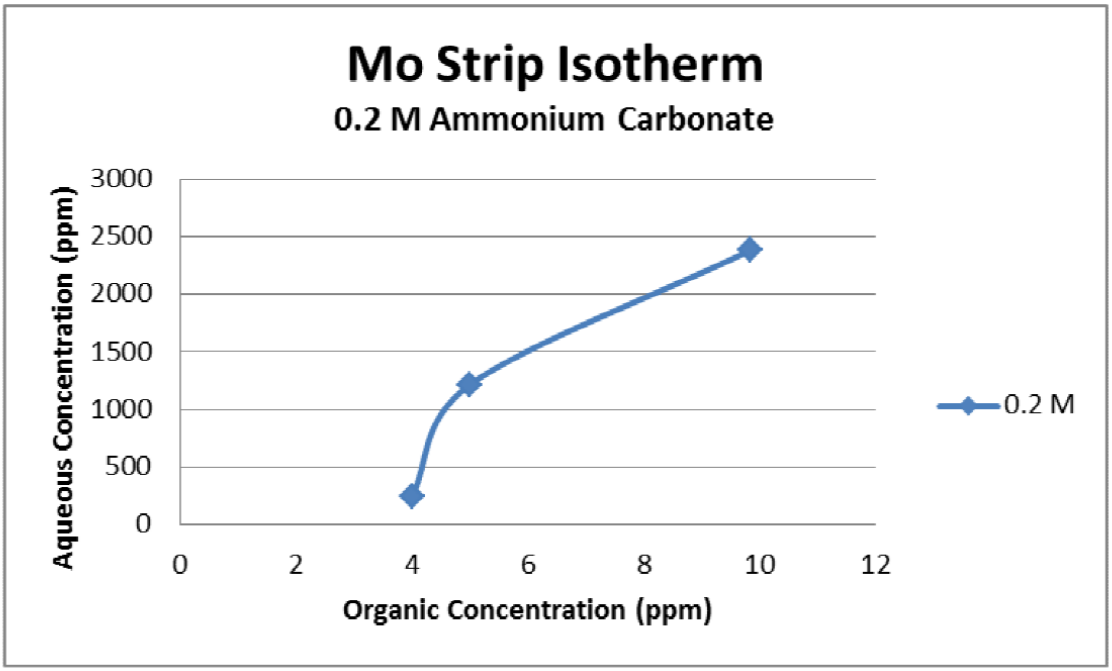


Figure 12-28: Molybdenum strip isotherm

The molybdenum strip was carried out using 0.2M ammonium carbonate, several other alkali strip reagents could potentially be used. The optimum strip reagent will be determined based on the cost of the reagent delivered to site and several processing aspects, including:

- Safety Health and Environmental (SHE);
- Mo recovery;
- impurities co-stripping;
- and U co-stripping.

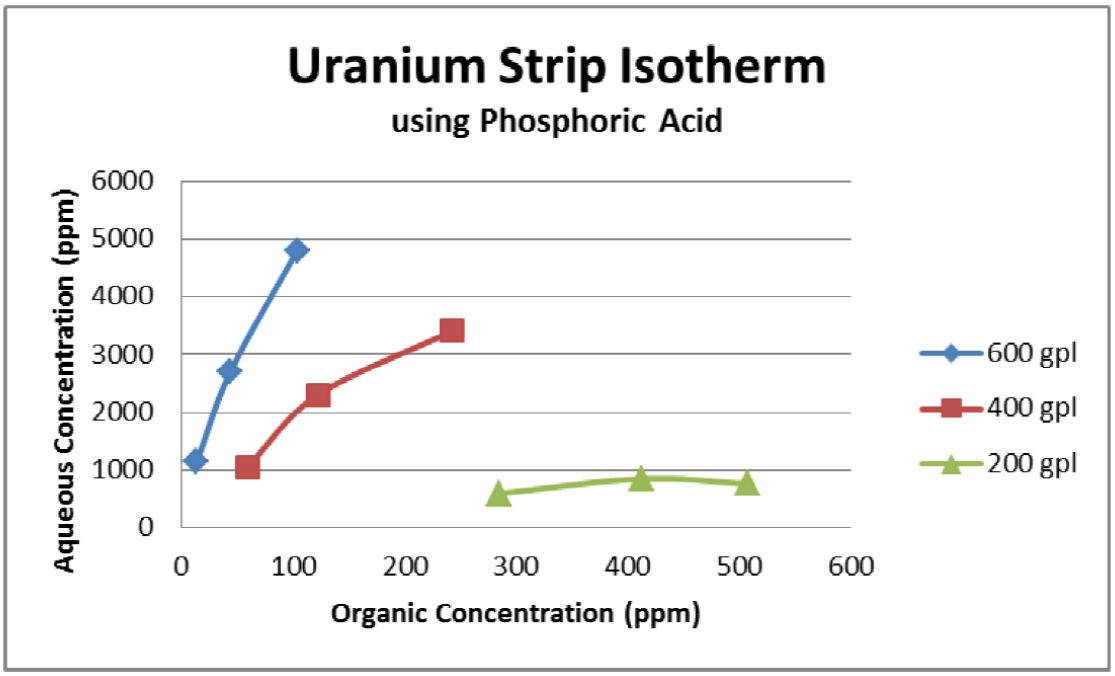


Figure 12-29: Uranium strip isotherm

The optimum phosphoric acid concentration was 600 g/L which resulted in approximately 4800 ppm of uranium in the aqueous phase when in contact with 100 ppm of U in the oraganice phase. The iron contained in the organic phase (approximately 51ppm) was co-stripped with the uranium irrespective of the acid concentration used. The cost benefit analysis of the solvent extraction methodology is still to be finalised, although Cyanex 600 is more expensive than conventional SX reagents (USD 45 versus USD 30 respectively) the Cyanex 600 process losses are lower and it is more efficient. Further testwork and process optimisation is required (including additional stripping stages and improved raffinate uranium recovery) to optimise the recovery of uranium, these improvements are likely to increase the recovery to approximately 99%.

12.12.4 Resin in Pulp ("RIP")

A commercially available RIP-grade resin was selected for the test work, namely, Purolite A500/ 2788. The selectivity series of these resins as provided by the supplier is as follows:

$V_2O_7^{4-} > Mo_8O_{28}^{4-} > UO_2(SO_4)_4^{4-} > UO_2(SO_4)_2^{2-} > Fe(OH)(SO_4)_2^{2-} > SO_4^{2-} > Fe(SO_4)^{2-} > NO_3^- > HSO_4^- > Cl^-$

Uranium recovery was investigated using resin in pulp ("RIP") technology on unclarified leach pregnant leach solution (pulp) at a pH range of 1.5 – 3.5. To attempt to replicate potential large scale operation conditions the resin was pre-fouled synthetically to a level of approx. 12.5 % m/m SiO_2.

The optimal pH value to minimise impurity loading onto the resin and maximise uranium loading was 3.5 pH units. A maximum loading of 74 g/L U_3O_8 on the resin was observed in equilibrium with a barren solution of 690 mg/L U_3O_8. The co-loading of Mo was 1 g/L. After 8 hours of adsorption, 84 % of the U_3O_8 was adsorbed onto the resin. The resin was pre-loaded in a batch with pH 3.5 leached pulp, to produce loaded resin for elution test work. At pH 3.5, the impurities such as Si and Mo which foul the resin, precipitate but a portion of the U_3O_8 also precipitates due to the high concentration present in the PLS. However during the

adsorption stage, the U_3O_8 that precipitated was found to re-dissolve attributing to no U_3O_8 loss.

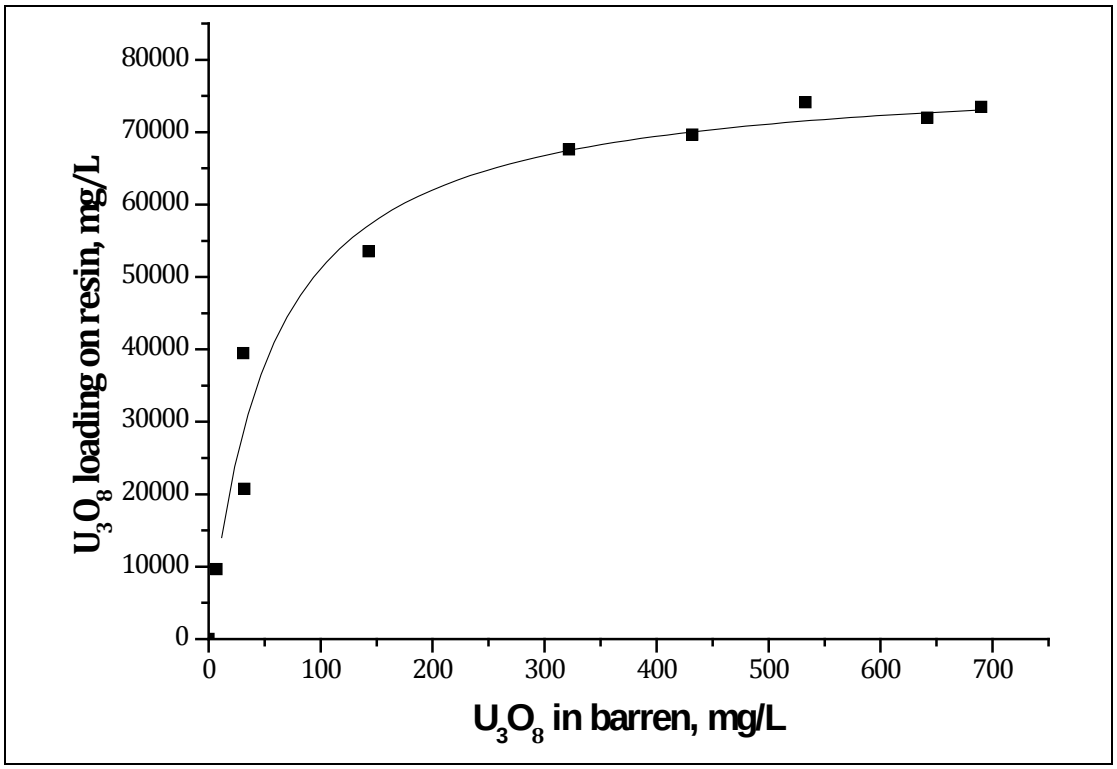


Figure 12-30: Resin in pulp - U_3O_8 equilibrium adsorption

Figure 12-30 above illustrates that the resin loading was fairly constant between 350 and 700 mg/L U_3O_8 in the barren solution (65 – 75 g/l U_3O_8 loading on resin). Further testwork is required to establish the exact conditions necessary for optimum recovery. Equilibrium stripping isotherms were conducted by contacting uranium pre-loaded resins with 110 g/L H_2SO_4 in batch. The eluant to resin ratios used during the test work is shown in Table 12-3.

Table 12-15: Eluant to resin ratios

Point no	Eluant/resin ratio (volume)	Resin volume, mL	Eluant volume, mL
1	10	2	20
2	20	2	40
3	40	2	80
4	80	2	160
5	160	2	320
6	320	2	640
7	640	2	1,280
8	1,000	2	2,000

McCabe Thiele constructions were fitted to the isotherm and it was found that at an eluant to resin ratio of 6.5, a U_3O_8 eluate grade of 12 g/L can be produced in 6 elution stages with a residual U_3O_8 resin concentration of 0.45 g/L. The rate of elution is relatively fast with 100 % elution achieved after a residence time of 4 hours. An elution test was conducted with resin a column to determine the stripping profile of the pre-loaded resin. Within the first 5 BV of

eluate, 14 g/L U_3O_8 was eluted which accounted for 87 % of the U_3O_8 loaded on the resin. The remaining U_3O_8 was eluted from the resin within the next 10 BV of eluant.

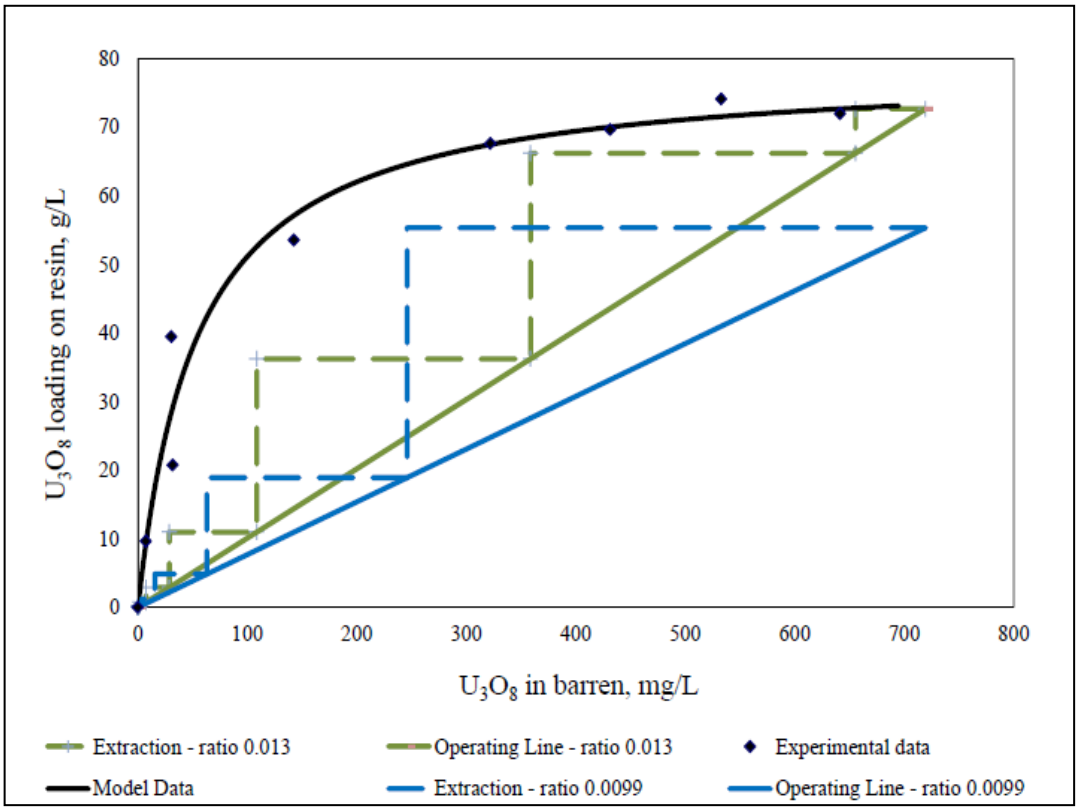


Figure 12-31: Equilibrium adsorption isotherm for U_3O_8 fitted to McCabe Thiele diagram

McCabe Thiele constructions fitted to the equilibrium adsorption isotherm indicate 5 stages of adsorption would be required to obtain resin loadings of 73 g/L from a feed U_3O_8 concentration of 719 mg/L with U_3O_8 in the barren solution of < 7 mg/L. It may be possible to achieve U_3O_8 recoveries in 4 stages and achieve a barren U_3O_8 concentration of 4 mg/L in stage 4, however larger resin volumes would be required. The cost benefit analysis and impact on downstream elution is still to be evaluated.

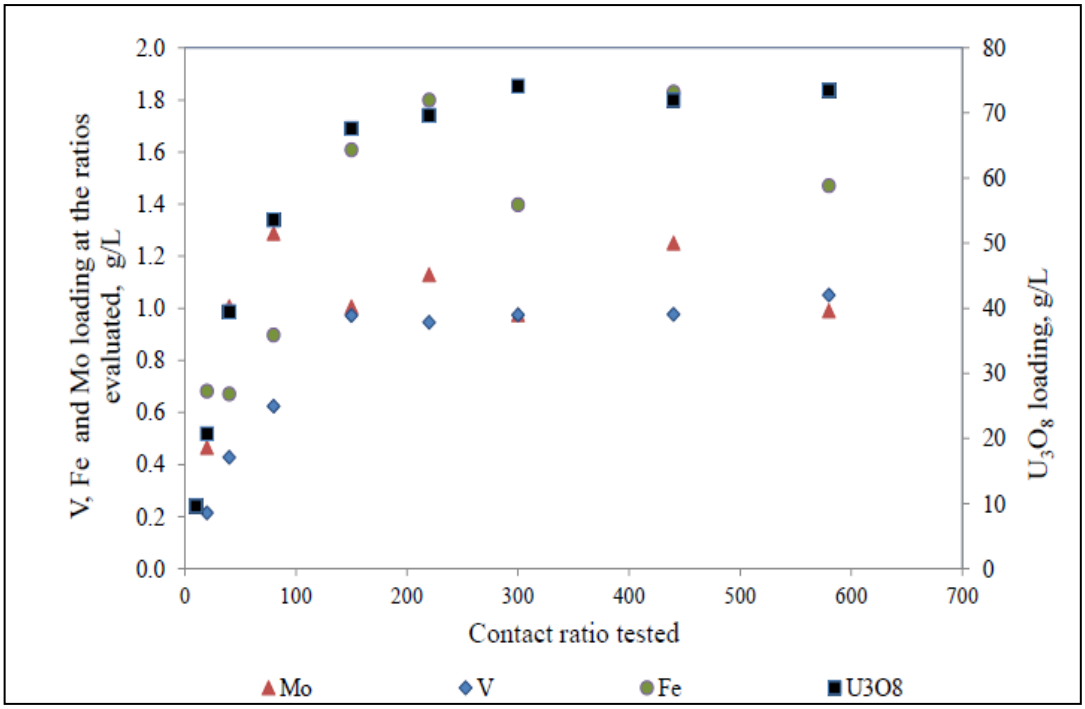


Figure 12-32: Impurity loadings versus U_3O_8 loadings for the equilibrium ratios evaluated

Based on the results obtained from the equilibrium adsorption test and assuming a RIP plant operation of 1 % resin v/v 1 g Mo / kg resin will remain on the resin after the acid strip, further testwork is required to assess the build-up of the impurities over the long term.

12.12.5 Column Elution

The column elution test was conducted by transferring a portion the U_3O_8 loaded resin into a glass elution column and pumping 110 g/L H_2SO_4, as the eluant, through the resin bed in a downward plug flow manner. The eluant was pumped at a rate of 2 Bed Volumes/hr ("BV/h") until a total of 31 Bed Volumes ("BV") was pumped through the resin bed.

Solid Liquid Separation Testwork Objectives

The objective of the testwork was to investigate the static settling-thickening response of the Bulk Leach Final Pulp slurry, as well as the filtration (vacuum and pressure) properties. The Bulk Leach slurry was prepared per the test conditions of Leach Test 32 using Comp.

Sample Receipt and Description

The Bulk Leach Final Pulp test sample contains NORM and was produced through an acid leaching process performed at the SGS Minerals Services' Lakefield facility.

Particle size determinations were performed on representative aliquots retrieved from the samples using a Malvern Mastersizer 2000 laser particle sizer (Table 12-16).

The "as received" Bulk Leach Final Pulp sample was partially thickened (without the use of reagents) to collect sufficient liquor for dilutions in the subsequent liquid-solid separation (settling) tests. A sample aliquot was retrieved, filtered and washed; the wet cake produced during the filtration was analyzed for solids content in a Halogen Moisture Analyzer HR83 (Mettler-Toledo), and in a conventional oven for comparison purposes. One of the dried

samples was submitted for specific gravity ("SG") determination using a nitrogen-purged "Model 1305 Micromeritics" multi-volume pycnometer.

Table 12-16: Sample characterization

Sample	K_{80} micron
Final Bulk Leach Pulp	218

Static Settling-Thickening Tests

Methodology

The static settling-thickening tests involved three distinct stages:

- Flocculant screening, selection and confirmation;
- Static settling-thickening tests-thickener feed solids optimization; and
- Static settling-thickening tests-flocculant dosage optimization.

Reagent Screening and Selection

Commercially available flocculation reagents were tested ("screened") by performing small scale settling tests. Settling behavior of the resulting slurries was examined in terms of relative effectiveness of the reagents with respect to particle aggregation, flocc formation, resulting structure characteristics and supernatant clarity

Static Settling Tests

Standard Kynch settling tests (with rakes) were performed; the sample was tested at variable feed slurry solids densities and at variable reagent additions as selected based on the reagent screening results. The test data produced consisted of interface height and feed slurry density profile under a given flocculant dosage. Thickener sizing was performed using the Tallmadge and Fitch method. The thickener unit area and hydraulic capacity were calculated.

Test Results

Flocculant Screening Tests Results

The flocculant screening tests results indicated that the Bulk Leach Final Pulp sample required the use of a low-charge-density anionic flocculant – a relatively common reagent called Magnafloc 10, manufactured by CIBA was tested. The sample settled fast with a clear supernatant using a reagent addition of 120 g/t.

The reagent additions mentioned in this section were as used during the screening and selection process, hence not optimized.

Static Settling-Thickening Tests Results

The following settling-thickening flocculant dosage conditions reported were determined during the static settling-thickening testing. These conditions produced the best settling-thickening results and were deemed optimized on that basis, with reference to the test procedures applied as described in this report. It is possible that further evaluation of the data, particularly with a view to the commercial application and prospective equipment, could provide opportunities for further optimization. It is recommended in this context that the test data that are summarized and reported in this report be reviewed prior or during the design as to ensure realistic sizing with respect to the commercial plant throughput requirements. The results of the settling-thickening (best conditions-as tested) are summarized in Table 12-17.

Bulk Leach Final Pulp slurry sample settled well in the presence of 120 g/t Magnafloc 10 producing a ~51 % (wt/wt) solids underflow from a ~4 % (wt/wt) (autodiluted) thickener feed. Relevant thickener data included: 0.09 m^2/tpd thickener underflow unit area (TUFUA), 0.03 m^2/tpd thickener hydraulic unit area ("THUA") and 790 m^3/ m^2 /day initial settling rate ("ISR"). The resulting supernatant was clear.

Note: It should be noted that the static settling tests results and subsequent design criteria do not incorporate any safety or scale up factor. Depending on factors such as throughput, cost, risk tolerance, etc., the equipment suppliers may require additional data, including continuous thickener testing, rheology study, etc. in order to establish the final design criteria.

Table 12-17: Summary of results - best conditions (as tested)

Sample	Flocculant CIBA	Dosage g/t	Feed[1] %wt.	U/F[2] %wt	TUFUA[3] m^2/t/day	THUA[4] m^2/t/day	ISR[5] m^3/m^2/day	Supernatant[6] visual
Final Bulk Leach Pulp	Magnafloc 10	120	4	51	0.09	0.03	790	Clear

[1]Autodiluted Thickener Feed [2]Ultimate Underflow Density [3]Thickener Underflow Unit Area thickener Hydraulic Unit Area [5]Initial Settling Rate [6]Clarity after 60 minutes

12.12.6 Vacuum Filtration

Test Procedures

Vacuum filtration testing was to be carried out on the Bulk Leach Final Pulp sample produced by thickening the "as received" test-samples under conditions deemed optimized through static settling-thickening testing. The supernatant was then decanted from the thickened pulp and only the thickened "underflow" was used as filter test-feed.

The testwork was to be carried out using a filter cloth selected through preparatory scoping tests.

Standard vacuum filter ("leaf-pour-on") testing was to be carried out using a 113 mm diameter (Pannevis) Leaf Filter. The tests were intended to be performed with varying slurry volumes to yield a range of filter cake thicknesses from ~10 mm to ~24 mm. The tests were intended to be performed with varying slurry volumes to yield a range of filter cake thicknesses from ~7 mm to ~13 mm. These cake thicknesses were to be determined through preparatory scoping tests. Drying times in both cases were to be varied to obtain a range of cake moistures. Test data to be produced included vacuum level during form time and dry time, form time, drying time, cracks/separations from the filter wall, filtrate volume, cake thickness, wet cake weight, dry cake weight and tare weight.

Test Results

A selection of filter cloths was tested to determine a suitable cloth to use in the standard "leaf-pour-on" vacuum filtration testing program. Bulk Leach Final Pulp slurry sample was fed as a pulp that was thickened to ~45 % (wt/wt) solids using Magnafloc 10 at 120 g/t, but the sample was not amenable to vacuum filtration due to limited cake formation after an extended period of elapsed form time (>12 minutes) when filtered on a selection of filter cloths of varying pore sizes; therefore, no sizing data were calculated or reported.

Test Procedures

Pressure filtration testwork was performed on the Bulk Leach Final Pulp slurry, aiming to determine the lowest achievable cake moisture levels and to produce baseline data for general information purposes only. Pressure filtration cake washing was not performed.

The pressure filtration testing was performed using a filter cloth selected through preliminary scoping tests.

The test sample was produced in bulk using conditions resulting from optimum static settling-thickening testing. The pressure filtration tests were performed at 60 psig and 100 psig.

The test apparatus consisted of a "Shriver Unit 3" filterpress (Eimco) and necessary ancillaries: feed tanks, pumps, valves, pressure gauge, etc.

A pressure of 15 psig pressure was instantly applied (time zero), then increased to 60 psig or 100 psig accordingly as the test-design pressure. The filtrate collection rate was recorded automatically using an electronic analytical balance.

The scoping test criteria were:

- Filtrate flow rate ~ 10-20 % of the previous reading (i.e. deep decrease); or
- Filtrate flow rate ~ less than 5 % of the initial (i.e. test-start) value.

At the end of the test the pressure was released. The filtrate was collected, weighed, and the volume (hence density) was determined.

The solids were collected, weighed wet then dried to constant weight.

The data were entered and the test quality was assessed based on mass balance accuracy.

Dry solids capacity and filter capacity were calculated using generally accepted procedures.

Test Results

The bulk pulp feed to the pressure filter for Bulk Leach Final Pulp was targeted at 51 % (wt/wt), which was based on the ultimate underflow density achieved through settling-thickening testing. The actual solids density was determined later to be 45 % (wt/wt) after flocculation.

The results of the Shriver pressure filtration tests are summarized in Table 12-18. .

Test 1 was performed at 60 psi of air pressure. The cake moisture was 24.3 % (wt/wt) for a 3.7 cm cake thickness. The predicted dry solids capacity was 52 kg/m^{2}h.

Test 2 was performed at 100 psi of air pressure. The cake moisture was 24.7 % (wt/wt) for a 3.7 cm cake thickness. The predicted dry solids capacity was 51 kg/m^{2}h.

Test 3 was also performed at 100 psi of air pressure. The cake moisture was 24.7 % (wt/wt) for a 4.4 cm cake thickness. This cake was semi-cohesive. The predicted dry solids capacity

was 55 kg/m^{2}h.

There was a significant amount of solids present in the filtrate from the pressure filtration tests. There was an average of 9 % (wt/wt) of solids in the filtrate. However, the solids were released only in the initial 10 seconds of elapsed pumping time, so filtrate clarity may not be a problem during continuous plant operation.

The pulp sample was not compatible with the smallest (1.4 cm) test chamber size because of plugging due to the coarse particle size.

Table 12-18: Summary of pressure filtration results

Test ID	Test 1	Test 2	Test 3
Test Conditions and Data Summary			
Testing Pressure, psig	60	100	100
Filter diameter, m	0.0725	0.0725	0.0725
Actual cake thickness, m	0.037	0.037	0.044
Filter area, m^2	0.0083	0.0083	0.0083
Actual cake volume, L	0.153	0.153	0.182
Cake wet weight, kg	0.285	0.282	0.303
Cake dry weight, kg	0.216	0.212	0.228
Wet cake solids, % wt.	75.7	75.3	75.3
Wet cake density, kg/L	1.865	1.848	1.670
Dry cake density, kg/L	1.412	1.391	1.257
Filtrate volume, L	0.222	0.250	0.298
Filter loading, L/m^2	45.4	48.8	58.1
Feeding time, h	0.50	0.50	0.50
Filter Throughput versus Conditions			
Feed Pressure, psi	60	100	100
Cake thickness, cm	3.7	3.7	4.4
Feed solids, % wt.	44.6	44.6	44.6
Final wet cake moisture, % wt.	24.3	24.7	24.7
Dry solids capacity, kg/m^2/h	52	51	55
Filter capacity, L/m^{2}h	91	98	116

Cloth: Micronics Engineered Filtration, Cloth No. 798180, Polyester, Warp/Weft: Mono/Mono, Weave: Satin, Finish: Calendered, Weight: 9.77 oz/yd^2, Permeability: 190 cfm

12.12.7 Solvent Extraction & Precipitation Testing

Bulk PLS (both from the solid-liquid separation program and combined PLS from the leach tests) was used to generate a uranium extraction isotherm using tertiary amine extractant, specifically a solvent consisting of 4 % Alamine 336, 2 % isodecanol in Isopar M diluent. Extraction contacts were carried out at various phase ratios in hand-shaken separatory funnels for 4 minutes. The phases were allowed to separate and samples of organic and aqueous were submitted for analysis.

The feed PLS composition is listed in Table 12-19. At an A/O of 2, three extraction stages are predicated with an estimated loaded organic concentration of 2,600 mg/L from 1,300 mg/L in the aqueous feed.

Table 12-19: SX feed PLS composition

Extraction Feed	U mg/L	Mo mg/L	Fe mg/L	V mg/L
	1,300	340	14,000	69

The relevant data from the extraction isotherm test are listed in Table 12-20; no significant iron (or other impurities such as vanadium) was found to co-load in the organic phase, however molybdenum extraction was significant.

A stripping isotherm was also generated, using 1 m sodium carbonate (Na_2CO_3). Generally the stripping was efficient with a steep isotherm: at an O/A of 10, a strip liquor containing 28 g/L U is expected with 2 stripping stages. There was considerable uranium left in the organic phase, even at low O/A ratios, meaning a high concentration of uranium in the recycled organic. The data for the series of shakeout tests conducted are listed in Table 12-21, Mo was found to strip in the same relative proportion to uranium.

A single uranium precipitation test was conducted from a bulk strip liquor using sodium hydroxide. The mixture was allowed to react for 6 hours with solution samples taken at 1, 3 and 6 hours. The final solution was fully analyzed by ICP and the recovered precipitate analyzed for uranium by XRF and an ICP scan was performed to quantify deportment of impurities.

Table 12-20: Extraction isotherm data

O/A Ratio			Feed	1/8	1/4	1/2	1/1	4/1
Temperature, °C				22	22	22	23	23
Equilibrium pH			1.24	1.3	1.28	1.26	1.27	1.31
Organic IN Organic OUT		mL		30	50	100	100	200
		mL		29	50	99	98	200
Aqueous IN Aqueous OUT (equil)		mL		240	200	200	100	50
		mL		240	200	200	98	49
Initial Disengagement[2] Final Disengagement		sec sec		0	0	5	5	10
				30	30	60	45	90
Emulsion Crud		mm		none none	none none	none none	none none	none none
Phases	Aqueous	color clarity	green clear	green slt hazy	green hazy	green hazy	grn yel stl hazy	grn yel clear
	Organic	color clarity		green clear	green clear	green slt hazy	slt grn hazy	white slt hazy
Sample ID			**EF**	**1**	**2**	**3**	**4**	**5**
Uranium	Organic Calc'd Org. Aqueous	mg/L mg/L mg/L	1300	2,800 2,480 990	3,120 2,880 580	2,560 2,410 96	1,280 1,290 12	320 320 <5
	Loaded	%		24	55	93	99	100
	Distribution Coeff. (O/A) Calc. Aq. In			3 1328	5 1360	27 1363	107 1266	64 1285
Mo	Organic Calc'd Org. Aqueous	mg/L mg/L mg/L	340	768 880 230	576 560 200	536 520 78	312 310 28	75 80 80 15
	Loaded	% mg/L		32	41	77	92	96
	Distribution Coeff. (O/A) Calc. Aq. In			3.3 230	2.9 200	6.9 78	11.1 27	5 15
Iron	Organic Aqueous	mg/L mg/L	14,000	1.6 14,000	0.7 14,000	1.3 14,000	4.8 14,000	16 14,000
	Loaded	%						
	Distribution Coeff. (O/A) Calc. Aq. In	mg/L		<0.00 14,000	<0.00 14,000	<0.00 14,001	<0.00 13,725	<0.00 13,784

Table 12-21: Stripping isotherm data

O/A Ratio			Feed	10/1	5/1	2/1	1/1	1/3
Temperature, °C				19.5	19.4	20.4	20	20
Equilibrium pH				8.4	9.47	10.18	10.55	11
Organic IN Organic OUT		mL mL		200 202	150 148	60 60	50 51	25 24
Aqueous IN Aqueous OUT (equil)		mL mL		20 18	30 31	30 30	50 49	75 76
Initial Disengagement[2] Final Disengagement		sec sec		5 20	0 10	0 30	5 25	5 50
Emulsion Crud				none grn i/f	none slt i/f	none none	none none	none none
Phases	Aqueous	color clarity		yel slt hazy	yel clear	yel clear	lt yel clear	slt yel slt hazy
	Organic	color clarity	yel clear	none hazy	none slt hazy	none clear	none clear	none clear
Sample ID				**1**	**2**	**3**	**4**	**5**
Uranium	Organic (calc'd) Aqueous	mg/L mg/L	3,750	1,490 25,000	1,080 13,000	1,050 5,400	1,080 2,700	870 960
	Stripped	%		59.9	71.6	72	70.6	77.7
	Distribution Coeff. (A/O)			16.8	12	5.1	2.5	1.1
	Separation U/Mo			0.9	0.4	0.4	0.6	0.3
Molybdenum	Organic (calc'd) Aqueous	mg/L mg/L	700	260 4,900	100 2,900	100 1,200	130 580	60 210
	Stripped	%		62.7	85.9	85.7	81.1	91.7
	Distribution Coeff. (A/O)			18.8	29	12	4.5	3.5

The composition of the feed solution to the precipitation test is listed in Table 12-22 and the barren solution at the completion of the test is listed in Table 12-23. Based on the feed and barren uranium concentrations the precipitation efficiency was 99.5 % it is possible that with longer duration even more complete uranium precipitation would have occurred, further testwork has indicated that 99.8% is achievable.

Table 12-22: Precipitation test feed solution composition

Feed Strip Solution			
Ag mg/L	< 2	Mo mg/L	1,000
Al mg/L	< 5	Na mg/L	48,000
As mg/L	< 3	Ni mg/L	< 1
Ba mg/L	< 0.07	P mg/L	< 8
Be mg/L	< 0.2	Pb mg/L	< 10
Bi mg/L	< 2	Sb mg/L	< 3
Ca mg/L	4.7	Se mg/L	< 3
Cd mg/L	< 0.2	Sn mg/L	< 2
Co mg/L	< 0.8	Sr mg/L	< 0.2
Cr mg/L	< 0.5	Ti mg/L	< 2
Cu mg/L	< 3	Tl mg/L	< 3
Fe mg/L	8.2	U mg/L	4,400
K mg/L	< 4	V mg/L	< 0.2
Li mg/L	< 2	W mg/L	< 2
Mg mg/L	15	Y mg/L	< 0.4
Mn mg/L	0.9	Zn mg/L	< 2

Table 12-23: Precipitation barren solution composition

Final Barren			
Ag mg/L	< 2	Mo mg/L	890
Al mg/L	< 5	Na mg/L	64,000
As mg/L	< 3	Ni mg/L	< 1
Ba mg/L	< 0.07	P mg/L	< 8
Be mg/L	< 0.2	Pb mg/L	< 10
Bi mg/L	< 2	Sb mg/L	< 3
Ca mg/L	< 0.9	Se mg/L	< 3
Cd mg/L	< 0.2	Sn mg/L	< 2
Co mg/L	< 0.8	Sr mg/L	< 0.2
Cr mg/L	< 0.5	Ti mg/L	< 2
Cu mg/L	< 3	Tl mg/L	< 3
Fe mg/L	< 0.2	U mg/L	23
K mg/L	9	V mg/L	< 0.2
Li mg/L	< 2	W mg/L	< 2
Mg mg/L	< 0.07	Y mg/L	< 0.4
Mn mg/L	< 0.04	Zn mg/L	< 2

The precipitate analysis is listed in Table 12-23 above. The uranium content was determined to be 59 %, equivalent to ~80 % sodium diuranate (or 70 % U_3O_8).

12.12.8 Tailings Mineralogy

A mineralogy program was completed on the Test 18 leach residue to speciate the unleached uranium. The sample was found to consist mainly of quartz (~72.5 %), K-feldspar (~16.9 %), clays (~4.4 %), pyrite (~2.7 %) and plagioclase (~1 %). U-bearing minerals accounted for ~0.1 %. Uranium was hosted by U-Ti-P-Fe oxide mixed phase (~54 %) and by U-bearing clay (~46

%). Liberation of uranium minerals was very poor at ~7 %, and such particles are very fine (< 30 µm). The main association of uranium minerals is as complex particles, and middlings with quartz. The mineralogical observations suggest that potential uranium loss can be attributed to the fine-grained nature of uranium minerals (~44 % are less than 30 µm), their occurrence as complex middling particles, and the presence of discontinuous rims and fine-grained inclusions in quartz; and substantial presence of U in the uranium-bearing clay.

12.13 Conceptual Flowsheets

Four alternative options for the metallurgical processing have been considered and based on the results from the testwork one conceptual flowsheet has been developed.

13 MINERAL RESOURCE ESTIMATES (ITEM 14)

13.1 Introduction

For the narrow flat lying mineralisation that occurs in most of Madaouela Uranium Project deposits (MSNE, MSCE, MSEE, and Maryvonne), a 2D approach has historically been used as an appropriate resource modelling technique for the potential Underground mining method. A more recent update of Marilyn/Marianne (M&M) has used a 3D unfolding approach allied with conventional block kriging, in order to smooth the process to mine scheduling. Both approaches are suitable for the flat lying mineralisation. For the potential Open Pit mining at Miriam a 3D recoverable resource approach (UC) was utilised.

Mineral resources in this report are defined using the convention of "eU" in kg/t for grade. The eU distinquishes concentration as being uranium concentration largely defined from gamma spectrometry with only a small portion of core chemically assayed. This does not compromise the integrity of the result but only seeks to demonstrate source of the information. This value can be related to eU_3O_8 concentration by a factor of 1.17924. Uranium concentrations determined by chemical assay are shown simply as U or U_3O_8.

13.2 Methodologies

13.2.1 2D Estimation (Underground: MSNE, MSCE, MSEE, and Maryvonne)

For each retained intersection within a drillhole, three variables (traditionally called "Service Variables") are created:

- Thickness Th (length of the intersection),
- average eU grade G, and
- Accumulation GTh (product of G and Th)

These three service variables Th, G and GTh are correlated, so an estimation of each variable separately by Ordinary Kriging is not optimal, and can be biased.

The best linear estimation method based on historical assessment is co-kriging of the two additive variables Th and GTh, followed by an estimation of the grade, obtained by dividing the estimated GTh by the estimated Th. The co-kriging method is powerful, but can be sometimes complex, because it requires the use of the co-variogram between Th and GTh.

In some cases, when Th and G are not spatially correlated, it is possible to simplify the co-kriging approach, by evaluating the regression between GTh and Th. Co-kriging of GTh and Th is replaced by the independent Ordinary Kriging of Th and R which are spatially uncorrelated. This simplified method is generally known as the Residuals method.

GoviEx has used the Residuals method in previous estimations, with the UTIMINE software after careful checking of the assumption of non-correlation between G and Th and between Th and R. They also used as validation the full co-kriging method with the Isatis® software.

SRK has updated the Madaouela resources using the Co-Kriging method with the Isatis® software. The published resources reported herein are estimated by co-kriging. However, in the case of MSNE, MSCE, MSEE, and Maryvonne, the residual method was used for checking and validation.

GoviEx defined for each hole the mineralised intersections, based on a eU cut-off grade of 0.4 kg/t applied to 20 cm intervals, with a minimum thickness of 40 cm. In general there is

only one intersection, generally located at or just above the Talak/Guezouman contact. When two or more intersections exist in one hole, only one is retained, based on the geological interpretation of the continuity of the mineralisation. In most cases the retained intersection is the lowermost closest to, but still above, the Talak/Guezouman contact. Depending on the future mining method, some of these "secondary" intersections may well be minable. By ignoring them, the present estimation is considered slightly conservative. Low grades are also recorded within the Guezouman/Akokan Channel Unit (UA) and at the UA/Talak contact. They are taken into account in the present update, when they appear to be minable together with the main "seam".

For each retained intersection, the three Service variables Th, GTh and G are created: When no intersection is found in a hole located within the mineralised envelope, Th is set to 0.4 m (minimum thickness) and grade and accumulation are given a 0 value, so that the continuity of the mineralisation is maintained. (An alternative approach would be to create internal waste zones within the envelope).

The interpreted mineralised envelope for all deposits, with the exception of MM and Miriam, was defined by GoviEx, as a polygonal contour, and corresponds approximately to a cut-off of 0.09 m kg/t on GTh. The polygonal limits are considered as hard limits for the estimation process with only data from within a contour considered for the estimation within that contour.

13.2.2 3D Estimation (Underground: M&M)

For the updated M&M Mineral Resource model, initially a similar approach was taken in terms of data pre-processing to the 2D modelling described above with mineralised intersections defined for each hole, based on the 20 cm sample interval and a eU cut-off grade of 0.4 kg/t, with a minimum thickness of 40 cm. In general there is only one intersection, generally located at or just above the Talak/Guezouman contact. When two or more intersections exist in one hole, only one is retained, based on the geological interpretation of the continuity of the mineralisation. In most cases the retained intersection is the lowermost closest to, but still above, the Talak/Guezouman contact. Depending on the future mining method, some of these "secondary" intersections may well be minable. By ignoring them, the present estimation is considered slightly conservative. Low grades are also recorded within the Guezouman/Akokan Channel Unit (UA) and at the UA/Talak contact. They are taken into account in the present update, when they appear to be minable together with the main "seam".

The M&M estimate differs in that the hangingwall (HW) and footwall (FW) intercepts defined by the mineralised intersections, were used to define a physical 3D "seam" model describing the upper and lower extent of mineralisation (Figure 13-1). Modelling was carried out using the Vein Modelling tool in the Leapfrog Geo® software. Following the 3D modelling of the mineralised domain and the main geological contacts, an unfolding routine was used in the Isatis® software using the Guezouman -Talak contact as the unfolding surface. Estimation of Co-variogram models and Co-kriging were carried out as for the other models and the final 3D block model was then "re-folded" back to the true 3D space for Mineral Resource reporting and export to the mine planning software.

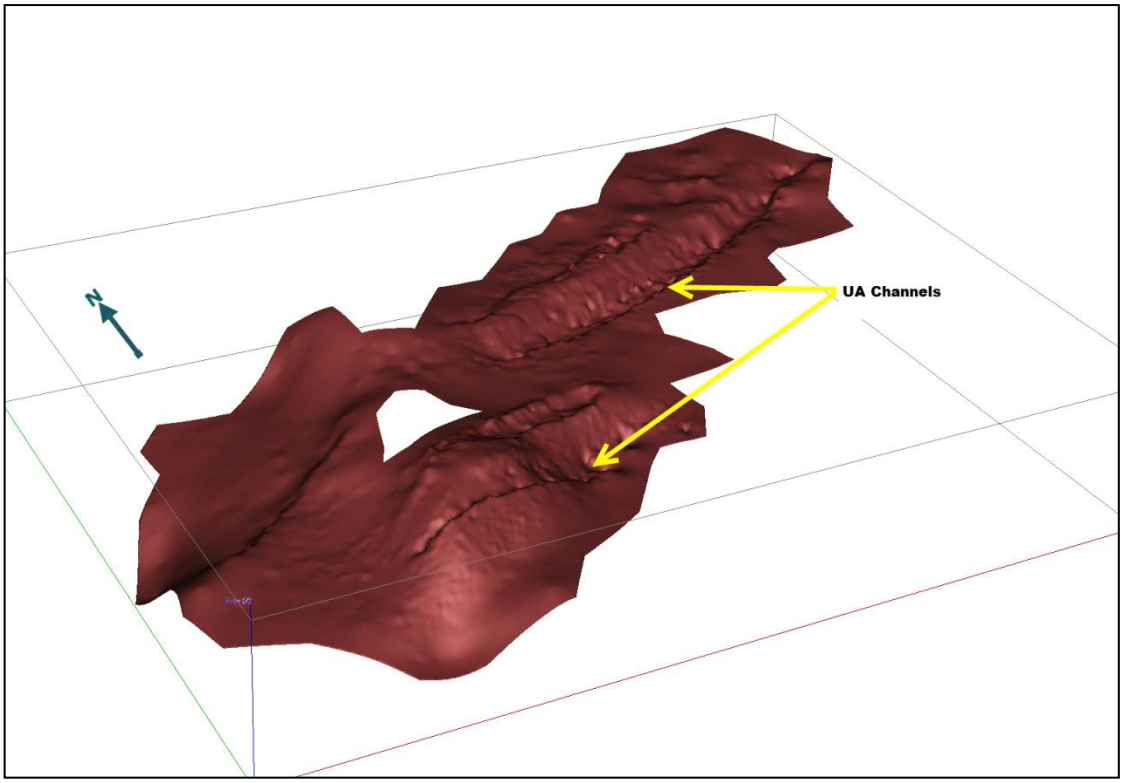


Figure 13-1: 3D model for the MM deposit (5X vertical exaggeration)

13.2.3 3D Estimation (Open Pit: Miriam)

The case of Miriam is different from the other deposits in that the Uranium mineralisation is present at various levels; therefore a 3D approach is more appropriate, and given the relative depth of the mineralisation, an open pit approach to mining is considered appropriate. The method used with GoviEx was to define 20 cm eU composites from the radiometric measurements, and to model both the high-grade mineralisation at a 0.4 kg/t cut-off and the lower-grade envelope at a 0.2 kg/t cut-off. Modelling was carried out using the Leapfrog® geological modelling package. The resulting wireframes were intersected with a 25 m x 25 m x 0.4 m block model, thus defining the mineralised envelopes. Within these envelopes, the resource was estimated by Uniform Conditioning ("UC") with a Selective Mining Unit ("SMU") of 5 m x 5 m x 0.4 m (high selectivity assumption). For each 25 m x 25 m x 0.4 m block, the UC estimate gives, for a range of cut-off values, the recovered ore tonnages and grades corresponding to the selectivity level considered.

13.3 Marilyn and Marianne

13.3.1 Data

Significant infill drilling has been carried out in the central and southern areas of the deposit down to a spacing of approximately 30 m along strike. A map of the drilling used for the Mineral Resource estimate reported here is given in Figure 13-2. The drilling density is relatively consistent, except at the borders where the distance between drillholes varies to up to over 100 m.

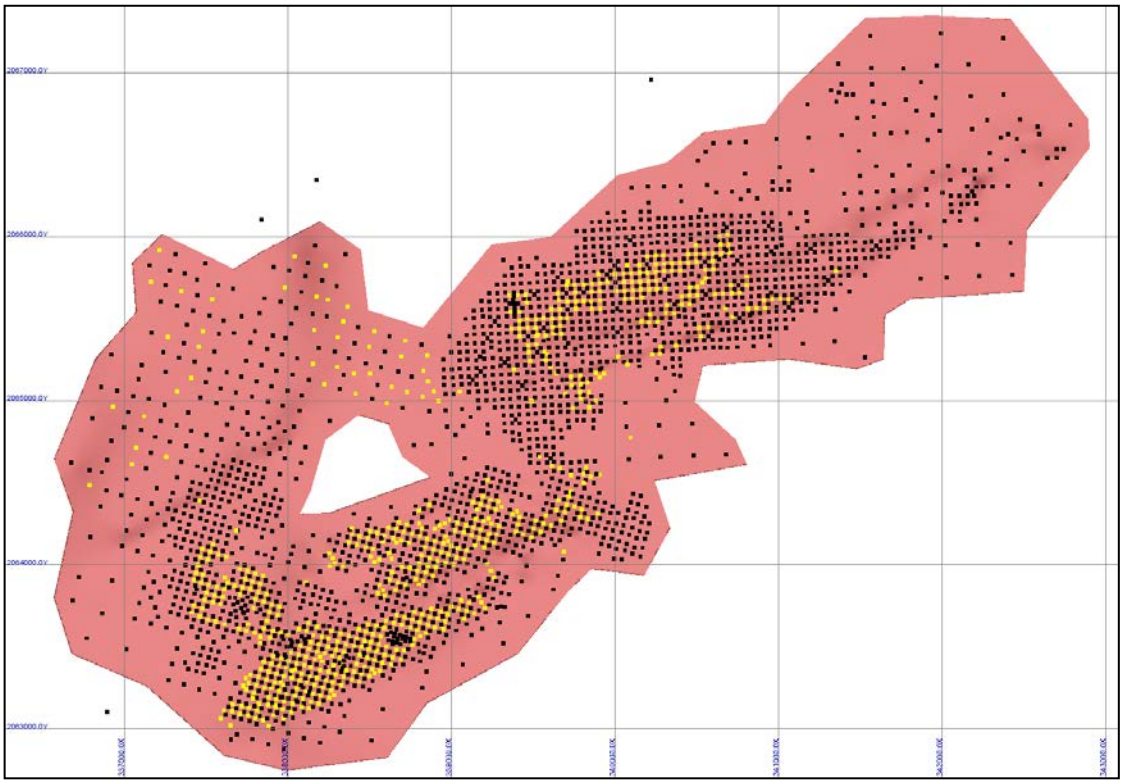


Figure 13-2: M&M drill collar locations and mineralised envelope (March 2015), yellow collars represent additional holes drilled through 2013.

13.3.2 Statistics

Pre-processing of the data consisted of producing a single but variable length composite for each hole, which intersects the mineralisation model. These composites were produced by intersecting the wireframe model produced in Leapfrog® with all drillholes. The composite length is captured in the database and the length weighted average grade over that interval is estimated from the raw data values encountered between the HW and FW intersections. The GTh value is then estimated for each interval.

The resulting database consisting of the three service variables is then transferred to Isatis® as a desurveyed 3D model along with the 3D surfaces representing the geological contact between the Guezouman –Talak and Guezouman-UA. This surface was then used to run an unfolding routine within Isatis®, which produced a 2D grid model onto which the grade service variable data was transferred. This allows the deposit to be evaluated as a single domain for production of semi-variograms and kriging based on the assumption that, mineralogicaly and in terms of deposit genesis, the deposit is a single entity. The following figures (Figure 13-3 and Figure 13-4) show the degree of local dip variability across the deposit due to underlying topographic and sedimentological features, these are effectively “smoothed” by the unfolding routine. Following estimation and block modelling the resulting grid is post processed back to the original 3D location for export to mine planning.

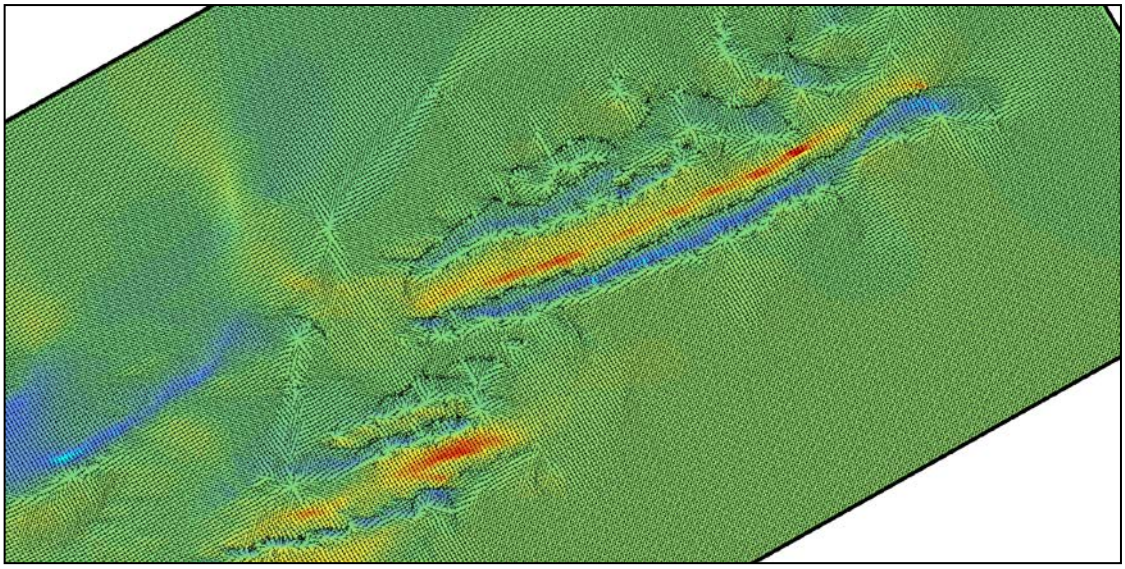


Figure 13-3: Image capture from the unfolded grid model for M&M showing the variation in local dip across the central area of the M&M deposit, colour intensity denotes intensity of local dip variation.

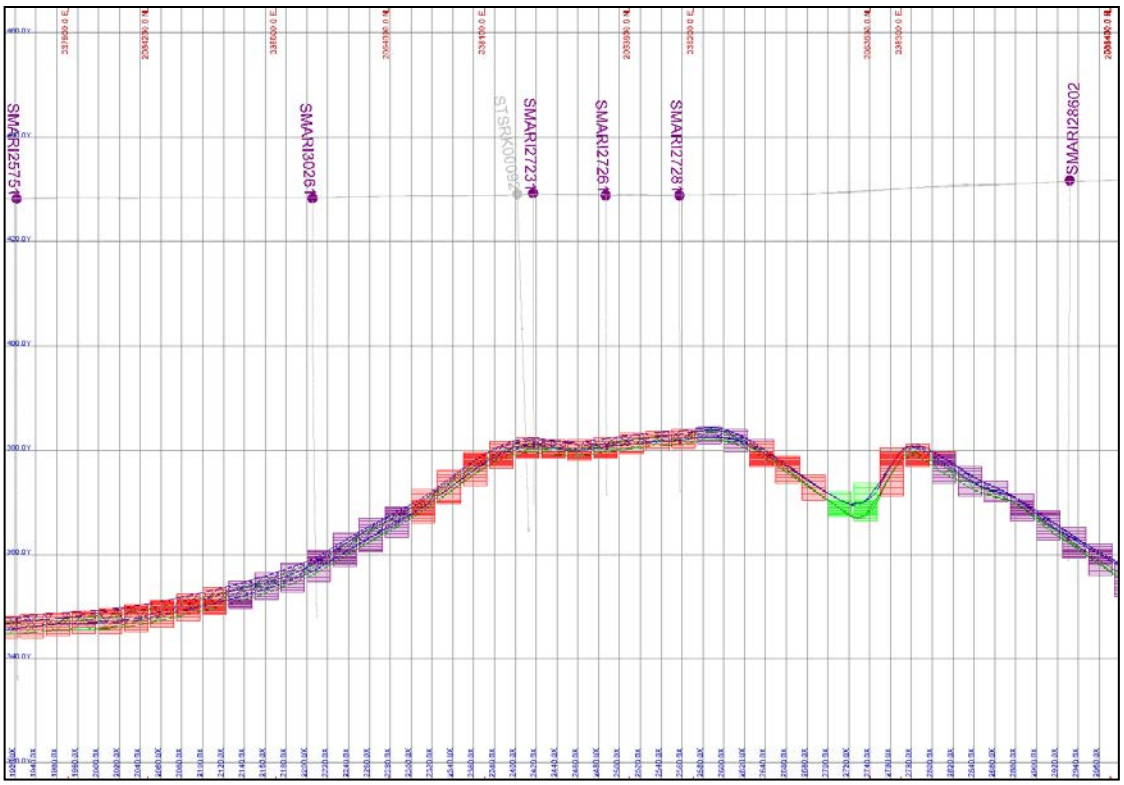


Figure 13-4: Image capture cross section from the original 3D model after re-folding showing variation in dip across the orebody (section extent 1000 m horizontally, 5x vertical exaggeration).

Although the distribution of the data is fairly regular, there is some variability that warrants the use of declustering weights. These were defined in Isatis® after several tests, the final preference being given to a 78 m x 78 m declustering cell.

Raw and declustered statistics for the 3 Service Variables are given in Table 13-1 to Table 13-3.

Table 13-1: Raw statistics for Marianne and Marilyn

variable	count	min	max	mean	stdev
Th (m)	2752	0.14	5.0	1.05	0.48
GTh	2752	0.00	12.99	1.39	1.07
eU (KgT)	2752	0.00	6.12	1.19	0.66

Note that the statistics for eU are not very meaningful because eU is not an additive variable; therefore the real average grade is more realistically given by the ratio of the average GTh to the average Th, which is equal to 1.32 kg/t.

Table 13-2: Declustered statistics for Marianne and Marilyn

variable	count	min	max	Mean	stdev
Th (m)	2752	0.14	5.0	0.94	0.52
GTh	2752	0.00	12.99	1.14	0.95
eU (KgT)	2752	0.00	6.12	1.07	0.60

For the declustered statistics, the ratio of the average GTh to the average Th, is equal to 1.21 kg/t

Table 13-3: Raw correlation coefficients for Marianne and Marilyn

variable	Th	GTh	G (eU)
Th	1.00	0.73	0.32
GTh	0.73	1.00	0.82
G (eU)	0.32	0.82	1.00

The impact of declustering is moderate, with a maximum drop in the mean GTh value of about 12%. As expected the correlation coefficients between GTh and Th, and GTh and G are relatively high, while the correlation between eU and Th is low. The global declustered histograms of Th and GTh are represented in Figure 13-5.

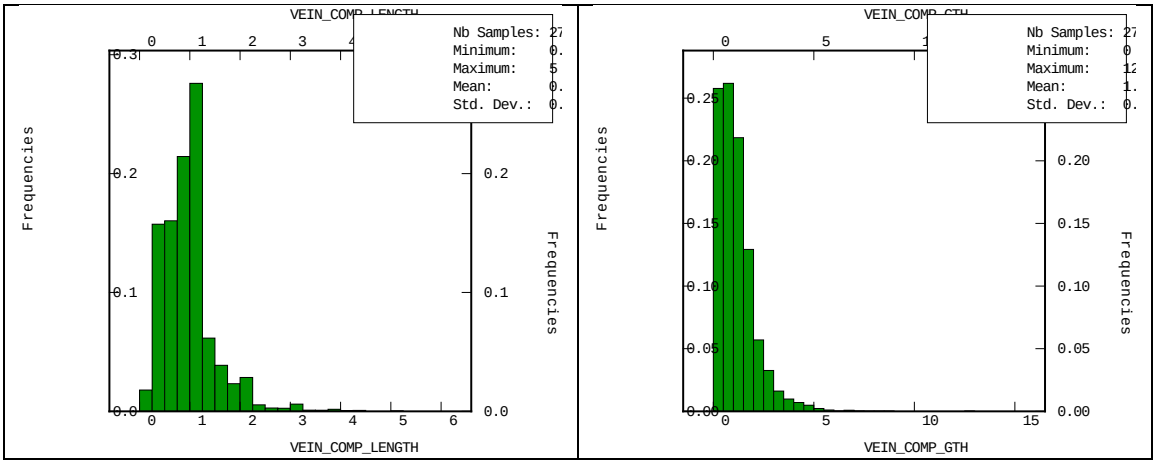


Figure 13-5: Declustered histograms of Th and GTh for Marianne and Marilyn

There are a small number of isolated high grade values, however the close infill hole spacing and chosen neighbourhood search parameters means that the weighting applied to these will be reduced significantly and they are not considered worthy of grade capping.

13.3.3 Estimation by Co-Kriging

The bivariate model makes use of the components of the Th model. The variograms of Th, GTh as well as the cross-variogram GTh – Th are modelled by linear combination of those components. The experimental variograms, the cross-variograms and their models are given in Figure 13-6. The variogram model has a nugget variance of approximately 11% and ranges of 460 m and 150 m in the major (060) and semi-major (150) directions although the majority of the variogram occurs below the second structure with ranges of approximately 180 m and 75 m respectively.

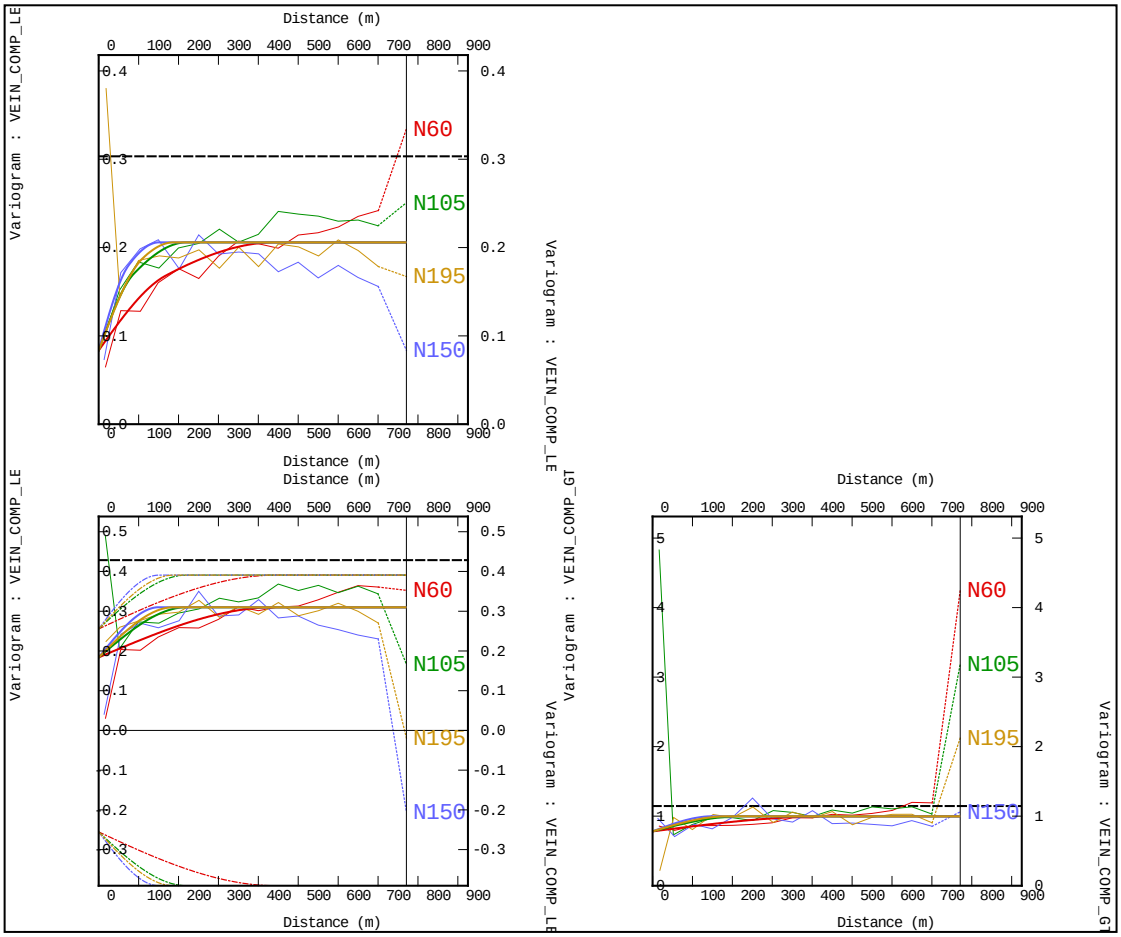


Figure 13-6: GTh and Th bivariate variogram model for Marianne and Marilyn

Th and GTh are estimated by Ordinary Co-Kriging. The kriging parameters are summarised in Table 13-4. Once the co-kriging of Th and GTh is completed, the average block grade is obtained by dividing GTh^* by Th^*. A graphical representation of eU estimates can be found in Figure 13-7.

Table 13-4: Co-Kriging neighbourhood parameters for Marianne and Marilyn

Variable	GTh/Th
Major Ellipsoid Axis Radius (m)	400
Semi-major Ellipsoid Axis Radius (m)	100
Minor Ellipsoid Axis Radius (m)	10
Rotation (counter clockwise)	30°
Minimum composites required for estimate	4
Number of sectors	4
Optimum number of composites per sector	8
Threshold for large values	N/A
Distance	N/A
Discretization (XYZ)	5x5x2

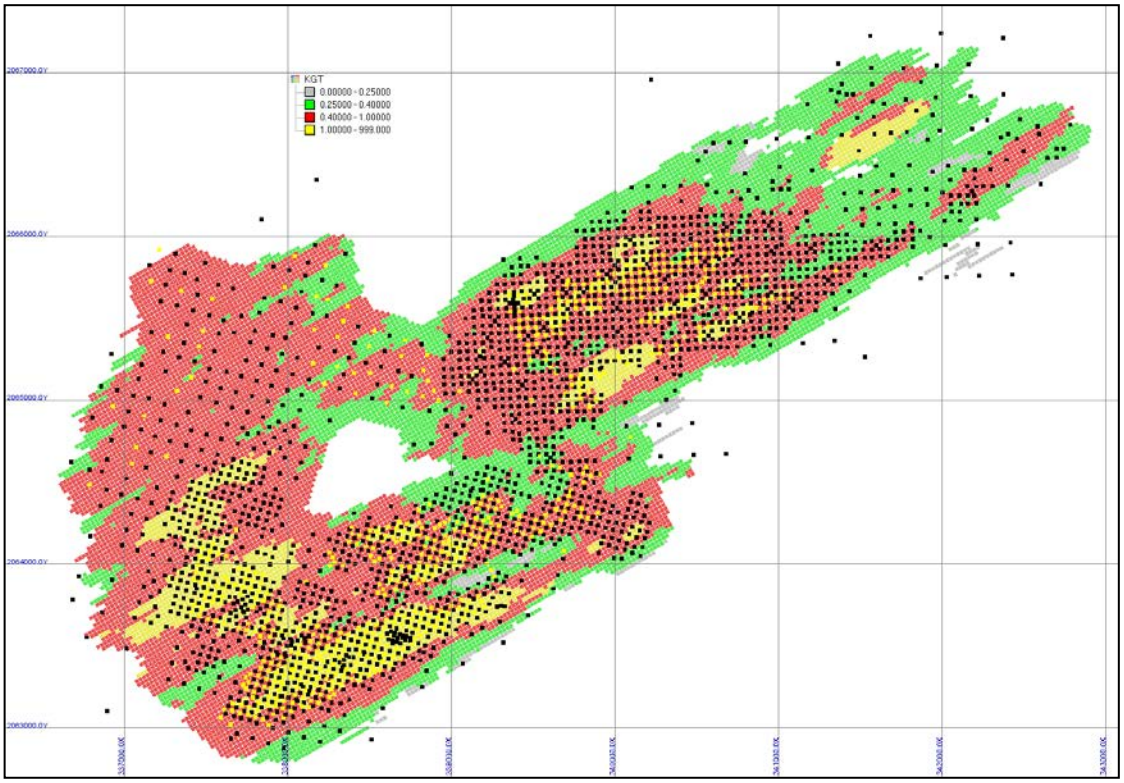


Figure 13-7: eU (kg/t) estimated after co-kriging of Th and GTh for M&M

The quality of the estimation of individual blocks is illustrated by the histograms of the regression slopes (Figure 13-8). Th appears to be better estimated than GTh, as 62 % of the blocks have a regression slope better than 0.9 for Th against 32 % for GTh. The global estimates are very good. The following figures show the slope of regression values for the Co-Kriged blocks.

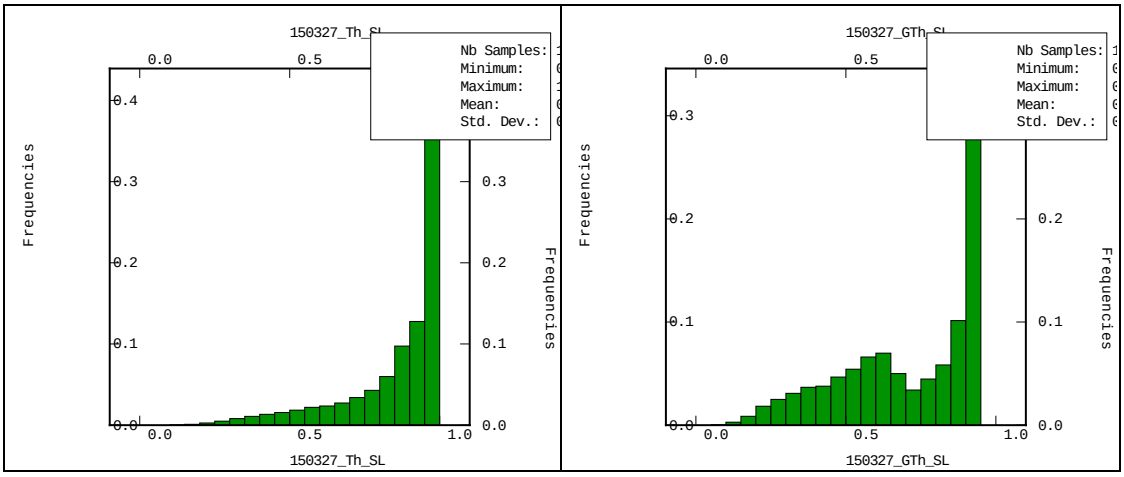


Figure 13-8: Histograms of Th and GTh COK regression slopes for M&M

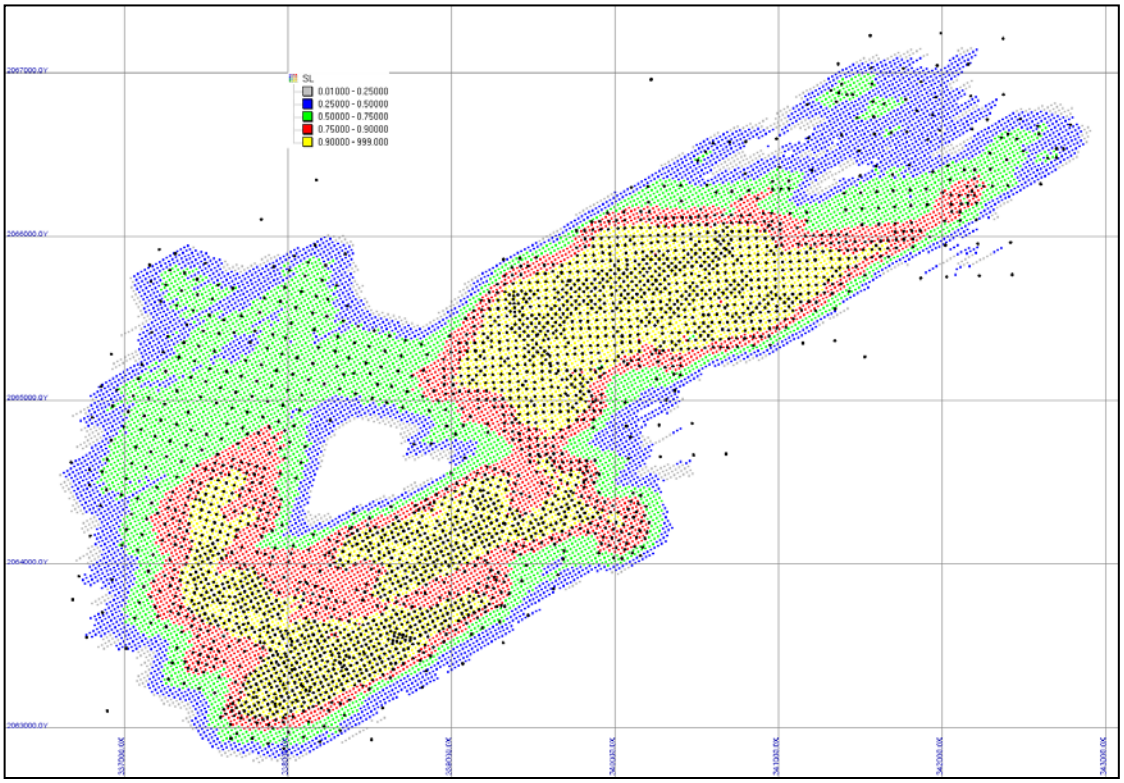


Figure 13-9: Slope Regression results for GTh variable at M&M showing collar locations

Cross validation was carried out for both the GTh and Th variables to attempt to quantify the quality of the kriging. This is a method (also known as jack knifing) whereby individual sample nodes are estimated using the kriging parameters and the result compared directly with the actual known value for the interval. The results are summarised in the following figures.

The cross validation shows that, although the mean values for the estimates and actual data populations in both the Th and GTh were very close, the range of grades of the estimates is much lower. Where the actual values are high (GTh > 5.0) there is generally an underestimation of the estimated values when compared with the actual values. However the number of points where this occurs is very limited. The minimal number of samples affected

and the grade profile indicate that this may be producing a locally conservative grade estimate in isolated high grade areas and is effectively capping the influence of high grades by assigning them low weighting values during the kriging process.

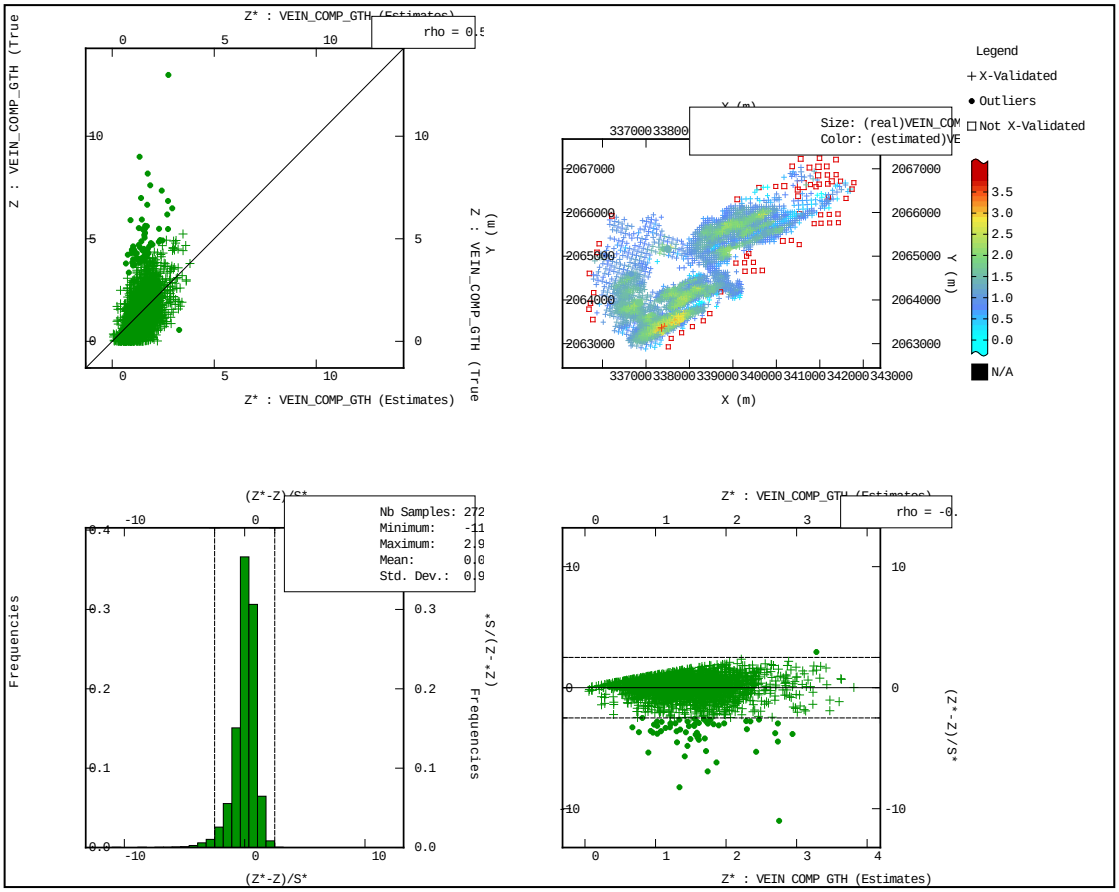


Figure 13-10: Cross validation results from estimation of the GTh variable

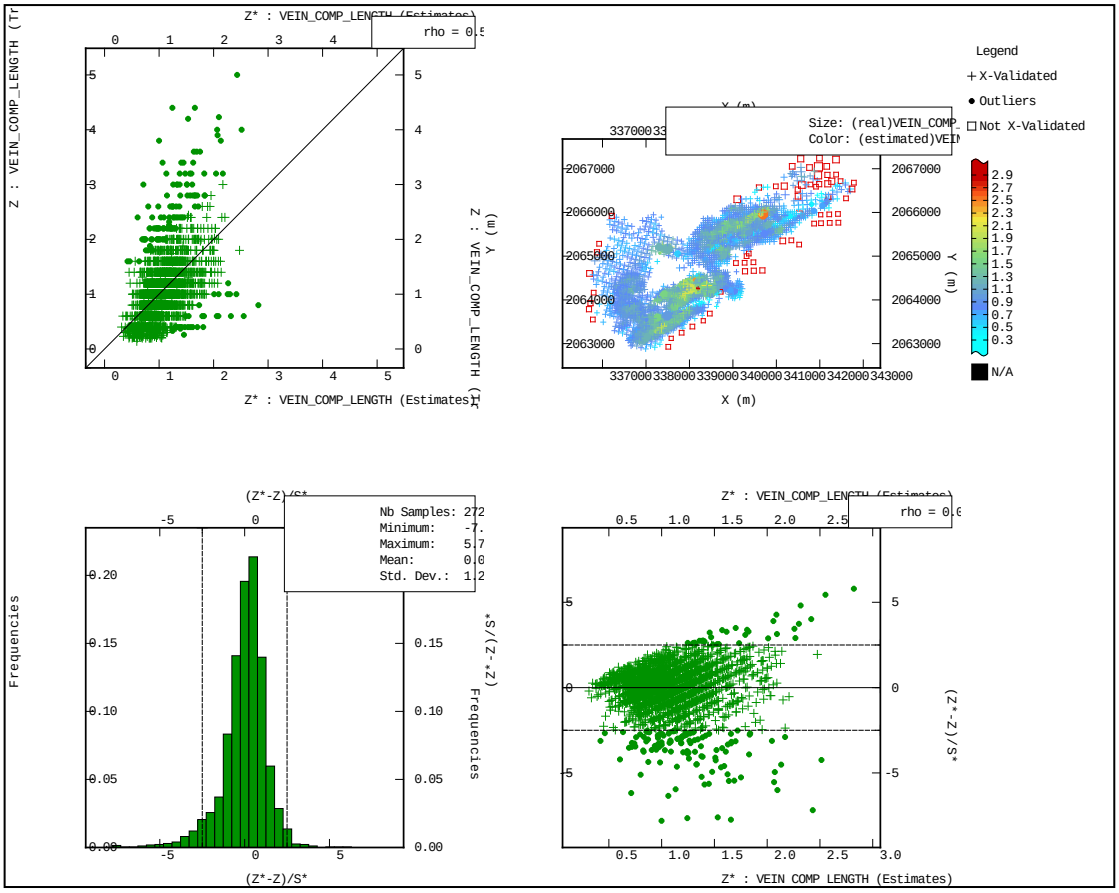


Figure 13-11: Cross validation results from estimation of the Th variable

13.3.4 Model Parameters and Classification

As the drillhole density averages around 50 m x 50 m, and the variograms show reasonable structures at this spacing, it is appropriate to use 25 m x 25 m cells for the block model. The parameters chosen are given in Table 13-5.

Table 13-5: Block model parameters for Marianne and Marilyn

	X	Y
Origin (block centroid)	337029.405	2060916.905
Block Size	25	25
Number of Blocks	360	200

In addition the block model is rotated anticlockwise by 30° around its origin to align it approximately with the strike of the deposits. The resource model limits were set using the Leapfrog® derived wireframe model. The HW and FW contacts are hard boundaries obtained directly from the wireframe. The lateral extents were defined by creating a 75m boundary beyond the limits of the outermost drillhole intersections in order to prevent smoothing of grades into areas of no drilling. This distance is approximately half the range of the main structure as derived from the semi-variogram models. Figure 13-12 shows the limits of the resource model in relation to the drilling and the classification applied to the final blocks.

The density used for the tonnage is 2.3 t/m^3.

The classified blocks are represented in Figure 13-12. As expected, the Inferred material is at the periphery of the orebody, where the drilling density is lower. Classification category is derived from a combination of kriging quality (slope of regression, Figure 13-9) and hole spacing. Inferred Mineral Resources are generally limited to the outer extremity of the deposit where drilling density drops to over 100 m. The majority of the deposit is considered to be Indicated Resources given the slope of regression results are consistently greater than 50 %. A small area of Measured is defined in both the Marilyn and Marianne domains, generally in areas of consistent infill (30 m spacing) drilling. Measured Resource blocks were excluded from any areas considered as part of the UA contact domain.

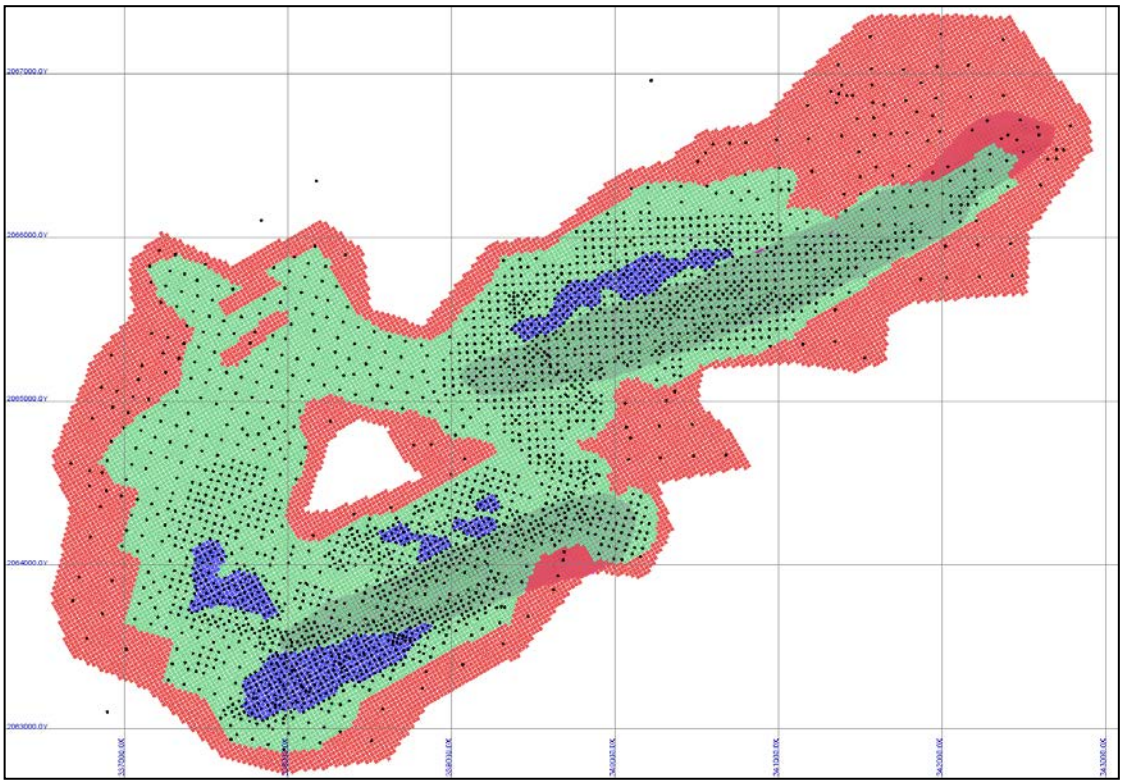


Figure 13-12: Classification (Measured=blue; Indicated=green; Inferred=red) for Marianne and Marilyn and showing area underlain by the UA channels.

Table 13-6: Mineral Resource Statement reported using CIM guidelines, Marilyn/Marianne reported at 0.4 Kg/t eU.

Classification	Tonnes (Mt)	Grade (kg/t eU_3O_8)	eU_3O_8 (t)	eU_3O_8 (Mlb)
Measured	2.14	1.79	3,835	8.45
Indicated	14.72	1.43	21,000	46.30
Inferred	5.04	1.17	5,908	13.02

13.4 Madaouela South North East (“MSNE”)

The estimation process followed for MSNE is similar to the one used for Marianne and Marilyn with the exception that the estimation was carried out in a 2D model and a check estimate was carried out using the residuals method.

13.4.1 Data

The drilling density at MSNE varies, from about 100 m x 100 m to over 200 m x 200 m in the West and South (Figure 13-13).

While, as indicated, the data was prepared by GoviEx, SRK took part in the process and checked the results in detail. In the end, the total number of 2D intersections retained for the estimation is 234.

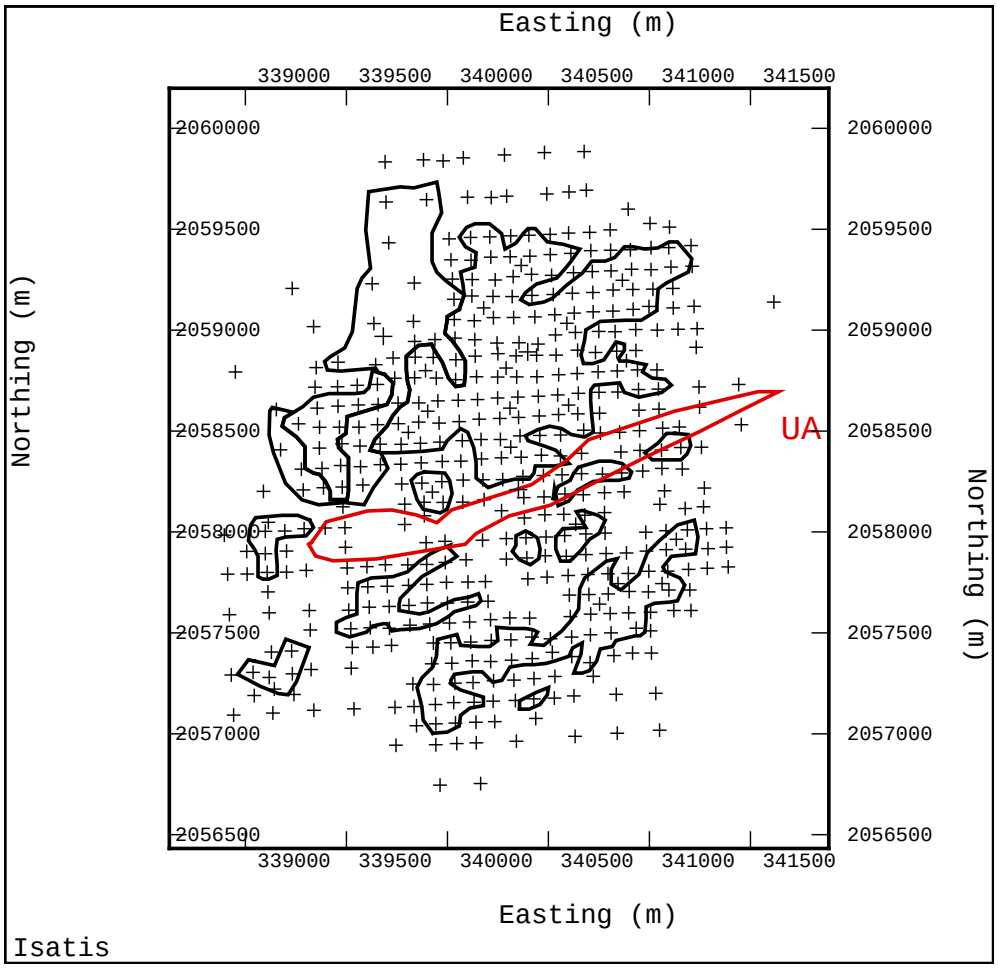


Figure 13-13: MSNE: data location and mineralisation envelopes

13.4.2 Statistics

Although the distribution of the data within MSNE is fairly regular, with the central and eastern part showing a 100 m x 100 m grid and the periphery from North to South drilled at a 200 m x 200 m grid, there is some variability which warrants the use of declustering weights. These were defined in Isatis® after several tests, the final preference being given to a 200 m x 200 m declustering cell.

Statistics for the three Service Variables are given in Table 13-7 to Table 13-10.

Table 13-7: Raw statistics for MSNE

Variable	Count	Minimum	Maximum	Mean	Median	Std.Dev.	Variat.Coef
Th (m)	234	0.4	2.40	0.97	1.00	0.38	0.39
GTh (mkg/t)	234	0.0	3.88	1.35	1.25	0.74	0.55
G (eU KG/T)	234	0.0	4.48	1.38	1.28	0.65	0.48

Note that the statistics for G are not very meaningful because G is not an additive variable; therefore the real average grade is more realistically given by the ratio of the average GTh to the average Th, which is equal to 1.39 kg/t

Table 13-8: Declustered statistics for MSNE

Variable	Count	Minimum	Maximum	Mean	Median	Std.Dev.	Variat.Coef
Th (m)	234	0.4	2.40	0.96	0.80	0.40	0.41
GTh (mkg/t)	234	0.0	3.88	1.28	1.19	0.74	0.58
G (eU KG/T)	234	0.0	4.48	1.29	1.21	0.65	0.50

The ratio of the average GTh to the average Th, is equal to 1.33 kg/t

Table 13-9: Raw correlation coefficients for MSNE

Variable	Th	GTh	G (eU)
Th	1.00	0.58	0.07
GTh	0.58	1.00	0.81
G (eU)	0.07	0.81	1.00

Table 13-10: Declustered correlation coefficients for MSNE

Variable	Th	GTh	G (eU)
Th	1.00	0.61	0.16
GTh	0.61	1.00	0.83
G (eU)	0.16	0.83	1.00

As expected the correlation coefficients between GTh and Th, and GTh and G are high, while the correlation between G and Th is low.

The declustered global histograms of Th and GTh are represented in Figure 13-14.

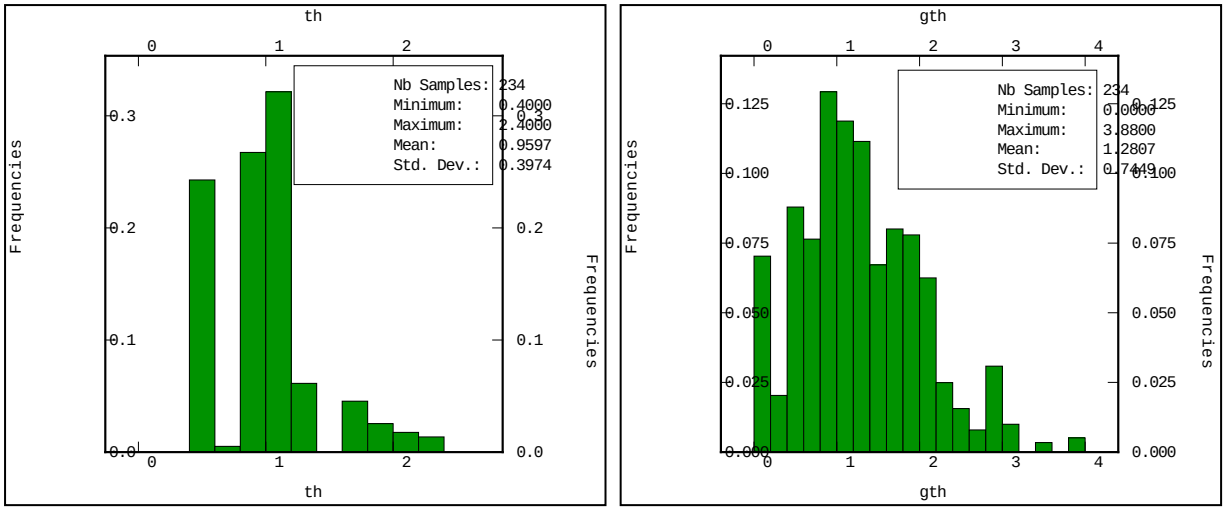


Figure 13-14: Declustered histograms of Th and GTh for MSNE

There are no real outliers in these histograms.

13.4.3 Estimation using Residuals

In order to be able to use the Residuals method, it is required to check the correlation between Th and G. Figure 13-15 shows the scatter diagram and cross-variograms between these two variables: the assumption of lack of spatial correlation is reasonably confirmed with a low correlation coefficient. The cross-variograms show some structure, but overall they are fairly flat.

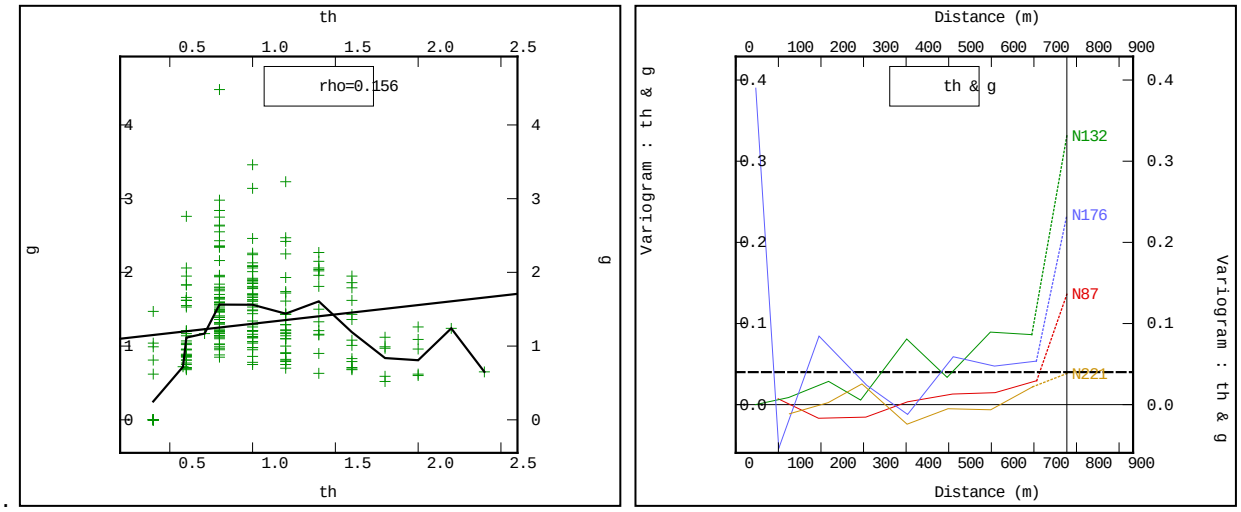


Figure 13-15: Th – G scatter diagram and cross-variograms for MSNE

The resulting regression between GTh and This given as: GTh = 1.357456 Th + R

Th and R are estimated by Ordinary Kriging. The kriging neighbourhoods were optimised in Isatis®. A critical parameter to define is the number of composites to be used for the estimates. The choice was guided by optimising the slope of regression. The kriging parameters used are summarised in Table 13-11.

Table 13-11: Th and R kriging neighbourhood parameters for MSNE

		Th	R
Major Ellipsoid Axis Radius (m)		500	500
Minor Ellipsoid Axis Radius (m)		300	300
Rotation (counter clockwise)		-41.5°	-41.5°
Minimum composites required for estimate		1	1
Number of sectors		4	4
Optimum number of composites per sector		8	8
Threshold for large values		0	0
Distance		0	0
Discretisation	X	10	10
	Y	10	10

There is no need for a special handling of the high grades, as the histograms do not show excessive skewness.

Once the kriging of Th and R is done, the accumulation is estimated by: $GTh^* = 1.357456\ Th^* + R^*$ and the average block grade is obtained by dividing GTh^* by Th^*.

377 blocks are estimated. A representation of eU estimates can be found in Figure 13-16.

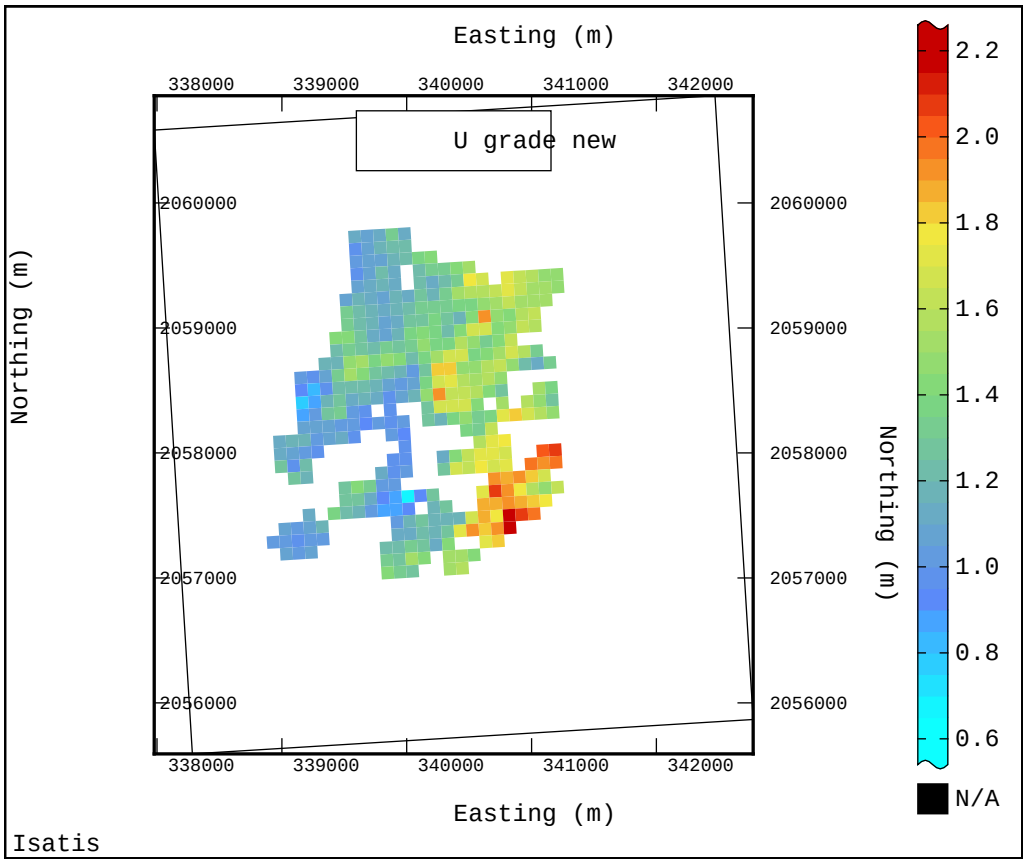


Figure 13-16: eU (kg/t) estimated by residuals method for MSNE

The quality of the estimation of individual blocks is considered good, but is dependent on the drilling density. This is well illustrated by the histograms of the regression slopes (Figure 13-17). Th appears to be better estimated than R, as about 70 % of the blocks have a regression slope better than 0.8 for Th against 28 % for R. Overall the estimation is correct and the global estimates are good.

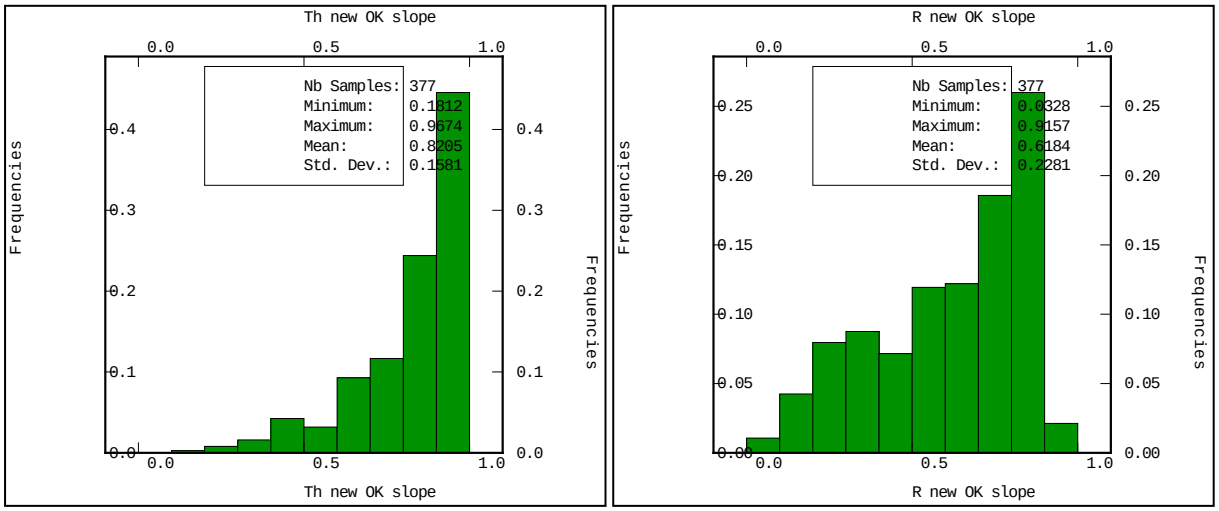


Figure 13-17: Histograms of Th and R OK regression slopes for MSNE

13.4.4 Estimation by Co-Kriging

The bivariate model makes use of the components of the Th model showed in Figure 13-18. The variograms of Th, GTh as well as the cross-variogram GTh – Th are modelled by linear combination of those components. The experimental variograms, the cross-variograms and their models are given in Figure 13-18.

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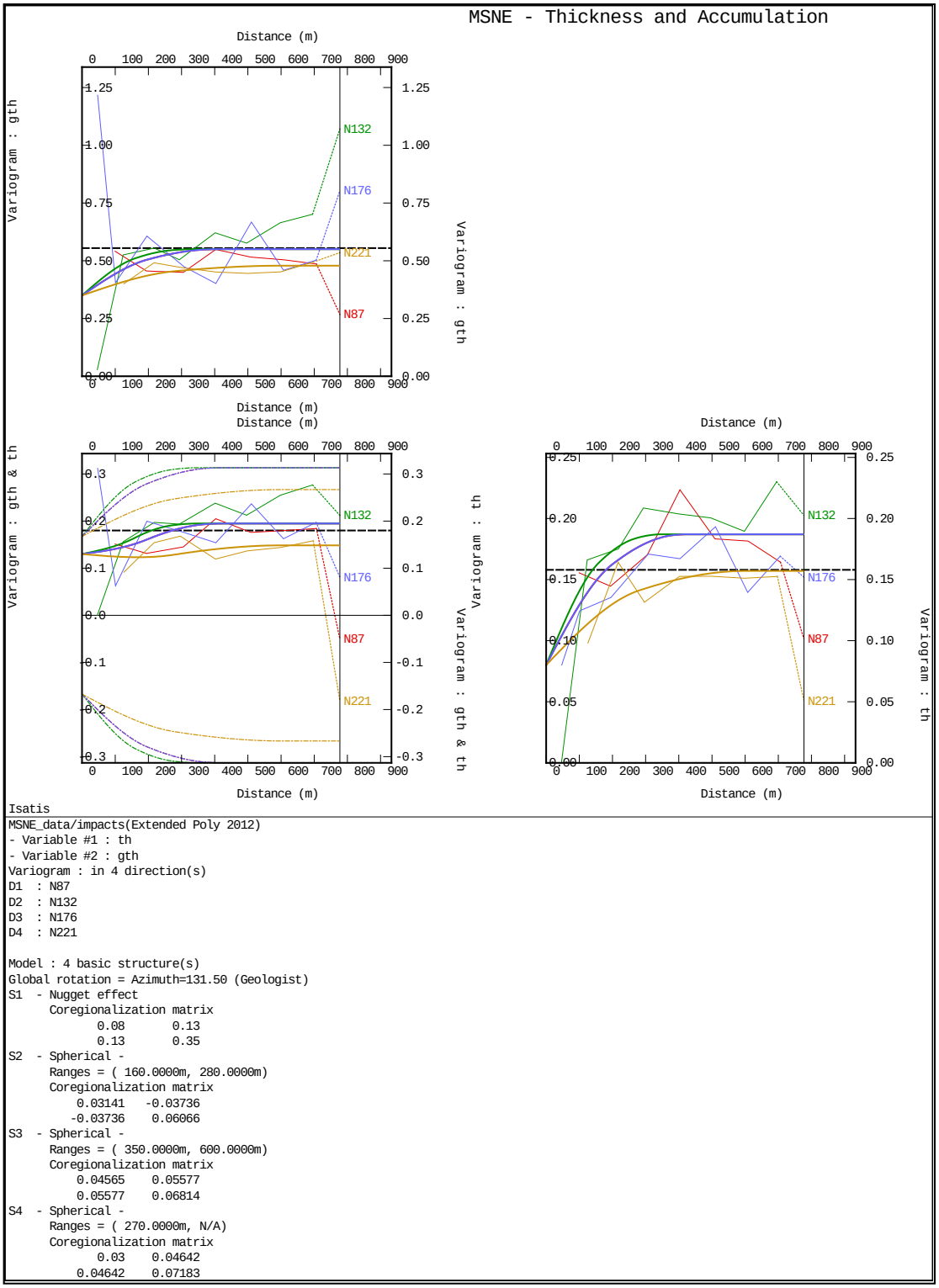


Isatis

```
MSNE_data/impacts(Extended Poly 2012)
- Variable #1 : th
- Variable #2 : gth
Variogram : in 4 direction(s)
D1  : N87
D2  : N132
D3  : N176
D4  : N221

Model : 4 basic structure(s)
Global rotation = Azimuth=131.50 (Geologist)
S1  - Nugget effect
      Coregionalization matrix
             0.08       0.13
             0.13       0.35
S2  - Spherical -
      Ranges = ( 160.0000m, 280.0000m)
      Coregionalization matrix
          0.03141   -0.03736
         -0.03736    0.06066
S3  - Spherical -
      Ranges = ( 350.0000m, 600.0000m)
      Coregionalization matrix
          0.04565    0.05577
          0.05577    0.06814
S4  - Spherical -
      Ranges = ( 270.0000m, N/A)
      Coregionalization matrix
             0.03    0.04642
          0.04642    0.07183
```

Figure 13-18: GTh and Th bivariate model for MSNE

13.4.5 Estimation by Co-kriging:

Th and GTh are estimated by Ordinary Co-Kriging. The kriging neighbourhoods are the same as for the Residuals methods. The kriging parameters are the ones summarised in Table 13-11.

Once the co-kriging of Th and GTh is done, the average block grade is obtained by dividing GTh^* by Th^*. A representation of eU estimates can be found in Figure 13-19.

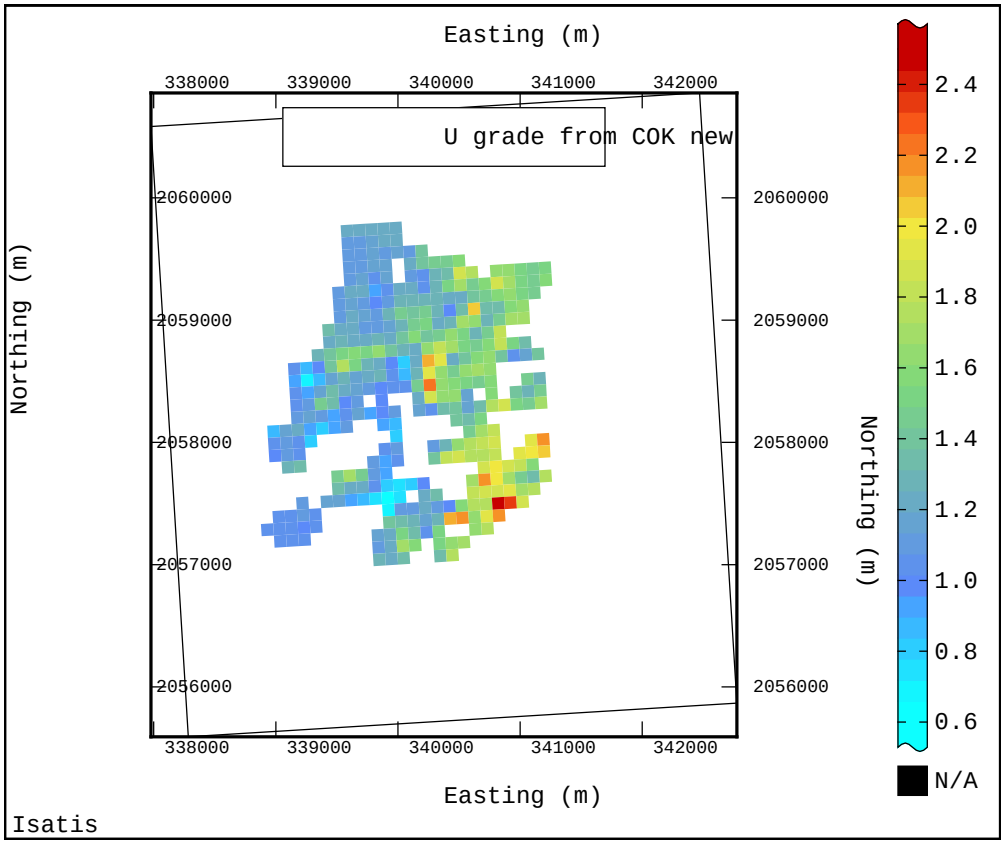


Figure 13-19: eU (kg/t) estimated after co-kriging of Th and GTh for MSNE

13.4.6 Quality of the Estimation

The quality of the estimation of individual blocks is similar to that of the Residuals method. This is again illustrated by the histograms of the regression slopes (Figure 13-20). Th appears to be better estimated than GTh, as over 70 % of the blocks have a regression slope better than 0.8 for Th against 48 % for GTh. Overall the estimation is again correct, the global estimates are good. Like for the Residuals method, there is a strong smoothing effect.

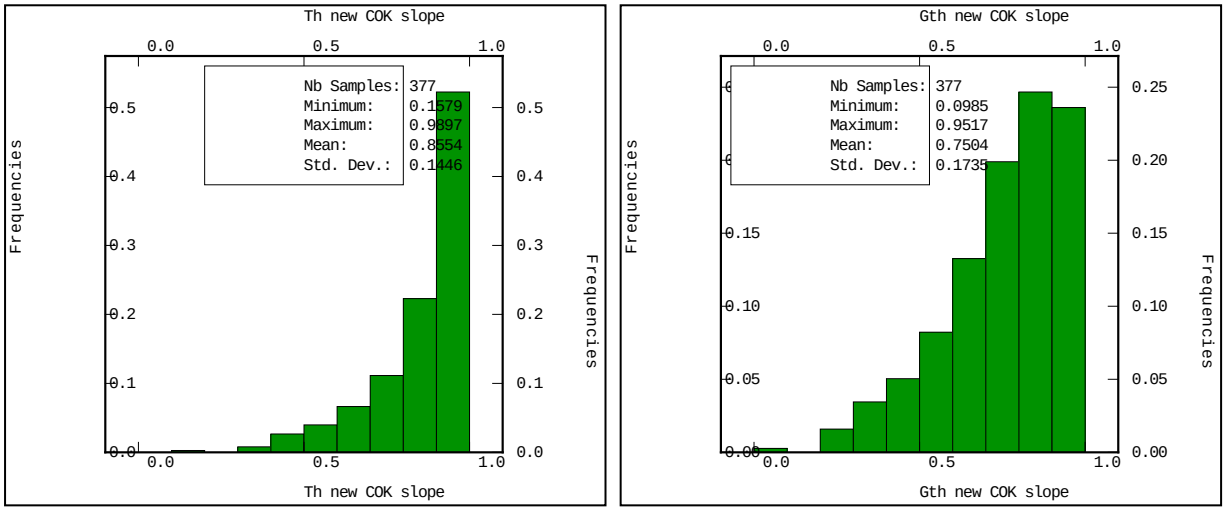


Figure 13-20: Histograms of Th and GTh COK regression slopes for MSNE

13.4.7 Model Parameters and Classification

As the drillhole density in MSNE is 100 m x 100 m in the central part, it makes sense to use the same dimensions for the cells of the block model. The parameters chosen are given in Table 13-12.

Table 13-12: Block model parameters for MSNE

	X	Y
Origin (block centroid)	338330	2055645
Block Size	100	100
Number of Blocks	45	50

In addition the block model is rotated anticlockwise by 3.5 ° around its origin to align it approximately with the drilling grid (and the strike of the deposits). This block model is intersected by the polygons of the mineralised envelope, which gives for each block its proportion within the mineralised envelope. This proportion will be used to define the mineralised tonnage.

The density used for the tonnage is 2.3 t/m^3.

In view of the uncertainties related to the grade-radiometric relationship (in large part due to the lack of DDH which can be used to calibrate it) and also because the short distance continuity of the Service Variables is not well known, no Measured material can be defined. Nevertheless, from a statistical viewpoint, the estimation is of a reasonable quality, so part of the Resources can be classified as Indicated.

The process used for the classification is exactly the same as the one used for Marianne and Marilyn, based on Th estimated by co-kriging.

The classified blocks for MSNE are represented in Figure 13-21. As expected, the Inferred material is at the periphery of the orebody, where the drilling density is generally lowest, 200 m x 200 m.

The Classified Mineral Resources, based on Th COK, are given in Table 13-13.

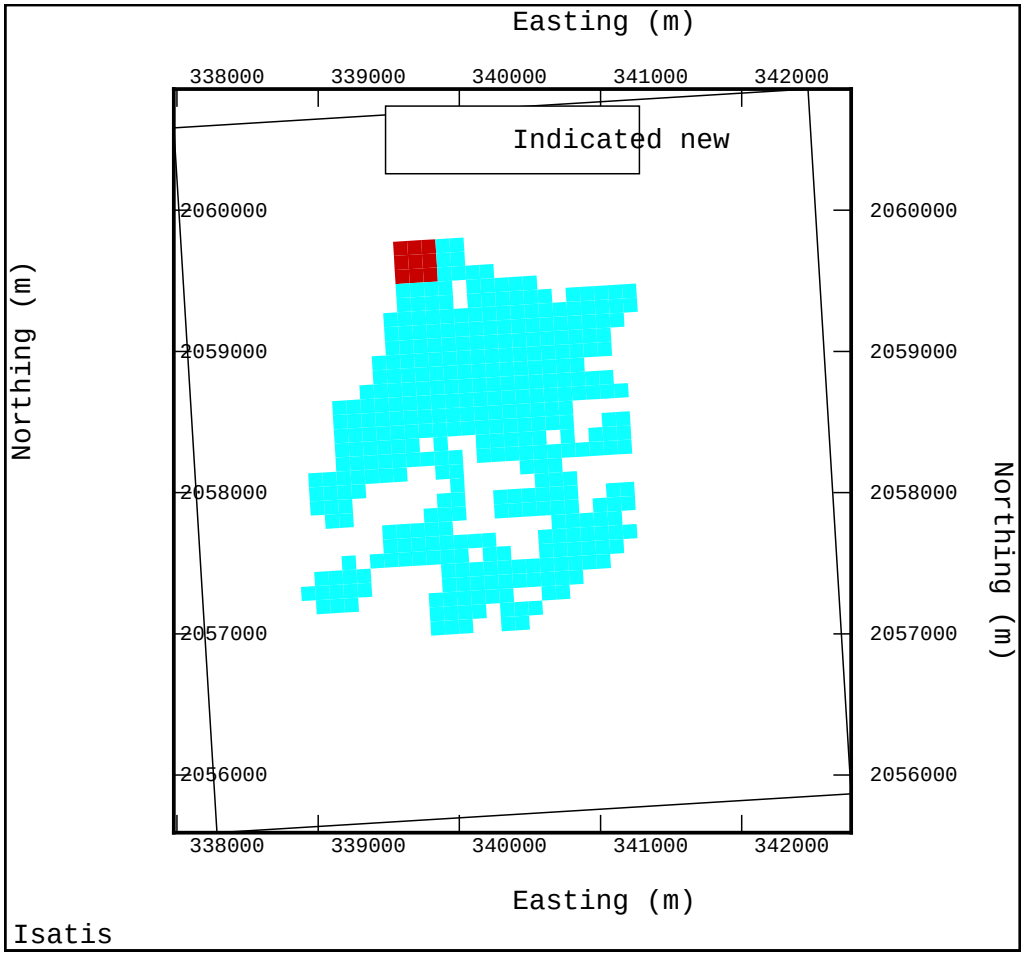


Figure 13-21: Classification (Indicated=blue, Inferred=red) for MSNE

Table 13-13: Mineral Resource Statement reported using CIM guidelines, MSNE reported at 0.4 Kg/t eU.

Classification	Tonnes (Mt)	Grade (kg/t eU_3O_8)	eU_3O_8 (t)	eU_3O_8 (Mlb)
Measured	-	-	-	-
Indicated	5.05	1.61	8,111	17.88
Inferred	0.10	1.34	131	0.29

13.5 Madaouela South Central East ("MSCE")

The estimation process followed for MSCE is identical to the one used for MSNE.

13.5.1 Data

The drilling density at MSCE is wide spaced but fairly regular at 200 m x 200 m (Figure 13-22).

While, the data was prepared by GoviEx, SRK took part in the process and checked the results in details. In the end, as the mineralised envelope is very small, the total number of 2D intersections retained for the estimation is 13.

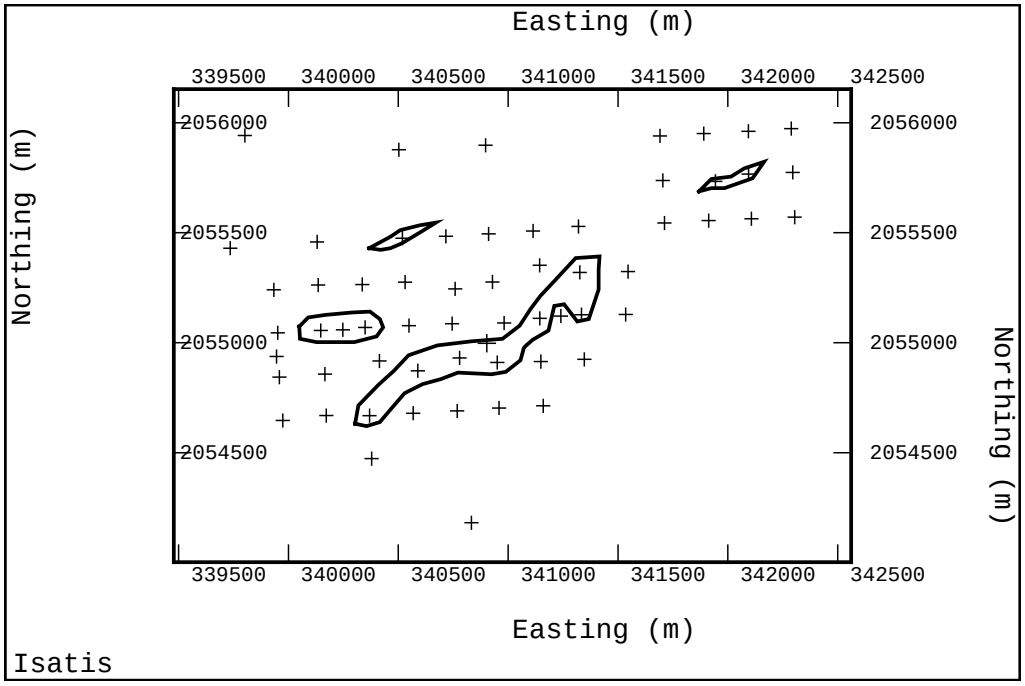


Figure 13-22: MSCE: Data location and mineralisation envelopes (August 2012)

13.5.2 Statistics

Declustering is not required as the data density is fairly regular.

Statistics for the 3 Service Variables are given in Table 13-14 and Table 13-15.

Table 13-14: Raw statistics for MSCE

Variable	Count	Minimum	Maximum	Mean	Median	Std.Dev.	Variat.Coef
Th (m)	13	0.4	2.52	1.09	0.80	0.74	0.67
GTh (mkg/t)	13	0.0	4.46	1.60	1.30	1.21	0.76
G (eU KG/T)	13	0.0	3.01	1.44	1.58	0.70	0.48

The real average grade is more realistically given by the ratio of the average GTh to the average Th, which is equal to 1.46 kg/t

Table 13-15: Raw correlation coefficients for MSCE

Variable	Th	GTh	G (eU)
Th	1.00	0.83	0.04
GTh	0.83	1.00	0.47
G (eU)	0.04	0.47	1.00

As expected the correlation coefficients between GTh and Th, and GTh and G (to a smaller extent) are high, while the correlation between G and Th is low.

The global histograms of Th and GTh are represented in Figure 13-23.

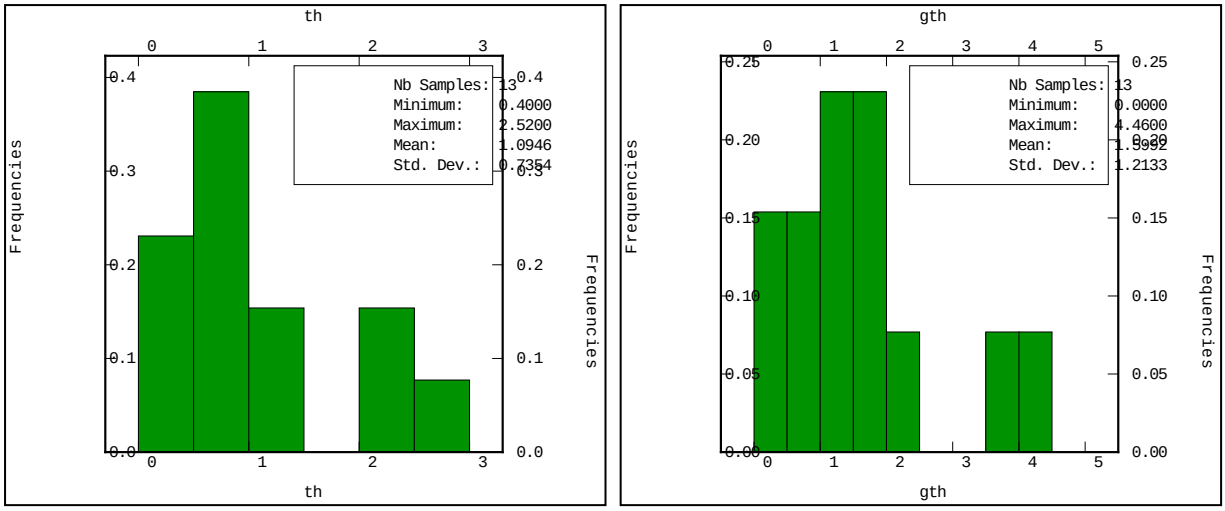


Figure 13-23: Histograms of Th and GTh for MSCE

There are no real outliers in these histograms, although some high values do exist.

13.5.3 Estimation using Residuals

In order to be able to use the Residuals method, we need to check the correlation between Th and G. Figure 13-24 shows the scatter diagram and cross-variograms between these two variables: the assumption of lack of spatial correlation is reasonably confirmed with a low correlation coefficient. The cross-variograms show some structure, but overall they are fairly flat.

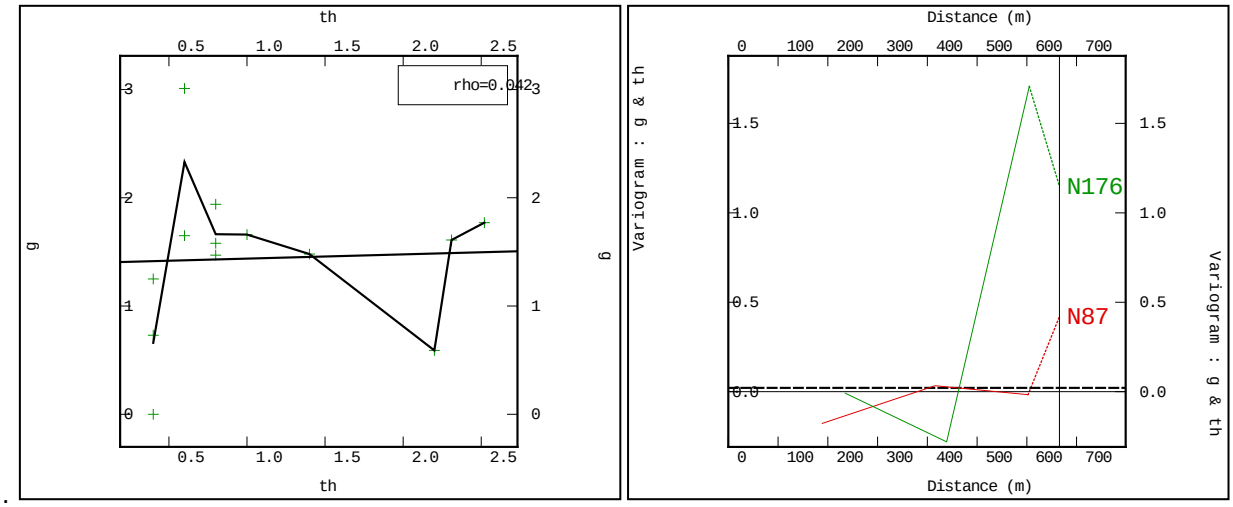


Figure 13-24: Th – G scatter diagram and cross-variograms for MSCE

The resulting regression between GTh and Th is given as: GTh = 1.434247 Th + R

Th and R are estimated by Ordinary Kriging. The kriging neighbourhoods were optimised in Isatis. A critical parameter to define is the number of composites to be used for the estimates. The choice was guided by optimising the slope of regression. The kriging parameters used are summarised in Table 13-16.

Table 13-16: Th and R kriging neighbourhood parameters for MSCE

		Th	R
Major Ellipsoid Axis Radius (m)		600	600
Minor Ellipsoid Axis Radius (m)		400	400
Rotation (counter clockwise)		3.5°	3.5°
Minimum composites required for estimate		1	1
Number of sectors		4	4
Optimum number of composites per sector		8	8
Threshold for large values		0	0
Distance		0	0
Discretisation	X	20	20
	Y	20	20

There is no need for a special handling of the high grades, as the histograms do not show excessive skewness. Once the kriging of Th and R is done, the accumulation is estimated by: $GTh^* = 1.434247\ Th^* + R^*$ and the average block grade is obtained by dividing GTh^* by Th^*. 20 blocks are estimated. A representation of eU estimates can be found in Figure 13-25.

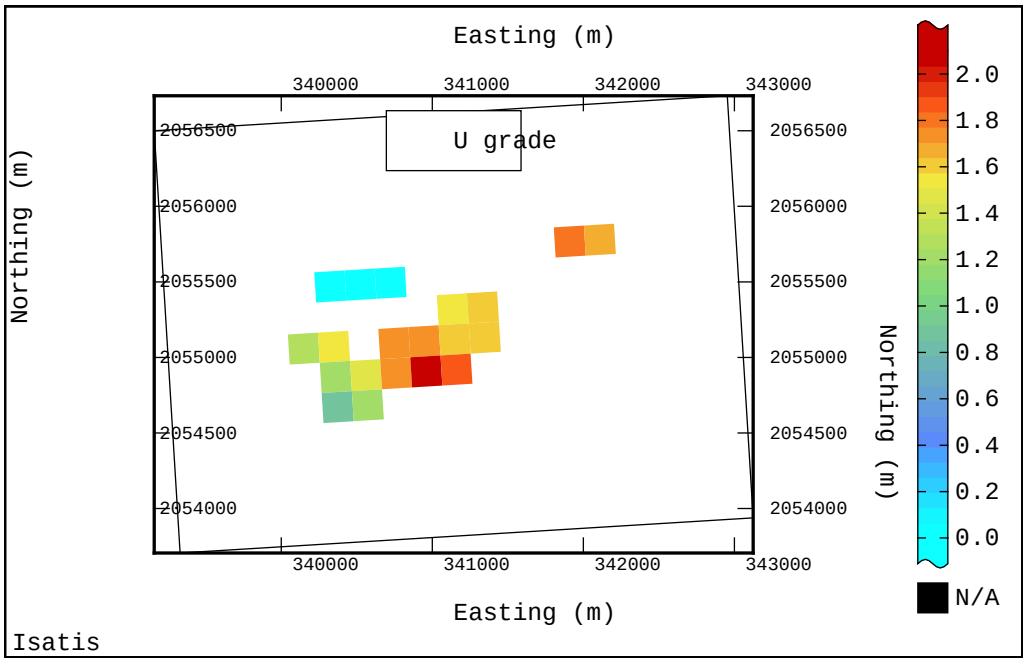


Figure 13-25: eU (kg/t) estimated by residuals method for MSCE

The quality of the estimation of individual blocks is average, depending on the drilling density, and lower than for Marianne and Marilyn. This is well illustrated by the histograms of the regression slopes (Figure 13-26). Th appears to be better estimated than R, as about 50 % of the blocks have a regression slope better than 0.7 for Th against 40 % for R. Overall the estimation is correct. The global estimates are reasonable.

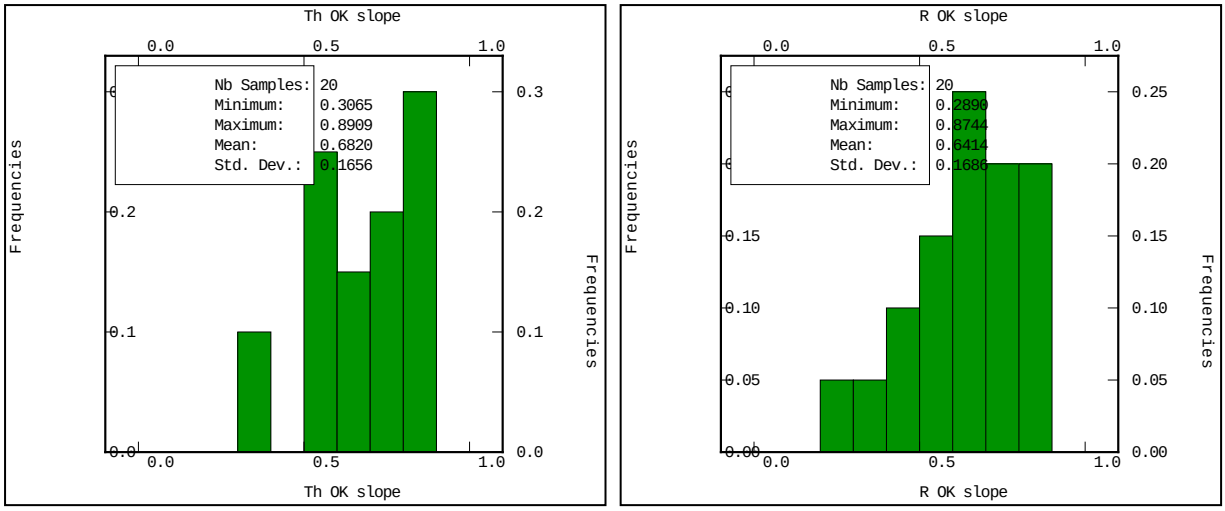


Figure 13-26: Histograms of Th and R OK regression slopes for MSCE

13.5.4 Estimation by Co-Kriging

The bivariate model makes use of the components of the Th model. The variograms of Th, GTh as well as the cross-variogram Gth – Th are modelled by linear combination of those components. The experimental variograms, the cross-variograms and their models are given in Figure 13-27.

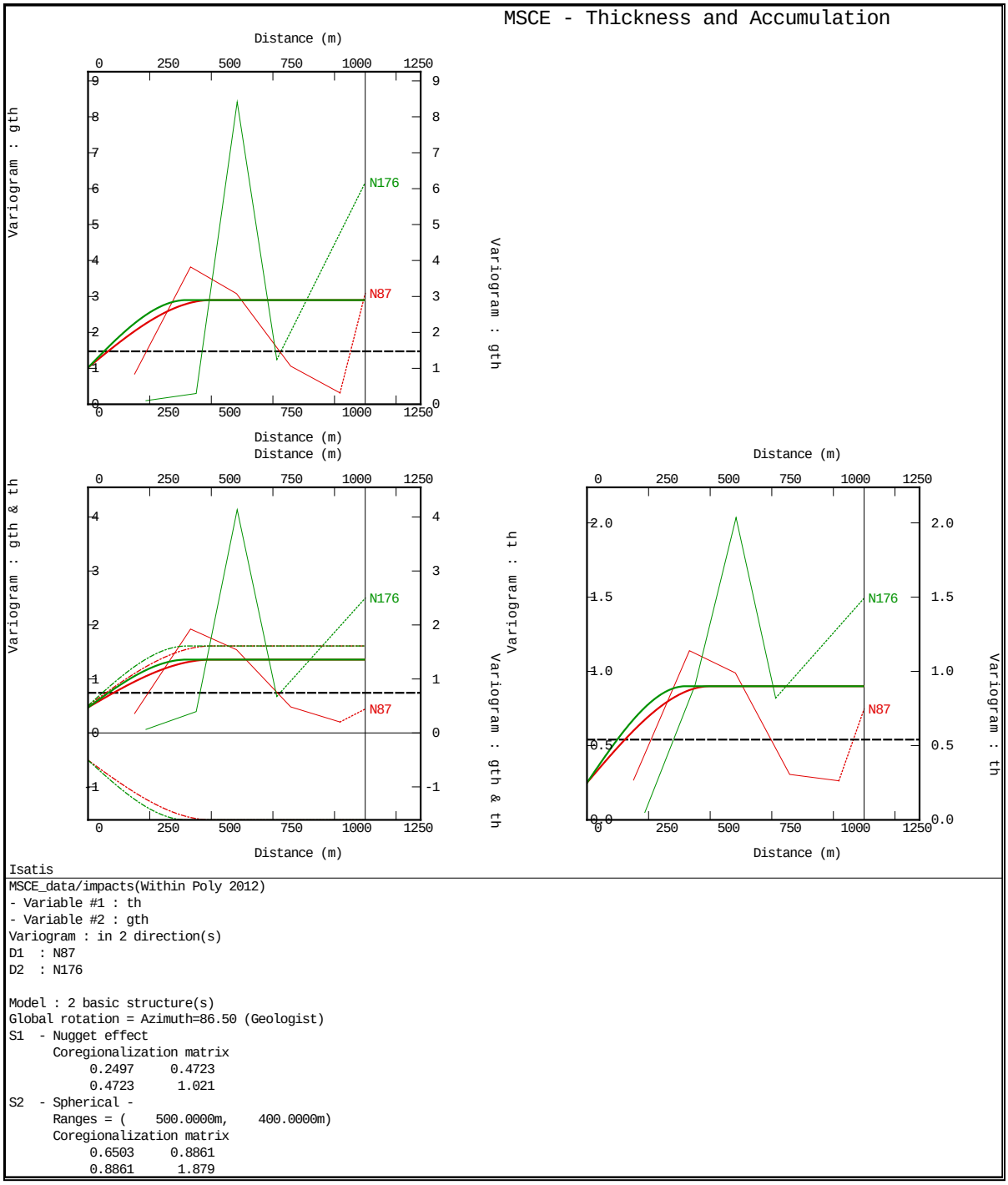


Figure 13-27: GTh and Th bivariate model for MSCE

Th and GTh are estimated by Ordinary Co-Kriging. The kriging neighbourhoods are the same as for the Residuals methods. The kriging parameters are the ones summarised in Table 13-16. Once the co-kriging of Th and GTh is done, the average block grade is obtained by dividing GTh^* by Th^*. 20 blocks are estimated. A representation of eU estimates can be found in Figure 13-28.

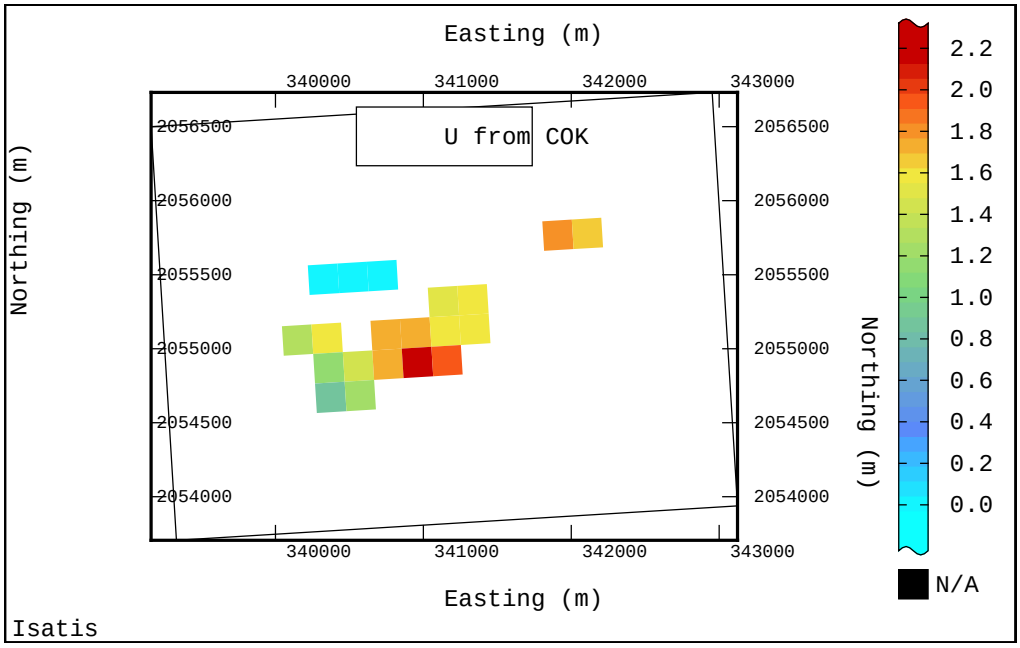


Figure 13-28: eU (kg/t) estimated after co-kriging of Th and GTh for MSCE

The quality of the estimation of individual blocks is similar to that of the Residuals method and acceptable. This is again illustrated by the histograms of the regression slopes (Figure 13-29). Th appears to be better estimated than Gth, as over 45 % of the blocks have a regression slope better than 0.7 for Th against 35 % for GTh. Overall the estimation is again acceptable, the global estimates are reasonable. Like for the Residuals method, there is a strong smoothing effect.

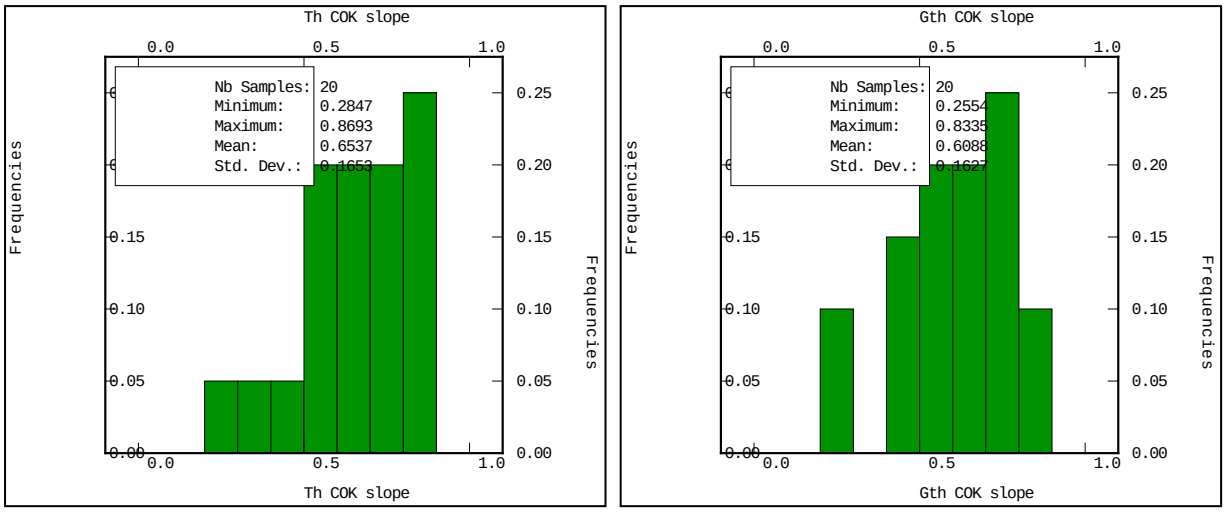


Figure 13-29: Histograms of Th and GTh COK regression slopes for MSCE

13.5.5 Model Parameters and Classification

As the drillhole density in MSCE is 200 m x 200 m in the central part, it makes sense to use the same dimensions for the cells of the block model. The parameters chosen are given in Table 13-17.

Table 13-17: Block model parameters for MSCE

	X	Y
Origin (block centroid)	339424.31	2053811.45
Block Size	200	200
Number of Blocks	19	14

In addition the block model is rotated anticlockwise by 3.5 ° around its origin to align it approximately with the drilling grid. This block model is intersected by the polygons of the mineralised envelope, which gives for each block its proportion within the mineralised envelope. This proportion will be used to define the mineralised tonnage.

The density used for the tonnage is 2.3 t/m^3.

Because of the lack of data and the wide spacing, the estimation is of relatively poor quality and the resources can only be classified as Inferred.

Table 13-18: Mineral Resource Statement reported using CIM guidelines, MSCE reported at 0.4 Kg/t eU.

Classification	Tonnes (Mt)	Grade (kg/t eU_3O_8)	eU_3O_8 (t)	eU_3O_8 (Mlb)
Measured	-	-	-	-
Indicated	-	-	-	-
Inferred	0.72	1.81	1,308	2.88

13.6 Madaouela South Extreme East (“MSEE”)

The estimation process followed for MSEE is identical to the one used for MSNE.

13.6.1 Data

The drilling density at MSEE is wide spaced but fairly regular at 200 m x 200 m with some in-filling in the Eastern part (Figure 13-30). The total number of 2D intersections retained for the estimation is 51.

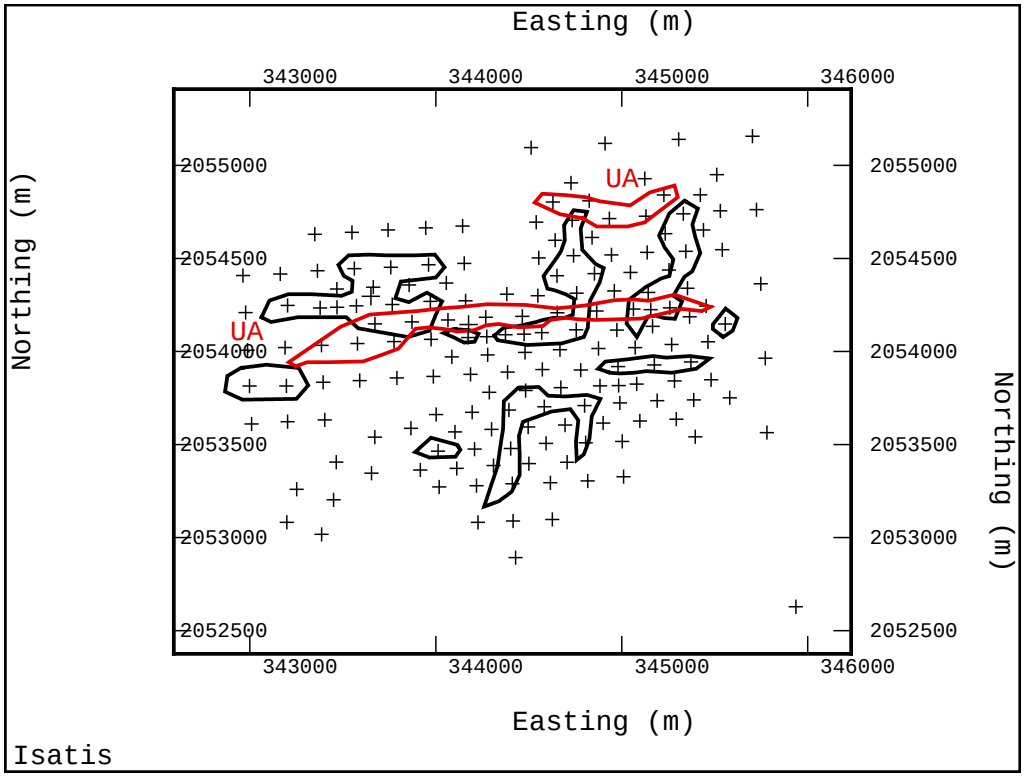


Figure 13-30: MSEE: data location and mineralisation envelopes

13.6.2 Statistics

Although the distribution of the data within MSEE is fairly regular, at a 200 m x 200 m grid with some in-fill at 100 m spacing, there is some variability that warrants the use of declustering weights. These were defined in Isatis® after several tests, the final preference being given to a 200 m x 200 m declustering cell.

Statistics for the 3 Service Variables are given in Table 13-19 to Table 13-22.

Table 13-19: Raw statistics for MSEE

Variable	Count	Minimum	Maximum	Mean	Median	Std.Dev.	Variat.Coef
Th (m)	43	0.4	2.00	0.96	0.80	0.35	0.37
GTh (mkg/t)	43	0.0	3.60	1.38	1.18	0.80	0.58
G (eU KG/T)	43	0.0	3.00	1.38	1.25	0.59	0.42

The real average grade is more realistically given by the ratio of the average GTh to the average Th, which is equal to 1.44 kg/t.

Table 13-20: Declustered statistics for MSEE

Variable	Count	Minimum	Maximum	Mean	Median	Std.Dev.	Variat.Coef
Th (m)	43	0.4	2.00	0.93	0.80	0.31	0.33
GTh (mkg/t)	43	0.0	3.60	1.37	1.15	0.80	0.59
G (eU KG/T)	43	0.0	3	1.41	1.26	0.59	0.42

The ratio of the average GTh to the average Th is equal to 1.47 kg/t.

Table 13-21: Raw correlation coefficients for MSEE

Variable	Th	GTh	G (eU)
Th	1.00	0.71	0.27
GTh	0.71	1.00	0.84
G (eU)	0.27	0.84	1.00

Table 13-22: Declustered correlation coefficients for MSEE

Variable	Th	GTh	G (eU)
Th	1.00	0.69	0.31
GTh	0.69	1.00	0.87
G (eU)	0.31	0.87	1.00

The global declustered histograms of Th and GTh are represented in Figure 13-31.

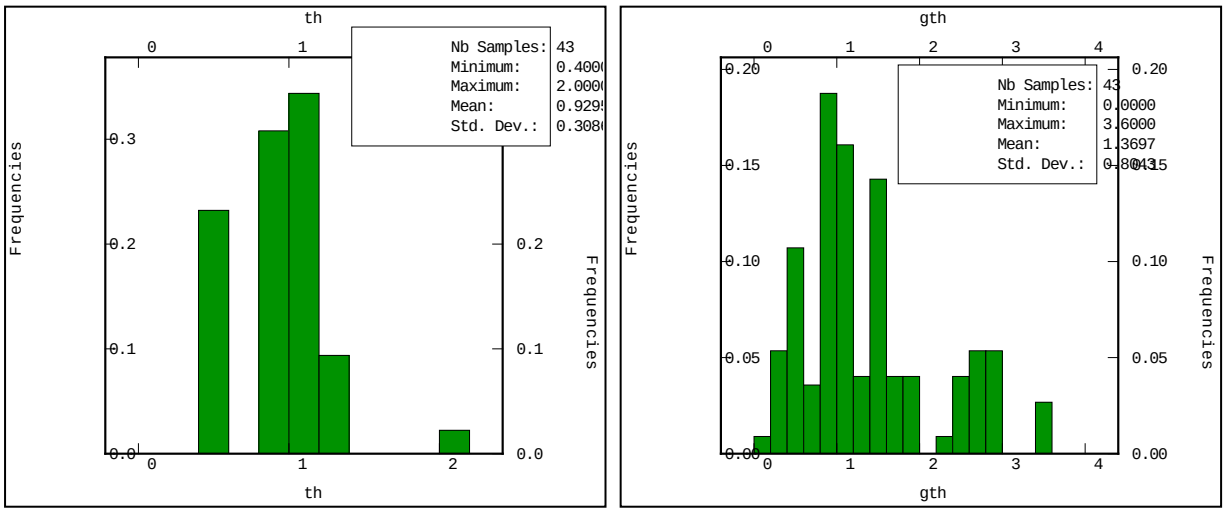


Figure 13-31: Histograms of Th and GTh for MSEE

There are no real outliers in these histograms, although some high values do exist.

13.6.3 Estimation using Residuals

In order to be able to use the Residuals method, the correlation between Th and G was calculated. Figure 13-32 shows the scatter diagram and cross-variograms between these two variables: the assumption of lack of spatial correlation is reasonably confirmed with a low correlation coefficient.. The cross-variograms show very little structure.

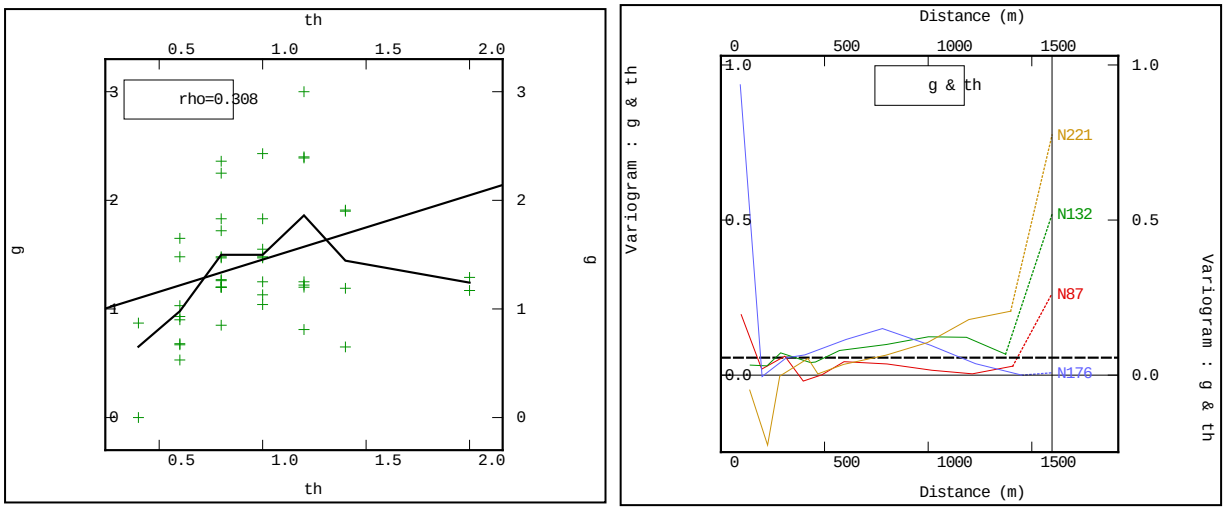


Figure 13-32: Th – G scatter diagram and cross-variograms for MSEE

The resulting regression between GTh and Th is given as: GTh = 1.458616 Th + R

The kriging parameters used are summarised in Table 13-23.

Table 13-23: Th and R kriging neighbourhood parameters for MSEE

		Th	R
Major Ellipsoid Axis Radius (m)		500	500
Minor Ellipsoid Axis Radius (m)		800	800
Rotation (counter clockwise)		3.5°	3.5°
Minimum composites required for estimate		1	1
Number of sectors		4	4
Optimum number of composites per sector		8	8
Threshold for large values		0	0
Distance		0	0
Discretisation	X	10	10
	Y	10	10

There is no need for a special handling of the high grades, as the histograms do not show excessive skewness. Once the kriging of Th and R is done, the accumulation is estimated by: $GTh^* = 1.458616\ Th^* + R^*$ and the average block grade is obtained by dividing GTh^* by Th^*. 167 blocks are estimated. A representation of eU estimates can be found in Figure 13-33.

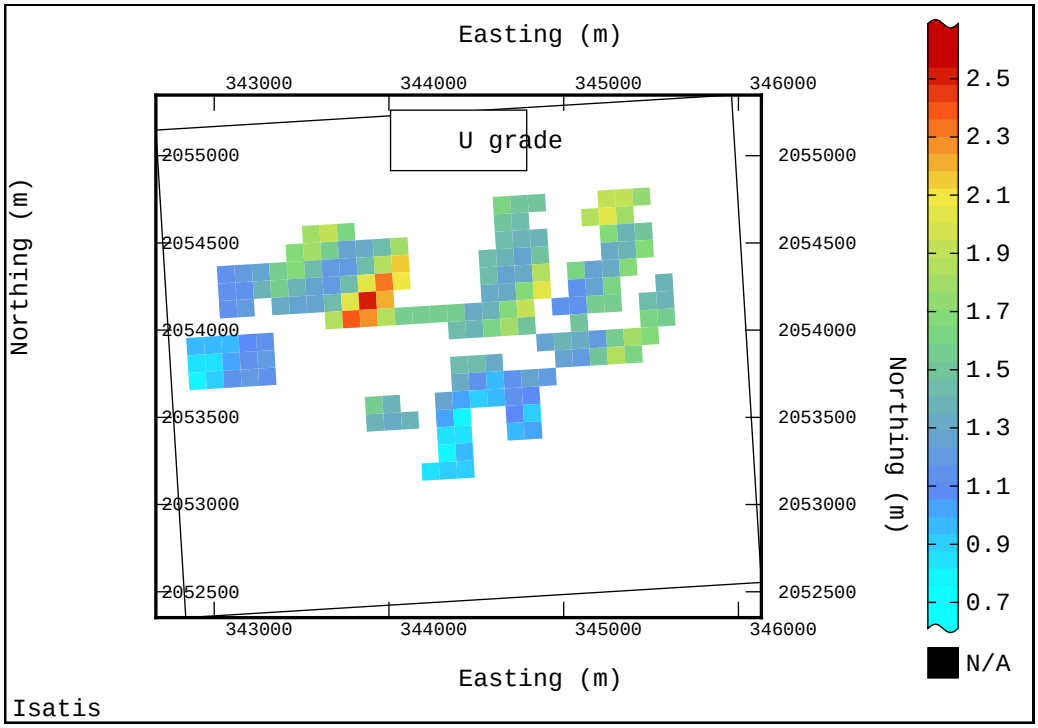


Figure 13-33: eU (kg/t) estimated by residuals method for MSEE

The quality of the estimation of individual blocks is good, depending on the drilling density. This is well illustrated by the histograms of the regression slopes (Figure 13-34). Th and R appear to be equally well estimated, as about 54 % of the blocks have a regression slope better than 0.9 for both variables. Overall the estimation is good. The global estimates are quite good.

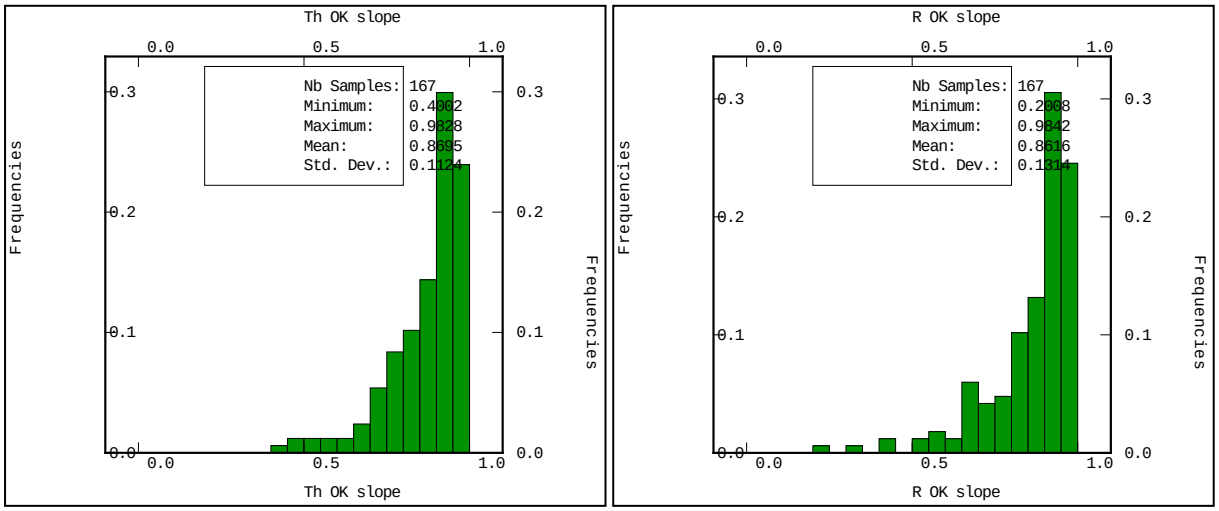


Figure 13-34: Histograms of Th and R OK regression slopes for MSEE

13.6.4 Estimation by Co-Kriging

The variograms of Th, GTh as well as the cross-variogram GTh – Th are modelled by linear combination of those components. The experimental variograms, the cross-variograms and their models are given in Figure 13-35.

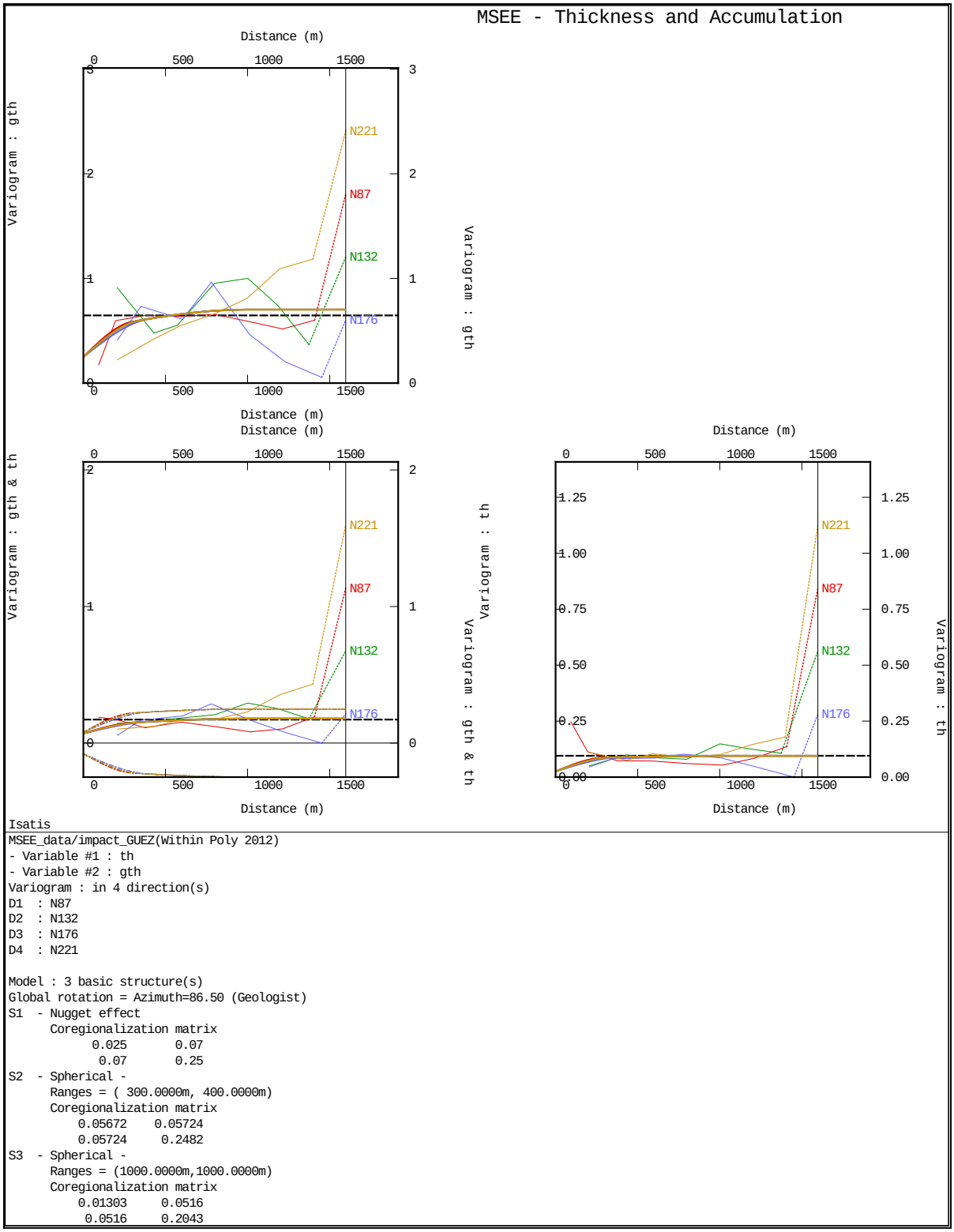


Figure 13-35: GTh and Th bivariate model for MSEE

Th and GTh are estimated by Ordinary Co-Kriging. The kriging neighbourhoods are the same as for the Residuals methods. The kriging parameters are the ones summarised in Table 13-23. Once the co-kriging of Th and GTh is done, the average block grade is obtained by dividing GTh^* by Th^*. 167 blocks are estimated. A representation of eU estimates can be found in Figure 13-36.

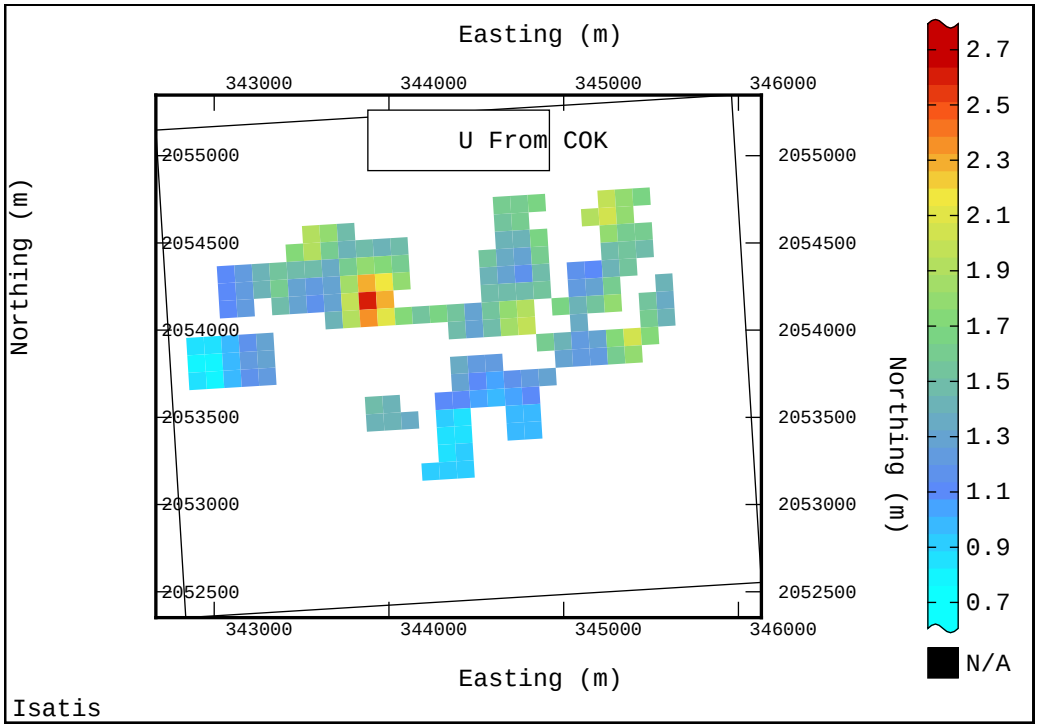


Figure 13-36: eU (kg/t) estimated after co-kriging of Th and GTh for MSEE

The results from co-kriging (1.45Mt; 1.39 kg/t) are very close to those obtained by the Residuals method (1.44 Mt; 1.37 kg/t), confirming the robustness of the estimation.

The quality of the estimation of individual blocks is similar to that of the Residuals method and quite good. This is illustrated by the histograms of the regression slopes (Figure 13-37).Th and GTh appear equally well estimated, as about 52 % of the blocks have a regression slope better than 0.9 for both variables. Overall the estimation is again good; the global estimates are good too.

As there is for the Residuals method, there is a strong smoothing effect when using the COK method.

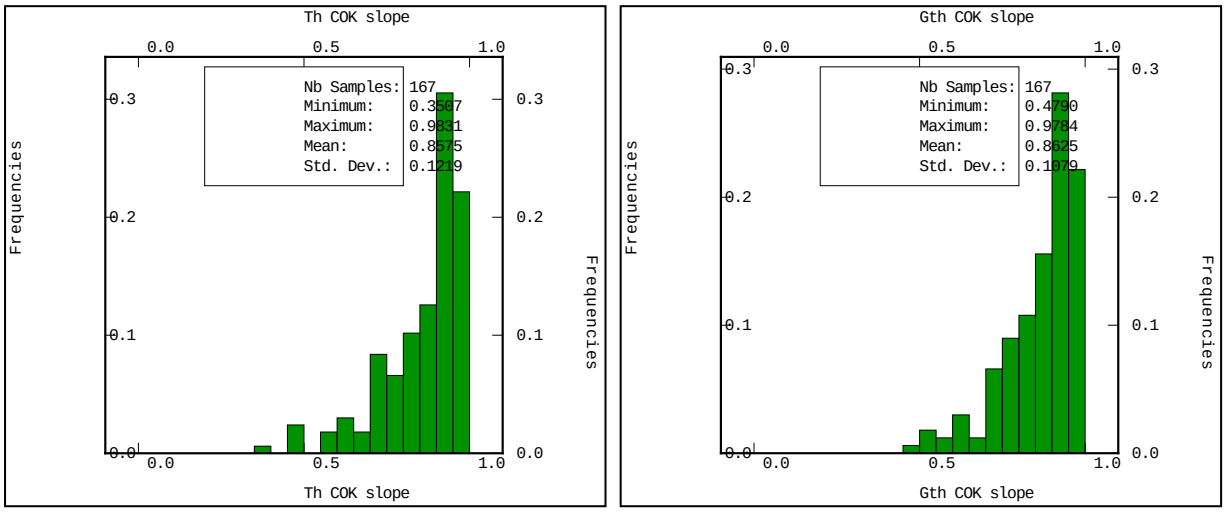


Figure 13-37: Histograms of Th and GTh COK regression slopes for MSEE

13.6.5 Model Parameters and Classification

As the drillhole density in MSEE is 200 m x 200 m, it makes sense to use a smaller dimension for the cells of the block model. SRK decided upon 100 m x 100 m blocks. The parameters chosen are given in Table 13-24.

Table 13-24: Block model parameters for MSEE

	X	Y
Origin (block centroid)	342884.22	2052404.91
Block Size	100	100
Number of Blocks	33	28

In addition the block model is rotated anticlockwise by 3.5° around its origin to align it approximately with the drilling grid. This block model is intersected by the polygons of the mineralised envelope, which gives for each block its proportion within the mineralised envelope. This proportion will be used to define the mineralised tonnage.

The density used for the tonnage is 2.3 t/m^3.

On a purely statistical basis, there would be a case to classify some of the MSEE resource as Indicated. Nevertheless, because of the lack of data and the wide spacing (well over 100 m), the continuity of the mineralisation is hard to assess and the variograms to a large extent are somewhat arbitrary, As a consequence, the resources can only be classified as Inferred.

Table 13-25: Mineral Resource Statement reported using CIM guidelines, MSEE reported at 0.4 Kg/t eU.

Classification	Tonnes (Mt)	Grade (kg/t eU_3O_8)	eU_3O_8 (t)	eU_3O_8 (Mlb)
Measured	-	-	-	-
Indicated	-	-	-	-
Inferred	1.45	1.64	2,373	5.23

13.7 Maryvonne (“MYVE”)

The estimation process followed for MYVE is identical to the one used for MSNE.

13.7.1 Data

The drilling density at MYVE is very uniform at 100 m x 100 m (Figure 13-38). The total number of 2D intersections retained for the estimation is 234.

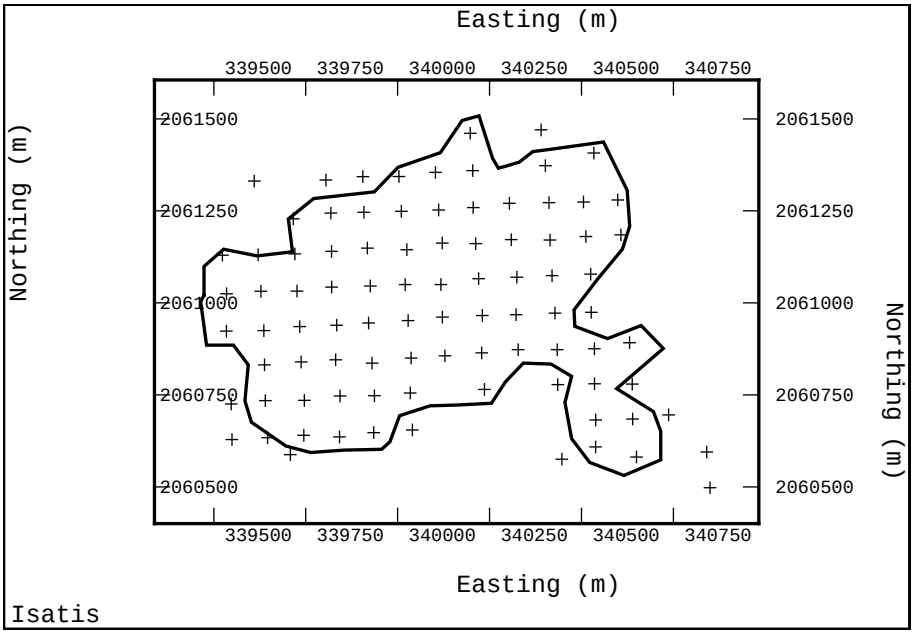


Figure 13-38: MYVE: data location and mineralisation envelopes

13.7.2 Statistics

Because the distribution of the data within MYVE is extremely regular, there is no need for declustering. Statistics for the 3 Service Variables are given in Table 13-26 and Table 13-27.

Table 13-26: Raw statistics for MYVE

Variable	Count	Minimum	Maximum	Mean	Median	Std.Dev.	Variat.Coef
Th (m)	73	0.4	2.20	0.98	0.80	0.34	0.34
GTh (mkg/t)	73	0.0	3.16	1.44	1.31	0.69	0.48
G (eU KG/T)	73	0.0	3.08	1.46	1.44	0.53	0.36

Note that the statistics for G are not very meaningful because G is not an additive variable; therefore the real average grade is more realistically given by the ratio of the average GTh to the average Th, which is equal to 1.47 kg/t

Table 13-27: Raw correlation coefficients for MYVE

Variable	Th	GTh	G (eU)
Th	1.00	0.63	0.07
GTh	0.63	1.00	0.78
G (eU)	0.07	0.78	1.00

The global histograms of Th and GTh are represented in Figure 13-39.

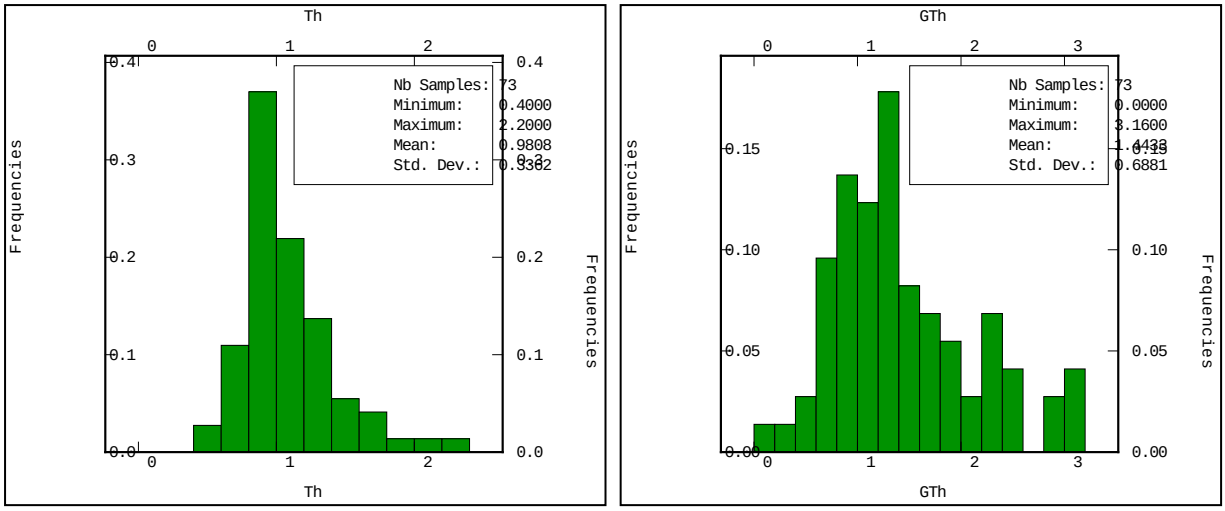


Figure 13-39: Declustered histograms of Th and GTh for MYVE

There are no real outliers in these histograms.

13.7.3 Estimation using Residuals

In order to be able to use the Residuals method, the correlation between Th and G was calculated. Figure 13-40 shows the scatter diagram and cross-variograms between these two variables: the assumption of lack of spatial correlation is reasonably confirmed with a low correlation coefficient. The cross-variograms show some structure, but overall they are fairly flat.

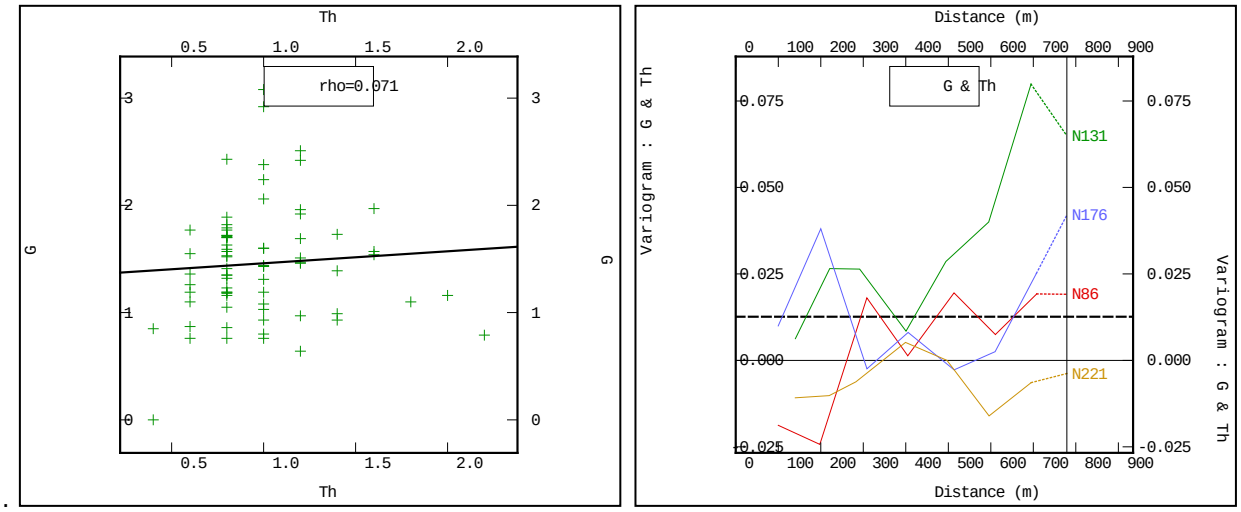


Figure 13-40: Th – G scatter diagram and cross-variograms for MYVE

The resulting regression between GTh and Th is given as: GTh = 1.452803 Th + R

The raw Th variograms show little structure. For this reason, we transformed Th into a Gaussian variable GTh with mean 0 and variance 1 (anamorphosis), and calculated the variograms of GTh which are better behaved than the raw ones, due to the symmetry of the GTh distribution. Finally, the model fitted to GTh is back-transformed through the anamorphosis function to give us the final model for Th.

The kriging parameters used are summarised in Table 13-28.

Table 13-28: Th and R kriging neighbourhood parameters for MYVE

		Th	R
Major Ellipsoid Axis Radius (m)		420	420
Minor Ellipsoid Axis Radius (m)		325	325
Rotation (counter clockwise)		3.58°	3.58°
Minimum composites required for estimate		1	1
Number of sectors		4	4
Optimum number of composites per sector		9	9
Threshold for large values		0	0
Distance		0	0
Discretisation	X	10	10
	Y	10	10

There is no need for a special handling of the high grades, as the histograms do not show excessive skewness. Once the kriging of Th and R is done, the accumulation is estimated by: $GTh^* = 1.452803Th^* + R^*$ and the average block grade is obtained by dividing GTh^* by Th^*. 345 blocks are estimated. A representation of eU estimates can be found in Figure 13-41.

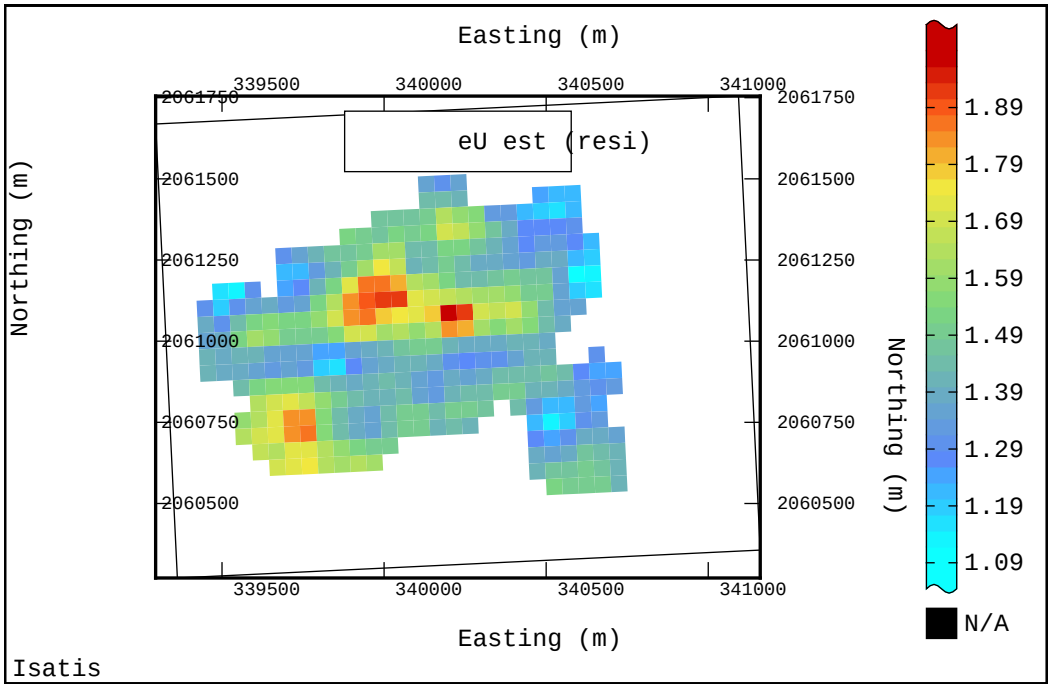


Figure 13-41: eU (kg/t) estimated by residuals method for MYVE

13.7.4 Quality of the estimation:

The quality of the estimation of individual blocks is average, depending on the drilling density. This is well illustrated by the histograms of the regression slopes (Figure 13-42). R appears to be better estimated than Th, as about 63 % of the blocks have a regression slope better than 0.8 for R against less than 50 % for Th. Overall the estimation is correct. The global estimates are reasonable.

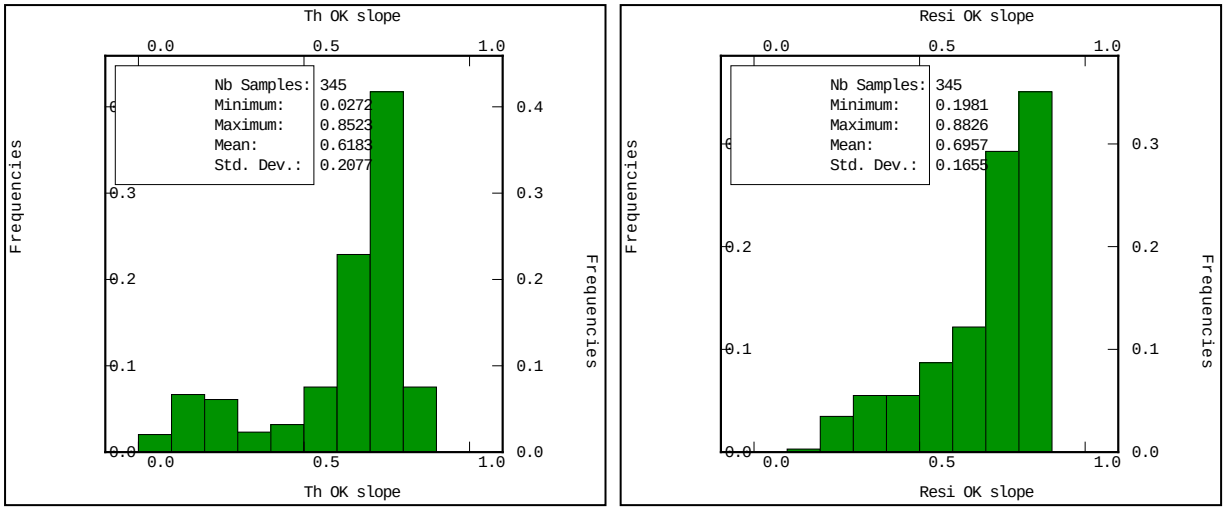


Figure 13-42: Histograms of Th and R OK regression slopes for MYVE

13.7.5 Estimation by Co-Kriging

The variograms of Th, GTh as well as the cross-variogram GTh – Th are modelled by linear combination of those components. The experimental variograms, the cross-variograms and their models are given in Figure 13-43. The back-transformed bivariate model of GTh and Th is given in Figure 13-44.

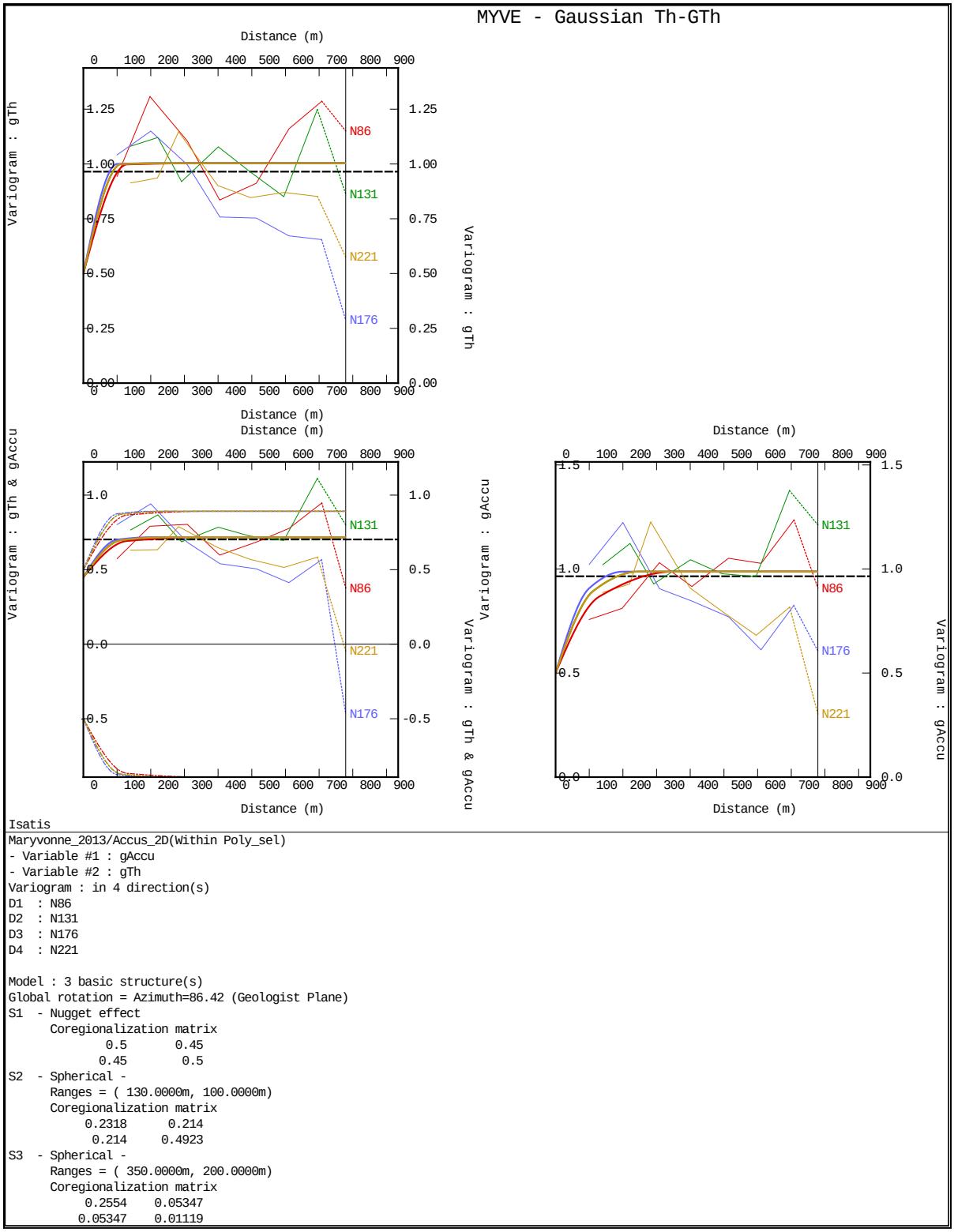


Figure 13-43: Gaussian GTh and Th bivariate experimental variograms and model for MYVE

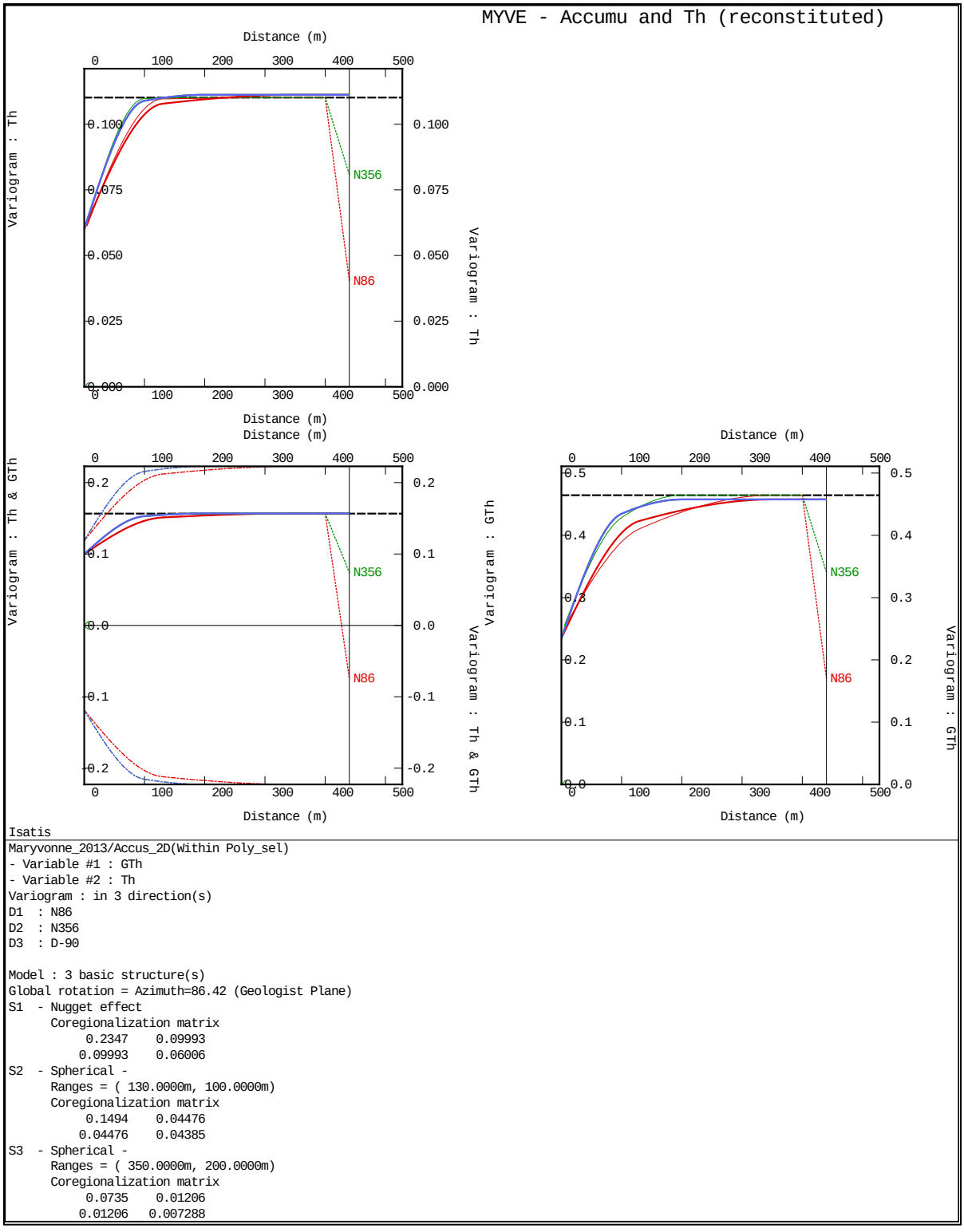


Figure 13-44: GTh and Th back-transformed bivariate model for MYVE

Th and GTh are estimated by Ordinary Co-Kriging. The kriging neighbourhoods are the same as for the Residuals methods. The kriging parameters are the ones summarised in Table 13-28. Once the co-kriging of Th and GTh is done, the average block grade is obtained by dividing GTh^* by Th^*. 345 blocks are estimated. A representation of eU estimates can be found in Figure 13-45.

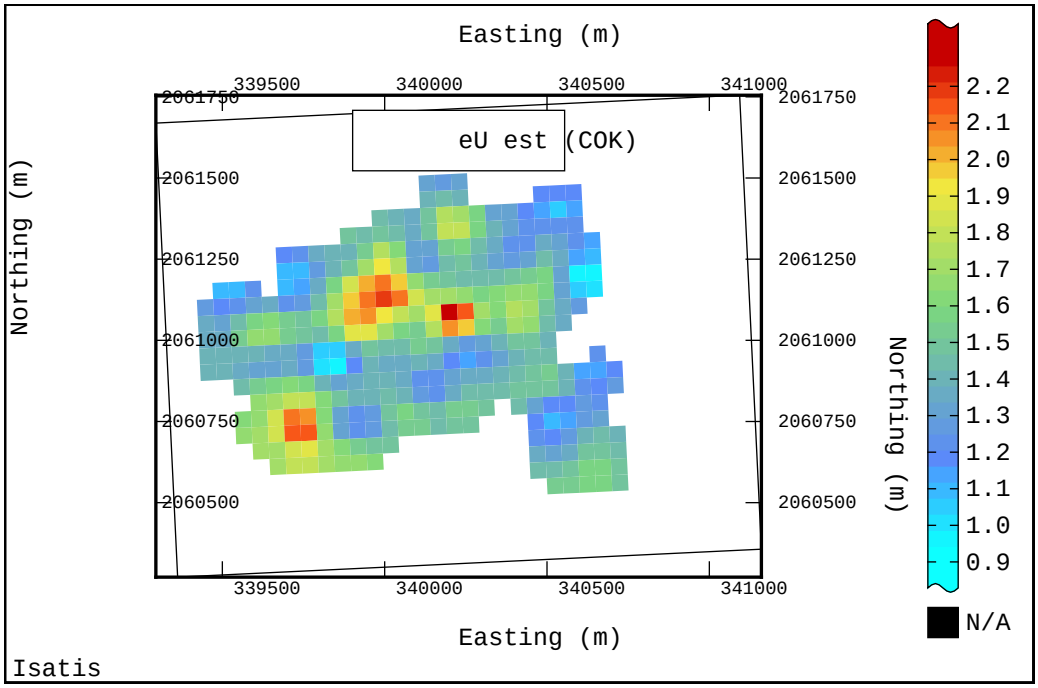


Figure 13-45: eU (kg/t) estimated after co-kriging of Th and GTh for MYVE

The results from co-kriging (1.65Mt; 1.49 kg/t) are very close to those obtained by the Residuals method (1.66 Mt; 1.48 kg/t), confirming the robustness of the estimation.

The quality of the estimation of individual blocks is similar to that of the Residuals method. This is again illustrated by the histograms of the regression slopes (Figure 13-46). GTh appears to be better estimated than Th, as over 55 % of the blocks have a regression slope better than 0.8 for GTh against 35 % for Th. Overall the estimation is again correct, the global estimates are good.

Like for the Residuals method, there is a strong smoothing effect.

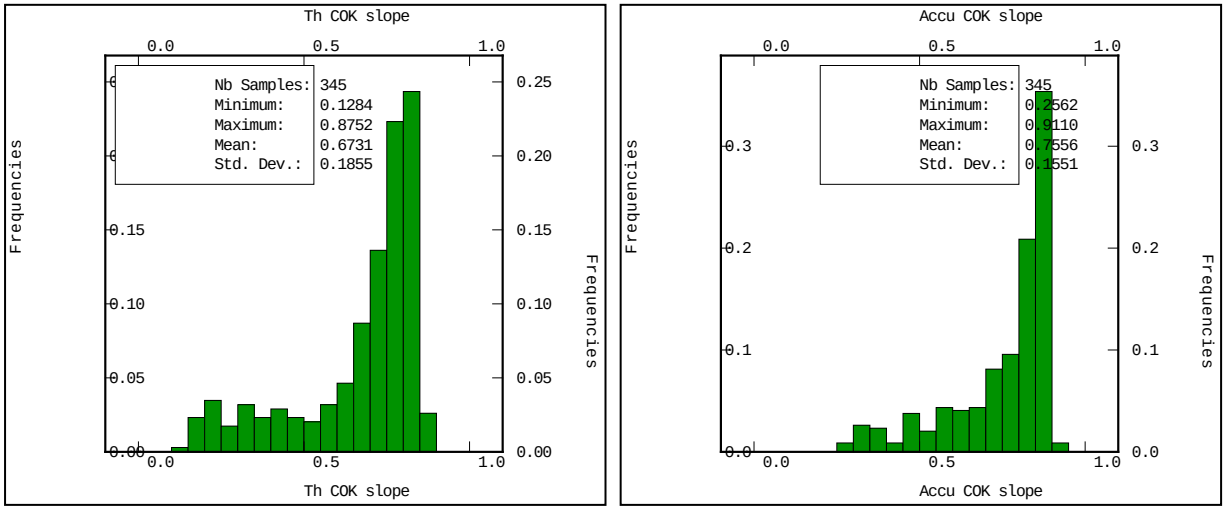


Figure 13-46: Histograms of Th and GTh COK regression slopes for MYVE

Estimation by Residuals method:

Th and R are estimated by Ordinary Kriging. The kriging neighbourhoods were optimised in Isatis. A critical parameter to define is the number of composites to be used for the estimates. The choice was guided by optimising the slope of regression.

13.7.6 Model Parameters and Classification

As the drillhole density in MYVE is 100 m x 100 m and very regular, it makes sense to use 50 m x 50 m cells for the block model. The parameters chosen are given in Table 13-29.

Table 13-29: Block model parameters for MYVE

	X	Y
Origin (block centroid)	339386	2060297
Block Size	50	50
Number of Blocks	36	28

In addition the block model is rotated anticlockwise by 2.74 ° around its origin to align it approximately with the drilling grid. This block model is intersected by the polygons of the mineralised envelope, which gives for each block its proportion within the mineralised envelope. This proportion will be used to define the mineralised tonnage.

The density used for the tonnage is 2.3 t/m^3.

In view of the uncertainties related to the grade-radiometric relationship (in large part due to the lack of DDH which can be used to calibrate it) and also because the short distance continuity of the Service Variables is not well known, no measured material can be defined. Nevertheless, from a statistical viewpoint, the estimation is of a reasonable quality, so part of the Resources can be classified as Indicated.

The classified blocks for MYVE are represented in Figure 13-47. As expected, the Inferred material is at the periphery of the orebody, where the drilling density is generally lowest, 200 m x 200 m.

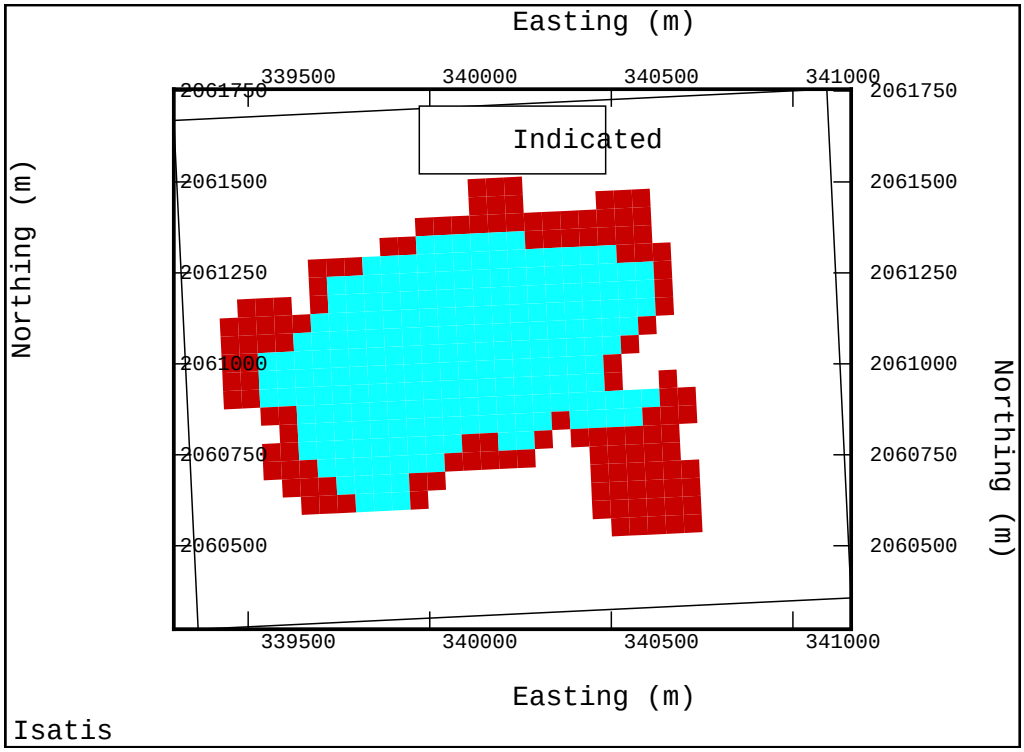


Figure 13-47: Classification (Indicated=blue, Inferred=red) for MYVE

Table 13-30: Mineral Resource Statement reported using CIM guidelines, MYVE reported at 0.4 Kg/t eU.

Classification	Tonnes (Mt)	Grade (kg/t eU_3O_8)	eU_3O_8 (t)	eU_3O_8 (Mlb)
Measured	-	-	-	-
Indicated	1.23	1.79	2,195	4.84
Inferred	0.42	1.66	703	1.55

13.8 Miriam

13.8.1 Data

Miriam mineralisation is shallower than in the other deposits and may be amenable to open pit mining. GoviEx interpreted the geology, based mainly on two features:

- The TM surface, a very important paleohorizontal horizon, which has been unfolded to facilitate the interpretation and modelling.
- The surface of the Guezouman/Talak contact. This surface does not host a lot of mineralisation in Miriam, but it is the main controlling RedOx surface.

The new drilling shows that the deposit is strongly controlled by a “structural line” trending N311E. The mineralisation within these corridors trends vertically, and accumulations of 25 to over 100 m eU kg/t are found. The width of these corridors is very important for the economy of the project

The drilling density at Miriam varies from 100 m x 100 m to 30 m x 30 m and has a total of 803 holes drilled (Figure 13-48).

SRK made use of GoviEx interpretation to build the 3D geological model, using LeapFrog® 3D. Two grade shells were built, one (nominally at 0.4 kg/t eU) was used to constrain the

richest part of the orebody (the main corridors, including the vertical “Christmas trees”), the other, much broader, at 0.2 kg/t eU, constrained the background flatter mineralisation, Figure 13-49 and Figure 13-50 show both Leapfrog® shells. These grade shells were used to flag the 20 cm composites involved in the estimation.

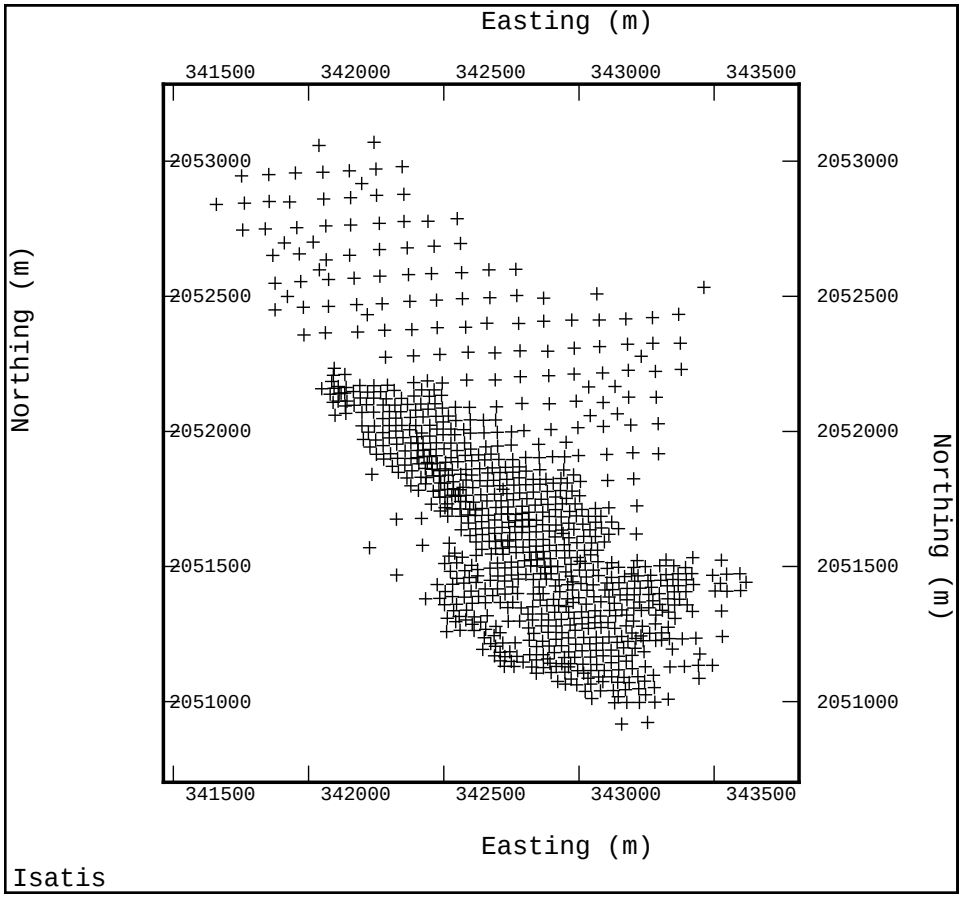


Figure 13-48: Miriam: data location

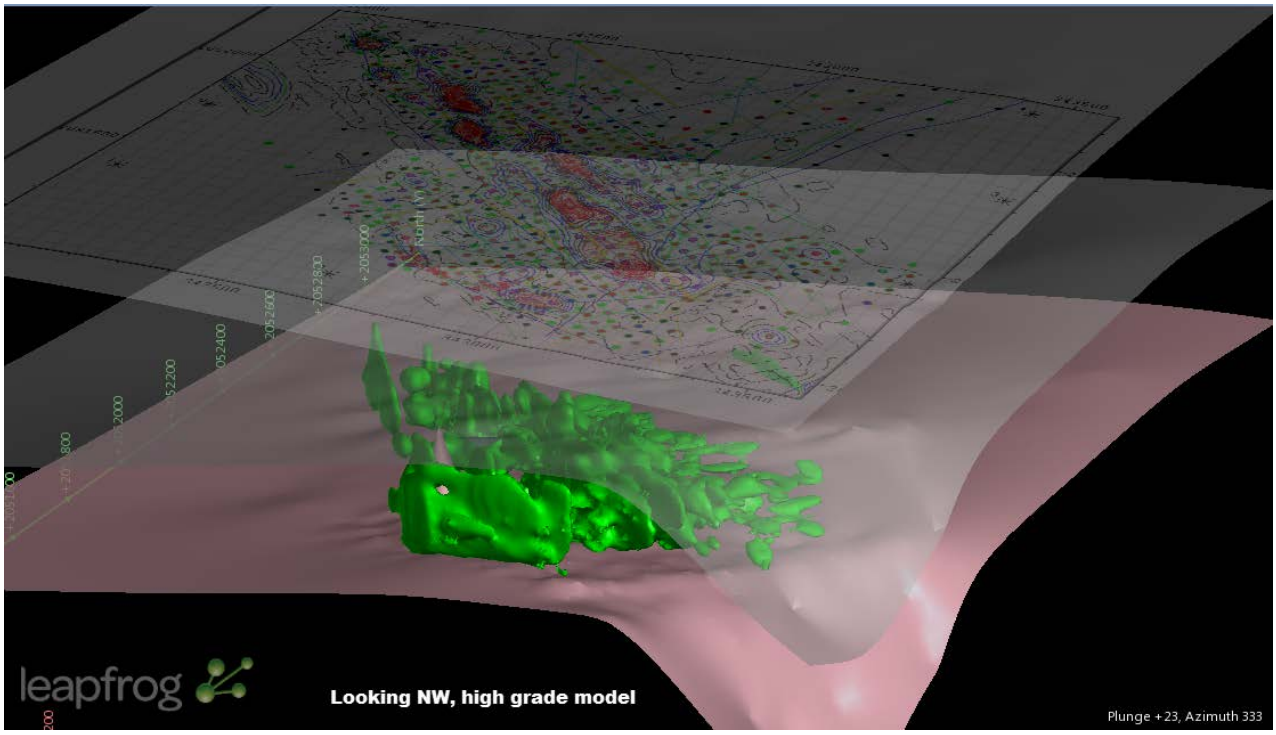


Figure 13-49: Miriam: high-grade Leapfrog model

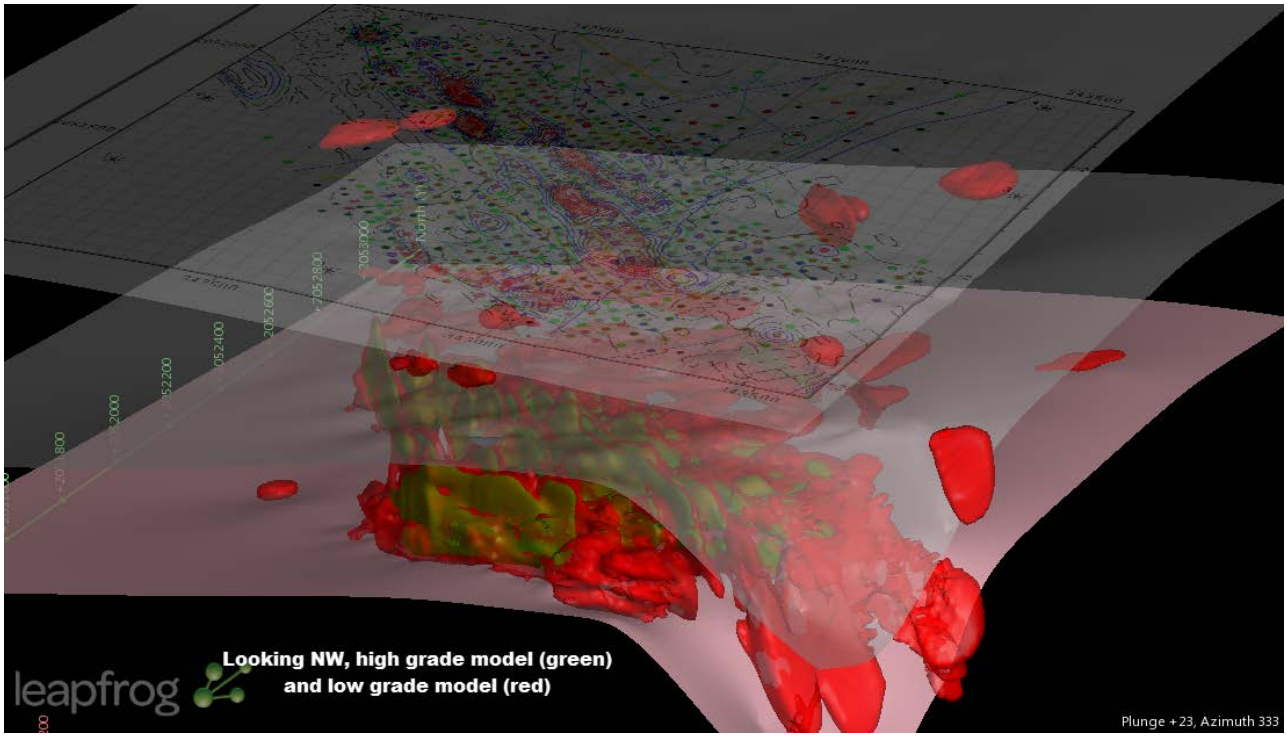


Figure 13-50: Miriam: low (red) and high (brown)-grade Leapfrog models

13.8.2 Statistics

The distribution of drillholes within Miriam's two domains: HG (0.4 kg/t eU shell) and LG (0.2 kg/t eU shell) is quite regular, which indicates that declustering is not required.

Statistics for flagged 20 cm composites are given in Table 13-31.

Table 13-31: Raw statistics of flagged 0.2 m composites for Miriam

Domain	Count	Minimum	Maximum	Mean	Median	Std.Dev.	Variat.Coef
HG eU (kg/t)	16987	0.0	43.26	0.90	0.48	1.61	1.79
LG eU (kg/t)	33305	0.0	17.89	0.30	0.22	0.36	1.18

As expected, the HG domain shows greater variability then the LG domain

In HG domain, 3860 composites have a grade eU over 1 kg/t and 81 composites have a grade over 10 kg/t. In LG domain, 921 composites have a grade eU over 1 kg/t and 3 composites have a grade over 10 kg/t. It is noted that even in the remaining composites (outside the models), there are a few isolated high grades (189 composites with a grade over 1 kg/t).

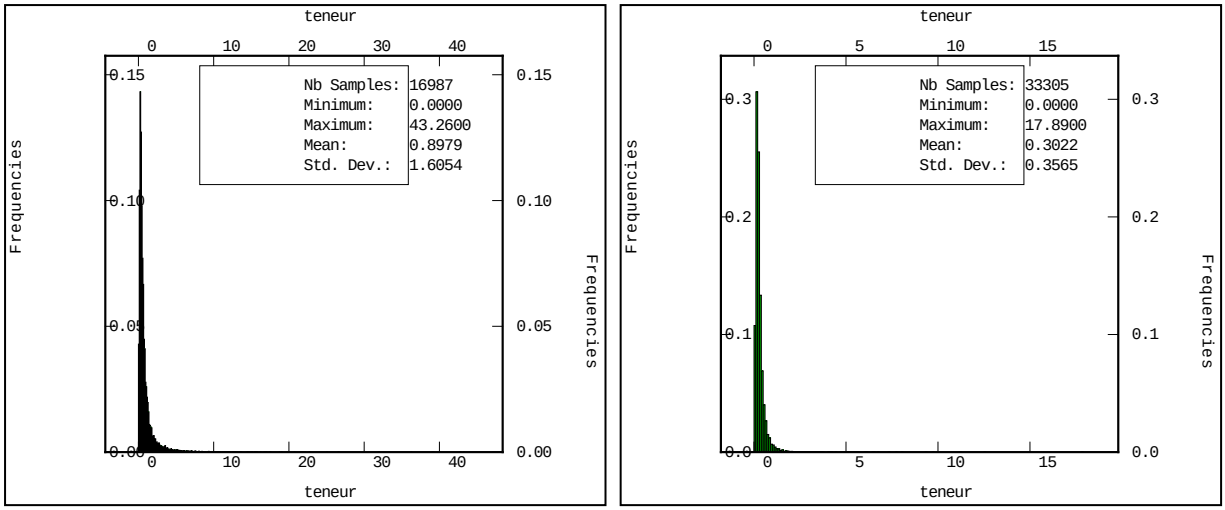


Figure 13-51: Histograms of eU for the HG (left) and the LG (right) domains in Miriam

There is a case for applying a top-cut, but SRK decided to limit the influence of the very high grades at the kriging stage.

13.8.3 Variography

Clearly, because of the presence of high-grade composites, the raw variograms are difficult to interpret. Grades were transformed into Gaussian distributed values after unfolding with respect to the TM surface.

By back-transforming the Gaussian variogram, we obtain the model adapted to the original 0.2 m composite grades.

The variograms show horizontal structures, albeit with high variability and little anisotropy. The down the hole variogram has been fitted with a very low nugget effect, due to the fact that we are dealing with reconstituted grades from radiometric measurements. There is a very clear vertical range of the order of 20 m.

13.8.4 Grade Interpolation

There are several considerations dictating the choice of estimation method:

Link with the mining method and mining selectivity: Open Pit mining is a possible option for Miriam. This means that at mining stage, grade control will be able to give some flexibility to

the search of ore.

Link between drilling density, mining selectivity and the continuity of the grade: ideally, one would like to estimate SMU size blocks directly by kriging. Unfortunately, with the current drilling density and the relatively poor grade continuity of U, this is not an option a priori, and we need to use a non-linear estimation method, where we estimate within suitably large panels, the proportion and grade of SMU parcels.

There are several possible non linear estimation methods described in the literature. Simple tests show that a Gaussian based method, Uniform Conditioning ("UC") is acceptable for Miriam.

The block model is linked to the current drilling density, and also to the orientation of the main mineralised corridors. The parameters chosen are given in Table 13-32.

Table 13-32: Block model parameters for Miriam

	X	Y	Z
Origin (block centroid)	343251.7822	2050388.394	300.000
Block Size	25	25	0.4
Number of Blocks	75	125	426

In addition the block model is rotated anticlockwise by 47° around its origin to align it approximately with the predominant HG mineralisation trend. The block model has been intersected by the HG and LG shells and contains the proportions of the respective mineralised envelopes. These proportions will be used to define the mineralised tonnage.

The density used for the tonnage is 2.3 t/m^3.

The kriging neighbourhoods were optimised in Isatis. A critical parameter to define is the number of composites to be used for the estimates. The choice was guided by optimising the slope of regression. Two kriging runs took place in each domain, so that all blocks could be estimated. In the second run, the search radii were doubled and the constraints alleviated. The kriging parameters used are summarised in Table 13-33.

Table 13-33: Kriging neighbourhood parameters for Miriam

	HG 1st run	HG 2nd run	LG 1st run	LG 2nd run
X Radius (m)	150	300	150	300
Y Radius (m)	5	10	5	10
Z Radius (m)	75	150	100	200
Rotation around Z	140°	140°	140°	140°
Rotation around Y	0°	0°	0°	0°
Rotation around X	90°	90°	90°	90°
Minimum composites required for estimate	5	1	5	1
Number of sectors	4	4	4	4
Optimum number of composites per sector	8	8	10	10
Maximum composites per drillhole	10	10	8	8
Threshold for large values	10	10	5	5
Distance (m)	2	2	2	2
Discretisation X	10	10	10	10
Discretisation Y	10	10	10	10
Discretisation Z	2	2	2	2

25,457 blocks are estimated for the HG domain and 85,499 for the LG domain. The Grade/Tonnage breakdown of the Mineral Resource for Miriam obtained by Ordinary kriging, accounting for the mineralised envelope and applying a density of 2.3 t/m^3 are summarised in Table 13-34.

Table 13-34: Miriam Grade/Tonnage breakdown for Ordinary Kriging derived Mineral Resource

eU cut-off (kg/t)	Tonnage (Mt)	eU (kg/t)	eU_3O_8 (kg/t)	eU_3O_8 (Tonnes)	eU_3O_8 (Mlb)
0	35.97	0.47	0.55	19798	43.65
0.2	34.91	0.48	0.56	19604	43.22
0.3	24.11	0.57	0.68	16311	35.96
0.4	14.12	0.74	0.87	12268	27.05
0.6	6.79	1.01	1.19	8090	17.84
0.8	3.61	1.30	1.53	5514	12.16
1	2.20	1.56	1.84	4037	8.90
1.2	1.47	1.79	2.11	3098	6.83
1.4	1.00	2.03	2.39	2381	5.25

Note these results correspond to a non selective mining method (25 m x 25 m x 0.4 m). They are therefore unrealistic, but will serve as a comparison to the UC results.

The quality of the estimation of individual blocks is variable, depending on the drilling density, but in general good. This is illustrated by the histograms of the regression slopes (Figure 13-52). The local estimation of individual blocks may be average to poor, but overall the estimation is reasonable. The global estimates are good. Clearly there is some smoothing effect. As discussed, the selectivity implied by this resource is low.

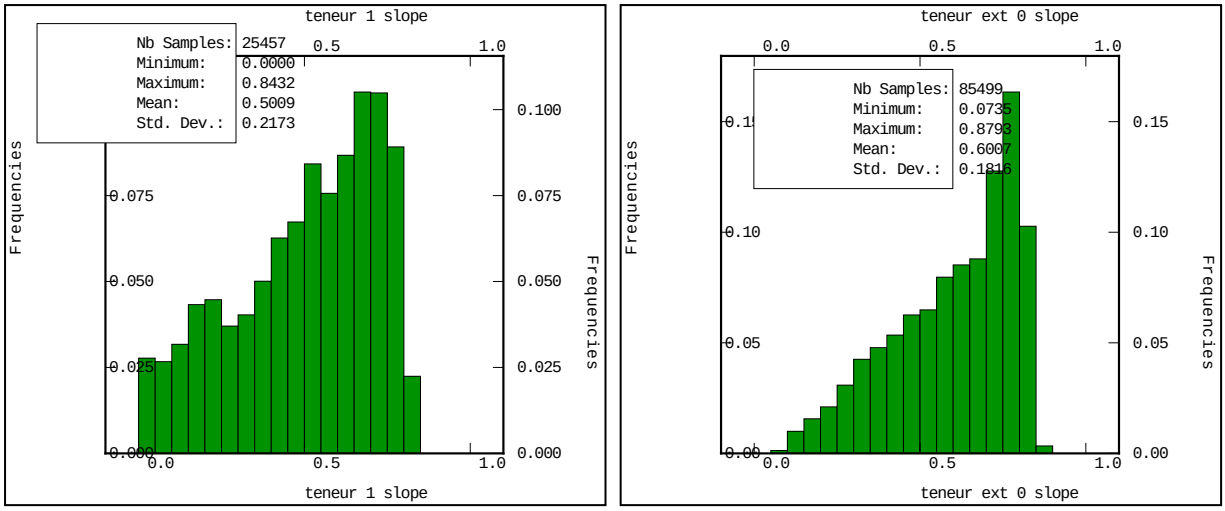


Figure 13-52: Histograms of eU regression slopes for HG (left) and LG (right) domains Miriam

13.8.5 Uniform Conditioning

To perform the UC, an SMU of 5 m x 5 m x 0.4 m was used, based on the variogram models. The Grade/Tonnage breakdown of the Mineral Resource for Miriam obtained by UC from the 0.2 m composites, accounting for the SMU and for the mineralised envelope and applying a density of 2.3 t/m^3 are summarised in Table 13-35.

Table 13-35: Miriam Grade/Tonnage Breakdown for the UC derived Mineral resource

eU cut-off (kg/t)	Tonnage (Mt)	eU (kg/t)	eU_3O_8 (kg/t)	eU_3O_8 (Tonnes)	eU_3O_8 (Mlb)
0	35.97	0.47	0.55	19797	43.65
0.2	28.07	0.56	0.66	18433	40.64
0.3	19.02	0.70	0.83	15784	34.80
0.4	12.88	0.87	1.03	13282	29.28
0.6	6.78	1.23	1.44	9796	21.60
0.8	4.14	1.57	1.85	7658	16.88
1	2.79	1.90	2.24	6236	13.75
1.2	2.00	2.21	2.61	5220	11.51
1.4	1.50	2.52	2.97	4457	9.83

As expected, a better selectivity leads to higher grades and higher metal content.

13.8.6 Classification

After the recent drilling campaign, a good deal of the resources is explored at or near a 25 m x 25 m grid. In Niger, based on a long exploration and mining history, such a drilling density is associated to a good knowledge of the geology and a good understanding of the continuity of the mineralisation. SRK therefore classified as Measured the zone of the orebody explored at a 25 m x 25 m grid. The surrounding LG zone was classified as Indicated, and isolated pods defined on the basis of little information were classified as Inferred.

13.8.7 Grade Sensitivity

The Classified Mineral Resources, based on UC and broken down by grade groups, are given in Table 13-36 to Table 13-38 to give an idea of sensitivity to various cut off grade boundaries.

Table 13-36: Measured resources obtained by UC (5 m x 5 m x 0.4 m SMU) for Miriam

eU cut-off (kg/t)	Tonnage (Mt)	eU (kg/t)	eU_3O_8 (kg/t)	eU_3O_8 (Tonnes)	eU_3O_8 (Mlb)
0	24.92	0.50	0.59	14751	32.52
0.2	19.80	0.59	0.70	13857	30.55
0.3	13.79	0.74	0.88	12099	26.67
0.4	9.62	0.92	1.08	10397	22.92
0.6	5.29	1.27	1.50	7918	17.46
0.8	3.32	1.61	1.90	6321	13.94
1	2.28	1.94	2.29	5223	11.51
1.2	1.66	2.27	2.67	4421	9.75
1.4	1.25	2.58	3.04	3809	8.40

Table 13-37: Indicated resources obtained by UC (5 m x 5 m x 0.4 m SMU) for Miriam

eU cut-off (kg/t)	Tonnage (Mt)	eU (kg/t)	eU_3O_8 (kg/t)	eU_3O_8 (Tonnes)	eU_3O_8 (Mlb)
0	10.19	0.35	0.41	4216	9.29
0.2	7.56	0.42	0.50	3756	8.28
0.3	4.57	0.53	0.63	2883	6.36
0.4	2.68	0.67	0.79	2112	4.66
0.6	1.05	0.96	1.13	1191	2.63
0.8	0.50	1.26	1.49	749	1.65
1	0.28	1.56	1.84	514	1.13
1.2	0.17	1.85	2.19	376	0.83
1.4	0.11	2.14	2.52	288	0.64

Table 13-38: Inferred resources obtained by UC (5 m x 5 m x 0.4 m SMU) for Miriam

eU cut-off (kg/t)	Tonnage (Mt)	eU (kg/t)	eU_3O_8 (kg/t)	eU_3O_8 (Tonnes)	eU_3O_8 (Mlb)
0	0.86	0.82	0.96	831	1.83
0.2	0.71	0.98	1.15	819	1.81
0.3	0.65	1.04	1.23	802	1.77
0.4	0.58	1.12	1.33	773	1.70
0.6	0.44	1.34	1.58	686	1.51
0.8	0.32	1.58	1.86	589	1.30
1	0.23	1.83	2.16	499	1.10
1.2	0.17	2.08	2.46	423	0.93
1.4	0.13	2.33	2.75	360	0.79

Table 13-39: Mineral Resource Statement reported using CIM guidelines, Miriam reported at 0.4 Kg/t eU.

Classification	Tonnes (Mt)	Grade (kg/t eU_3O_8)	eU_3O_8 (t)	eU_3O_8 (Mlb)
Measured	9.62	1.08	10,397	22.92
Indicated	2.68	0.79	2,112	4.66
Inferred	0.58	1.33	773	1.70

13.9 Summary of the Resources for Madaouela

The Classified Mineral Resources at a 0.4 kg/t eU cut-off (approximately 0.047 % eU_3O_8) and a minimum thickness of 0.4 m (except for Miriam, where a Mining selectivity unit (SMU) of 5 m x 5 m x 0.4 m is assumed) are summarised in Table 13-40.

Table 13-40: Summary of the classified mineral resources in accordance with CIM guidelines for Madaouela (cut-Off: 0.4 kg/t eU) as of April 20, 2015

Classification	Tons (Mt)	Grade (kg/t eU_3O_8)	eU_3O_8 (t)	eU_3O_8 (Mlb)
Marianne/Marilyn				
Measured	2.14	1.79	3,835	8.45
Indicated	14.72	1.43	21,000	46.30
Inferred	5.04	1.17	5,910	13.02
Miriam				
Measured	9.62	1.08	10,397	22.92
Indicated	2.68	0.79	2,112	4.66
Inferred	0.58	1.33	773	1.70
MSNE				
Indicated	5.05	1.61	8,111	17.88
Inferred	0.10	1.34	131	0.29
Maryvonne				
Indicated	1.23	1.79	2,195	4.84
Inferred	0.42	1.66	703	1.55
MSCE				
Inferred	0.72	1.81	1,308	2.88
MSEE				
Inferred	1.45	1.64	2,373	5.23
La Banane				
Indicated	1.57	1.64	2,589	5.71
Inferred	1.15	1.18	1,358	2.99
TOTAL MEASURED	**11.76**	**1.21**	**14,232**	**31.37**
TOTAL INDICATED	**25.25**	**1.43**	**36,007**	**79.39**
TOTAL INFERRED	**9.46**	**1.33**	**12,556**	**27.66**

**On November 22, 2012, GoviEx submitted to the Niger authorities a license application covering certain portions of the original Madaouela I and IV licences surface areas that were excluded from the renewed licences granted on November 2, 2012. GoviEx has been advised that the two applications for the excluded areas of Madaouela I and IV will be approved, but is awaited at the time of writing its applications, and hence has not adjusted its resources to account for any potential changes. However, it should be noted that resources associated with MSEE, and La Banane would be materially affected, and to a limited extent Miriam's resources would be affected should GoviEx not be successful in its application.*

The Company's mineral resources as at April 20, 2015 are classified in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum's "CIM Definition Standards - For Mineral Resources and Mineral Reserves" in accordance with the requirements of National Instrument 43-101 "Standards of Disclosure for Mineral Projects" (the Instrument). Mineral reserve and mineral resource estimates reflect the Company's reasonable expectation that all necessary permits and approvals will be obtained and maintained.

Mineral resources that are not mineral reserves do not have to demonstrate economic viability. Mineral resources are subject to infill drilling, permitting, mine planning, mining dilution and recovery losses, among other things, to be converted into mineral reserves. Due to the uncertainty associated with inferred mineral resources, it cannot be assumed that all or any part of an inferred mineral resource will ever be upgraded to indicated or measured mineral resources, including as a result of continued exploration.

13.10 Conclusion

The resource estimation performed at Madaouela confirms that both the 2D approach used by GoviEx, and the 3D approach for the larger and more complex deposits (MM and Miriam) are valid and that, more specifically, both the co-kriging and the Residuals method work very well. The choice between the methods is a matter of convenience, although SRK gave preference to co-kriging, because fewer assumptions are required.

The present resource update has ignored the small number of "secondary" intersections

higher up in the sequence, the continuity of which is difficult to assess. Some of them might give rise to minable ore in the future, depending on the mining method. In that sense the estimation can be considered as conservative. In addition, there is scope for additional mineralisation outside the currently recognised zones, particularly in the southern licenses and along the redox front.

The quality of the estimation is good where the drilling grid is at or below 50 m x 50 m, and reasonable at 100 m x 100 m, which allows a significant portion of the Mineral Resources to be classified as Indicated with selected areas of MM and Miriam to be classified as Measured where detailed infill drilling is available (30 x 30 m).

The current estimation suggests that in-filling to at least 50 m x 50 m as a minimum, is necessary to better define the envelope of the mineralisation and where the geology appears complex closer spaced drilling is required. Miriam has been the subject of a very intensive drilling campaign, which has clarified the geological model and allowed SRK to upgrade the classification significantly.

14 MINERAL RESERVE ESTIMATE (ITEM 15)

The Mineral Reserve estimate is based on those Mineral Resources classified as Measured and Indicated (as of May 20, 2015) and incorporate technical and economic studies that justify economic extraction on the information currently available. All Mineral Reserves are classified as Probable in accordance with international guidelines for classification.

The Madaouela Mineral Reserve estimate incorporates Mineral Resources to be mined in three areas: by open pit method from the Miriam deposit and by underground methods from the M&M deposit and the combined MSNE-Maryvonne deposits

The Mineral Reserve Statement reported for the M&M deposit is based on the Madaouela Mineral Resource Estimate by SRK (April 2015). The Mineral Reserve Statement for MSNE-Maryvonne and Miriam are based on the Madaouela Mineral Resource Estimate by SRK (March 2013).

SRK's Mineral Reserve Statement for M&M and MSNE-Maryvonne as at May 20, 2015, was prepared under the direction of Tim McGurk FIMMM who is a Qualified Person as defined by the CIM Code. SRK's Mineral Reserve Statement for Miriam as at May 20, 2015, was prepared under the direction of Rick Skelton MIMMM who is a Qualified Person as defined by the CIM Code.

The M&M and MSNE-Maryvonne deposits will be mined by underground Room and Pillar mining methods utilising low profile mining equipment. The Miriam deposit will be mined by open pit. The Mineral Reserve estimate for Miriam has been calculated by applying a global ore loss of 2% and dilution of 5% with 0 kg/t eU diluting grade on the in-situ reported ore tonnages and grades. The Mineral Reserve estimate includes the dilution associated with mining at a 1.8 m height and an addition 2.5% dilution, both with 0 kg/t eU diluting grade.

Ore processing by two-stage acid leach is used to extract the uranium from the ore after pre-concentration using radiometric ore sorting and ablation.

SRK has applied the following cut-off grades:

- 0.48 kg/t eU for the material it considers has potential to be mined underground from the Marianne-Marilyn and MSNE-Maryvonne deposits;
- 0.40 kg/t eU to define the Miriam Mineral Reserve; and
- based on a metal price of USD70/lb U_3O_8 (USD 154 /kg U_3O_8) and a metal recovery of 83.0%, without considering revenues from other metals.

The U_3O_8 equivalents have been calculated based on the formula presented below.

$$U_3O_8 \text{ equivalent} = \text{eU (uranium metal)} \times 1.17924$$

Table 14-1 shows the Mineral Reserve Estimate for the three deposits.

Table 14-1: Mineral Reserve Statement (May 20, 2015)

Deposit	Cut-Off Grade eU (kg/t)	RoM	Uranium Metal		Uranium Oxide	
		Tonnes (Mt)	Grade eU (kg/t)	Contained U (t)	Grade eU_3O_8 (kg/t)	Contained U_3O_8 (t)
Marianne-Marilyn (M&M)*						
Probable	0.48	14.1	0.79	11,164	0.93	13,165
MSNE-Maryvonne*						
Probable	0.48	7.8	0.76	5,938	0.89	7,002
Total Underground (Probable)	**0.48**	**21.9**	**0.78**	**17,102**	**0.92**	**20,167**
Miriam Open Pit**						
Probable	0.40	7.5	0.82	6,192	0.97	7,302

* *Underground Mineral Reserves for Marianne-Marilyn and MSNE-Maryvonne are reported at a cut-off grade of 0.48 kg/t eU. Cut-off grades are based on a price of USD 70/lb of U_3O_8 (USD 154/kg U_3O_8) and uranium recoveries of 83.0 %, without considering revenues from other metals. Note Mineral Reserves include both Measured and Indicated Resources.*

***Open Pit Mineral Reserves are reported within the MAD I licence and within a designed pit shell at a cut-off grade of 0.4 kg/t eU. Cut-off grades are based on a price of USD 70/lb of U_3O_8 (USD 154/kg U_3O_8) and uranium recoveries of 83%, without considering revenues from other metals. Mining modifying factors are 2% ore loss and 5% dilution at 0 kg/t grade. Note Mineral Reserves include both Measured and Indicated Resources.*

Most of the baseline data used for the Mineral Resource estimate has been obtained through gamma probe results that are considered of lower confidence than assayed results from drill core. Consequently a portion of the intersections have had verified results from chemical assays and the database of probe results regressed against equivalent chemical assay grade.

The underground mine plan is based on the geological interpretation that mineralisation is located directly above the contact between the Guezouman Sandstone and the Talak Shale. The mine design and mining methods applied are based on the interpretation of this geological contact.

The three room and pillar mining methods applied to the underground deposits at varying orebody dips are planned to be mined in a similar fashion with additional waste development and larger pillars applied in the design to compensate for their differences. There is a risk that at steeper dips, mining becomes more complicated, which incurs more ore losses and higher mining costs than is allowed for in the Mineral Reserve estimate.

Limited metallurgical tests have been made on material from the Miriam deposit compared to M&M, however preliminary results suggest the physical and metallurgical properties of the Miriam deposit are similar to those in the M&M deposit, and these have been applied to the technical studies on Miriam. While considerable metallurgical tests have been completed there remains a risk that the results of future metallurgical tests diverge from existing studies.

15 MINING METHODS (ITEM16)

15.1 Introduction

Underground mining method selection has been determined by the geological and geotechnical characteristics of the deposits and the productivity/production rate assessed as being required from the mine operations. The selection criteria used to evaluate the underground mining method to be applied to the Madaouela Mine falls into four categories:

- Physical Characteristics
  - depth of orebody;
  - orebody geometry;
  - grade (or quality) distribution; and
  - geomechanical characteristics.
- Production Considerations
  - production tonnage;
  - grade/quality of feed; and
  - maximising Resource recovery.
- Environmental Factors
  - subsidence allowed/not allowed;
  - waste production;
  - effect on groundwater; and
  - skills level of existing workforce.
- Economic Factors
  - operating costs; and
  - capital costs for establishment.

15.2 Benchmarking

There are currently two producing mines in the Arlit region that are on concessions immediately adjacent to the Madaouela Uranium Project. Both mines are operated by subsidiaries of Areva:

- SOMAÏR's Tamgak Mine is a surface operation; and
- COMINAK's Akouta Mine is an underground operation.

These two operations exploit the same target horizon explored on the Madaouela concessions and where the Madaouela Mineral Resources are found, so they are considered the most appropriate benchmarks for potential operations. Underground mining at Akouta Mine (COMINAK) began in 1974 and currently extracts approximately 0.45 Mtpa of ore. SRK and GoviEx visited the Akouta Mine and held discussions with the COMINAK engineers in March 2012.

At Cominak ore is extracted using conventional room and pillar mining. Six metre wide rooms are initially mined using leaving 18 m by 18 m pillars before being reduced to 6 m by 6 m pillars on retreat from the exhaust raise.

Where required, resuing is used to minimise dilution being sent into the ore stream. The entire

face is drilled in a single pass and probes are placed into each hole to determine which holes are to be taken in the first firing. The minimum mining height is 1.7 m with a final mining height between 2.5 and 4 m, based on the mineralised thickness.

In areas of high grade, backfill is used to maximise extraction. When pillars are mined on retreat, a single 6 m room is taken through the middle of the initial 18 m pillars and backfilled using cemented fill. The remaining 6 m pillars are then extracted and backfilled using unconsolidated fill (Figure 15-1).

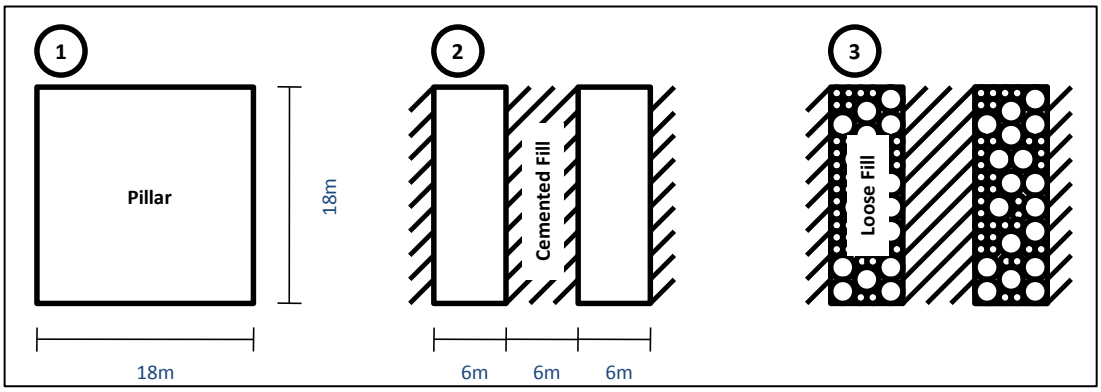


Figure 15-1: Backfill strategy used at COMINAK's Akouta Mine

Akouta Mine has previously trialled longwall mining using drill and blast. 100 m panels were developed with hydraulic supports providing ground control for the working areas (the mined areas were allowed to cave naturally). The trial lasted two years, however, the method was not integrated into the long-term plan for the following reasons:

- Ground control – trouble initiating failure in the mined out areas resulting in stress concentrations at the working face;
- Grade continuity – required mining areas of low grade that would otherwise be left as pillars;
- Equipment damage – the hydraulic supports required regular maintenance as a result of being struck by flyrock from the blasts; and
- Staff turnover – the mine experienced a high turnover of key technical staff at the time of the trial, including individuals involved in the trials.

In addition, a cut and fill method was also trialled, again using the 100 m panel framework. The method followed a similar approach to that used on a smaller scale for pillar extraction. Ventilation of the secondary drives proved to be difficult and combined with the aforementioned continuity issues, the method was not pursued.

15.3 Madaouela Deposits

Four main geological formations exist at Madaouela; Tchinezogue, Guezouman, Unite d'Akokan (UA) and Talak formations. Structurally the lithology is generally sub-horizontal with minor domes and some fault structures causing minor localised increases in bedding dip and slight increases in fracture frequency. Rock mass properties vary by lithology as outlined in the table below.

Table 15-1: Overview of Geology

Lithology Formation	Comments
Tchinezogue – upper 40m (weathered)	Reduced intact strength (UCS≈50 MPa) and increased fracture frequency seen above water table due to weathering/oxidation of the bedrock.
Tchinezogue – below 40m (unweathered)	Moderate strength anisotropy, likely due to the thinly bedded nature of the rock mass. UCS≈70 MPa. Low fracture frequency and high RQD is common.
Guezouman Sandstone	Comprises fine to coarse grained sandstones with minor conglomerates. Intact strength is more isotropic since unit is more massive. High void ratio results in relatively low UCS values (≈40MPa). Low fracture frequency, increasing slightly around mineralized zone where mineralization has slightly damaged rock mass.
Unite d'Akokan	Occurs as localized channels. Consists of interbedded mudstones and siltstones resulting in strength anisotropy (UCS values from 30 to 140 MPa). Localised zones of higher fracture frequency exist, likely related to syn-sedimentary fault structures.
Talak	Consists of fine-grained mudstones and claystones containing mafic minerals. Rock mass strength is relatively isotropic and the intact rock strength of this unit is considered extremely weak (≈25 MPa); some anisotropy is observed due to mudstone laminations on a micro scale. Fracture frequency is low and RQD is high despite the weak intact strength.

The Madaouela deposits being considered for mining comprise four distinct deposits:

- Marianne-Marilyn (M&M);
- MSNE;
- Maryvonne; and
- Miriam.

All four deposits feature in the Guezouman sandstones, a series of flat-lying sedimentary beds with varying mineralised thicknesses and dips up to a maximum of around 20°.

The footwall of the mineralisation, known as the Talak Shale, has a clay like nature. Mineralisation is formed along the Guezouman-Talak contact and concentrates on the edges of the UA channel and along structures within the rock.

Miriam is to be mined using open pit methods due to the larger mineralised thicknesses and proximity to the surface. Whilst there is potential for highwall mining to improve the Resource extraction, no engineering work has been undertaken on this option.

Figure 15-2 demonstrates that the gradient of Guezouman-Talak contact that forms the footwall of the Marianne-Marilyn mineralisation is variable in nature. The steeper gradients reflect the edges of the UA channels and structures within the rock mass. This variation in gradient is common to all Madaouela deposits targeted for underground mining and means a single approach to mining is unlikely to be applied consistently throughout the deposit.

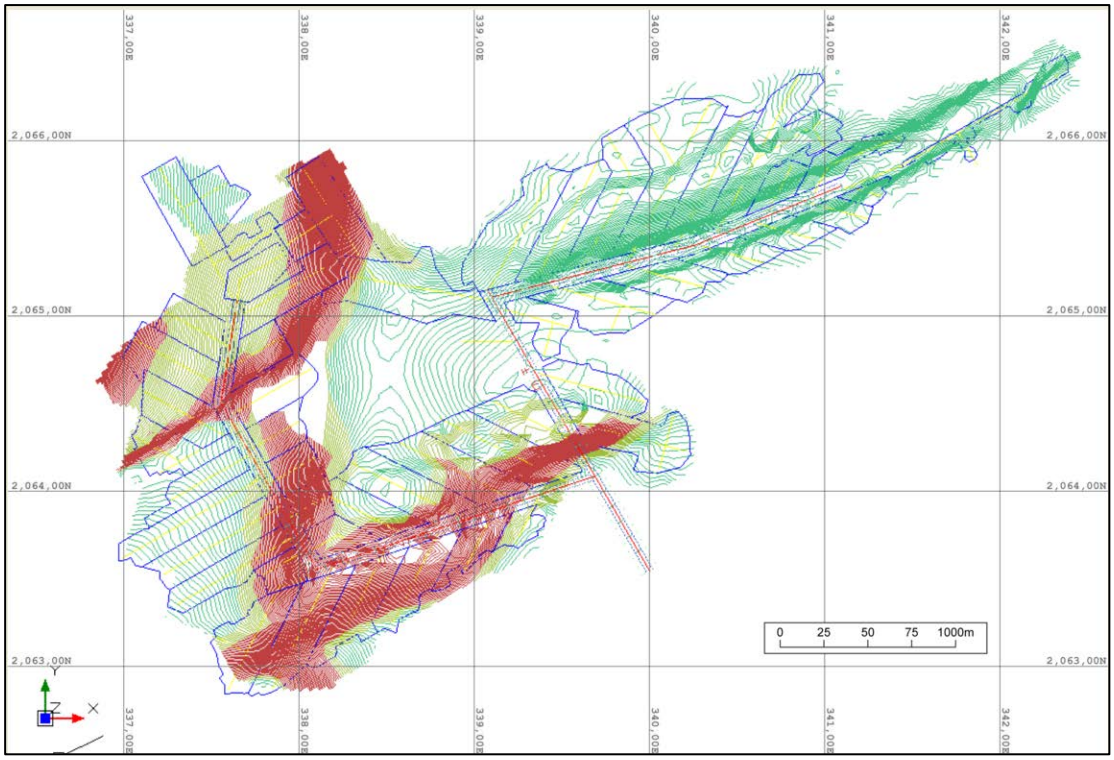


Figure 15-2: Gradient of footwall contact of Marianne-Marilyn resource (in degrees)

Grade continuity is an important factor when assessing a mining method. Selective mining methods will be required where continuity is inconsistent. The grade distribution within the block model suggests that continuity will only be a minor consideration in the mine design process, and demonstrates the falloff in grade within the UA channels. Outside of this, the grade distribution is relatively consistent.

15.4 Underground Mining

15.4.1 Introduction

SRK recommends that the Marianne-Marilyn (M&M) and MSNE-Maryvonne deposits are mined by room and pillar mining methods for the following reasons:

- Demonstrated as being an effective method in similar conditions;
- Allows height of mining to be adjusted to reflect the variable nature of the mineralisation;
- Allows pillars to be left in areas of low grade;
- Flexibility to react to localised variations in the deposit;
- No specialised equipment required; and
- Reduced training requirements as local workers already familiar with the mining technique.

As indicated (Figure 15-2) the gradient of the footwall contact is variable and therefore SRK has applied a conservative approach to the application of mining methods. To account for the variability in gradient (θ), three variations of room and pillar mining are to be implemented:

- Where θ<7° - traditional room and pillar mining (“TR&P”);
- Where 7°<θ<10° - apparent room and pillar mining (“AR&P”); and
- Where 10°<θ - step room and pillar mining (“SR&P”).

AR&P is similar to TR&P, but rooms and pillars are angled at 45° to the dip direction to

minimise the effective gradient of mining. Though conventional conditions would allow for mining on a gradient steeper than 7°, the clay like nature of the Talak shale footwall will impact negatively on roadway conditions and as a result GoviEx and SRK has taken a conservative approach to limit the impact of gradient.

10° is used as a cut-off for SR&P as at this dip angle the gradient of development at 45° to the dip direction is above 7°.

To simplify the transition between TR&P and AR&P, the panels are to be oriented at 45° to the main structures intersecting the Mineral Resources that create the zones of higher dips.

The SR&P method is to be applied in the isolated zones of higher dip angles. Where this occurs, the room and pillar pattern will require reorientation in line with the strike and dip of the geology.

A common approach to mining has been assumed for all underground deposits to allow the same skills and equipment to be applied across all operations. The modular nature of room and pillar mining in panels allows the inputs to production costs and productivities to remain relatively stable across all operations, simplifying the analysis and ultimate operations. The configurations selected for the underground mine design and proposed operations is based on an evaluation of the options available in a number of areas. The options assessment incorporates:

- Mine access;
- Extraction method (drill and blast or mechanical excavation);
- Vertical development method;
- Materials handling;
- Ground control;
- Equipment;
- Services; and
- Labour.

The options are benchmarked against the adjacent underground mine, Akouta, operated by COMINAK.

15.4.2 Mine Access

Access to and from an underground mine needs to account for:

- “Men and materials”: access for workers, equipment, and supplies (including explosives) to the underground workings; and
- “Materials handling”: removal of ore, and waste where required, from underground to surface.

Though shaft hoisting can be applied across the full spectrum of production rates, traditionally it has been favoured at higher production rates. However, recent advances in conveyor technology made declines more appealing at higher rates. The proposed Madaouela mines will have medium production rates making both options viable. Analysis of material handling is further discussed in Section 15.4.5.

Due to the shallow depth of mineralisation, which is typically around 100 metres below the surface for the Madaouela deposits, decline access has been adopted. This has the following additional advantages;

- The use of conveyors to bring the mineral ore to surface.
- Simplifies materials handling to bring spare parts and other consumables into the mine.
- Simplifies movement of men during shift change-overs.
- The processing plant is envisaged to be at a distance from the deposits meaning any lateral distance covered by a decline will reduce the transportation distances on the surface.

A three-tunnel decline has been applied to be consistent with the selected mine layout (Section 15.5.5) and material handling method (Section 15.4.5). The three tunnel approach allows for two access roads either side of a central conveyor increasing the carrying capacity of the mine infrastructure in terms of air, services and vehicles thereby increasing the number of panels that can be operated simultaneously and provides for more egress options from the mine for employees in case of an emergency.

The central conveyor drive is to have a profile of 3.2 m high by 4 m wide and the two flanking access drives, 4.5 m high by 5 m wide. Only the outside drives are large enough to be used for trucking so it is envisaged that all three drives would be developed simultaneously.

The gradient will be dictated by the Nigerien legislation at 1:7. No concreting of the floor has been allowed for. A boxcut has been designed to a depth of 20 m below the surface to provide a fresh face for the portal. All three declines would share the same boxcut

The strata overlying the mineralised horizon has low strength properties. As a result, the low cross-sections of the decline method are considered a lower risk than the larger shaft dimensions.

A visualisation of the planned boxcut, decline portal and surface infrastructure can be seen in Figure 15-3. This includes the process plant, mine surface facilities and the dry-stack TSF facility in the background.

Figure 15-4 depicts the plant and RoM stockpile looking South-East from the M&M boxcut.

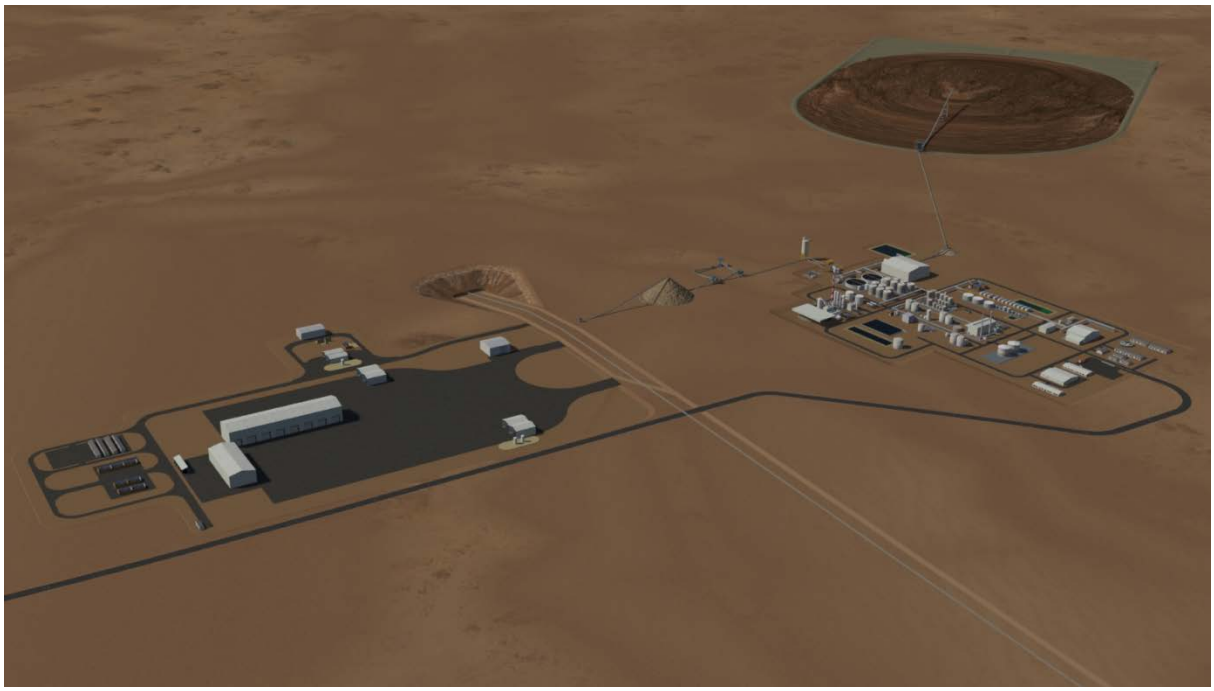


Figure 15-3: View of the M&M portal looking North-East.

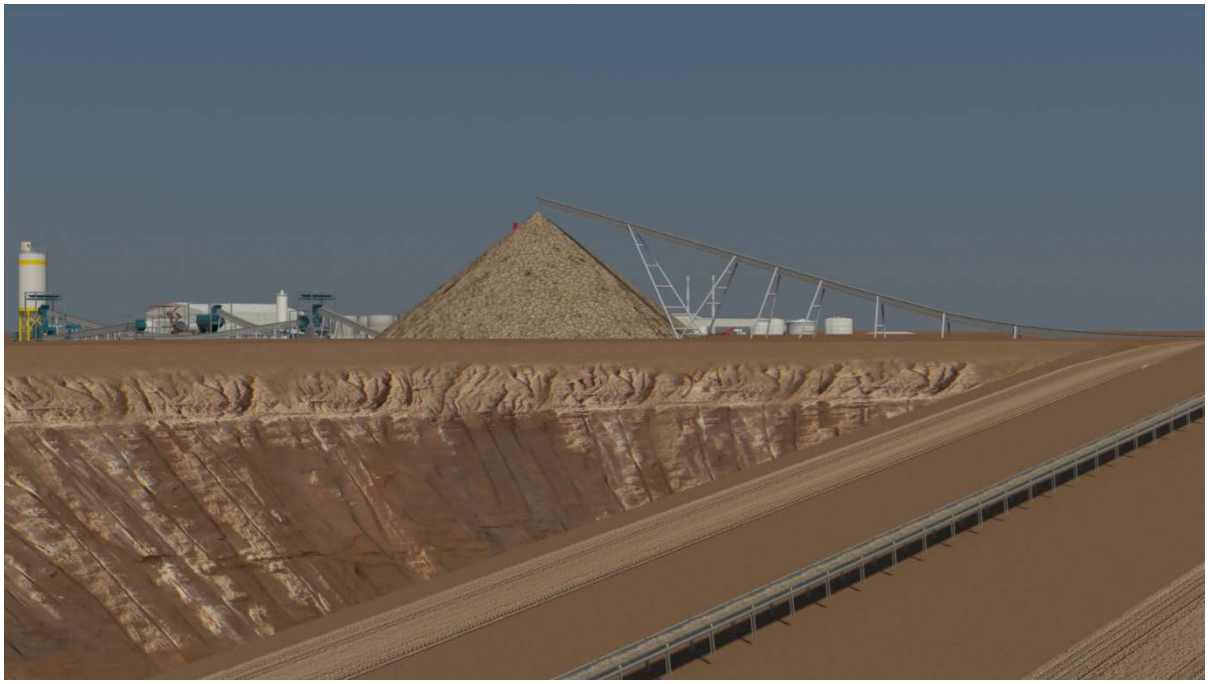


Figure 15-4: View of the RoM stockpile and plant looking South-East

Benchmarking

The nearby Akouta Mine uses a twin decline access, with a conveyor in one access and transportation in the second. The declines run parallel and daylight in the same boxcut portal. The decline is at a 1:5 gradient, constructed prior to regulations limiting declines to 1:7 gradient. Communication with COMINAK staff suggests that any future decline requirements would be at 1:7, even without the legislative change, due to the practical challenges of such a steep access. The floor of the decline has been concreted, which assists with traction on the steep decline.

15.4.3 Extraction Method

Introduction

Room and pillar mining uses the same equipment used for mine development. Therefore, much of the discussion in this subsection applies to both mine development and production.

There are two basic options for underground excavation; drill and blast or mechanical excavation.

Benchmarking

At Akouta Mine the majority of mining is by conventional drill and blast. Production mining headings are 6 m wide with a height ranging between 2.5 m and 4 m depending on the mineralised thickness. On average, around 150 t of rock is mined each blast.

Where the mineralised thickness is less than 2.5 m, resuing is used with a minimum height of excavation of 1.7 m. Drilling is completed in a single pass and probing (grade control) is used to define the blasted height. The ore is excavated and a second firing for waste then takes the development to the full height.

Drillholes are 40 mm diameter (with 5 uncharged void holes used) and are charged with 32 mm diameter packaged emulsion explosive. Five burn holes (uncharged) are drilled but

not reamed. COMINAK has a preference not to use ANFO due to the presence of water from drilling, though this issue can resolved by angling drill holes to allow the water to drain where possible and blowing out the holes with air after drilling.

A key issue relating to drill and blast is explosive storage requirements. Explosives are delivered once every six months, due to the logistics and security required during transportation to site. This requires large storage facilities and appropriate security on site.

In 1984, COMINAK trialled mechanical excavation methods; the technology was not implemented due to the high abrasivity of the material; however there is little data from the trials available.

Applied Design

SRK has ruled out mechanical excavation due to the abrasive nature of the deposit, and the high capital cost of the equipment. Consequently, the preferred approach to mining is to develop the rooms are developed using low profile drill and blast equipment.

Room and pillar mining is generally applied to tabular deposits with shallow dips and deposits with a large footprint (several kilometres) can enable multiple independently active mining districts. This mining method is therefore ideally suited to the Madaouela geology. Rooms are developed within the mineral horizon to leave pillars in-situ. These pillars act as primary, regional support for the roof and overlying strata. The target mineral horizon is located within the Guezouman sandstones and consists of sandstone beds interspersed with thin bands of mineralisation.

The scale of mining is such that a 1.8 m cut will be required to facilitate the mining equipment and workforce. The 1.8 m mining cut is planned to extract all the mineralised bands within the Guezouman host and this will therefore introduce a significant portion of waste dilution material into the RoM ore.

The mining equipment selected has an effective operating range of 1.7 m – 2.5 m seam thickness. In thicker areas of the deposit benching can be used to extract the mineral (Figure 15-5), however as less than 1% of the mineable area of the deposit has a mining height greater than 1.8 m this is therefore considered negligible in its impact.

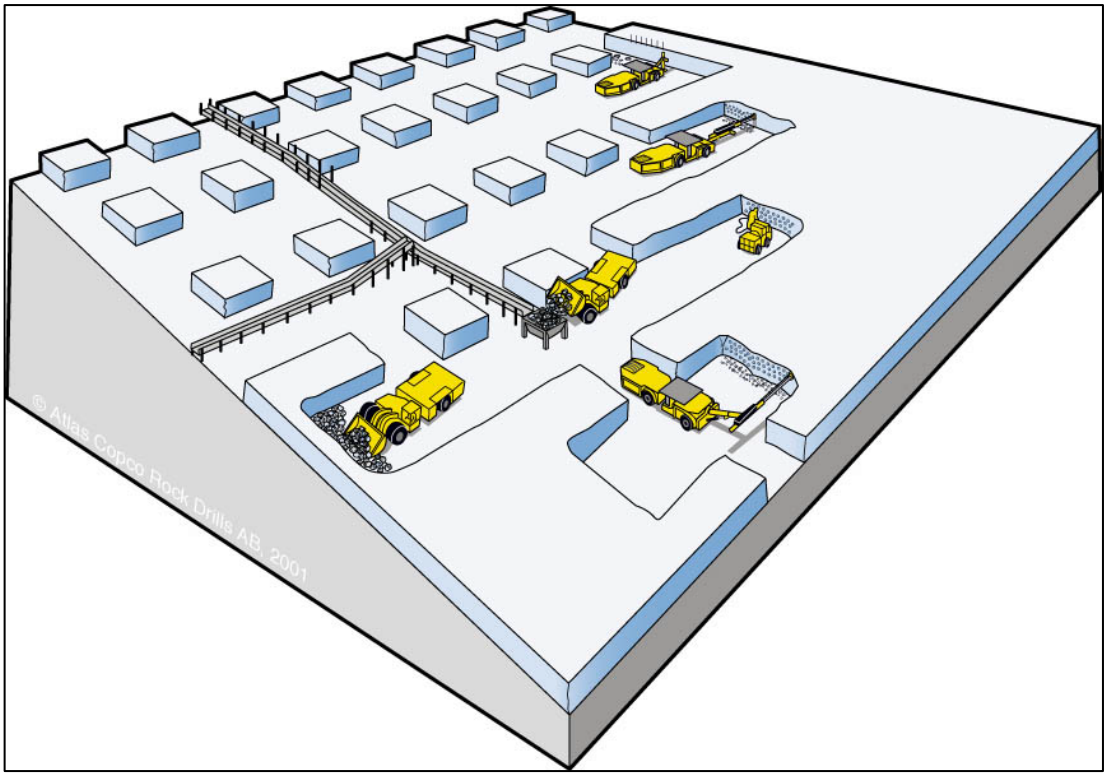


Figure 15-5: Schematic of room and pillar mining (Atlas Copco 2008)

Benefits of Room and Pillar mining

- Thicker sections of the orebody can be benched;
- Allows for selective mining where larger pillars can be left in-situ in areas of thin seam thickness or low grade;
- Most infrastructure development can be included within the orebody itself by maintaining pillars limiting the need for waste rock development; and
- Development requirements can be largely standardised as often the same equipment is used for both development and production producing operational efficiencies and cost savings.

Step Room and Pillar Mining

There are many variants on the basic room and pillar method that can be applied in areas of variable geology. For example in thicker seams, post-pillar room and pillar is a viable alternative or for thin seams with a steeper dip, step room and pillar is an option.

Step Room and Pillar is a variant of the traditional room and pillar method that allows the mining to take place at dips between 8 and 30°. Traditional room and pillar methods struggle in this environment as the gradients become too great for the efficient operation of LHD's to clean the faces. Where the dip is greater than 30°, there is a risk of roadways deteriorating adjacent to the excavated rooms causing a safety hazard.

In step room and pillar mining, the orebody is developed along strike, with rooms mined connecting the parallel drives down-dip. Where geotechnical conditions allow, multiple drives can be developed along strike to increase the room width (Figure 15-6).

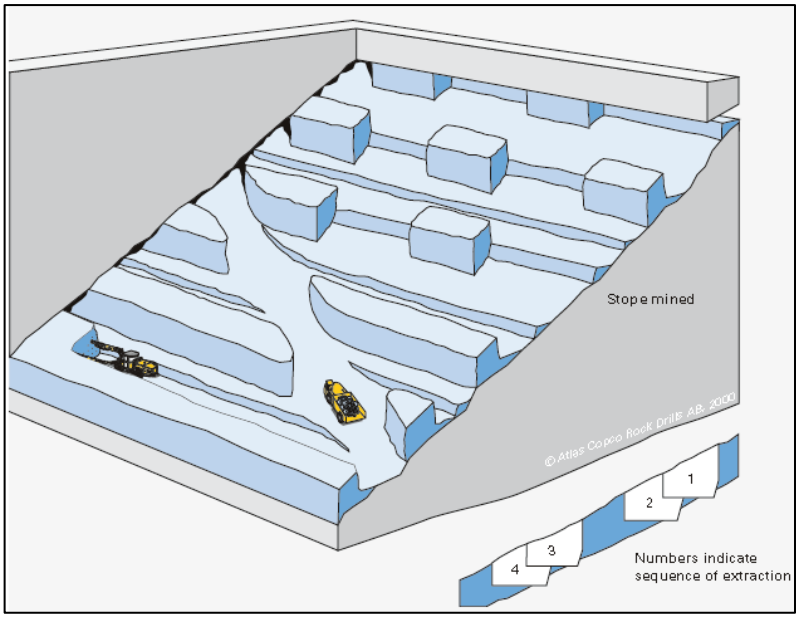


Figure 15-6: Schematic of step room and pillar mining (Atlas Copco 2008)

The advantage of step room and pillar is that it is compatible with other forms of room and pillar allowing the same fleet of equipment to be used in varying conditions and shares many of the advantages of traditional room and pillar.

However, the method has greater mining losses due to the need for a flat floor for tramming and pillar recovery can be difficult. Most notably, the angle of dip prevalent with this method is such that blasted pillar material tends not to rill naturally into the development below for loading.

Drill and Blast

The key advantage of drill and blast is its wide application in different types of ground, with drill and blast patterns being changed to suit rock types and ground conditions. The process is cyclical, which enables equipment to be used to advance multiple headings concurrently, though the advance rates of individual headings will depend on the number activities in the production cycle, activities duration and the amount of time between activities (face utilisation).

Explosives

Three types of explosives have been considered for the proposed Madaouela operations;

- Ammonium nitrate-fuel oil (“ANFO”);
- Bulk emulsion; and
- Packaged emulsion.

ANFO and bulk emulsion are blasting agents that require primers to initiate, whereas packaged emulsions is essentially the same product as a bulk emulsion but can be made to be ‘detonator sensitive’ so can be used as a primer or as an explosive without the use of primers.

Emulsion provides a higher velocity of detonation (“VOD”) than ANFO which provides better fragmentation after blasting. However, ANFO allows for better propagation of gases after the initial shock (“heave”), which can assist with the generation of void during a blast (Table 15-2). It is for this reason that ANFO is favoured for softer rock . Bulk emulsion can be adjusted

when loading into a blasthole to change the physical properties of the explosive to provide different blasting properties.

Table 15-2: Characteristics of underground bulk explosives (Orica 1998)

Explosive Type	Density (t/m^3)	VOD (km/s)	Relative Energy	
			Fragmentation	Heave
ANFO (Poured)	0.8	3.7	100	100
ANFO (Blow-Loaded)	0.95	4.3	134	125
Bulk Emulsion	1.2	6.2	140	115

Water

The presence of water within a blasthole will affect the performance of a blast when loaded with ANFO as ammonium nitrate is water soluble. The emulsions provide an oil-based matrix for the ammonium nitrate providing water resistance. Emulsions are therefore preferred in wet conditions, especially when residence times are higher. Plastic hole liners can be used to isolate the ANFO from the water, however, these can be difficult to use and will affect the productivity of loading. Conventionally, for ANFO mines with minor water issues, wet holes will use packaged emulsion with dry holes using ANFO.

Transportation and Storage

The putty-like nature of packaged emulsion makes them a simpler product to handle, though the pump-able nature of ANFO and bulk explosives allows for faster loading of the blastholes. Packaged emulsion comes in boxes so are easy to transport around a site.

ANFO is generally transported underground as either 25 kg bags or in a 1 t hessian bag. ANFO would then be loaded into a venturi loader for loading either in the magazine for the 1 t bags (a crane will be required) or at the face for the smaller bags. When the 1 t bulk bags are used, the ANFO can be mixed on site allowing for the ammonium nitrate to be transported as a chemical, rather than an explosive. Bulk emulsion has a similar advantage over packaged emulsion.

Emulsion is loaded using re-pump units, also referred to as mobile mixing units (“MMU”). The emulsion doesn’t become an explosive until it is mixed with the gassing agent as it is being loaded into a hole. MMUs range in size from 2 to 10 t so can be difficult to apply in smaller mines.

Cost

As a general rule, ANFO is the cheapest option, followed by bulk emulsion and then packaged emulsion.

From an operational perspective, Madaouela is seen as a reasonably dry deposit with low strength rock properties. Conventionally, ANFO would be expected to perform better than emulsions under these conditions as it has greater heave. Packaged explosives have been assumed for the primers, lifters and an allowance has been made for wet holes

ANFO has been selected as the bulk explosive for both production and development. The use of ANFO means the constituents chemicals are transported separately greatly simplifying the delivery and storage of the explosives on site as the reduced security requirements allow for more regular deliveries and therefore reduced storage capacity.

Conventional Blasting

Room and pillar production can be divided into two zones, swings and slabbing (also known as slashing). The swings are where there is only one free face so mining is undertaken in a similar style as conventional development. Slabbing is where there are two free faces so a benching-style blast pattern can be applied, reducing the required powder factors. In a well scheduled room and pillar mine, around half of the production can be mined using slabbing.

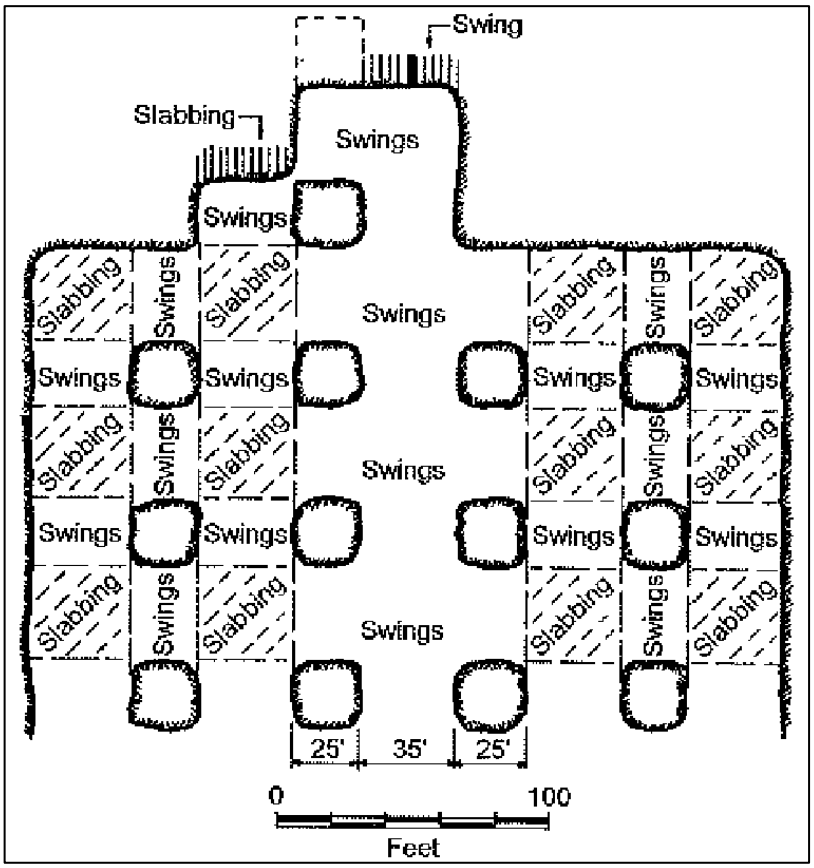


Figure 15-7: Plan view of panel advance using swings and slabbing (Bullock 1998)

There are two common drill patterns used for mining the swings in room and pillar mining; v-cuts and burn cuts.

V-cuts are most commonly applied in softer rocks. It involves angled blasting in a development/production face to create an initial free face without the use of void holes (Figure 15-8:). Blasthole drilling is minimised, however, as there is a poor distribution of explosives, it can lead to poor fragmentation. Drilling v-cuts rather than burn-cuts reduces cycle times improving advance rates. Although, fly rock can be thrown over 100 m from the face consequently drilling and bolting equipment needs to be largely independent of nearby services (air, water and electrical distribution).

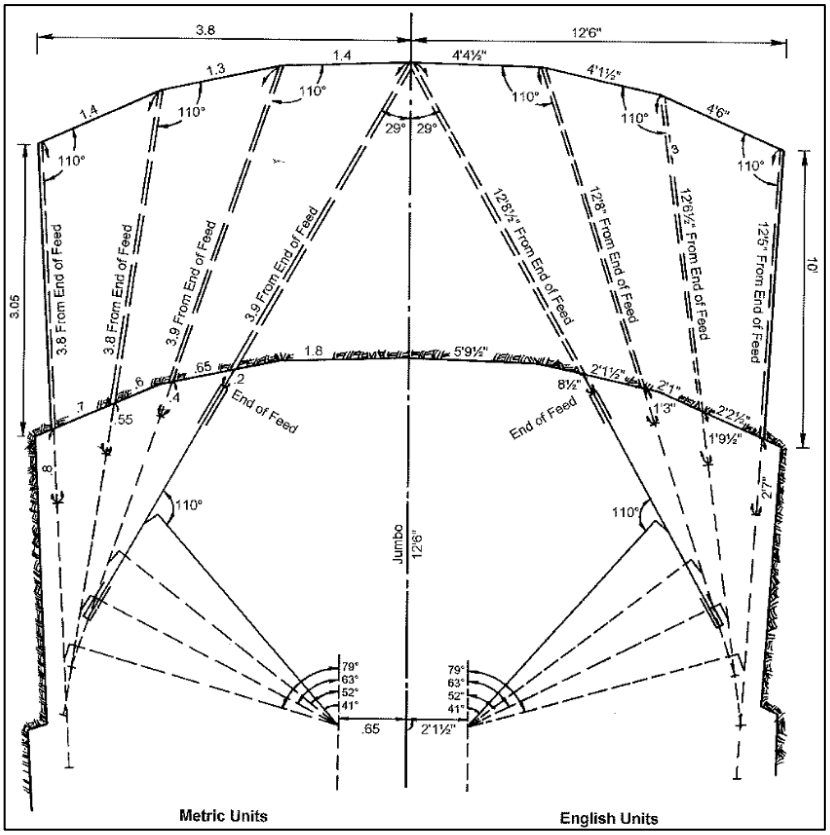


Figure 15-8: Typical v-cut design (Bullock 1998)

Due to the soft nature of the rock and the tight drilling required in the confined development, GoviEx and SRK has assumed that 25% of the drillholes are only half loaded (the perimeter holes) to prevent excessive fines and overbreak.

Alternative - Resuing

Resuing is a two stage blasting technique that allows some of the waste to be separated from the mineralised material as part of the mining process. It involves undercutting a face in one blast and returning to strip the roof of the tunnel for rockbolting. Typically, each blast would be drilled individually, however, this depends on the ground conditions. The advantages of resuing are:

- Higher head grades to the processing plant;
- Reduced RoM requiring processing; and
- Allows large mechanised equipment to be used in a narrow vein/seam.

Disadvantages include:

- Multi-pass system increasing the number of steps for each cycle;
- More material to be moved per tonne of RoM;
- Reduced productivity and advance rates for production headings;
- Increase in mining losses as visibility for loading is reduced and there is a risk mineralised material remains in the unfired portion of the face;
- Waste and ore must be campaign loaded onto the conveyor on a regular basis to prevent mixing of RoM and waste;
- Increased cost per tonne of RoM; and
- Potential for rock falls in temporary roof prior to second blast and supporting.

- Increased total tonnes mined as 2.5 m face height required to achieve the two cuts (given minimum blast height of 1.5 m required for ore blast to accommodate mining equipment) whilst conventional mining is based on 1.8 m face height.

A resue mining method is not being adopted at Madaouela and the full mining face will be loaded to surface and processed through the plant. Waste will be separated at the plant using radiometric sorting and the ablation process as it is a more efficient and cost-effective method of separating the waste dilution from the ore.

15.4.4 Vertical Development

Introduction

As mine access is proposed via decline for each operation, the need for vertical development is limited to providing ventilation (Section 15.6). The proximity of the mine workings to the surface and the generation of radon daughters that require primary ventilation of working areas makes the use of many small diameter raises regularly distributed throughout the orebody a practicable alternative to fewer, more isolated, large diameter raises. The requirements for vertical development are therefore anticipated to be relatively high.

Raiseboring is assumed for vertical development in the Madaouela deposits using owner-operated equipment. The speed of advance is expected to be high due to the soft nature of the rock and the continuous nature of the method.

In addition, there is anticipated to be a reduction in geotechnical risks as no blasting is undertaken to damage the excavation walls and the circular shape maximises natural support.

15.4.5 Materials Handling

An options analysis for the total cost of materials handling from mining districts demonstrated that those districts from where large tonnages, greater than 400,000 t are being mined, conveyor haulage has the lowest cost overall cost (combining operating and capital costs). At tonnages lower than 400,000 t, the preferred method is dependent on the haulage distance and the ability to recycle the equipment.

The use of conveyor haulage rather than diesel trucks greatly benefits the ventilation requirements for the mine as the needs to dilute and remove diesel fumes and the heat produced by engines is removed. Conveyor belts can produce significant heat loads which, in mines with a high thermal gradients and high ambient air temperature, can become an issue however this is not considered to be a problem at Madaouela. .

The mine layout has been laid out using a conveyor haulage strategy and consists of a central conveyor network carrying large tonnages to the surface fed by trucks hauling from the panels (Figure 15-9 and Figure 15-10).

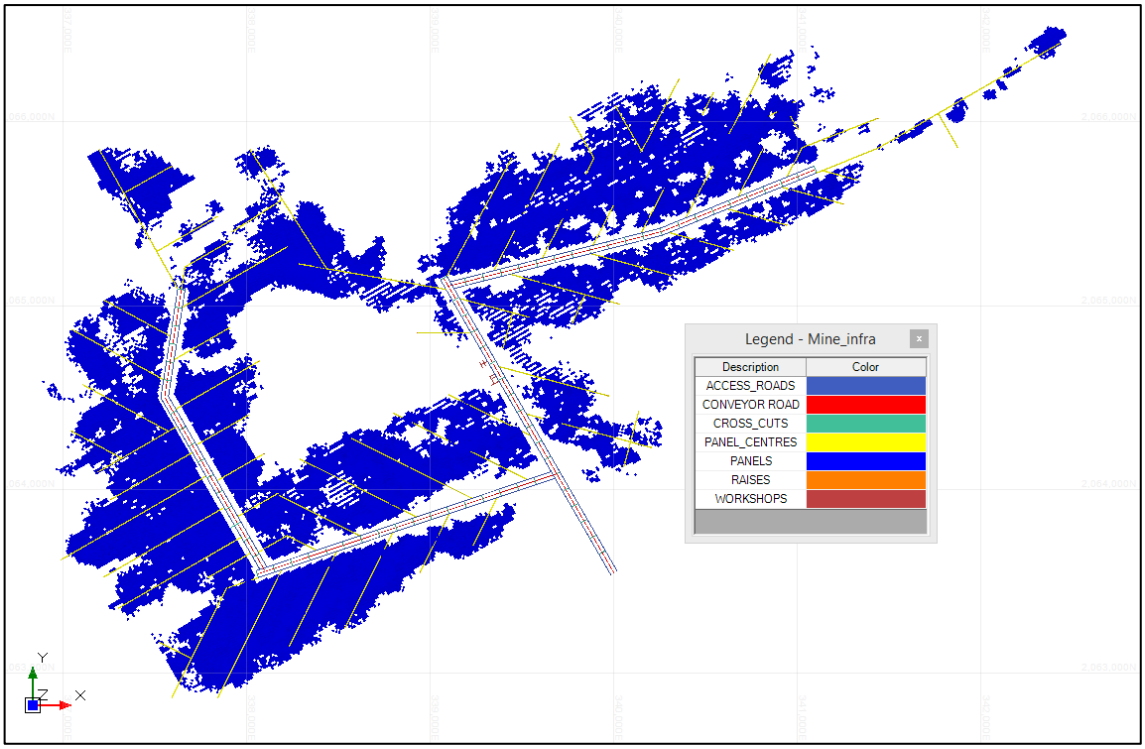


Figure 15-9: Mine Materials Handling Infrastructure - overview

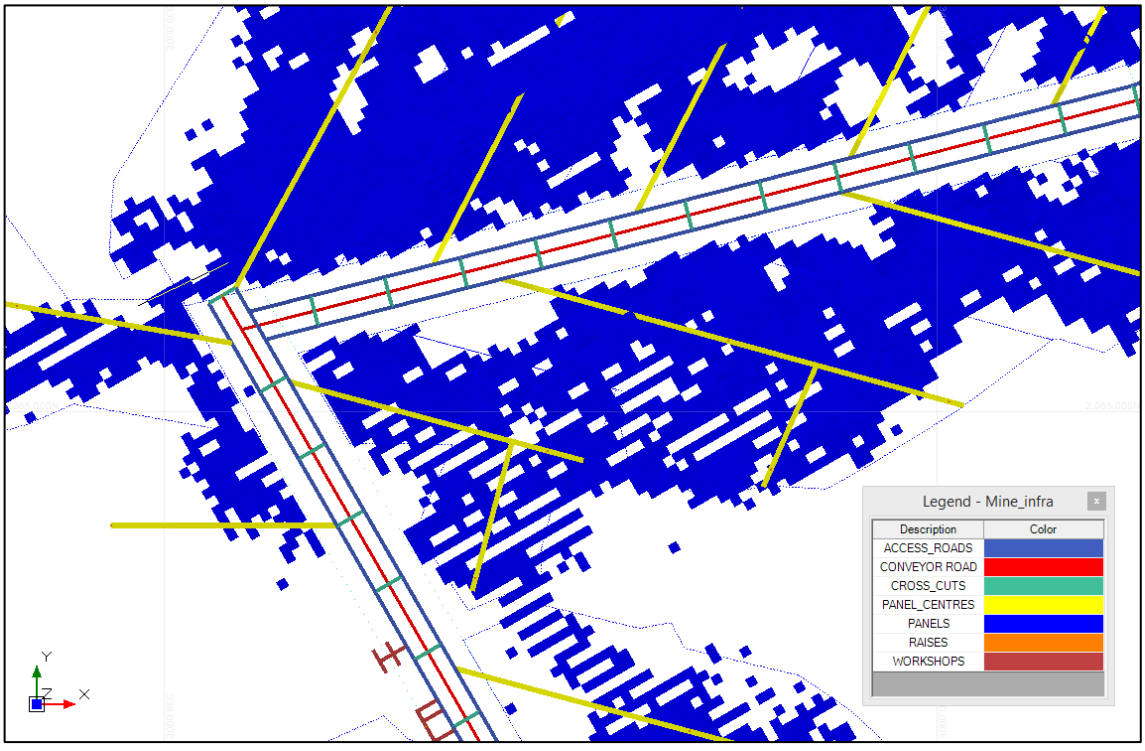


Figure 15-10: Mine Materials Handling infrastructure - detailed

This approach greatly reduces the number of trucks required on the mine and improves the ventilation efficiency of the operation. The use of trucks to haul from the panels to the central conveyors is a very flexible approach which allows ore production to transition between panels without the need to establish conveyor infrastructure. As high ventilation volumes are required to dilute radon emissions, this will also provide sufficient ventilation for diesel

equipment operating in the panel. Where mining districts have relatively small tonnages, no conveyor is installed and a single end access into the panel is developed.

Conveyor specifications are outlined in Table 15-3 and Figure 15-11.

Table 15-3: Conveyor belt specifications

	Unit	
Effective operating rate	%	58.5
Design Capacity	tph	380
Designed Belt Speed	m/s	1.0
Max feed size	mm	F100% - 250

It is anticipated that conveyors will be floor mounted (Figure 15-11)

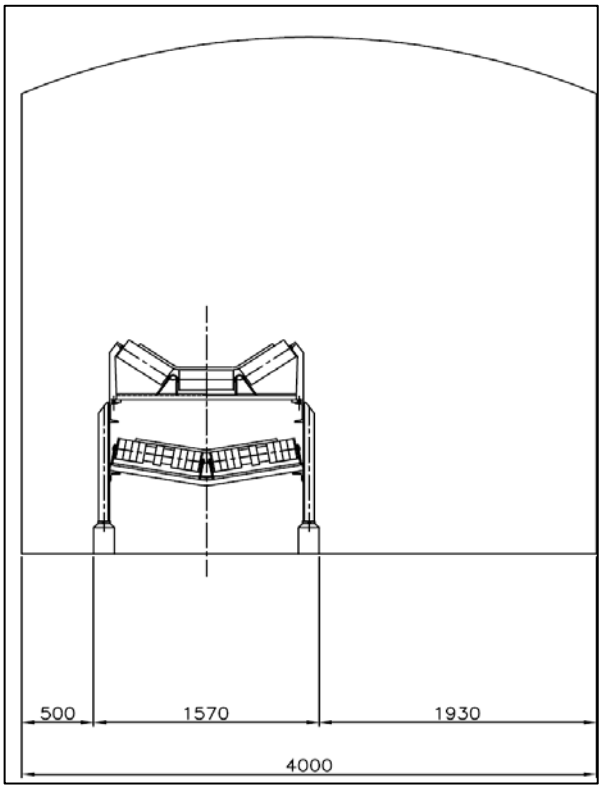


Figure 15-11: Conveyor dimensions

Loading

A balance has to be achieved between maximising the size of the loader for production efficiency and minimising operating height to account for the thin mineralisation and minimise the amount of waste that will be mined to accommodate equipment. Low-profile equipment is required to minimise the waste rock excavated from underground, however, their smaller bucket sizes result in longer truck loading times which reduces the productivity of the trucking fleet.

To mitigate this issue, faces are to be cleaned to stockpiles in central locations in each panel, from where ore and waste are to be rehandled by a larger loader into the trucks. The room and pillar method will create many stockpiling locations close to the load points and the

trucking fleet is decoupled from the production fleet increasing their productivity and the flexibility of scheduling. The trucking fleet will also transfer between production panels and development/waste mining, without affecting the productivity of mining from the face (this assumes the trucking fleet productivity is larger than the low-profile loading).

Trucks will tip onto feeder breakers which will load the conveyors. Feeder breakers are mobile to allow transfer points to be moved quickly and simply as production areas advance and conveyors are extended, to minimise trucking distances. They can be used to load conveyors at any point along its length. Discussions with Joy Mining have confirmed that the properties of the Madaouela rock are able to be handled using a conventional feeder breaker. As the projected passing size from the feeder breaker is in the order of 150 mm, no further crushing is required prior to transporting to the surface.

Waste Handling

The continued generation of radon and its daughters from the rock presents a hazard to mine workers that is mitigated by well controlled ventilation flows and by barricading mined out panels to prevent any leakage into the fresh air network. SRK has therefore assumed there to be limited underground storage of waste and the conveyor system design is capable of handling all ore and waste production to surface.

There is potential to store waste in mined out areas of active panels, however, this is expected to be an ad-hoc activity by LHD only, as trucks will only be able to travel in the central panel access.

Ore and waste will need conveying to the surface separately to prevent waste being fed into the plant. For this reason, it is important that the trucking fleets are independent from the production fleet, as previously discussed.

At the surface, the conveyor sends RoM to a stockpile, and waste is offloaded to a temporary stockpile near the portal. A surface trucking fleet will rehandle the waste to nearby waste dumps and manage the RoM stockpile as required.

Benchmarking

Akouta Mine primarily uses conveyors for underground transport. Ore is delivered to the conveyor from the production face using a loader. Crusher stations are used to transfer the material onto the crusher and ensure no oversize. Trucks are used for development where conveyor systems are not yet extended.

The primary belt is 1 m in width, 1,450 m long with a belt speed of 2 m/s. Secondary conveyors are used to connect the panels and generally are limited to 50 m in length. Conveyor feed is via a rolls crusher to remove oversize; the main conveyor feeds a central jaw crusher before ore is conveyed to the surface.

Trends in underground material handling have been identified by Pratt (2008) which benchmarks numerous operations around the world (Figure 15-12).

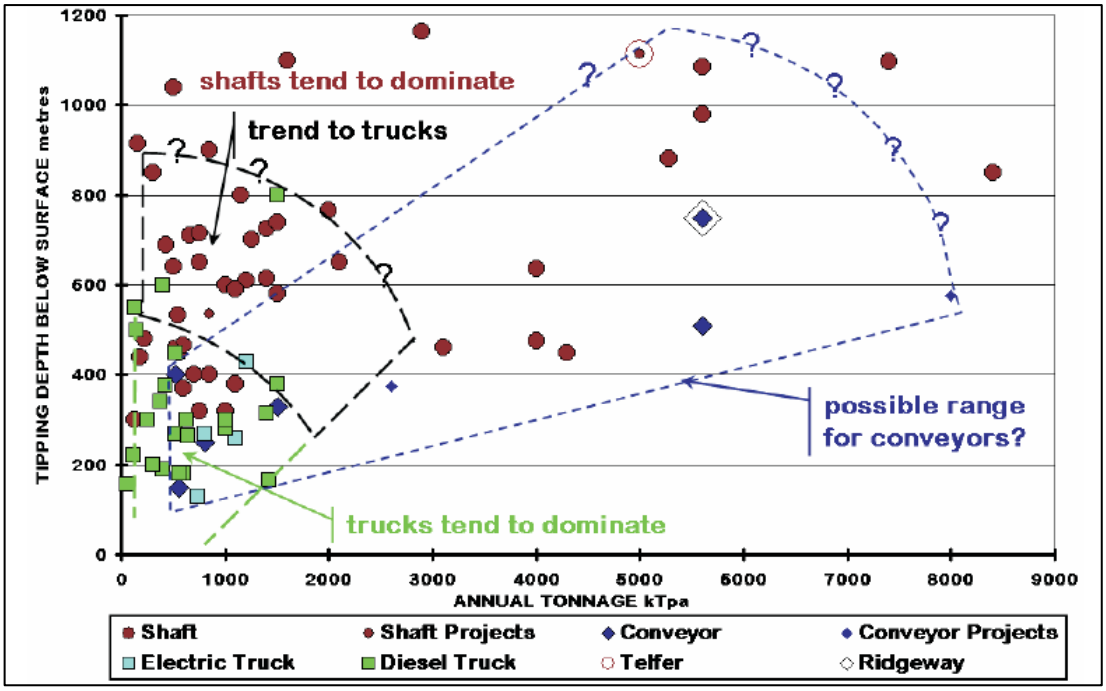


Figure 15-12: International trends for materials handling from underground to surface (Pratt 2008)

15.4.6 Ground Control

Introduction

Failure within a rock mass is generally caused by one of three mechanisms:

- Fallout of blocks from existing (or blast induced) fractures or planes of weakness driven by gravity;
- Shear failure caused by overloading of a rock mass from an existing stress field (including arching); and
- Failure from concentrations of high stress (from spalling to rockbursts).

Ground control measures aim to prevent unplanned failures. The issues to be considered when selecting a method of ground control fall broadly into three categories:

- Geotechnical considerations;
  - Rock strength;
  - Joint spacing;
  - Joint direction;
  - Joint roughness;
  - Bedding orientation; and
  - Rock density.
- Economic considerations;
  - Unit cost for support method;
  - Spacing/regularity of application;
  - Capital and operating cost of any equipment requirements; and
  - Labour intensiveness of the method.

- Practical considerations;
  - Excavation span;
  - Excavation height;
  - Hole diameter;
  - Impact on mining rates; and
  - Time required for excavation to be open/safely accessible.

Adequate ground control is essential to the safety of the work environment, and as room and pillar is a man entry method, all workings will be supported. The primary support for a room and pillar mine is provided by the pillars. A detailed review of the geotechnical designs of the underground mines can be found in Section 26

Whilst there are numerous forms of ground control available, SRK has considered the following:

- Scaling;
- Rockbolting;
- Shotcreting; and
- Backfilling.

Benchmarking

As with the Madaouela deposits, the Akouta Mine is composed of flat-lying, bedded sandstones with some parting between layers. COMINAK installs 1.8 m tensioned anchor bolts on a regular one metre square pattern. The bolts require re-tensioning after the face has advanced some 20 metres due stress relaxation of the ground and loosening caused by blast vibrations.

Swellex bolts had previously been trialled, but were found to fracture the rock surrounding the holes and thereby deteriorating the ground conditions. There are also areas of the mine where interstitial clay is present between the sandstone layers. 1.8 m resin bolts and grouting of the anchor bolts were found to be most successful in these zones.

Caving of the roof at the Akouta Mine is not allowed due to the potential disturbance of overlying aquifers. In areas of high grade, backfill is used to recover the high grade pillars. The backfill comprises 75% crushed sandstone, 3 to 6% cement, sand and water. The fill is mixed on the surface and transported underground via a purpose-built raise. Loaders transport and place the fill at the required locations, pushing tools are used to get the backfill close to the roof and to maximise the rill angle of the fill.

Trials were undertaken on pumping the fill from the surface, however, the results suggested this method which requires installation pipes to the fill site was labour intensive and took too long to establish.

Applied Design

Scaling

Scaling of roof and sidewall is required for all work places following blasting and subsequent check scaling to test the integrity of the ground over time. Scaling is assumed to be undertaken using manual methods due to the low-profile production areas limiting the application of mechanical scaling. The sub-horizontal nature of the sedimentary bedding should result in scaling being simple and effective.

Rock Bolting

A bolting pattern has been applied to underground mining using 1.8 m resin bolts with a bolt spacing of 1 m and 1.5 m between rings. For production areas, the resin bolt lengths are reduced to 1.5 m to account for the limited roof height. No surface support (mesh or shotcrete) is applied as the sub-horizontal bedding planes provide favourable roof conditions.

Resin bolts have been selected due their suitability for weaker rock conditions. Their ability to hold higher stresses means that bolting density can be decreased which partially offsets their higher unit costs. The bolts will be installed with face plates to provide localised support. No cablebolting is assumed for intersections.

Shotcreting

The low mining height of the production areas means that a conventional shotcreter would not be able to access the production areas of the mine. However, a shotcreter and low-profile transmixer have been allowed for.

It is expected that roof conditions around the mine will be good due to the sub-horizontal parting of the bedding planes, and that there will be little need for shotcreting of the roof.

The availability of a shotcreter is expected to add value to the project in the following areas:

- Areas of poor ground conditions such as near the regional faults;
- Areas of high radon emission in the central access drives and panel accesses;
- Construction of ventilation barricades to prevent leakage from previously mined panels; and
- Provide protection for pillars (adjacent to panel drives) where truck traffic erodes the shale floor, exposing or undercutting shale in the pillars.

Protection against roof failure

SRK considers it important to protect against roof failure and possibly related subsidence issues at the Madaouela deposits for a variety of reasons, including:

- Impact on local aquifers: The Guezouman Sandstone hosts a number of water-bearing units. There is a risk that a caved zone would extend through these units which would significantly increase water inflows into the mine and other users of the local aquifers. Of particular concern is the Tarat Aquifer, which is a main source of water supply in the area.
- Ventilation: As the roof fails, radon/radiation would emit from low grade material within the overlying strata and unrecovered mineralisation. Radon gas would collect in roof voids where the ventilation will be poor and ultimately ‘leak’ into active workings which would be difficult to control. There is also a risk that the failures will open fractures in the regional pillars providing a pathway for radon gas to leak into neighbouring panels.
- Poor understanding of the failure mechanism of local rock mass: No assessment of how the rock mass would fail has been undertaken as part of this study to consider the hydraulic radius (area divided by perimeter) at which failure is induced and the stress concentrations that would arise in the surrounding pillar zones. This also presents a risk that pillars would penetrate into the softer shale floor.

Pillar Recovery

No pillar recovery has been included in the assessment. The low average mining height will make the distribution of backfill difficult. The study does not include a cost estimation or cut-off

grade estimate for pillar recovery using backfill or shotcrete pillars which provides upside potential for the higher grade sections of the deposit.

15.4.7 Equipment

As the planned approach to mining is consistent for the proposed underground operations, equipment is common to all operations. The concept is for the fleet of equipment to move from one operation to the next to maintain a constant production rate in the ramp up and down of the operations and to limit purchase of additional equipment during the life of the operations.

The mobile fleet requirements are split into four categories:

- Development fleet;
- Production fleet;
- Materials handling; and
- Auxiliary fleet.

The productivities for all development and production equipment has been built up from first principles applying both the experience of GoviEx and SRK, and information provided from the equipment suppliers.

Development Fleet

The central access, infrastructure and panel access development are treated separately from the production activities due to the higher mining heights used. This allows a conventional fleet of development equipment to be used. It is estimated that the maximum annual development requirements are in the order of 10,200 m per year (though a typical year would be closer to 7,500 m per year), which requires one development suite for the equipment fleet. A summary of the required equipment for development mining is summarised in Table 15-4.

Table 15-4: Overview of development mining fleet

Equipment	Make	Model	Units	Unit Cost[1] (USDk)	Productivity (m/month)
Jumbo	Atlas Copco	Boomer 282	1	994	1,000
Rockbolter	Atlas Copco	Boltec 235	1	837	1,000
ANFO Loader	Normet	Charmec 605 DA	1	418	1,000

[1]Includes additional 10% for transport to site

Production Fleet

The production fleet is constrained by the working height of the proposed excavations. As the majority of the mineralisation has a thickness less than the minimum mining height, production mining will be largely by low-profile equipment. In most cases, the minimum mining height will be determined by the equipment selection. Low profile mining equipment used extensively in South African platinum and chrome operations is specifically designed to reduce the total mining height and reduce waste dilution in low seam environments.

The annual production rate sent to the processing plant is 1,447,000 tpa, which equates to 4,020 tpd, for conventional mining. A summary of the estimated production fleet requirements is shown in Table 15-5.

Table 15-5: Overview of production mining fleet

Equipment	Make	Model	Units	Unit Cost[1] (USDk)	Productivity (t/month)
Jumbo	Atlas Copco	Boomer S1 L	4	994	37,500
Low-Profile Loader	Atlas Copco	Scooptram ST7LP	5	724	32,000
Rockbolter	Atlas Copco	Boltec 235	4	683	matched to jumbo
ANFO Loader	Normet	Charmec 605 DA	3	418	-

[1]Includes additional 10% for transport to site

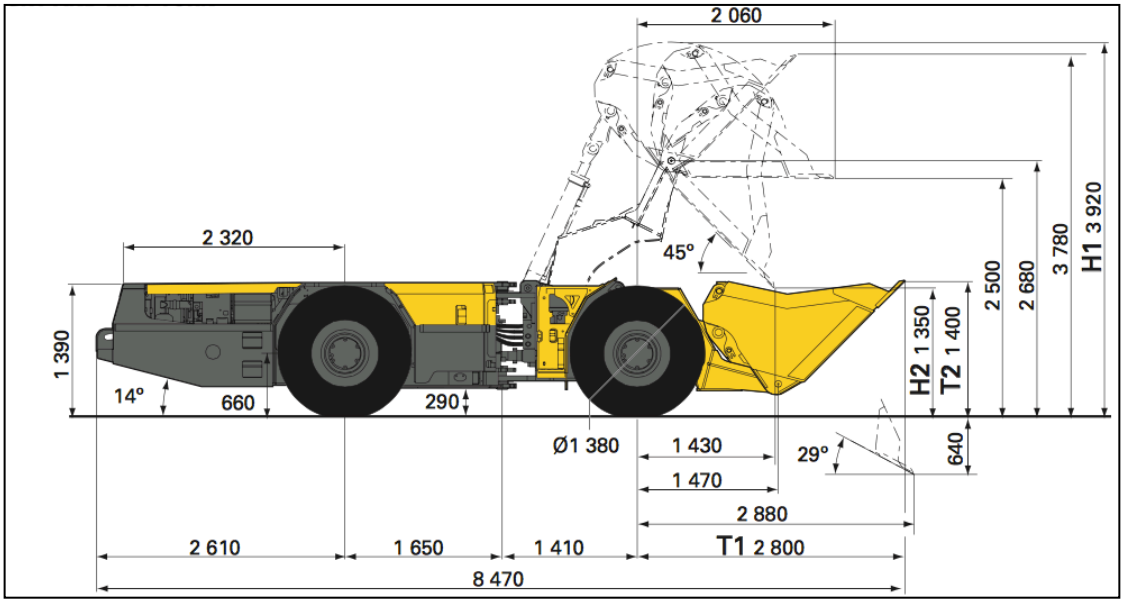


Figure 15-13: Scooptram ST7LP Low Profile Loader

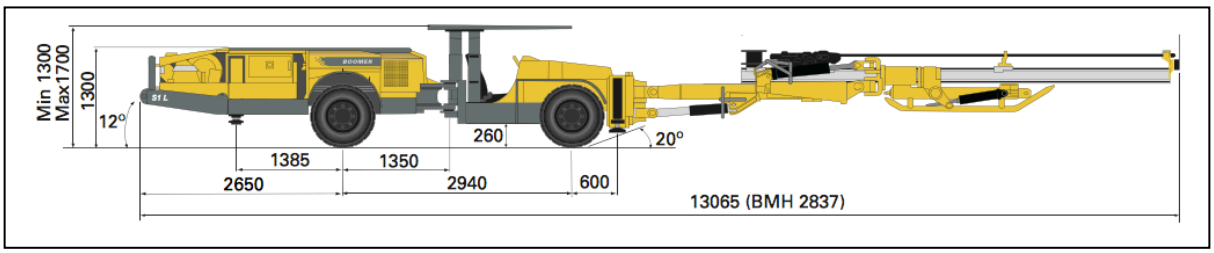
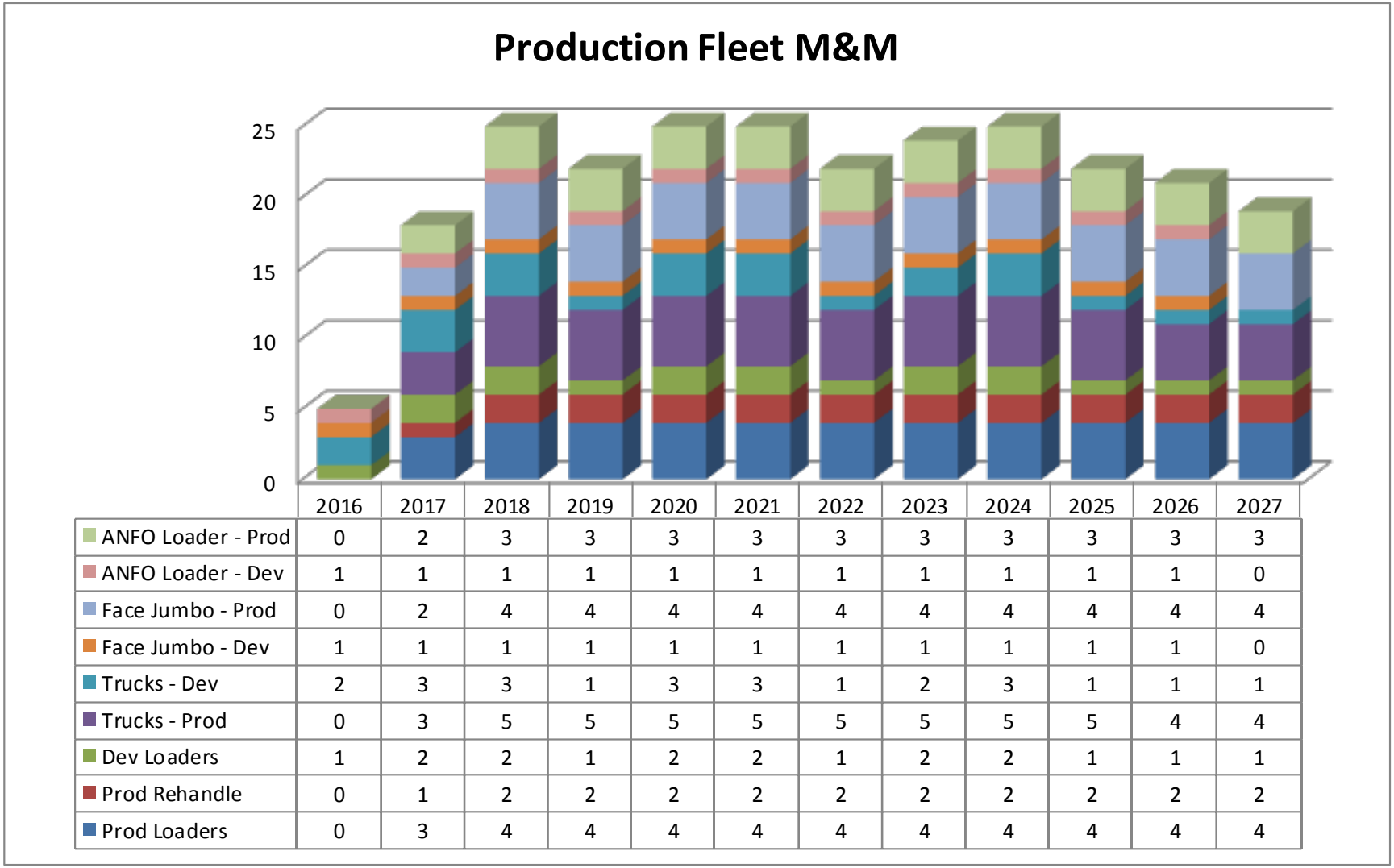


Figure 15-14: Low Profile Drill Jumbo Boomer S1 L



	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027
ANFO Loader - Prod	0	2	3	3	3	3	3	3	3	3	3	3
ANFO Loader - Dev	1	1	1	1	1	1	1	1	1	1	1	0
Face Jumbo - Prod	0	2	4	4	4	4	4	4	4	4	4	4
Face Jumbo - Dev	1	1	1	1	1	1	1	1	1	1	1	0
Trucks - Dev	2	3	3	1	3	3	1	2	3	1	1	1
Trucks - Prod	0	3	5	5	5	5	5	5	5	5	4	4
Dev Loaders	1	2	2	1	2	2	1	2	2	1	1	1
Prod Rehandle	0	1	2	2	2	2	2	2	2	2	2	2
Prod Loaders	0	3	4	4	4	4	4	4	4	4	4	4

Figure: 15-15: Production Fleet M&M

Materials Handling Fleet and Infrastructure

The materials handling fleet comprises a rehandle loader and truck fleet and conveyors to transport RoM from the production face to the processing plant, and the waste from the development face to the surface stockpile. As discussed in Section 15.4.5, RoM is to be rehandled in the panel to decouple face loading with low-profile equipment from the trucking activities. Material movement to surface is:

- Conventional mining;
  - 4,020 tpd of RoM;
  - a maximum of 1,825 tpd of waste;

A loader and truck fleet is to be used on surface to haul waste to the waste dump.

A study to assess the conveyor requirements for the Marianne-Marilyn and MSNE-Maryvonne deposits has been undertaken by Sandvik Mining. A summary of the mobile fleet and conveyors are provided in Table 15-6 and Table 15-7, respectively.

Table 15-6: Overview of materials handling mobile fleet

Equipment	Make	Model	Units	Unit Cost[1] (USDk)	Productivity (t/month)
Rehandle Loader	Atlas Copco	Scooptram ST1030	2	717	32,500
Development Loader	Atlas Copco	Scooptram ST1030	2	717	44,000
Truck	Atlas Copco	Minetruck MT436LP	8	1,000	30,500
Feeder Breaker	Joy	UFB-14B	4	880	32,500
Surface Loader		6.9 m^3 Loader	2	873	70,000
Surface Truck		24.5 m^3 Truck	3	776	45,000

[1]Includes additional 10% for transport to site

Table 15-7: Overview of conveyor network

Branch	Make	Belt Length (m)	Lift (m)	Unit cost[2] (USDM)	Installed Power (kW)
Marianne-Marilyn					
Decline	Sandvik	820	115	4.62	195
Central	Sandvik	1,250	55	6.70	147
West 1	Sandvik	825	35	4.65	110
West 1	Sandvik	885	20	5.00	96
Northwest 1	Sandvik	920	40	5.90	119
NorthWest 2	Sandvik	585	40	0.77	64
East 1	Sandvik	1,200	20	6.43	108
East 2	Sandvik	900	10	5.07	86
MSNE-Maryvonne[1]					
Decline	Sandvik	740	105	1.44	182
Central	Sandvik	990	20	1.92	100
West 1	Sandvik	1,000	25	1.90	106
West 2	Sandvik	1,090	10	2.05	93

Branch	Make	Belt Length (m)	Lift (m)	Unit cost[2] (USDM)	Installed Power (kW)
South	Sandvik	500	10	1.00	71
Overland	Sandvik	3,250	0	13.07	231
North	Sandvik	1,230	35	6.70	147
Maryvonne	Sandvik	900	15	1.79	100

[1]Assumed that Marianne-Marilyn will be moved to MSNE-Maryvonne

[2] Including installation and electrical

Though the mobile fleet is anticipated to remain relatively stable over the life of mine, the difference in the layout between the Marianne-Marilyn deposit and the MSNE-Maryvonne deposit means that haulage requirements required for production will change over time. This is reflected in the equipment schedule over the life of the mine.

The conveyor belt design (Sandvik) is based on a 1 m wide belt travelling at a speed of 1 m/s with the exception of the overland conveyor that has a conveyor speed of 2.5 m/s. The system is rated to deliver 380 t/h. A schematic of the two conveyor networks are shown in Figure 15-16 and Figure 15-17.

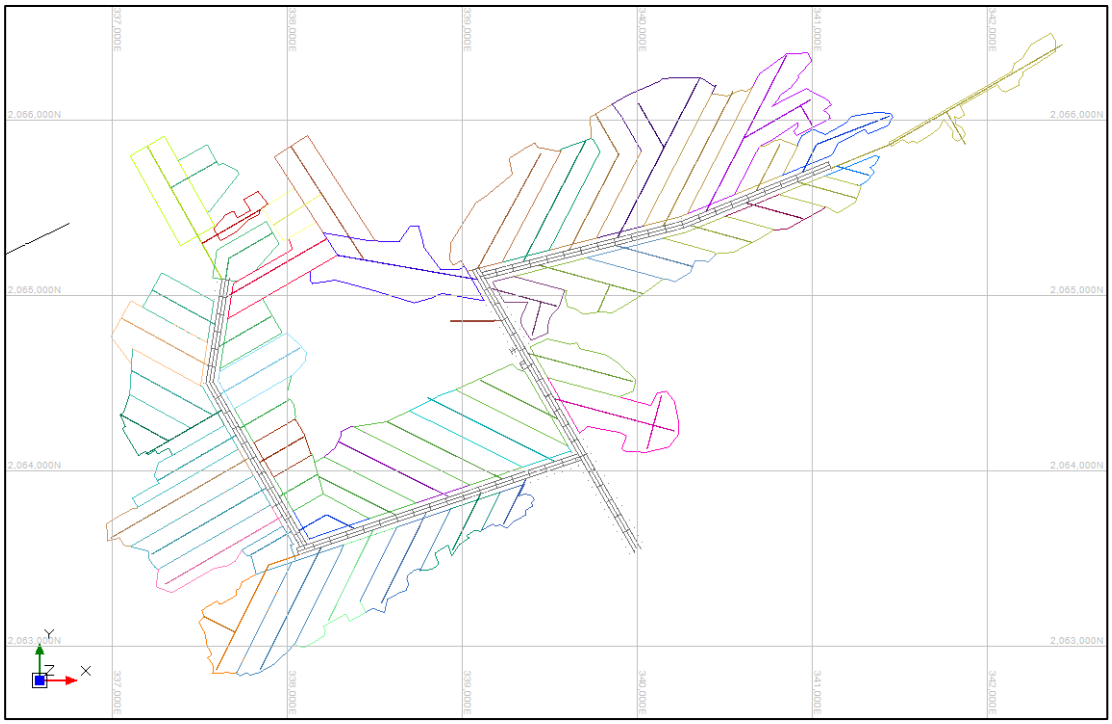


Figure 15-16: Schematic of Marianne-Marilyn conveyor network

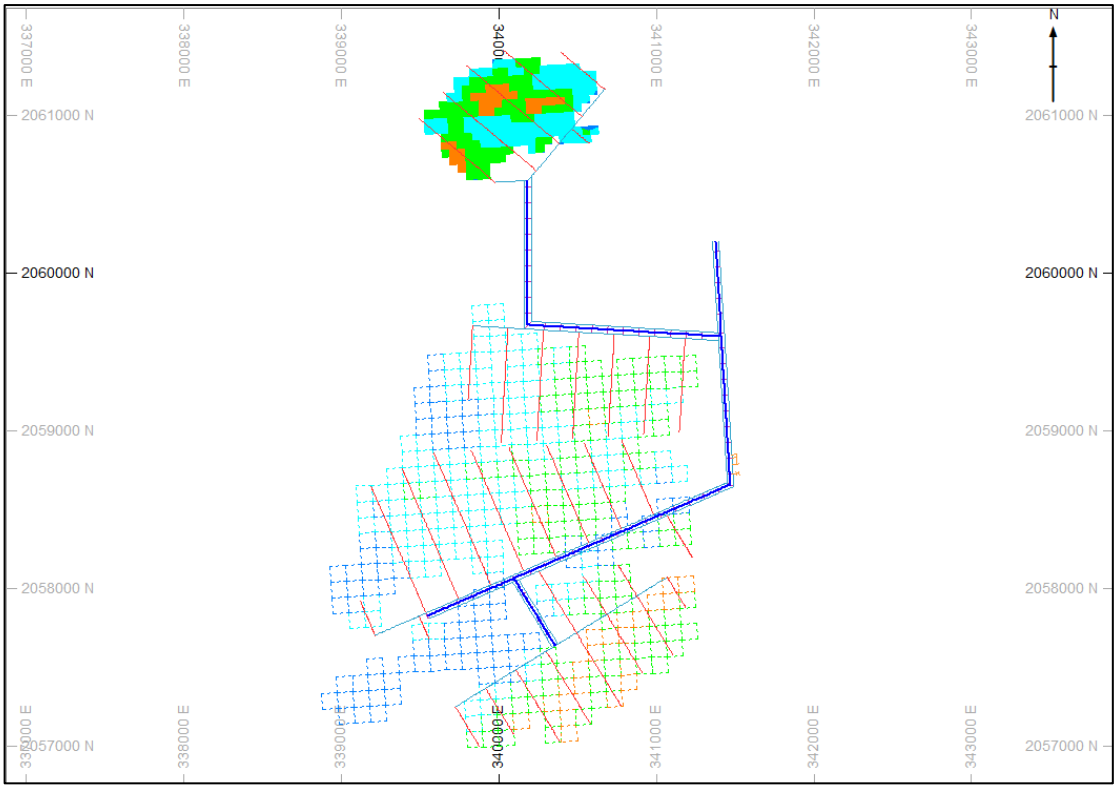


Figure 15-17: Schematic of MSNE-Maryvonne conveyor network

Auxiliary Fleet

To facilitate the development and production activities, numerous auxiliary processes need undertaking, including (but not limited to) the installation of air and water services, communications, ventilation controls, raiseboring and materials supply the underground stores. A fleet of auxiliary equipment necessary to undertake these services is provided for (Table 15-8).

Table 15-8: Overview of auxiliary equipment fleet

Equipment	Make	Model	Units	Unit Cost[1] (USDk)	Productivity (m/month)
Shotcreter	Normet	Spraymec 6050 WP	1	495	-
Transmixer	Normet	Ultimec LF500	1	297	-
Toolcarrier/MPV		Loader with Platform	2	160	-
Grader	Caterpillar	120H	1	269	-
Light Vehicle	Toyota	Landcruiser 70	15	69	-
Lubricant Truck	Toyota	Landcruiser 70	1	69	-
Store Truck		Flat Bed Truck	1	281	-
Raiseborer	Robbins	91RH C	1	6,710	150

[1]Includes additional 10% for transport to site

Two tool carriers are provided for, one for the service crew and another for the ventilation crew. As much of the production areas will be 1.8 m high it is assumed that this work in the production areas will be undertaken from ground level or using work platforms on light vehicles.

15.4.8 Services

Air

Based on the equipment discussed in Section 15.4.7, the maximum air consumption from the mining fleet is estimated to be 223 L/s (Table 15-9, note that this assumes all equipment operates simultaneously, actual demand is likely to be much lower). 10 % is allowed for miscellaneous consumption and 15 % for pipe leakage.

Table 15-9: Estimated peak compressed air requirements

	Units	Consumption (L/s)	Total (L/s)
Production Jumbo	4	11.7	46.8
Development Jumbo	1	12.5	12.5
Production Rockbolter	4	11.7	46.8
Development Rockbolter	1	12.5	12.5
Production ANFO Loader	3	10.0	30.0
Development ANFO Loader	1	10.0	10.0
Shotcreting	1	20	20.0
Miscellaneous Consumption	10%		17.9
Leakage	15%		26.8
Total			**223.3**

Reticulation of compressed air is assumed to be via 63 mm high-density polyethylene (“HDPE”) piping and installed as part of the development cycle. 100 m length rolls of HDPE are assumed to reduce the time required to hang the pipe, reduce labour requirements and minimise the number of pipe connectors required.

Service air is assumed to be supplied via two 1,200 L/s air compressors located at the surface. Whilst this is substantially higher than the estimated requirements, two compressors are required to ensure 100% availability at all times (i.e. during maintenance periods) and no detailed calculations to account for leakage in pipes across the entire network has been undertaken.

Water

Potable Water

SRK has assumed that there is no requirement for potable water underground. Drinking water is assumed to be stored in a small tank underground, to be kept full by the stores workers, and worker to be supplied with drinking vessels.

Service Water

Based on the equipment discussed in Section 15.4.7, the maximum service water consumption from the mining fleet is estimated as 24 L/s (Table 15-10). 10% is allowed for miscellaneous consumption and 5 L/s for pipe leakage.

Table 15-10: Estimated peak service water requirements

	Units	Consumption (L/s)	Total (L/s)
Production Jumbo	4	1.1	4.4
Development Jumbo	1	1.7	1.7
Production Rockbolter	4	1.2	4.8
Development Rockbolter	1	1.3	1.3
Production Loader	5	0.5	2.5
Development Loader	4	0.5	2.0
Pipe Leakage	1	5.0	5.0
Miscellaneous Consumption	10%		2.2
Total			**23.9**

As with the compressed air, service water is to be distributed throughout the mine using rolls of HDPE piping, though with a larger diameter of 100 mm.

Dewatering

Control of nuisance water will be an important function to maintain safe working conditions in the proposed mine. The Talak shale footwall will readily deteriorate and could cause dangerous conditions in which to operate mobile equipment.

Dewatering in a room and pillar mine with an undulating floor can lead to many areas where water collects which each require small scale pumping. These can be simply dealt with using low-cost, small pumps driven by compressed air to move the water to the regional sumps.

The primary sumps is where the majority of the settling will take place before being transferred into a central dewatering network that leads the underground pumping station. Pumping from these sumps is assumed to be undertaken by 9 kW electrical pumps with switches to turn off when the sumps levels are low.

Pumping Stations are assumed to be centrally located and contain six 30 kW Mono pumps to rehandle the water to the surface. The Pump station would require significant water storage, which could be in the form of a traditional sump or large tanks to avoid the need to develop through to the Talak Shale. There is a risk that the dewatering system could result in concentrations of radiation derived from the uranium content within the sediments and potential for the groundwater itself to emit radiation.

Whilst no design has been undertaken for the pumping station and associated storage requirements, an allowance for the costs has been made. One challenge will be the need to have sumps in the Talak clay, which may be difficult to develop and manage.

There is potential to use fabricated sumps to settle material that when full are collected by truck, and taken to surface for cleaning.

Estimated dewatering requirements are provided in Table 15-11.

Table 15-11: Estimated dewatering requirements

	Total (L/s)
Water Production	
Equipment	23.9
Groundwater	55.0
Diesel Exhausts	1.0
Contingency (10%)	8.0
Total	**87.9**

Power

Main power is to be distributed at 10 kV to mine transformers located in mining districts to step down to 0.5 kV for distribution to mining equipment.

Estimation of the power requirements for underground mining are provided in Table 15-12. Note that these figures assume all equipment is operating simultaneously and that the typical instantaneous load will be less than this.

Table 15-12: Estimated peak power requirements

	Units	Power (kW)	Total (kW)
Production Jumbo	4	59	236
Development Jumbo	1	125	125
Production Rockbolter	4	63	252
Development Rockbolter	1	66	66
Feeder Breaker	4	84	336
Surface Fan	10	112	1120
Auxiliary Fans	17	30	510
Pump Station Pumps	6	30	180
Sump Pumps	36	9	324
Miscellaneous Consumption	10%		315
Total			**3,464**

In addition, significant power will be required for the conveying network. Table 15-7 provides an overview of the power required for these elements.

Future studies should include a detailed and cost estimation for the underground power reticulation network.

Communications

The mine communications system proposed for the Madaouela operations comprises a leaky feeder network throughout the mine with mobile radios located in each vehicle and a limited number of hand held radios. Communication will also be provided at designated fresh air bases (“FAB”) with an independent connection to the surface to provide backup communications in the event of an emergency.

The leaky feeder cable is to be installed and extended as the mine develops.

Road Works

To overcome poor road conditions expected in the Talak Shale, the central access development is to be raised above the sandstone-shale contact providing a better floor upon which to build a road.

Road building will use crushed road base spread and levelled with a grader. In panel development, located on the Talak-Guezouman contact, a road construction material is essential to provide a good floor for trucking to take place. Waste rock can be taken from development blasting and crushed waste rock from the surface can be transported underground through ventilation raisebores that are no longer used.

Within the production zones the limited working height will mean production loaders will be responsible for maintaining roads in these areas. The limited amount of traffic and relatively short time that production development areas will remain active limits the amount of road maintenance required in these areas. However, maintaining a dry floor along transportation routes. Should the roads in the production zone start to wear, they risk exposing the Talak shale at the base of the mining pillars, which will have a detrimental impact on the geotechnical integrity of the pillars.

Grade Control

The Akouta Mine uses radiation detector hanging from the roof of the development to estimate the grade of each loader bucket before it is loaded into the underground crusher. In this way, any excessive waste material could be identified prior to being sent to the plant. This approach limits the need to sample and assay production faces. The use of radiation detectors is designed to inform the processing plant of the material that is being dispatched to the plant to allow it to divert it to the appropriate stockpile.

It is anticipated that this approach will not be as effective at Madaouela as there will be up to 40% waste per truck load which will impact the ability of the radiation detector to accurately determined the contained mineral. This could potentially result in one of two negative impacts;

- Incorrect grade assignation of mined mineral resulting in mineral discharge as grade material is sent to the waste stockpile.
- Underestimation of face grade resulting in the premature stopping of mining headings and a loss of resource.

Radiation detectors will not be used at Madaouela. The Madaouela process strategy is based on a single feed and waste material being separated during the radiometric sorting and ablation processes. It is therefore less important to accurately determine the mineral content of the trucks and more important that the face grade is accurately determined to allow for accurate dispatch to the waste or ore stockpiles and to ensure that faces are not stopped prematurely. It is therefore envisaged that the following methodologies be adopted as part of the grade control strategy:

- Probing of drillholes prior to loading; and
- Face assay.

15.4.9 Labour

Labour Sources

The Arlit region has an extensive history of mining and provides a workforce familiar with the techniques and skills required for the proposed underground operations. In addition, the limited remaining mine life at the Akouta Mine may result in trained workers becoming available for the proposed operations prior to the commencement of any operations.

Niger has its own mining school and the experience of the Akouta Mine is that it provides competent professionals ready for the industry. Consequently, only a small expatriate workforce is maintained at the two Areva subsidiaries and mainly in management roles.

GoviEx and SRK have assumed that the Madaouela project can be staffed entirely with Nigerien labour for both technical and labour functions. Any requirement for expatriate assistance is assumed to be covered by the general and administrative costs.

Shift Structure

The project schedule is based on 4 work crews working three 8 hr shifts per day.

This shift structure complies with Nigerien legislation and removes any requirement for overtime payments.

Office staff is assumed to work a standard 40 hour work week.

Office and Administrative Staff

All labour requirements for the administration of the underground operations, including the Mine Manager are provided for in the general and administrative costs so are not included in the evaluation.

Technical Staff

Technical staff requirements are derived from SRK's experience in similar operations and discussions with the Client (Table 15-13).

Table 15-13: Technical staff requirements

Position	Shift Structure	Workers per Shift	Total Workers
Senior Mining Engineer	Office Hours	1	1
Mining Engineer	Office Hours	2	2
Geotechnical Engineer	Office Hours	1	1
Senior Mechanical Engineer	Office Hours	1	1
Senior Electrical Engineer	Office Hours	1	1
Electrical Engineer	Office Hours	1	1
Senior Engineer/Geologist	Office Hours	1	1
Engineer/Geologist	Office Hours	6	6
Surveyor	Office Hours	3	3
Radiation/Ventilation	Shift	2	6
Draftsman	Office Hours	1	1
Grade Control	Shift	3	12
Total			**36**

Underground Workforce

Estimates of underground workforce requirements are derived from SRK's experience in similar operations, estimated equipment requirements and discussions with the Client and COMINAK. Consideration has been given to local conditions that include the following:

- Annual leave requirements for workers;
- Additional leave taken by some workers for pilgrimage during the Eid festival that is traditionally taken in addition annual leave allowance in the Arlit region;
- Reduced worker productivity during Ramadan; and
- High levels of sick leave to account for workers who may contract malaria whilst on leave in the south of Niger (whilst not a malaria region itself, this is a noticeable trend in the nearby mines).

An overview of the envisaged underground workforce at full production, including provision for leave, is provided in Table 15-14.

Table 15-14: Underground production workforce

Position	Shift Structure	Workers per Shift	Total Workers
Mine Captain	Shift	1	4
Shift Boss	Shift	1	4
Loader Operators	Shift	9	36
Truck Drivers	Shift	10	40
Dev. Drillers	Shift	1	4
Production Drillers	Shift	4	16
Nippers	Shift	3	12
Services	Shift	3	12
Ventilation	Shift	2	8
Mechanics	Shift	6	24
Electricians	Shift	2	8
Rockbolters	Shift	5	20
Scaling Crew	Shift	3	12
Shotcreter	Shift	1	4
Transmixer	Shift	1	4
Charge Up	Shift	8	32
Belt Runner	Shift	2	8
Stores	Shift	2	8
Road Crew	Shift	1	4
Medical	Shift	1	4
Miscellaneous Crew	Shift	10	40
Total		**76**	**304**

Ramp Up

The labour schedule required for underground mining is estimated over the life of the proposed operations based and allows for the proposed ramp up activities and changes in equipment levels over the life of the underground operations.

Maintenance

No workshop is included in underground as equipment can be quickly trammed to surface due to the shallow depth of the working. Workshops on surface have the advantage of additional space, better light and more potential to establish clean rooms for high quality maintenance of sensitive components.

Equipment would only be taken to the surface workshop for planned maintenance and when broken down. In the event that equipment is unable to transport itself to the workshop, development loaders will tow the equipment.

Shift servicing (i.e. lubricating, oil refills) and refuelling of the equipment is to be undertaken underground either near the fuel bay or in the production areas using a dedicated service vehicle. The fuel bay is to have an independent exhaust and be centrally located in each mine.

15.5 Mine Planning

15.5.1 Introduction

Having established a preferred approach to mining, this section develops the selected options into a coherent mine design. The ultimate objective being the construction of a life of mine plan for each scenario to be applied to the technical-economic model.

Mine design for the M&M mine is undertaken using the Deswik suite of software to interrogate the resource model, develop the mine design and create the mine schedule. The Maryvonne mine is undertaken using CAE Mining's Datamine Studio 3, 5-D Planner, Enhanced Production Scheduler. VentSim Visual software is used to develop and evaluate the mine ventilation network.

15.5.2 Design Parameters

SRK has applied four development profiles to the mine design (Table 15-15). The dimensions of the profiles have been selected based on the requirements of the equipment to be used underground and ventilation requirements.

Table 15-15: Development profile design parameters

Profile	Height (m)	Width (m)	Drillholes[1]	Powder Factor (kg/m^3)
Access	4.5	5	37	0.68
Conveyor	3.2	4	25	0.75
Panel	3.5	7	40	0.68
Production (Conventional)	1.8	7	30	0.78

[1]Includes uncharged void holes

Access development is applied to all of the central infrastructure development in the mine, including declines, workshop, central accesses and loading bays throughout the mines.

Conveyor development will only serve the conveyor network applied underground. As the conveyor development is too low to fit a truck, it will be developed with or immediately after associated access development to enable haulage of the waste from the headings.

Panel development is a central access for each panel to allow for trucking and **production development** provides a typical profile for the room and pillar mining. SRK notes that due to

the variability of the mineralised thickness, there will be sections of the mines where the mining height will be larger than discussed in the report. However, for the purposes of producing estimates for cost and productivity, the single design has been assumed.

The mining height for the mining panels is assumed to be 1.8 m high. These parameters are based on the operational requirements of the equipment applied to the production mining. Where the mineralised thickness is greater than the estimated thicknesses, the overall height is assumed to increase in line with the mineralised thickness.

Room/drive widths of 7 m are applied to both the production and panel development to remove the need for slashing cuts when mining at 90° to an existing heading. Sufficient room will be available for the jumbo drill booms to position without affecting the blast pattern or reducing the advance per cut.

Corner drill holes are lowered in development profiles to give a small arched profile (cross-sectional area reduced by 8%) to improve longer term roof stability. Production development will have a square profile due to the smaller mining heights. 5% overbreak is also allowed for from both production and development profiles.

Powder factors are calculated assuming a blasthole diameter of 45 mm and five uncharged void holes. Blast patterns have not been designed for the individual headings, rather specific powder factors have been estimated based on experience. All holes are assumed to be blow-loaded and 25% of the blastholes half-loaded with ANFO. SRK notes that these estimates are considered conservative, as no adjustment has been made for the reduced drilling requirements of swings and slashing.

Based on a preliminary ventilation analysis (refer to Item 24), the vertical development is to have diameters of:

- Intake raise - 5 m diameter; and
- Exhaust raise - 3.5 m diameter.

15.5.3 Mining Factors

Mining Losses

Mining losses considered for the Madaouela design are:

- Planning losses;
- Geotechnical losses; and
- Operational losses.

Planned losses are incorporated into the design and include outliers (Resources that fall outside of the panel designs) and regional pillars around the central development. They are a function of the design processes rather than a factor applied to the RoM tonnage calculations.

Geotechnical losses result from the geotechnical design parameters estimated for the Madaouela deposits. The design requires three pillar sizes to be applied to the panels (depending on the geotechnical characteristics) as well as inter-panel pillars to provide regional stability. A summary of the geotechnical design parameters are shown in Table 15-16, Figure 15-18 and Figure 15-19. Detailed geotechnical analysis can be found in Section 26.

Table 15-16: Summary of geotechnical design parameters

Geotechnical Zone	Room Width (m)	Pillar Dimensions (m)	Mining Losses	Losses Inc. Inter-Panel Pillars
1	7	4 x 4	87%	78%
2	7	5 x 5	83%	74%
3	7	6 x 6	79%	70%

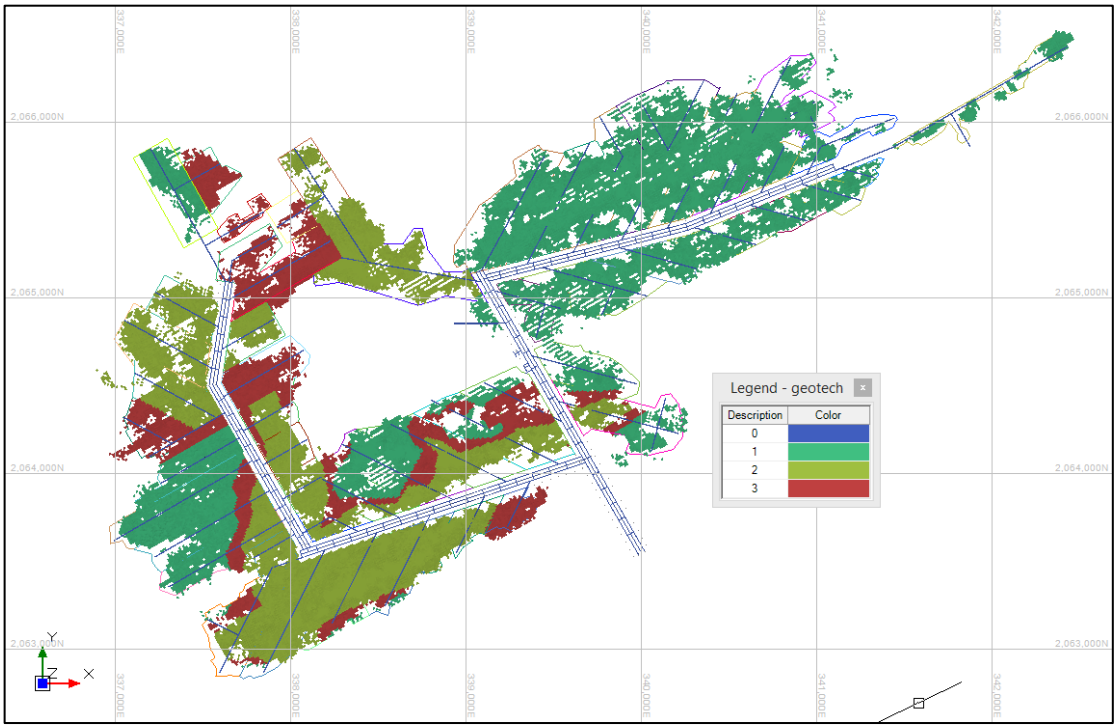


Figure 15-18: Geotechnical zones for Marianne-Marilyn deposit

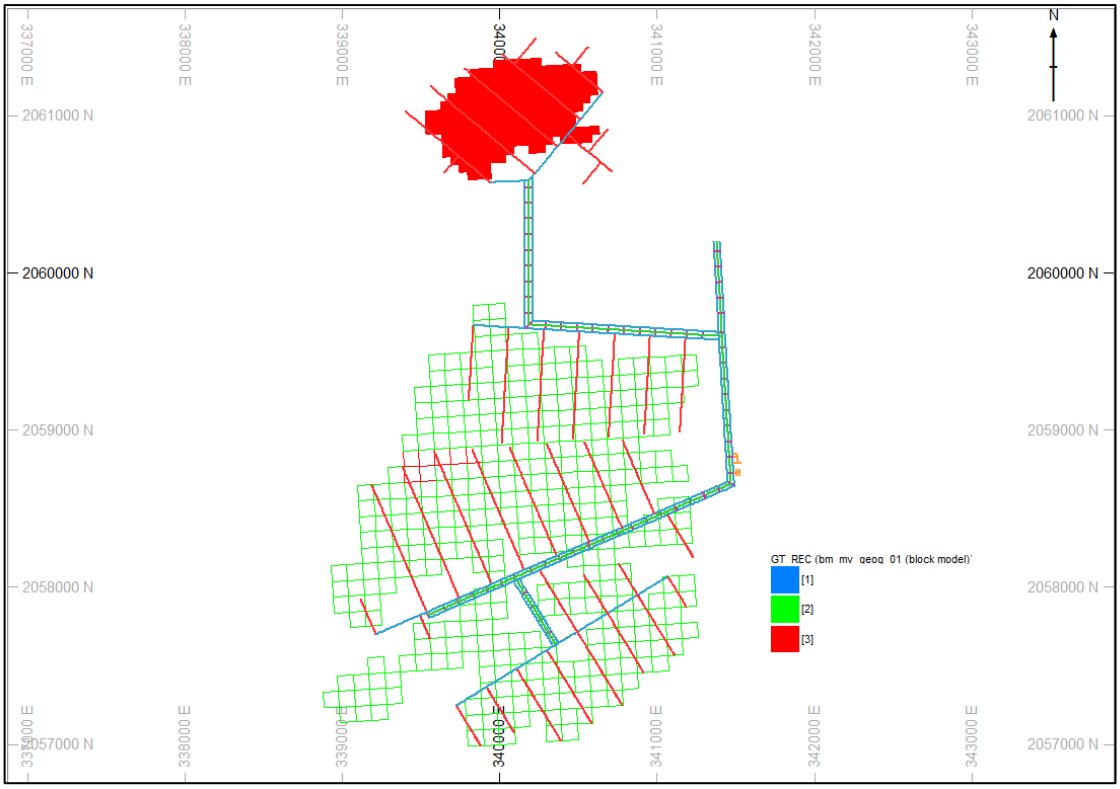


Figure 15-19: Geotechnical zones for MSNE-Maryvonne deposit

Operational losses assume that there will be losses of ore resulting from underbreak, flyrock and spillage that causes incomplete recovery of ore from the face and loss of ore into the waste stream during the course of mining.

SRK has assumed overbreak in the pillars that introduces mineralised dilution to mine production will balance out operational losses.

Dilution

Room and pillar mining in a bedded deposit is considered a selective mining method. Bedding planes can be used as natural partings to minimise the amount of overbreak from development. The underlying Talak Shale is, however, soft and dilution caused by digging into this layer when loading is possible. External dilution is assumed to be minimal with a 2.5% applied to the designed tonnages.

Internal dilution will be a significant contributor to the overall production tonnages with internal dilution coming from the interstitial layers of waste within the mineralised horizon, and waste rock that make the difference between thickness of mineralisation and mining height (1.8 m for conventional mining and 1.5 m for resuing). This designed dilution will be variable and is incorporated into the panel tonnes for the M&M mine as part of the mineral optimisation process. The internal dilution for Maryvonne has been calculated at a localised level for each column of blocks within the model.

Cut-Off Grade

Cut-off grades for conventional mining and as applied to the Madaouela deposits are derived from the preliminary input parameters outlined in Table 15-17.

Table 15-17: Cut-off grade inputs

Parameter	Units	
Conventional Mining		
Mining	USD/t_{RoM}	29.80
Processing	USD/t_{RoM}	36.83
General and Administrative	USD/t_{RoM}	3.45
Total Production Cost	USD/t_{RoM}	70.08
Uranium Price	USD/lb U_3O_8	70.00
Recovery – Radiometric Sorting		93.5%
Recovery – Leaching		93.0%
Recovery – Precipitation		97.0%
Royalty		5.5%
Transportation	USD/t_{metal}	205.00
Land Royalty	USD/t_{metal}	158.76

The cut-off grade calculations assume that the conventional mining method is mined to 1.8 m high. The waste introduced into the production feed is dealt with by the ablation process in the process plant.

Based on the above parameters, the estimated cut-off grade is set at:

- Conventional mining - 0.48 kg/t_{RoM} eU.

The cut-off grade is calculated independent of dilution and hence is applied to the production tonnage calculations after adjusting for mining factors.

15.5.4 Mine Optimisation

The Resource block model used for the M&M deposit estimation is a sub-blocked Datamine model which was created from the SRK resource model. The Deswik mine stope optimiser was used to determine the resources available to mine and to create 3D mining shapes within the Deswik software which could be applied to the mine design and scheduler. The stope optimiser was run to produce in-situ tonnage shapes 12.5 m wide x 12.5 m long. A mining height of 1.8 m was selected at a cut-off grade of 0.48 kg/t which included the internal dilution for the panels. A further modifying factor of 2.5% was added to allow for mining dilution.

The Resource block model used for MSNE-Maryvonne is regularised using a percentage for both the mineralised thickness within the each block and the footprint within each block included in the Resources. The blocks have the following dimensions:

- MSNE – 100 m x 100 m x 0.4 m; and
- Maryvonne – 5 m x 5 m x 0.4 m.

For MSNE-Maryvonne no underground optimisation software is used to identify the mineable zones. Instead, each column of blocks in the block model is consolidated and expanded to the minimum mining height 1.8 m assuming non-mineralised waste (internal dilution). The cut-off grade is applied to the resulting mining unit leaving only the areas to be targeted for mining. The results of this exercise are shown in Figure 15-20 and Figure 15-21

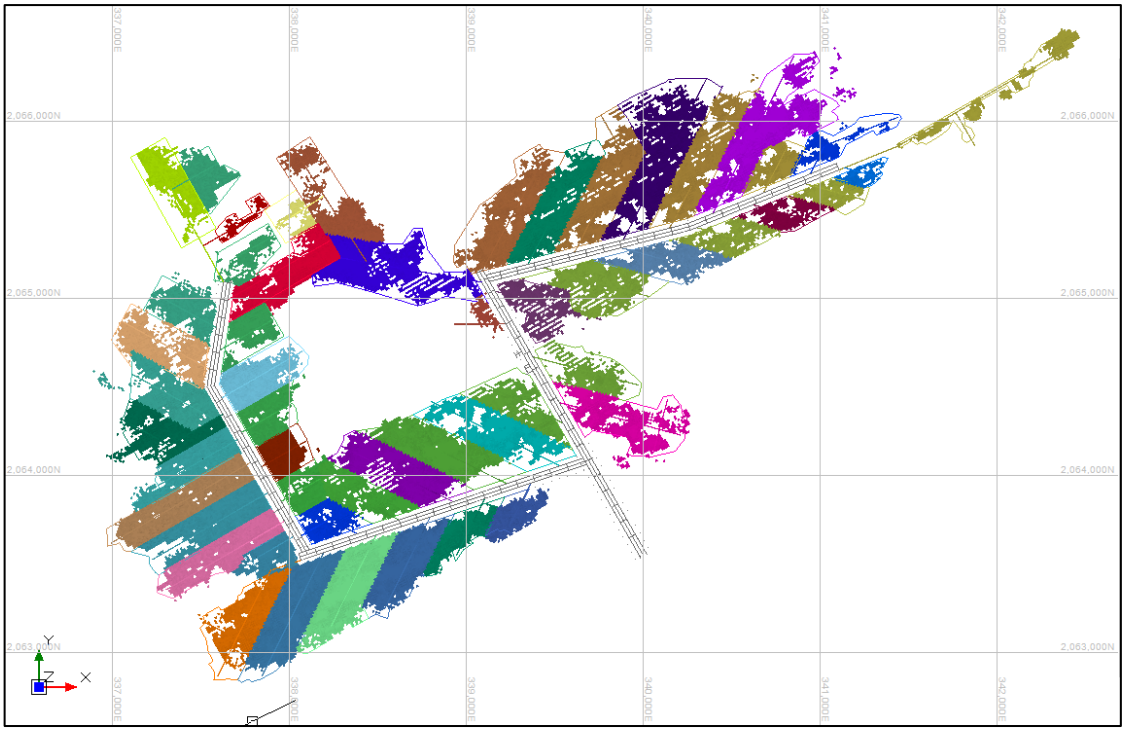


Figure 15-20: Mining units above cut-off grade for Marianne-Marilyn at a mining height of 1.8 m

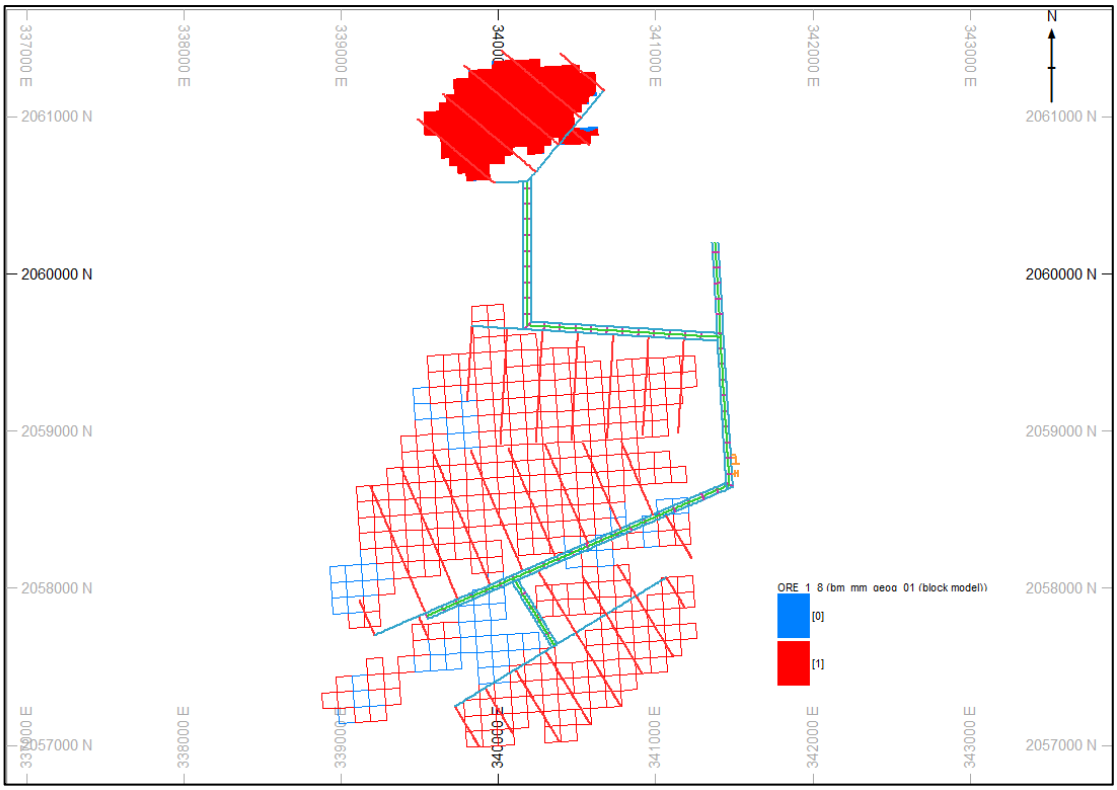


Figure 15-21: Mining units above cut-off grade for MSNE-Maryvonne at a mining height of 1.8 m

15.5.5 Mine Layout

The mine layout stems from a central conveyor network that links the production panels to the surface via a decline. Conveyor drives are located in waste where possible (e.g. the UC channels in the Marilynn-Marianne deposit) but in places it is located more centrally to increase the number of panels accessible and to maximise scheduling flexibility.

Each side of the central conveyor drive is an access drive for equipment movement and truck haulage (Figure 15-22 and Figure 15-23). The twin accesses, either side of the conveyor are used to remove the need for regular crossing of the conveyor. As a secondary advantage, the triple heading development maximises the cross-section available to deliver fresh air throughout the mine.

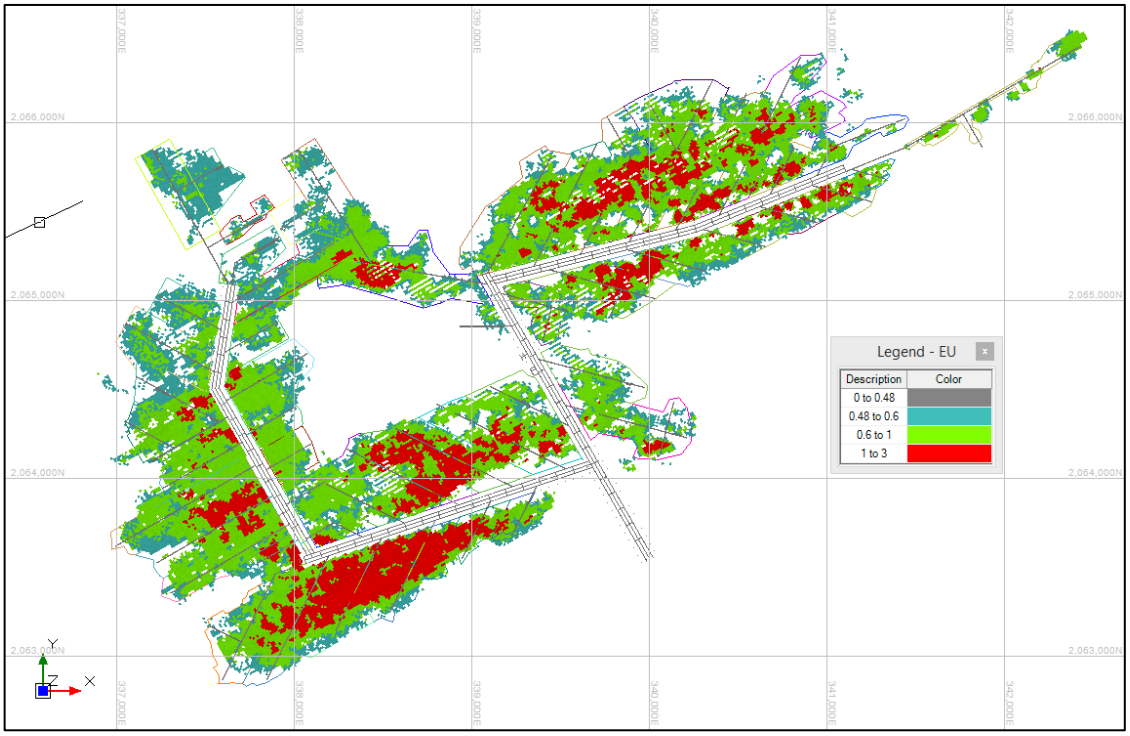


Figure 15-22: Plan view of proposed mine layout for Marianne-Marilyn deposit showing grades when diluted to 1.8 m mining height

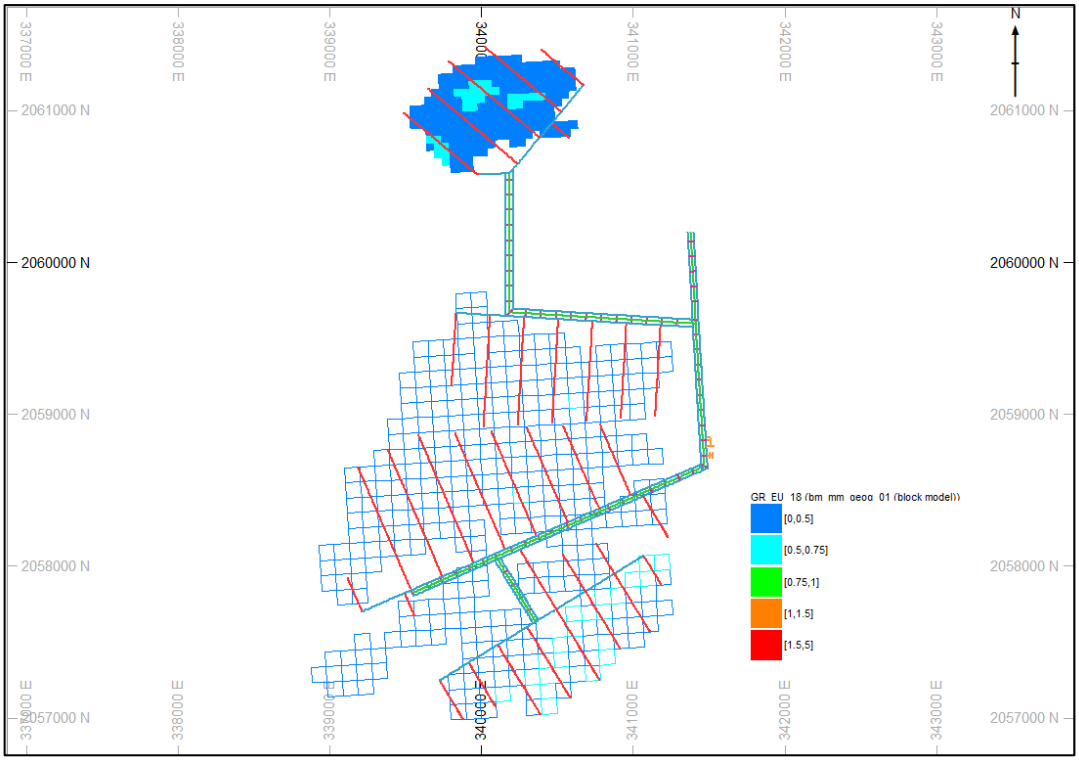


Figure 15-23: Plan view of proposed mine layout for MSNE-Maryvonne deposit showing grades when diluted to 1.8 m mining height

Access and conveyor development profiles are assumed to be located above the contact with the Talak Shale as it:

- Prevents the Talak Shale from affecting road conditions in the major transportation routes; and
- Locates the development above the ore zone, minimising the generation of radon daughters and its by-products, simplifying the ventilation design and increasing the allowable residence time of air underground.

Where possible, the accesses are located within the UA channels to provide a central access whilst minimising the Resources sterilised by the regional pillars.

To access production panels development will ramp down to the Talak/Guezouman contact (the mineralised zone) in the regional pillars surrounding the central access and conveyor infrastructure development (Figure 15-24).

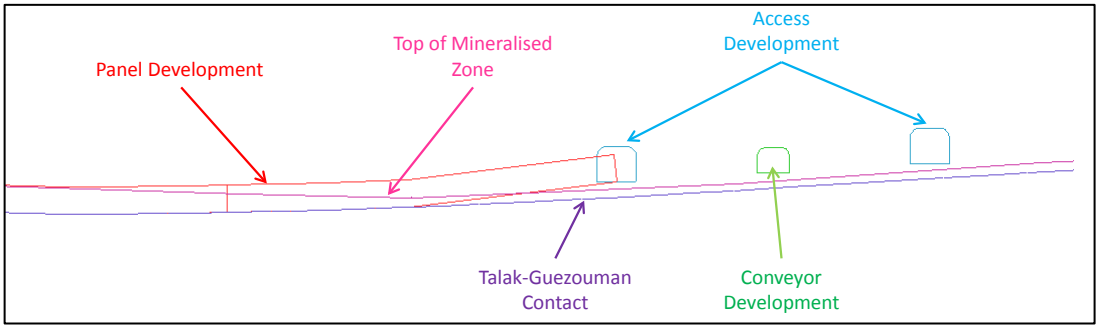


Figure 15-24: Schematic cross-section showing the central infrastructure outside of the mineralised horizon

Mining panels accessed from the central infrastructure form the production zones of the proposed mines (Figure 15-25 and Figure 15-26). Each panel is 225 m wide, comprising 200 m for production mining and 25 m for inter-panel pillars.

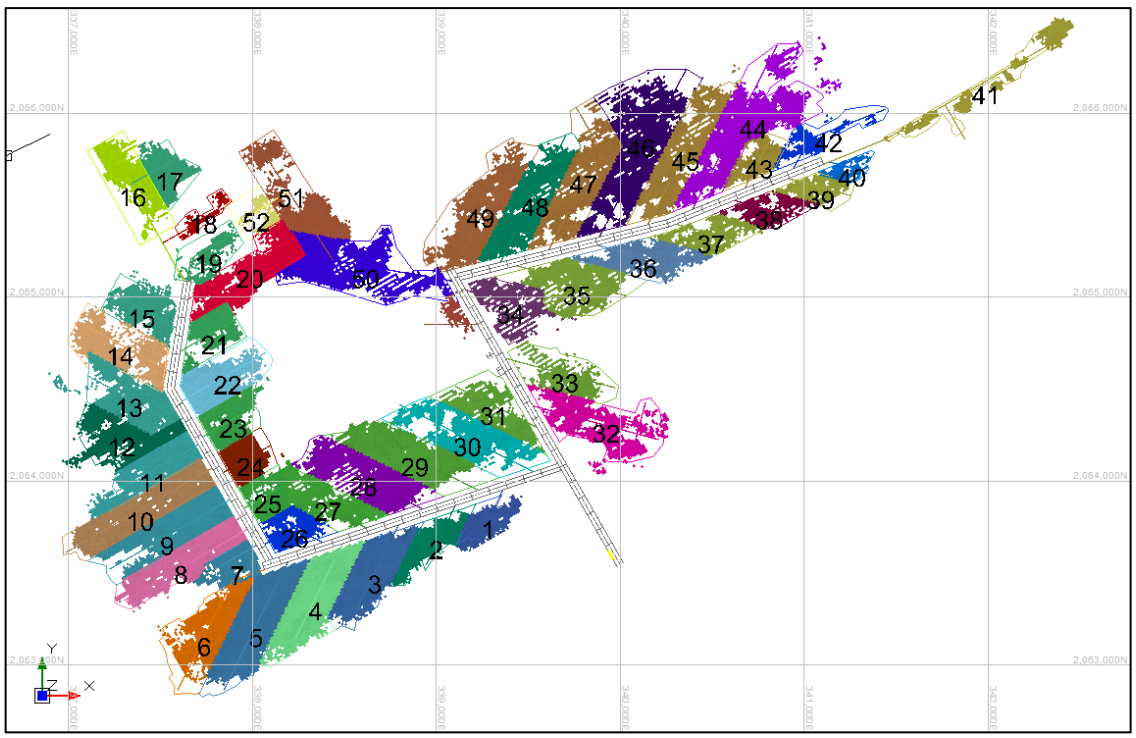


Figure 15-25: Plan view showing a proposed production panel layout for Marilynn-Marianne

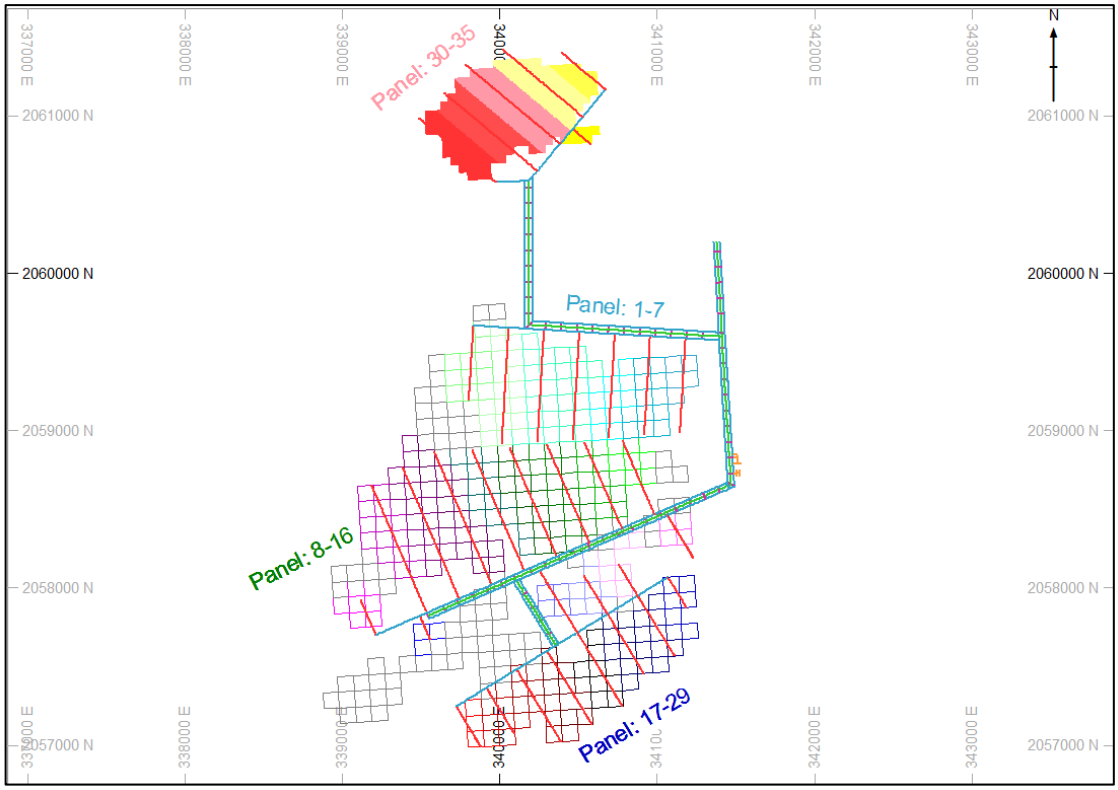


Figure 15-26: Plan view showing a proposed production panel layout for MSNE-Maryvonne

Each panel is to be split into two production zones divided by a 10 m pillar to make 100 m wide production zones. A 15 m pillar also is to be left between each panel to both provide a regional pillar for ground support and to separate ventilation circuits from adjacent panels. The panel access drive is to be located adjacent to the central pillar providing access to both production zones (Figure 15-27).

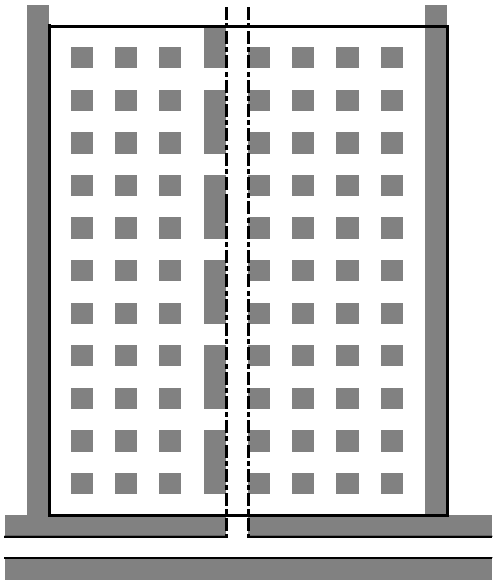


Figure 15-27: Plan view showing a schematic of an individual production panel layout

Panel access development has a higher profile than the production zones to allow the trucking crews to reload the mined material. 'A' pillars connect each panel access to the central infrastructure to negate the impact of angled drives on equipment turning circles and create passing zones along the haulage routes.

Production panels accessed from the central conveyor development gives much of the design a fishbone appearance. This is because, the panels are designed at 45° to the maximum dip of the mineralised horizon to minimise the need to change the mining direction due to localised zones of high gradient (Figure 15-28 and Figure 15-29).

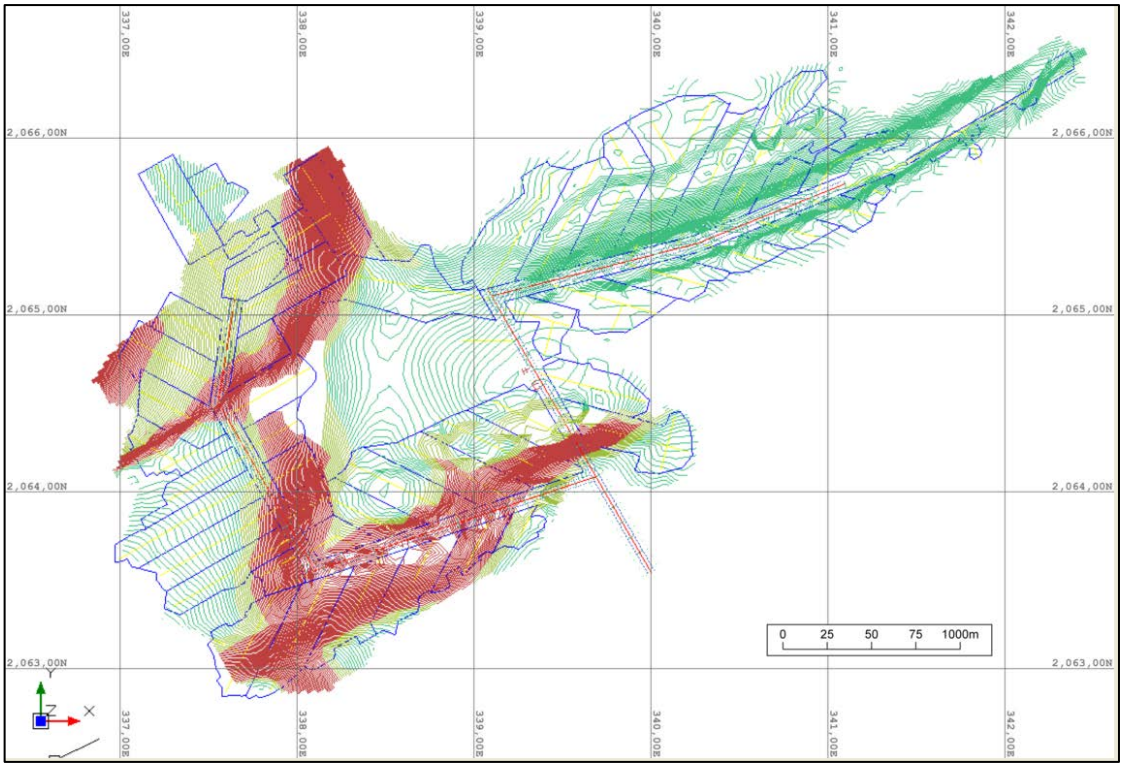


Figure 15-28: Plan view of development and geological dip of the Talak-Guezouman contact for Marianne-Marilyn

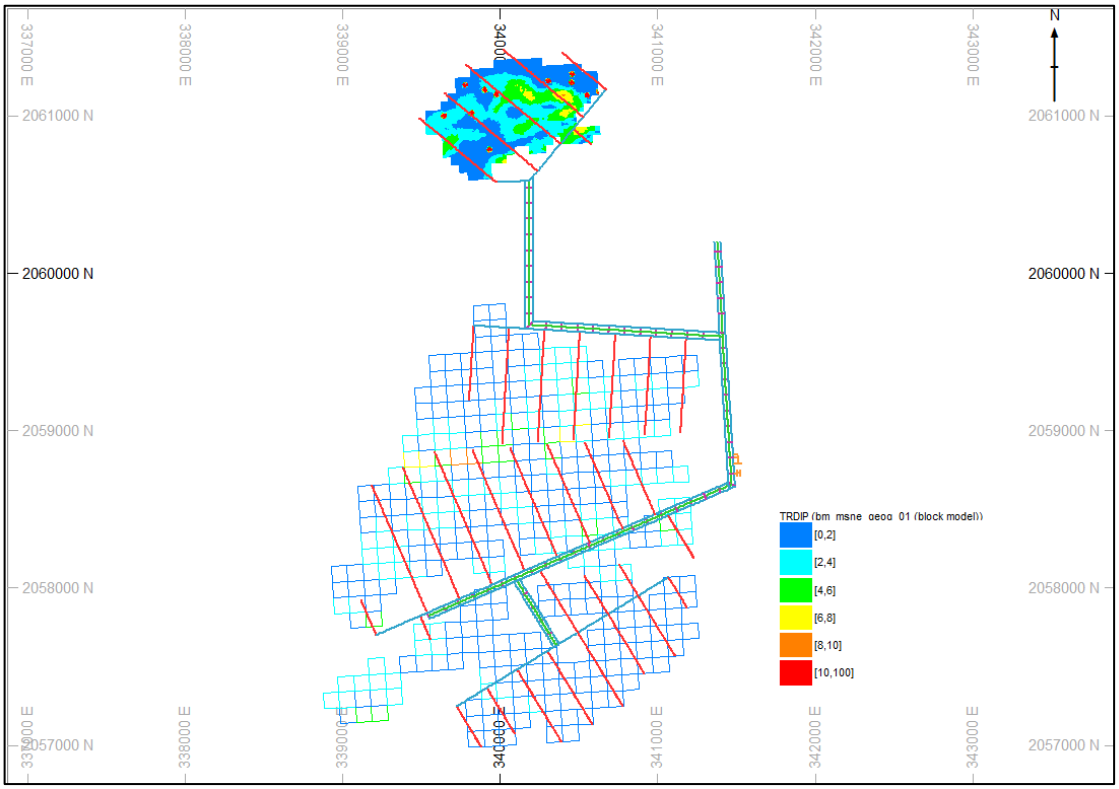


Figure 15-29: Plan view of development and geological dip of the Talak-Guezouman contact for MSNE-Maryvonne

Each panel is split into sections between 100 m and 300 m long with a ventilation raise at the end of each panel or section (Figure 15-30). This makes a primary ventilation circuit with return (used) air exhausted to surface without requiring the whole panel to be developed prior to starting production. Once the first section of the panel is mined, the extraction fan can be moved further along the panel to the next raise and the previous exhaust raise will provide fresh air from surface.

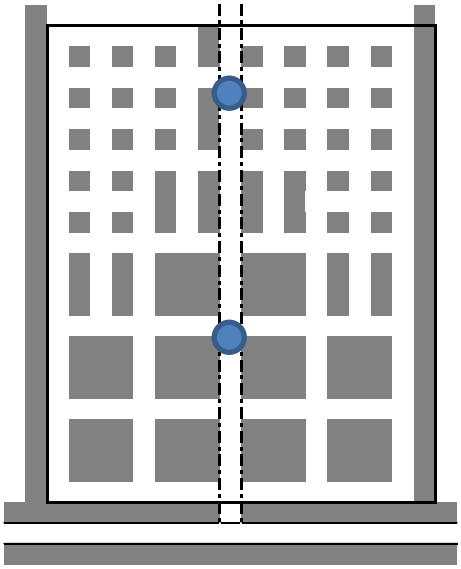


Figure 15-30: Schematic of the compartmentalised approach to mining of panels with nominal vent raise locations

15.5.6 Underground Infrastructure

Magazine

Underground storage of explosives is envisaged to be minimal with regular delivery from the surface magazine to underground. The magazine incorporates two development bays allowing a physical separation of the detonators, primers and bulk explosives. It is assumed that a hand operated chain crane would be mounted on the roof to assist with the handling of ANFO bulk bags within the magazine. A lockable gate would be mounted to the walls to restrict access to the underground magazine.

The magazine is to be centrally located with an independent exhaust raise is designed adjacent to the magazine in an attempt to isolate the impact of any catastrophic events.

Fuel Bay

The fuel bay is to be located within the same area as the underground stores and magazine. Diesel is envisaged to be stored at the surface and transported underground through lined boreholes with pressure reducers to control flow. Surface storage of the diesel allows for simple resupply and removes large reservoirs of diesel from the underground environment. Oils and lubricants would be pumped from drums to be stored within the fuel bay.

The fuel bay is to share the same exhaust raise as the magazine with a separation between the two of a safe distance. It is assumed that this is a configuration is viable as the diesel storage is on the surface. Confirmation is required to ensure such a configuration is consistent with Nigerien regulations. In the event this is not the case, an additional raise and exhaust fan will be required.

Workshops

Maintenance workshops are to be located at the surface. The shallow nature of the deposit allows the equipment to transport itself or in the event of breakdown to be towed to surface for maintenance and repair.

Simple repairs and checks can be undertaken underground at the work place or in a service bay that is equipped with lights and a good floor located near the fuel bay.

Stores

There will be minimal storage of consumables (i.e. pipe, rockbolts, fittings, etc.) underground as regular deliveries will be made and controlled from a central storage facility at the surface.

Offices and Lunch Rooms

Offices and lunch rooms will be located on the surface. A fleet of light vehicles and mine taxis has been allowed for to transport the workers to and from the underground workings. As the bulk of the production activities will be in a centralised region of the mine, the transportation times should allow for this.

Eating in the production areas or any other area without hand washing facilities or regular cleaning should be discouraged due to the risk of ingesting uranium minerals. A small allowance has been made for some small areas to be established at the fresh air bases to prevent excessive time lost for travelling, though these areas would require regular cleaning.

15.5.7 Run of Mine Estimation

For the M&M resource, RoM tonnage is determined for each individual mining block by applying mining losses determined by the panel layout to be applied i.e. R&P, AR&P and SR&P. Each block is then further diluted by 2.5% to allow for mining dilution such as floor over-dig and checked against the cut-off grade. The blocks are aggregated to determine the production potential for each panel. It is assumed that blocks that do not make the cut-off grade are not mined. This process is carried out by the Deswik Scheduler function.

RoM tonnage for MSNE and Maryvonne is estimated for each column of blocks intersecting the Mineral Resource. Each column is diluted to the minimum mining height before the mining factors are applied (i.e. mining losses, dilution and cut-off grade). Further adjustments are made to verify that the estimated revenue for each panel is not only sufficient to cover the operating costs, but also the cost of the panel access development and raises designed for each panel.

Estimated production ore tonnages for each panel are shown in Table 15-18 to Table 15-19, which include the relative production by each mining method.

Table 15-18: Estimated production panel tonnages for M&M

Panel	RoM (kt)	Metal (t eU)	Grade (kg/t eU)	R&P	AR&P	SR&P
1	146.07	104.43	0.71	0%	0%	100%
2	177.92	176.43	0.99	0%	100%	0%
3	348.37	430.47	1.24	3%	90%	7%
4	448.31	512.98	1.14	2%	86%	12%
5	471.16	446.91	0.95	1%	98%	2%
6	300.52	215.46	0.72	3%	74%	23%
7	146.12	125.58	0.86	0%	95%	5%

Panel	RoM (kt)	Metal (t eU)	Grade (kg/t eU)	R&P	AR&P	SR&P
8	361.15	266.41	0.74	55%	29%	16%
9	368.40	332.00	0.90	70%	13%	17%
10	448.09	338.99	0.76	92%	0%	8%
11	290.64	213.13	0.73	100%	0%	0%
12	276.04	194.49	0.70	0%	0%	100%
13	313.43	214.45	0.68	0%	100%	0%
14	295.21	170.34	0.58	0%	100%	0%
15	225.19	131.23	0.58	0%	100%	0%
16	240.00	131.31	0.55	100%	0%	0%
17	156.47	85.26	0.54	0%	0%	100%
18	38.69	20.96	0.54	0%	0%	100%
19	98.65	51.87	0.53	0%	0%	100%
20	342.75	211.97	0.62	0%	0%	100%
21	156.63	101.06	0.65	0%	100%	0%
22	244.80	172.40	0.70	0%	0%	100%
23	181.57	120.66	0.66	0%	63%	37%
24	146.55	104.78	0.71	0%	96%	4%
25	139.83	93.47	0.67	0%	97%	3%
26	140.70	98.76	0.70	0%	74%	26%
27	245.91	185.11	0.75	46%	25%	29%
28	393.59	372.24	0.95	58%	24%	18%
29	405.53	389.10	0.96	27%	41%	32%
30	377.11	325.69	0.86	26%	38%	35%
31	218.87	158.29	0.72	23%	13%	64%
32	350.28	238.56	0.68	49%	28%	23%
33	170.01	106.00	0.62	96%	0%	4%
34	226.84	147.19	0.65	100%	0%	0%
35	324.66	287.24	0.88	100%	0%	0%
36	260.53	216.92	0.83	100%	0%	0%
37	175.58	131.42	0.75	100%	0%	0%
38	142.95	111.76	0.78	100%	0%	0%
39	110.26	93.47	0.85	100%	0%	0%
40	61.03	45.74	0.75	100%	0%	0%
41	156.73	107.33	0.68	100%	0%	0%
42	101.54	71.97	0.71	100%	0%	0%
43	132.64	113.34	0.85	100%	0%	0%
44	632.88	509.21	0.80	100%	0%	0%
45	407.19	343.95	0.84	100%	0%	0%
46	619.12	496.67	0.80	100%	0%	0%
47	415.83	371.45	0.89	100%	0%	0%
48	395.72	322.97	0.82	100%	0%	0%
49	427.79	299.38	0.70	100%	0%	0%
50	506.98	391.41	0.77	0%	100%	0%
51	288.86	175.27	0.61	0%	100%	0%
52	49.71	25.60	0.52	0%	0%	100%
53	46.53	26.64	0.57	100%	0%	0%
Total	**14,147.91**	**11,129.70**	**0.79**	**49%**	**33%**	**17%**

Table 15-19: Estimated production tonnages for MSNE-Maryvonne

Panel	RoM (kt)	Metal (t eU)	Grade (kg/t eU)	R&P	AR&P	SR&P
1	73.6	56.9	0.77	100%	0%	0%
2	243.1	187.3	0.77	100%	0%	0%
3	264.1	230.8	0.87	100%	0%	0%
4	313.8	258.5	0.82	90%	10%	0%
5	393.5	284.2	0.72	100%	0%	0%
6	337.0	210.9	0.63	100%	0%	0%
7	303.8	169.2	0.56	100%	0%	0%
8	139.7	103.9	0.74	100%	0%	0%
9	252.6	181.6	0.72	100%	0%	0%
10	412.5	331.7	0.80	100%	0%	0%
11	491.9	413.6	0.84	100%	0%	0%
12	395.1	273.6	0.69	85%	15%	0%
13	460.8	277.3	0.60	95%	5%	0%
14	382.3	233.3	0.61	100%	0%	0%
15	326.4	176.4	0.54	100%	0%	0%
16	26.5	15.6	0.59	100%	0%	0%
17	36.1	28.9	0.80	100%	0%	0%
18	33.2	29.5	0.89	100%	0%	0%
19	80.3	70.4	0.88	100%	0%	0%
20	51.3	33.2	0.65	100%	0%	0%
21	34.4	19.1	0.55	100%	0%	0%
22	100.1	101.3	1.01	100%	0%	0%
23	154.7	135.0	0.87	100%	0%	0%
24	248.5	240.6	0.97	100%	0%	0%
25	139.0	130.6	0.94	100%	0%	0%
26	151.1	149.0	0.99	100%	0%	0%
27	110.2	86.5	0.79	100%	0%	0%
28	181.8	141.5	0.78	100%	0%	0%
29	44.4	32.6	0.74	100%	0%	0%
30	271.3	239.2	0.88	100%	0%	0%
31	385.8	297.3	0.77	98%	0%	2%
32	463.5	394.5	0.85	98%	0%	1%
33	361.9	283.6	0.78	97%	2%	1%
34	99.4	61.6	0.62	93%	2%	5%
35	49.3	34.5	0.70	100%	0%	0%
Total	**7,813.1**	**5,913.6**	**0.76**	**97%**	**2%**	**1%**

- Ore summary;
  - o Marianne-Marilyn - 14.1 Mt @ 0.79 kg/t eU;
  - o MSNE-Maryvonne - 0.8 Mt @ 0.76 kg/t eU;
  - o Total - 14.8 Mt @ 0.77 kg/t eU;

15.5.8 Sequencing

The M&M mine is sequenced based on the proximity to the shaft and the availability of services. The MSNE-Maryvonne deposits is sequenced according to the following principles:

- Highest grades (after dilution to the minimum mining height) are targeted first; and
- Mine areas/panels that have large tonnages, which maximise the use of the initial infrastructure installed, are prioritised over areas with smaller tonnages.

On this basis the following sequence is applied to the Madaouela underground deposits:

- Phase 1A – Western limb of the Marianne-Marilyn deposit;
- Phase 1B – Central limb of the Marianne-Marilyn deposit;
- Phase 2A – Eastern limb of the Marianne-Marilyn deposit;
- Phase 2B – North-Western limb of the Marianne- Marilyn deposit;
- Phase 3 – Northern limb of the MSNE-Maryvonne deposit; and
- Phase 4 – Southern limb of the MSNE-Maryvonne deposit.

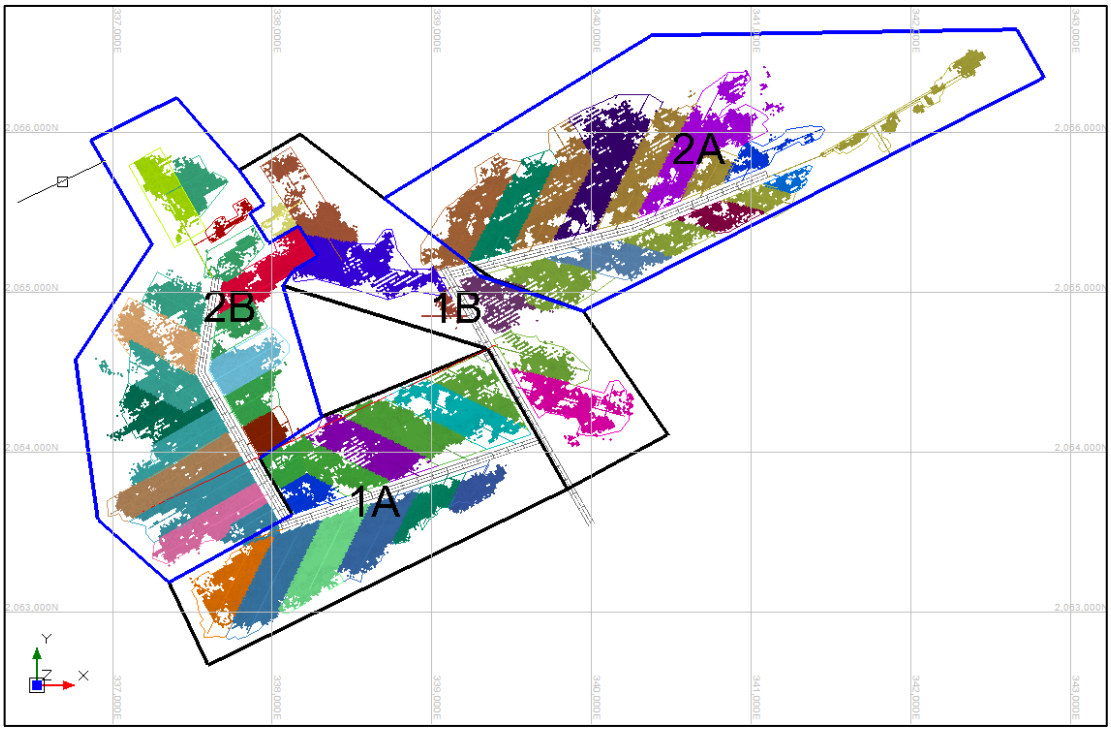


Figure 15-31: Visual Representation of the proposed mining sequence – M&M

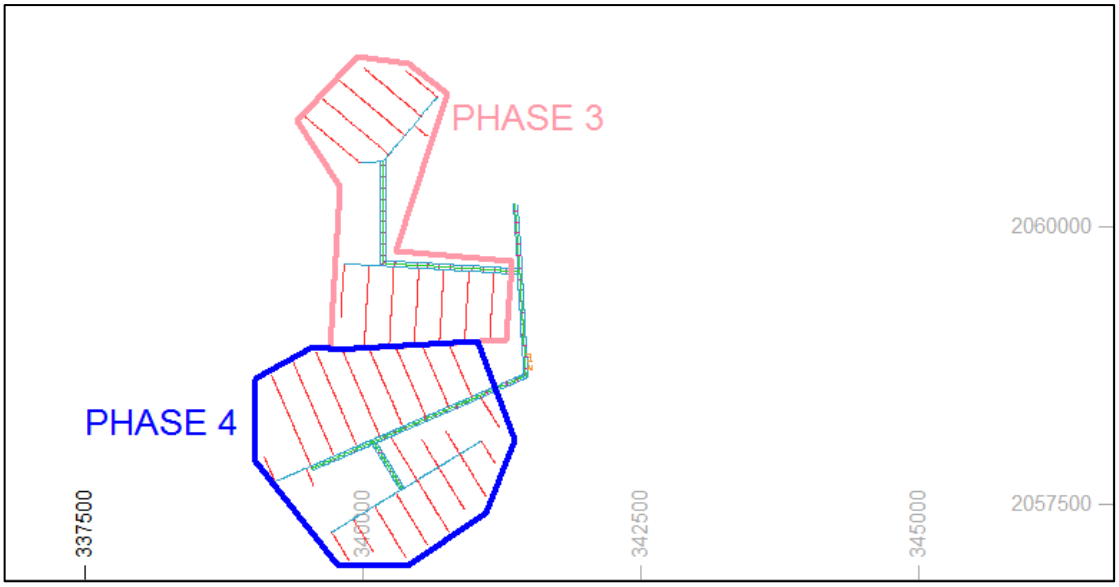


Figure 15-32: Visual representation of the proposed mining sequence - Maryvonne

In the early years of operation, the M&M panel sequencing is based on the panel proximity to the decline and the shortest duration possible to bring the panel into production. Once a district is established, production remains concentrated within that district. As more ore is required, production moves outside the district with the priority given as per the phasing sequence. For MSNE-Maryvonne the higher grade zones are targeted by the accesses so they occur early in the schedule, however as these are later in the mine life (+/- 14 years) less rigour is applied to the panel selection as it has less impact on the overall performance of the mine.

15.5.9 Production Rate

Production Constraints

A modular approach to mining of the Madaouela deposits is assumed for production modelling. Each panel is considered to be largely independent of the next as it can be mined with a dedicated mining crew and independent ventilation. Therefore, panel production rates are determined by the productivity of the mining crew, and so mine production is determined by the number of panels to be worked concurrently.

The advantage of this approach is that it provides flexibility in the scheduling, should there be an issue in one panel (e.g. ventilation), production can be sourced from a second panel quickly with a minimal delay (provided the panel is prepared in advance).

Material's handling is limited by the conveyor network to a capacity of 380 t/h (approximately 7,600 t/d based on a 20 operating hours per day).

Cultural aspects of the local workforce are considered in the estimation of productivity and labour requirements. Staff are expected to be predominantly Muslim and observance of prayer times and reduced worker productivities during fasting (Ramadan) are considered.

Benchmarking

COMINAK's Akouta Mine applies a similar modular approach to mining as proposed for Madaouela. Five production crews are producing approximately 90,000 tpa each, which includes an element of backfilling. A typical production rate for the mine is 1,500 t/d.

Applied Rate

Mine production rate is the result of an options analysis of preliminary production data for and considered:

- Mining costs;
- Processing costs;
- General and administrative costs;
- Mining and processing recoveries;
- Equipment productivities and requirements; and
- The number of panels required to meet production rates.

The analysis compared estimated mining costs per pound of uranium produced for conventional mining, resue mining, heap leaching and 2-stage leaching. Conventional mining and 2-stage leaching were the chosen options.

To deliver a feed rate of uranium to the processing plant, 4,020 tpd of RoM is required from the mining panels. The production rates are based on providing the same quantity of mineralised material in the RoM feed for each mine.

15.5.10 Ramp Up

The first two years of mining is to establish underground mine infrastructure, including central conveyors and an initial c.200 m of panel length to provide sufficient faces to start production at near full production rate. This will be schedule to coincide with the decline of ore production from the Miriam open pit. The ramp up is constrained by the panel availability. Once a panel is available, then it is mined at full production.

In order for a panel to be considered available the trunk conveyor servicing the panel and the first ventilation raise within the panle to allow for through ventilation need to be installed and commissioned.

Mining Equipment

Scheduling parameters for underground mining have been built up from first principles using information sourced from equipment suppliers, benchmark operations, GoviEx and SRK's own databases and experience.

The calculated productivities for selected equipment are shown in Table 15-20. The numbers assume three 8 hr shifts per day with a mechanical availability of 85% and utilisation of 90% for most equipment.

Table 15-20: Estimated equipment productivities

Equipment	Units	
Jumbo – Development	$m_{advance}$/month	1,050 – 1,250[1]
Jumbo – Production	t_{RoM}/month	37,500
Rockbolter		1 for each Jumbo
Loader – Development	t_{RoM}/month	44,000
Loader – Production	t_{RoM}/month	32,500
Truck	t_{RoM}/month	30,500

[1] Depending on profile being drilled

Penetration times for the drilling are anticipated to be high, and are in line with observations at the nearby Akouta Mine. Estimates applied to the productivity estimates are based on figures provided by the manufacturers for rock with a uniaxial compressive strength of 160 MPa and increased by 50% to allow for the low strengths of the Madaouela rock. A similar approach was applied to raiseboring advance rates.

Based on the estimated equipment productivities and cycle times, advance rates for the various development profiles are also estimated. Development drilled using the larger development jumbo is expected to achieve 3.6 m of advance per firing; the low-profile production jumbo will advance 3.4 m. Table 15-21 outlines the advance rates applied to the mining schedules.

Table 15-21: Estimated development advance rates

Development Profile	Units	
Decline/Access	$m_{advance}$/month	120
Conveyor	$m_{advance}$/month	120
Panel	$m_{advance}$/month	75
Production	$m_{advance}$/month	50
Raiseboring	$m_{advance}$/month	150

Mine Production

42 production faces are required to be blasted each day to meet the production targets for each mine. The productivity of the mining method is estimated to be 28.1 $t_{RoM}/m_{advance.}$

Construction

Construction of the portal is estimated to take six months from commencement of works before development of the decline can begin.

Conveyor construction is expected to be completed prior to mine production. In practice, truck haulage is available to commence production in the short term during construction, but no allowance for additional trucks to cover this additional haulage duty has been applied.

In-Panel Waste Development

Provision for additional waste development to account for potential discontinuity of the mineralisation within panels and the added complexity of mining steeper gradients has been made using a factor of metres developed per tonne in the panel.

The provision is only applied to mining of steeper areas of panels (>7°) as defined by the mining method. Shallower mining areas (<7°) have no allowance as any mining complexity or discontinuity is considered to be allowed for as by the assumption that the panel development is 100% waste when in reality, part of this will be within the mineralised horizon. As a result, there is a small double accounting of tonnages.

The provisions applied to the MSNE and Maryvonne deposits are slightly higher to reflect the less continuous nature of the deposits' mineralisation (Table 15-22).

Table 15-22: Operational waste development factors applied to LoMP

Method	Units	Provision
Marianne-Marilyn		
Room and Pillar	m/kt	0.00
Apparent Room and Pillar	m/kt	2.00
Step Room and Pillar	m/kt	3.00
MSNE-Maryvonne		
Room and Pillar	m/kt	0.00
Apparent Room and Pillar	m/kt	2.50
Step Room and Pillar	m/kt	3.75

15.5.11 Life of Mine Plan

Life of mine plans (LoMPs) are produced for each mining method considering low profile mining equipment. For the M&M deposit scheduling was undertaken using the Deswik scheduling software. For the MSNE-Maryvonne deposit, scheduling was undertaken using CAE Mining's Studio 5-D Planner ("5DP") and Enhanced Production Scheduler ("EPS") software. The LoMPs are based on the designed mine layout (Section 15.5.5), estimated panel tonnages (Section 15.5.7) and scheduling parameters.

Each LoMP integrates the underground designs from both Marianne-Marilyn and MSNE-Maryvonne into a single schedule. The schedule assumes only one development crew in operation at any time. All panel and infrastructure development in Marianne-Marilyn is scheduled to be completed before the decline commences in MSNE-Maryvonne. The LoMPs produced are independent of the influence of the proposed open pit mining at Miriam.

A summary of the LoMPs for each scenario is provided in Table 15-23 with a visual representation of the production profile provided in Figure 15-33.

Table 15-23: Summary of underground life of mine plan

Parameter	Units	
Production	Mt	21.945
Marianne-Marilyn	Mt	14.132
MSNE-Maryvonne	Mt	7.813
Grade	kg/t eU	0.776
Marianne-Marilyn	kg/t eU	0.787
MSNE-Maryvonne	kg/t eU	0.757
Waste Development	km	117.0
Marianne-Marilyn	km	73.9
MSNE-Maryvonne	km	43.1
Vertical Development	km	19.3
Marianne-Marilyn	km	8.9
MSNE-Maryvonne	km	10.4
Years of Production	years	15.8
Marianne-Marilyn	years	10.0
MSNE-Maryvonne	years	5.8

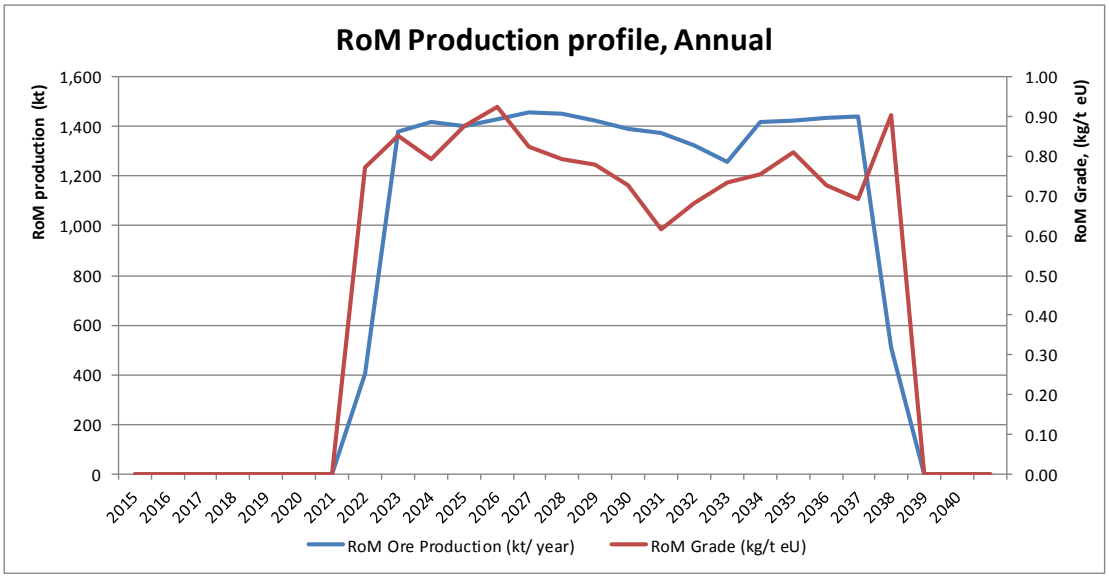


Figure 15-33: Production profile for the mining LoMP

Table 15-24: Life of mine production plan for conventional mining option; M&M and MSNE Maryvonne Combined

Activity	Units	Total	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	2038
Mine Source			M & M											Both	MSNE Maryvonne					
Lateral Dev.	**(m)**	100,017.2	2,609.9	10,929.1	8,230.6	1,800.6	8,366.8	7,748.0	1,794.8	3,667.1	8,918.4	3,367.3	12,075.9	10,766.7	11,307.3	8,435.0				
- Access	(m)	36,039.8	1,814.9	4,069.0	2,627.2		3,247.0	1,607.6		1,612.9	2,291.1	1,343.7	6,226.0	5,128.9	3,921.0	2,150.4				
- Conveyor	(m)	13,329.3	795.0	2,491.5	130.1		1,439.3	561.4		721.5	893.6	571.8	2,406.6	1,662.3	1,599.9	56.1				
- Panel	(m)	50,648.1		4,368.5	5,473.2	1,800.6	3,680.4	5,579.0	1,794.8	1,332.6	5,733.7	1,451.7	3,443.3	3,975.6	5,786.3	6,228.4				
Vertical Dev.	**(m)**	19,309.9		728.9	1,346.9	891.1	498.5	910.1	331.1	164.4	1,720.8	1,505.6	1,591.5	1,766.1	2,145.1	2,158.3	2,158.3	1,393.0		
- Raise	(m)	19,309.9		728.9	1,346.9	891.1	498.5	910.1	331.1	164.4	1,720.8	1,505.6	1,591.5	1,766.1	2,145.1	2,158.3	2,158.3	1,393.0		
- Intake	(m)																			
- Exhaust	(m)																			
Operational Waste Dev.	**(m)**	16,989.2		913.2	1,993.4	1,600.4	2,805.4	1,209.2		187.8	1,519.9	2,348.3	2,610.7	1,512.2	80.9			170.8	37.2	
- R&P	(m)																			
- AR&P	(m)	9,728.6		151.5	971.4	1,153.7	2,543.4	969.6		104.3	989.0	900.1	1,346.1	310.8	80.9			170.8	37.2	
- SR&P	(m)	7,260.6		761.7	1,022.0	446.6	262.0	239.5		83.6	530.9	1,448.2	1,264.6	1,201.4						
Operational Waste Dev.	**(kt)**	492.3		26.5	57.8	46.4	81.3	35.0		5.4	44.0	68.1	75.7	43.8	2.3			4.9	1.1	
- R&P	(kt)																			
- AR&P	(kt)	281.9		4.4	28.2	33.4	73.7	28.1		3.0	28.7	26.1	39.0	9.0	2.3			4.9	1.1	
- SR&P	(kt)	210.4		22.1	29.6	12.9	7.6	6.9		2.4	15.4	42.0	36.6	34.8						
RoM Resue Waste	**(kt)**																			
- R&P	(kt)																			
- AR&P	(kt)																			
- SR&P	(kt)																			
RoM Ore	**(kt)**	21,944.8		401.0	1,381.7	1,419.3	1,404.6	1,429.5	1,458.1	1,451.5	1,423.5	1,391.7	1,374.8	1,324.3	1,258.9	1,415.9	1,426.0	1,433.5	1,442.6	507.8
- R&P	(kt)	14,689.2		71.3	555.4	693.6	45.6	864.9	1,458.1	1,371.5	752.0	458.9	280.2	768.4	1,226.5	1,415.9	1,426.0	1,365.2	1,427.7	507.8
- AR&P	(kt)	4,835.4		75.7	485.7	576.9	1,271.7	484.8		52.1	494.5	450.1	673.1	155.4	32.3			68.3	14.9	
- SR&P	(kt)	2,420.2		253.9	340.7	148.9	87.3	79.8		27.9	177.0	482.7	421.5	400.5						
Ore Grade	**(kg/t eU)**	0.776		0.773	0.853	0.792	0.875	0.924	0.825	0.792	0.778	0.727	0.616	0.683	0.733	0.754	0.810	0.727	0.694	0.902
- R&P	(kg/t eU)	0.772		0.801	0.808	0.717	0.800	0.858	0.825	0.797	0.789	0.792	0.589	0.739	0.730	0.754	0.810	0.734	0.695	0.902
- AR&P	(kg/t eU)	0.816		0.829	0.918	0.865	0.874	1.068		0.713	0.761	0.667	0.625	0.639	0.826			0.600	0.592	
- SR&P	(kg/t eU)	0.722		0.749	0.835	0.857	0.934	0.771		0.726	0.774	0.721	0.620	0.591						

15.6 Ventilation

15.6.1 Introduction

Ventilation is a key consideration in any underground mine. However, for a uranium mine, the release of radon and its progeny introduces an additional consideration for the ventilation design. For this reason, specific attention is given to the ventilation considerations for the Madaouela project.

A preliminary ventilation study of the underground mine designs tests the assumptions and concepts applied to the underground operations in relation to the resulting airflows. This section provides an overview of this study and the full report can be found in Item 24.

15.6.2 Radiation

In addition to the conventional need to dilute diesel exhaust fumes and diesel particulate matter, the uranium content of the deposit introduces radon gas and its progeny into the mine ventilation stream. Because the rate of decay of radon into its radioactive daughter elements ("radon daughters") increases with time, very small quantities of radon gas can become problematic.

Removal of contamination at the source is regarded as the most effective strategy for dealing with radon progeny. If radon daughters are released into the mine atmosphere during operations, the methods currently available are to remove the contaminated air or dilute it to acceptable contamination levels. Effective management of the measurement and control system is critical, especially in working areas located near the mineralised zones. Where possible single-pass ("primary") ventilation should be used to minimise residence time of gases in the mine and limit the exposure of personnel to them.

Radon is highly soluble in water and there is usually a high occurrence of radon in the groundwater located near the orebody.

15.6.3 Legislation

Niger has a long history of underground mining of uranium and has an established regulatory environment for ventilation of radiation in underground mines. SRK has based the mine ventilation design on the requirements of the Niger government standards or on international best practice whichever was the more rigorous.

15.6.4 Monitoring

Management of mine ventilation requires an effective monitoring system. Monitoring can be continuous with fixed installations providing "live" feedback of ventilation parameters, or static requiring regular ventilation surveys and manual monitoring.

Ventilation monitoring should have the capacity to cover the following areas:

- Radiation;
- Airflow velocity;
- Noxious gases (CO, CO_2, etc.);
- Heat (wet and dry bulb);
- Humidity; and
- Air pressure.

Full time ventilation officers to undertake regular surveys throughout the mine and regulate airflows are included in the labour schedule.

15.6.5 Ventilation Controls

The principle means of controlling ventilation in the Madaouela underground deposits is to maximise the use of primary ventilation circuits and minimise the residency time of air underground. To manage air flows, a standard array of ventilation controls will be applied (i.e. brattices, parachutes, ventilation walls, regulators, etc.).

Local conditions where surface temperatures can reach 50°C during the summer, suggest that heat will require special consideration. Due to the shallow nature of the mine and the presence of groundwater, it is anticipated that working conditions underground should be more comfortable than on the surface. However, the possibility for extreme heat on the surface suggests that policies should be implemented to regulate working hours to minimise the risk of heat stroke, in line with Nigerien legislation.

To monitor exposure to radiation over time, workers are required to wear thermoluminescent dosimetry ("TLD") badges. These can be connected to cap lamps or self-rescuers to provide a continuous measurement of radiation exposure for each worker. Should the annual exposure limits be reached, individual workers would be removed from that environment for the remainder of the monitoring year. A similar system is in place at COMINAK's Akouta Mine. All attempts should be made to contain radiation exposure to levels where removal from the environment is not required.

Shotcrete is an effective way to contain radon emission at its source. The shotcrete acts as an airtight seal that can be sprayed on areas emitting high levels of radiation, creating a physical barrier and containing preventing the emission of radon gas.

Alpha radiation is simple to control as it has a low penetration distance. However, significant exposure can be experienced in the event that dust is ingested into the body. For this reason, smoking and eating underground should be discouraged unless in controlled areas, which are regularly cleaned and have facilities for washing hands. Dust control and respirators (in particular airstream helmets) will also play an important role in reducing the workers' exposure to radiation.

15.6.6 Design Parameters

The airflow requirements applied to the ventilation strategy for the underground designs are summarised in Table 15-25 and Table 15-26.

Table 15-25: Airflow requirements by area

Development Profile	Units	
Active Extraction Areas	m^3/s	7.1-27.2
Development Headings	m^3/s	15.0
Miscellaneous Loading	m^3/s	10.4
Ore Bins/Stockpiles	m^3/s	5.0
Active Travelways	m^3/s	15.0-27.2
Storage Areas	m^3/s	5.0
Sumps	m^3/s	3.8-5.0
Other	m^3/s	2.35

Table 15-26: Maximum recommended air velocities by area

Development Profile	Units	
Working Faces	m/s	4
Conveyor Drifts	m/s	5
Main Haulage Routes	m/s	6
Smooth-Lined Main Airways	m/s	8
Ventilation Shafts	m/s	20

In order to achieve the ventilation concept, SRK has applied the following design parameters:

- Access and conveyor development to be raised above the mineralised horizon to minimise the emission of radon gases
- Each panel is to have an independent exhaust raise to maintain positive pressure in the accesses, preventing radiation leakage from the panels into the fresh air drives;
- Regular fresh air raises to be located no more than 1,200 m apart;
- Upon depletion of a panel, shotcrete barricades are to be installed in the panel accesses to seal off the panel to prevent future interaction with the ventilation circuit;
- Fresh air sourced from a conveyor drive and its two parallel access drives can be used to ventilate no more than three active panels (a maximum of 5 panels is possible if fresh air is sourced from both ends of the conveyor and access drives); and
- No more than one active panel can be ventilated in areas where there is a single access drive providing fresh air.

15.6.7 Applied Design

Ventilation design, raise diameter optimisation and fan selection was undertaken using CAE Mining's Ventsim Visual 3 software. The ventilation design criteria discussed previously and general initial assumptions on mining sequence are applied to the Ventsim model. As a target for this evaluation, the maximum total residency time for the air to be underground is targeted at 15 minutes.

During the simulation of the staged airflow requirement, the following assumptions are made:

- Each mining panel requires approximately 125 m^3/s of fresh air during production, provided by a single exhaust fan in each panel;
- Airflow quantity is determined by the maximum instantaneous airflow requirement estimated for the panel; this approach allows single speed fans to be used; and
- Activities with the highest initial priority when designing the ventilation design stages are the installation of the first fresh air intake raise and the construction of the underground workshop; magazine; exhaust raise and fuel storage area.

Development of both the Marianne-Marilyn and MSNE-Maryvonne mines are phased to allow for sequential development of their LoMPs. A ventilation strategy for each phase was developed to confirm the concepts that are applied to the design over time, and to identify areas for further investigation. A detailed breakdown of each phase is provided in Item 24.

15.6.8 Ventilation Conclusions

The results from the iterative analysis of the ventilation design show that the proposed ventilation network can support five panels in production, at least two additional developed panels and multiple access development drives.

The exact volumetric flow into the mine will depend on the number and location of active panels and diesel-powered equipment. The design phases should be considered as conceptual guides to show that ventilation of the mine is achievable for the given ventilation design.

A primary ventilation system is proposed, where contaminated air is exhausted promptly through raises located within the panel developments which minimises the risk of radiation and other contaminants leaking back into the central access development.

Given the low residency times expected, single-pass primary ventilation system and contingency in airflow design the proposed ventilation concept for the Madaouela Project is considered technically feasible.

The approach to ventilation contamination control at the Madaouela underground uranium project should prioritise the removal of radon gas and radon progeny at its source. This is achieved by installing several exhaust raises located within the ore panels, and preventing the recirculation of contaminated air. Secondary ventilation should be used to dilute and direct potentially contaminated air to the exhaust system, and to supply fresh air to development headings.

Further reduction of radon progeny will be achieved by minimising the ingress of groundwater as well as maintaining effective ventilation management and operating practices, and with properly utilised and extensive real-time instrumentation monitoring.

The required airflow rates for each phase of the mine are dependent on the emissions of radon gases and the planned diesel-powered equipment operating at that stage. This study has taken into account the maximum instantaneous airflow requirements for diesel powered equipment, operating sumps and accounted for a ventilation contingency of 25%. Analysis determined a maximum total airflow of 1,320 m^3/s is required. The phased ventilation modelling shows that 882 m^3/s of airflow for Marianne-Marilyn and 1,150 m^3/s for MSNE-Maryvonne is required.

Effective scheduling of panels and development headings will be critical in maintaining acceptable air velocities in haulage drives, conveyor drifts and working areas. Analysis found that each intake raise can only provide fresh air to three active production panels operating concurrently if connected using three drifts (two access, one conveyor), and only one panel can be ventilated if a single access or panel development drift connects to the intake raise.

15.7 Operating Costs

15.7.1 Introduction

Operating costs have been developed from first principles. Where possible, equipment and consumable costs are sourced from suppliers, otherwise data is sourced from GoviEx, and SRK's own databases of similar operations.

A single mining unit cost is applied to both proposed operations as each will use the same mining method and the estimated weighted-average haulage distances are similar for both operations.

Operating costs are estimated for mining plan for inclusion in the TEM. The mining operating costs consist of three elements:

- Lateral development;
- Conveying to the processing plant; and
- Surface rehandling of waste.

15.7.2 Cost Drivers

The inputs used for the estimation of operating costs are provided in Table 15-27.

Table 15-27: Inputs used for operating cost estimation

Development Profile	Units	
Density	t/m^3	2.3
Operating Days per Year		360
Electricity	USD/kWhr	0.204
Diesel	USD/L	1.10
Waste Development per Year	km/a	10.0
Production Development per Year	km/a	51.5

15.7.3 Development Costs

Lateral Development

The cost of lateral development (including production development) is estimated by assessing the development cycle in its individual components and estimating the cost of each. Costs are split into three categories:

- Variable costs;
  - Development directs;
  - Drilling costs;
  - Loading to truck;
  - Loading to stockpile;
  - Truck haulage;
- Fixed costs;
  - Auxiliary equipment;
  - Crushing/feeder breaker; and
- Utilities;
  - Labour;
  - Ventilation;
  - Power; and
  - Dewatering.

Utilities and fixed costs are allocated to both development and production mining. The major difference between the key input parameters for the development and production costs is the average haulage distance; as the development areas are more likely to be remote from the central conveyor network a longer haulage distance has been applied.

A summary of the estimated operating costs for lateral development for the Madaouela underground operations are provided in Table 15-28.

Table 15-28: Operating cost for lateral development

Development Profile	Units	
Conventional Mining		
Decline/Access	USD/m	1,001
Conveyor	USD/m	792
Panel	USD/m	1,060
Production	USD/t_{RoM}	28.54

SRK notes that unit costs above include labour. When the unit costs are applied in the TEM, labour costs are calculated independently to reflect production rates, specifically production ramp up and down.

Annual waste development can vary considerably each year and 10 km per year is applied to the mining cost estimation to represent the maximum waste development at full production. SRK notes that less waste development per year will change allocations of fixed operating costs.

The three different types of room and pillar methods are proposed to mine the Madaouela deposits according to the dip of the Talak-Guezouman contact and mineralisation. Fundamentally, each method is similar and for this reason, the same unit cost has been applied to all three methods, although in practice it is expected that productivity will be lower and maintenance costs of mobile equipment will be higher where zones of steeper dip are being mined.

To account for the increasing difficulty of mining the steeper regions of the deposits, it is assumed that additional operational waste development is required to establish the required access and orientation for mining. This assumption is where the additional costs are incurred in the cost estimate. The assumed operational waste requirements have been previously outlined in Table 15-22.

Vertical Development

Operating costs for vertical development are derived on a unit cost basis. The operating costs comprise three categories:

- Equipment costs;
- Consumables; and
- Power.

Estimates of unit cost at different raise lengths were made from which a typical unit cost of USD1,540 /m was derived. Labour is excluded from this unit cost as it is applied separately in the technical-economic model.

15.7.4 Labour

Labour is applied in the TEM relative to mining activity. The labour schedule is linked to the equipment schedule to ensure that sufficient operators are allowed for to meet production requirements.

Labour figures are based on a simplification of GoviEx's existing pay structure (Table 15-54), which uses a sliding scale to account for years of experience and years with the company.

Table 15-29: Labour unit costs

Development Profile	Units	
Senior Engineers	USD/month	10,000
Engineers/Senior Technical	USD/month	5,500
Technical	USD/month	3,000
Skilled	USD/month	1,000
Unskilled	USD/month	600

General and administrative roles for the mine, such as mine manager, secretaries, etc. are included in the general and administrative costs applied to mining.

15.7.5 Conveying

Conveying costs are based on a cost analysis undertaken by Sandvik Mining Systems for Madaouela and includes a power consumption based on production rate. The conveyor design has been extended for the MSNE-Maryvonne Mine with the additional conveyors benchmarked against the Sandvik designs.

Table 15-30: Overview of conveyor network

Branch	Make	Belt Length (m)	Lift (m)	Mechanical (USD/month)	Power (USD/t)
Marianne-Marilyn					
Decline	Sandvik	820	115	7,503	0.0891
Central	Sandvik	1,250	55	9,063	0.0671
West 1	Sandvik	825	35	6,291	0.0499
West 1	Sandvik	885	20	5,399	0.0440
Northwest 1	Sandvik	1,100	40	6,900	0.0968
Northwest 2	Sandvik	585	40	4,388	0.0294
East 1	Sandvik	1,200	20	6,960	0.0494
East 2	Sandvik	900	10	5,400	0.0392
MSNE-Maryvonne					
Decline	Sandvik	740	105	6,690	0.0827
Central	Sandvik	990	20	5,940	0.0456
West 1	Sandvik	1,000	25	5,800	0.0483
West 2	Sandvik	1,090	10	6,322	0.0424
South	Sandvik	500	10	3,100	0.0327
North[1]	Sandvik	1,230	35	9,063	0.0671
Maryvonne[2]	Sandvik	900	15	5,940	0.0456
Overland	Sandvik	3,250	0	16,250	0.1052

[1]Based on Marianne-Marilyn central conveyor

[2]Based on MSNE-Maryvonne central conveyor

15.7.6 Surface Waste Rehandling

As waste is brought to the surface it will be side-tipped to stockpile from the conveyor near the portal. The waste is rehandled and hauled to a surface waste stockpile to be located within 1 km of the loading point. No specific designs for the waste stockpile have been undertaken.

The estimate for surface waste rehandling follows the same principles that are applied to the open pit materials handling cost estimation. As the waste is likely to be brought to the surface on a campaign basis, surface rehandling will not be required on a continuous basis. A number of fleet sizes and shift patterns were considered: the preferred option has an operating cost of USD 0.73/t of waste moved. There will be opportunity to dispose of some waste material underground during mining so this approach is considered conservative.

It is assumed that when the surface rehandling fleet is not moving waste, they will assist with RoM pad and stockpile management at the processing plant, roadworks and other miscellaneous tasks around the mine site.

Operating costs are calculated in the TEM on the basis of unit costs and production schedule. Labour costs vary over time as requirements change with the ramp up and down of the project and link in with the equipment schedule. Conveyor costs are calculated from a fixed and variable component, based on work undertaken by Sandvik Mining Systems.

Calculation of the production development costs and the unit costs for the other development profiles (allocated as capital costs) are integrated to prevent double counting of utilities and other fixed items when the unit costs are applied.

SRKs estimate for the total mining costs (including conveying, labour and surface waste rehandling) at full production is USD 28.6 /t_{RoM} at M&M and USD 28.5 /t_{RoM} at MNSE-Maryvonne.

15.8 Capital Costs

15.8.1 Introduction

Underground mining capital is split into four categories:

- Equipment;
- Infrastructure;
- Development; and
- Miscellaneous.

Where possible, estimates for major capital items are based on information sourced from suppliers.

15.8.2 Equipment

An equipment schedule is produced to reflect the monthly schedule and estimate the build-up of the fleet over time. The schedule is based on the monthly production schedule and allows for replacing old equipment.

The replacement profile assumes that equipment will have a relatively long mine life due to the soft rock conditions and relatively small water inflow. The assumed equipment lives are:

- Surface equipment - 8 years;
- Surface fans - 20 years;
- Fan motors - 10 years;
- Underground fleet (high utilisation) - 6 years;

- Underground fleet (low utilisation) - 8 years;
- Small equipment - 4 years;
- Major underground infrastructure - 12 years;
- Raiseborers - 12 years.

A summary of the capital cost inputs for the underground equipment are listed in Table 15-31.

Table 15-31: Equipment capital unit costs

Equipment	Model	Unit Capital (USD)
Loader - Surface	Caterpillar 725	872,700
Loader - Development	Atlas Copco Scooptram ST1030	717,000
Loader - Production	Atlas Copco Scooptram ST7	724,500
Truck - Surface	Caterpillar 962	776,300
Truck - Underground	Atlas Copco Minetruck MT436LP	1,041,800
Drill Jumbo - Development	Atlas Copco Boomer 282	993,500
Drill Jumbo - Production	Atlas Copco Boomer S1	683,400
Rockbolter - Development	Atlas Copco Boltec 235	836,900
Rockbolter - Production	Atlas Copco Boltec SL	793,800
ANFO Loader - Development	Normet Charmec 605 DA	418,000
ANFO Loader - Production	Light vehicle w/ portable kettle	99,000
Shotcreter	Normet Spraymec 6050 WP	495,000
Transmixer	Normet Ultimec LF500	297,000
Toolcarrier/Multipurpose Vehicle	Wheel loader w/ platform	99,000
Grader	Caterpillar 120H	269,000
Light Vehicle	Toyota LC70	68,600
Service Vehicle	Toyota LC70	68,600
Stores Truck	Flat-bed truck	281,000
Raiseborer	Robbins 91RH C	6,710,000
Airleg	-	10,000
Feeder Breaker	Joy UFB-14B	880,000
Radiation Monitor	Trekkopje	1,200,000
Surface Fan	Clemcorp L52C	138,200
Fan Motor	Variable speed motor	34,800
Auxiliary Fan	Twin stage 55 kW	25,000
Main Pump	Mono SE103MS1R8/H474	73,200
Secondary Pump	9 kW submersible pump	6,900

[1]Includes additional 10% for transport to site

15.8.3 Infrastructure

Much of the installed infrastructure (i.e. fans, pumps, etc.) are incorporated into the equipment schedule. The main infrastructure cost is the conveyors used for materials handling. A breakdown of the conveyor costs provided by Sandvik Mining Systems is provided in Table 15-32.

Table 15-32: Conveyor capital estimates for new conveyor

Branch	Make	Belt Length (m)	Capital Cost (USD)
Marianne-Marilyn			
Decline	Sandvik	820	4,622,000
Central	Sandvik	1,250	6,699,000
West 1	Sandvik	825	4,650,000
West 1	Sandvik	885	4,988,000
Northwest 1	Sandvik	1,100	5,895,000
Northwest 2	Sandvik	585	771,000
East 1	Sandvik	1,200	6,431,000
East 2	Sandvik	900	5,073,000
MSNE-Maryvonne			
Decline	Sandvik	740	4,103,000
Central	Sandvik	990	5,489,000
West 1	Sandvik	1,000	5,359,000
West 2	Sandvik	1,090	5,842,000
South	Sandvik	500	2,864,000
Overland	Sandvik	3,250	13,065,000
North	Sandvik	1,230	6,699,000
Maryvonne	Sandvik	900	5,100,000

[1]Includes additional 10% for transport to site

Whilst Sandvik has provided estimates for new conveyors, there is an opportunity to recycle some of the conveyors between mines. Where this is possible, only the erection costs are used.

Additional infrastructure includes the installations required for the underground fuel bay, magazine and small workshop for minor maintenance. An nominal allowance of USD 250,000 is provided for this. Construction of the portal is forecast to cost USD 500,000. No equipment purchase has been allowed for this as it is assumed that local contractors be brought in for the earthmoving required.

15.8.4 Development

The capital development for the underground mining includes all development with a profile of:

- Access;
- Conveyor;
- Panel; and
- Vertical.

The unit costs of development are discussed in Section 15.7.3 with the total lengths of development shown in Section 15.5.11.

Total capital development requirements applied to the technical-economic model are:

- Mining Capital;
  - Marianne-Marilyn - USD 64.5. million;
  - MSNE-Maryvonne - USD 53.5 million;

15.8.5 Miscellaneous

An allowance is made for miscellaneous equipment requirements. For the most part, these include smaller pieces of equipment and infrastructure vital for the success of the project, however, with insufficient impact on the overall costs to be engineered in detail for current investigations. In total, however, they add up to a material figure.

The allowance for Miscellaneous capital includes a consideration for the following elements:

- Pump station infrastructure (excluding pumps);
- Lunch room;
- Communication systems;
- Compressed air supply;
- Power substations and distribution boxes;
- Underground lighting;
- Survey equipment;
- Ventilation monitoring equipment;
- Cap lamp, safety equipment, self-rescuers and overalls;
- Escapeway ladders; and
- Miscellaneous worker tools.

The total allowed for these items collectively are USD 3.1 million.

15.8.6 Estimated Capital

The final capital schedule for underground mining is dependent on the option selected for the TEM. The differences are primarily based on adjustments to the capital development design (i.e. higher cut-off grades may make some panels uneconomic) and the ability to recycle conveyors in the various scenarios. A summary of the estimated capital costs for underground mining is provided in Table 15-33.

Table 15-33: Estimated capital costs for underground mining

Branch	Capital in 1st 3 Years of Underground Mining (USDm)	Total Capital Cost in LoMP (USDm)
Conventional Mining		
Marianne-Marilyn	92.5	181.5
MSNE-Maryvonne	0.0	120.8
Total	92.5	302.3

15.9 Conclusions

The combined schedule for both deposits provide a total mine life of 18 years with ore production over 17 years of 21.9 Mt at 0.78 kg/t eU by utilising Low Profile mining methods.

Mine production commences from the Marianne-Marilyn deposit then moves to MSNE-Maryvonne.

Underground mining of the Marianne-Marilyn and MSNE-Maryvonne is demonstrated to be technically viable applying three different room and pillar methods; traditional, apparent and step. The application of each mining method is dependent on the dip of the footwall contact between the Guezouman Sandstone and Talak Shale. Their depth below the surface, vein thickness and orientation precludes them from surface mining options.

Access to the mine is to be via a three-drive decline (at a gradient of 1:7), allowing a central conveyor and access to panels either side. The decline will be driven from a boxcut portal.

A drill and blast for the extraction methodology is proposed as the rock is believed to be too abrasive for mechanical excavation. ANFO is applied as the bulk explosive due to the lower unit costs, the relatively dry host rock, the advantages of additional heave in the softer rock and the ability to transport as a chemical provides upside potential for transportation and storage costs. Packaged emulsion is assumed for primers, lifters and any wet holes that may be encountered.

Raiseboring is assumed for vertical development in the Madaouela deposits using owner operated equipment. The speed of advance is expected to be high due to the soft nature of the rock and the continuous nature of the method.

The proposed material handling system for both ore and waste is as follows:

- Low-profile loaders from the face to panel stockpiles;
- Rehandling by larger loader into trucks;
- Truck haulage to feeder breaker;
- Rock is crushed and transferred onto central conveyor network;
- Conveyed to the surface;
- Ore is delivered to the RoM stockpile (RoM); and
- Waste is side tipped from where it is loaded into trucks for haulage to waste dumps.

A conventional bolting pattern using resin bolts is the preferred ground support methodology in conjunction with a well-designed room and pillar layout. Shotcreting equipment is to be available, however is not incorporated into the ground control patterns. No recovery of pillars is assumed so backfill is not required.

The equipment applied to the underground mining is broadly split into conventional and low-profile equipment. Selection of the low-profile equipment is a trade-off between productivity and minimum mining height. The aim being to minimise the amount of waste that needs to be extracted with the mineralised horizon, and which will be sent to the processing plant as RoM. Equipment selection therefore plays an integral part in the economics of mining at Madaouela. Taller production equipment lowers RoM grade (to which the cut-off grade must be applied) and increases the volume of material to be mined, transported and processed with associated costs.

Labour for the proposed underground operations is assumed to be sourced entirely from Nigerien sources. Allowances have been made for local conditions and customs, though

labour productivity is still assumed to be higher than the nearby Akouta Mine. This is a function of more modern equipment and mining practices including high ventilating volumes, non-return of air from working places and use of shotcrete to mitigate exposure to radon.

Two schedules are produced for underground mining, one for Marianne-Marilyn and the other for MSNE-Maryvonne. This enables a complete evaluation for the underground mine in the technical economic model ("TEM") and a sensitivity of applying different cut-off grades.

15.10 Miriam Open Pit

15.10.1 Introduction

An open pit study was conducted on the Miriam deposit including the following:

- Pit optimisation to determine the optimal pit shape to carry forward to design;
- Mine design of the selected pit shell and waste dump designs;
- Life of mine schedule;
- Equipment and labour requirements; and
- Cost estimate.

15.10.2 Site Layout

The deposits which are part of the Madaouela project are presented in Figure 15-34, together with the exploration license areas.

The Miriam deposit, which will be extracted with an open pit, is located 12 km to the south of the Marianne-Marilyn deposit where the processing plant is to be located.

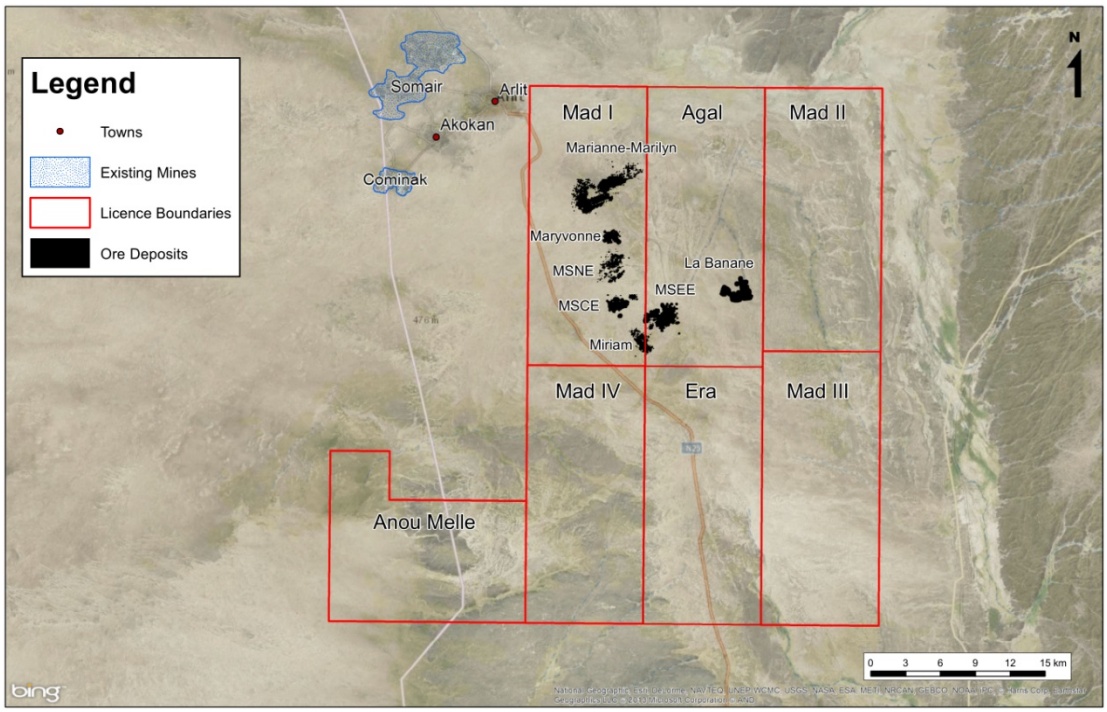


Figure 15-34: Location of Madaouela Deposits and Licenses

GoviEx has 100% ownership of Mad I, II, III and IV, and applications filed November 22, 2012 for Agal and Eral are pending. GoviEx has been advised by senior government officials that Agal and Eral can expect the applications to be approved.

15.10.3 Block Model

The geologic block model used in the study was created by SRK in April, 2013. The Miriam block model is a uniform conditioning ("UC") model where the selection of a cut-off grade ("CoG") is required before undertaking a pit optimisation to determine the amount of mineralisation available at a specified CoG, which is expected to be mined at the selective size of selective mining unit ("SMU"). Due to the drillhole spacing the resource model block size has been kept at 25 m x 25 m. The uranium grades in the model are quoted as eU, as the grades have mostly been determined by gamma ray probes and not by direct chemical analysis.

The Miriam block model parameters are shown in Table 15-34.

Table 15-34: Miriam Block Model Framework

Framework	Units	X	Y	Z
Origin (centroid)	**m**	343251.8	2050388.4	300.0
Block Size	**m**	25.0	25.0	0.4
Number of Blocks	**#**	75	125	426
SMU (grade estimation)	**m**	5	5	0.4
Rotation				47°

The following block model fields are present in the original model:

- X,Y,Z: Coordinates of the block centroids;
- Tons_xx: Tonnes at cut-off grades of 0, 0.2, 0.3, 0.4, 0.6, 0.8, 1.0, 1.2, and 1.4 (kg/t U);
- Grade_xx: Corresponding grades (kg/t U) ;
- Density: Constant at 2.3 t/m^3; and
- Classification: 1 - Measured, 2 - Indicated, 3 – Inferred.

Table 15-35 presents the quality and quantity of all material as estimated within the block model in relation to the UC CoG. This data is shown graphically in Figure 15-35.

Table 15-35: Block Model Quantities at Specific Cut-off Grades

Cut-off (kg/t U)	Metal (t U)	Tonnes (t)	Grade (kg/t U)
0.0	16,792	35,974,300	0.467
0.2	15,634	28,071,121	0.557
0.3	13,388	19,016,993	0.704
0.4	11,265	12,880,566	0.875
0.6	8,308	6,780,411	1.225
0.8	6,496	4,144,500	1.567
1.0	5,289	2,789,028	1.896
1.2	4,428	1,999,704	2.214
1.4	3,781	1,499,282	2.522

The sensitivity analysis indicates that the resource quantities at Miriam are more sensitive to the CoG whereas the grade is less sensitive when varying the CoG in the range of 0.2 to 0.4 kg/t. The grade-tonnage curve for the block model is shown in Figure 15-35.

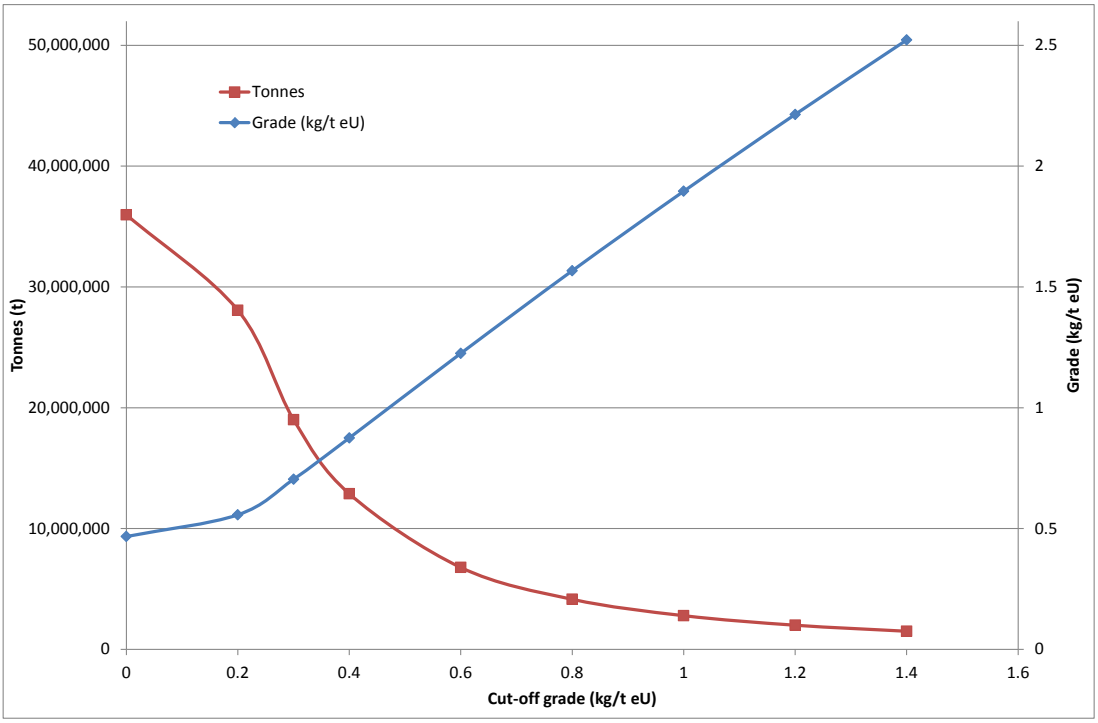


Figure 15-35: Miriam Block Model Grade Tonnage Curve

15.10.4 Pit Optimisation

Approach

The pit optimisation was carried out on the Miriam deposit using CAE's NPV Scheduler pit optimisation software ("NPVS"). NPVS uses the Lerchs-Grossmann algorithm for determining the optimal shape for an open pit to be used as a basis for design.

For the purposes of this study, SRK has considered that only Measured and Indicated classified Resources may contribute revenue in the optimisations. The classification has been carried through to the mining schedule and the economic analysis. The pit optimisation results form the basis of the mine design and schedule.

Pit Optimisation Parameters

The input parameters for pit optimisation are shown in Table 15-36. The Miriam block model has several CoGs available which allow optimisations at different cut-offs. Although the marginal CoG for the stated parameters is 0.28 kg/t U, the CoG applied to the model in order to run the optimisation was chosen by GoviEx at 0.4 kg/t U.

The optimisation was undertaken excluding any mineralisation located within the Agal licence (Figure 15-34) based on a provisional set of operating parameters and costs.

Table 15-36: Open Pit Optimisation Parameters

Parameters	Units	Value
Production		
Ore Production Rate	(ktpa)	1,175
Geotechnical		
Footwall Slope Angle	(°)	51
Hangingwall Slope Angle	(°)	51
Mining Factors		
Dilution	(%)	5.0
Recovery	(%)	98.0
Processing		
Recovery U	(%)	83.0
Cost Inputs		
Mining Cost	(USD/t_{rock})	2.14
Ore Transport Cost	(USD/t RoM)	2.21
Ore Re-handling Cost	(USD/t RoM)	0.16
Processing Cost	(USD/t RoM)	25.41
Infrastructure	(USD/t RoM)	0.59
Tailings	(USD/t RoM)	0.74
General and Administrative	(USD/t RoM)	3.56
Selling Costs	(%)	5.5
	(USD/kg U_3O_8)	8.47
Metal Price		
Uranium oxide	(USD/kg U_3O_8)	154
Cut-Off Grade		
Operating Costs	(USD/t RoM)	32.67
Marginal Cut-off Grade	(kg/t U)	0.28

Pit Optimisation Results

The results of the pit optimisation are shown in Figure 15-36, which shows the selling price sensitivity for the input parameters.

The 82% revenue factor (126 USD/kg U_3O_8) pit shell has been selected for final design and scheduling for Miriam. Table 15-37 shows summary results for the selected pit shell. The selected pit shell is shown in Figure 15-37 along with the mineralised blocks.

The selection of the optimal pit shell is based on following criteria:

- Maintaining average feed grade above 0.8 kg/t U;
- Optimising pushback selection with practical design criteria; and
- Minimum mine life of 6 years for optimum equipment utilisation.

An updated processing recovery was obtained subsequent to finalising the pit optimisation. A pit optimisation was undertaken with the updated processing recovery of 93.7% to ascertain the upside potential for future studies. The optimisation results indicate that the pit shell at the same revenue factor (82%) would have an increase in ore tonnage of 13% and an increase in U content of 10%.

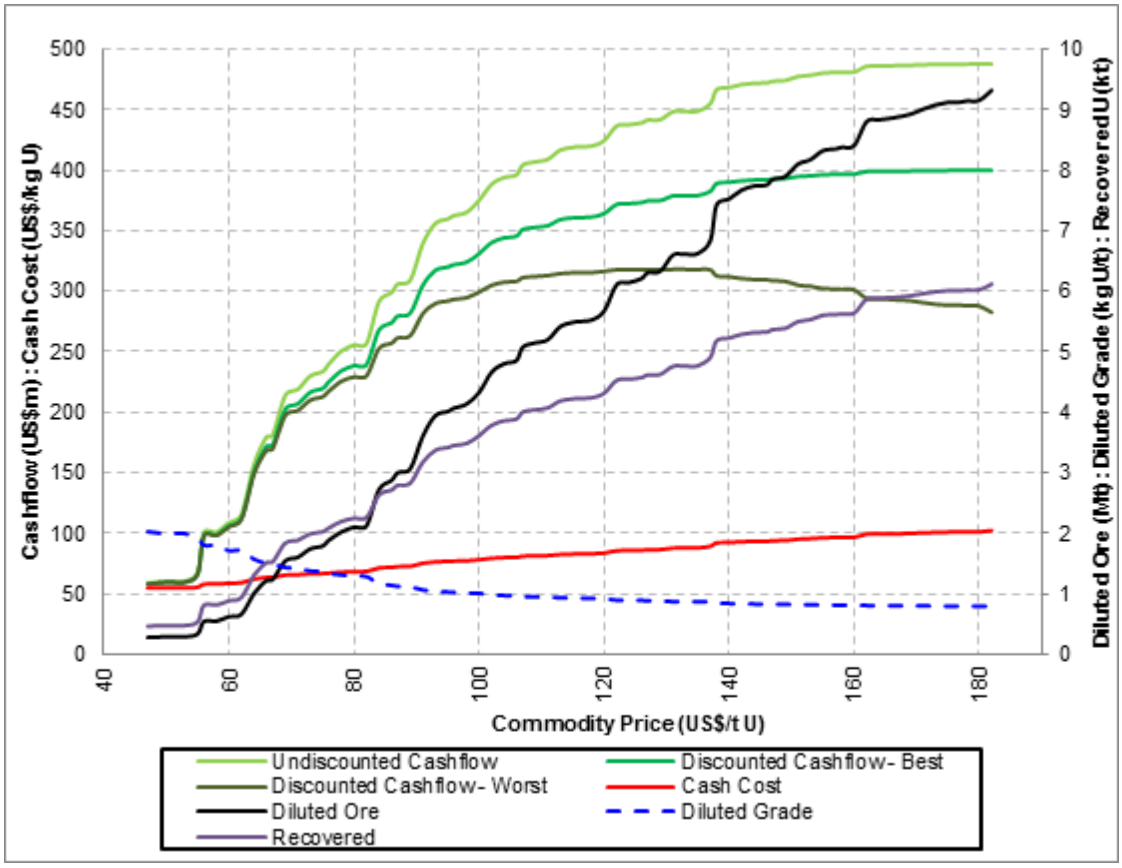


Figure 15-36: Miriam Pit Optimisation Results

Table 15-37: Selected Pit Shell Results

Optimisation Results	Units	Value
In-situ		
Inventory	(Mt)	7.7
	(kg/t U)	0.86
	(t U)	6,624
Modifying Factors		
Mining Dilution	(%)	5.0
Dilutant	(kg/t U)	0.0
Mining Recovery	(%)	98.0
U Metallurgical Recovery	(%)	83.0
Diluted		
Inventory	(Mt)	7.9
	(kg/t U)	0.82
	(t U)	6,492
Quantities		
Total Rock	(Mt)	90.8
Mineral Inventory	(Mt)	7.9
Waste	(Mt)	82.9
Stripping Ratio	(t:t)	10.5
Operating Expenditures		
Mining	(USD/t_{mined})	2.14
	(USD/t_{ore})	24.60
	(USD/kg U)	36
Processing and Other	(USD/t_{ore})	32.67
	(USD/kg U)	48
U_30_8 Selling Cost	(USD/kg U)	10
Total Cash Cost	(USD/kg U)	94
Product		
Recovered Metal	(t U)	5,388
Economic Summary		
Metal Price	(USD/t U_3O_8)	154
Revenue	(USDm)	981
Mining Costs	(USDm)	194
Processing and Other Costs	(USDm)	258
Selling Costs	(USDm)	54
Other Costs	(USDm)	
Cashflow	(USDm)	474
Discount Rate	(%)	10.0
Mill Rate	(Mtpa)	1.2
Discounted Cashflow - Best Case	(USDm)	393
Discounted Cashflow - Worst Case	(USDm)	308
Project Life	(years)	6.7

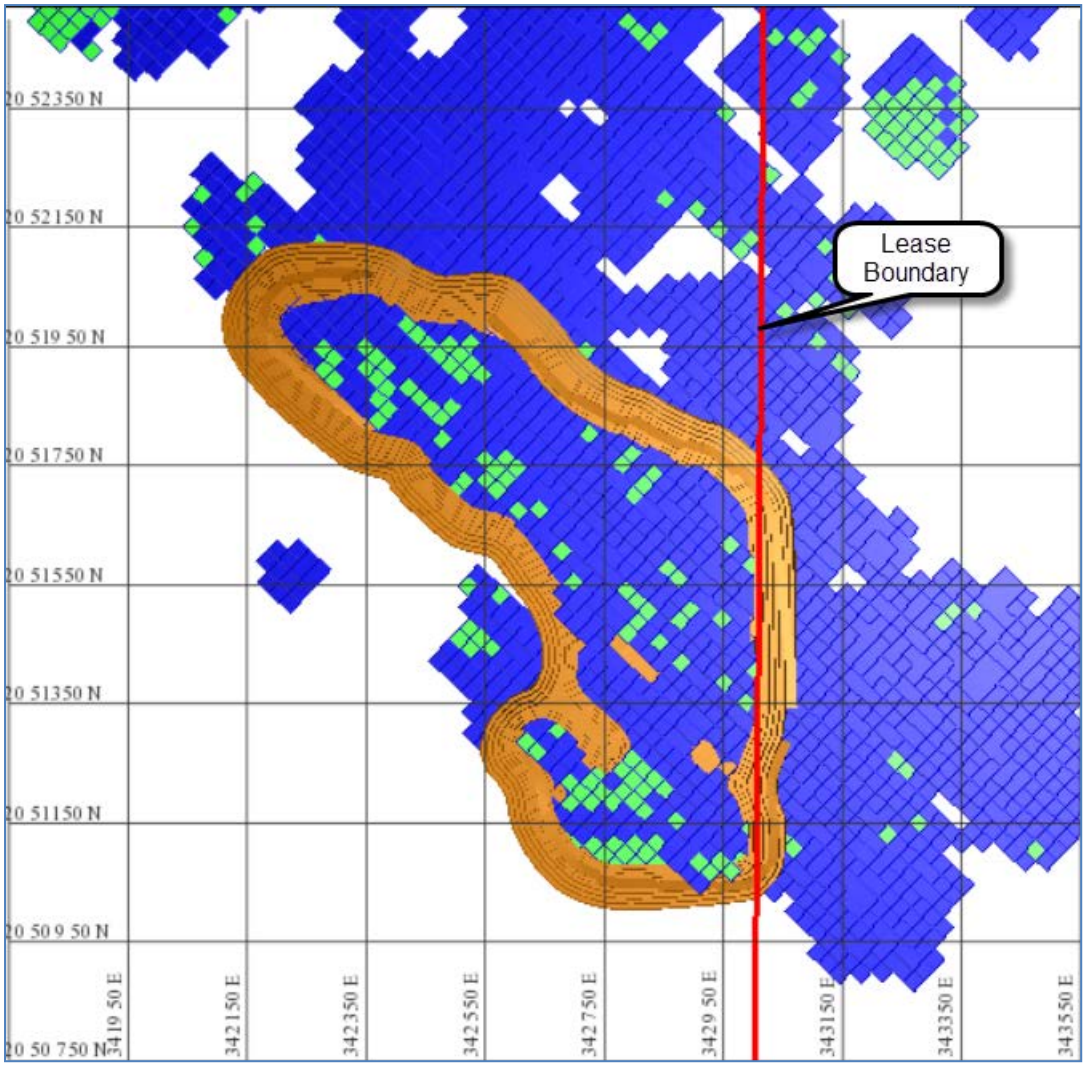


Figure 15-37: Miriam Selected USD 149/t U Pit Shell (blue: 0.4>U<=0.6 kg/t, green: U>0.6 kg/t)

15.10.5 Mine Design

Pit Design

The engineered final pit design for the Miriam deposit is based on the selected pit shell (82% revenue factor). Where possible, ramps are located where practical within the constraints of the location of the waste dumps and run of mine ("RoM") stockpile.

The open pit design parameters are summarised in Table 15-38.

Table 15-38: Open Pit Design Parameters

Parameter	Units	Value
Pit Depth ≤ 40 m		
Inter-Ramp Slope Angle	(°)	47
Face Angle	(°)	75
Berm Width	(m)	6.5
Bench Height	(m)	10
Pit Depth > 40 m		
Inter-Ramp Slope Angle	(°)	54
Face Angle	(°)	85
Berm Width	(m)	6.5
Bench Height	9m)	10
Bench and Road		
Ramp Width – Dual Lane	(m)	17
Ramp Width – Single Lane	(m)	12
Ramp Gradient	(m_{vert}/m_{hor})	1:10
Turning Circle	(m)	33
Minimum Mining Width	(m)	30

The ramp parameters are based on the characteristics of a Caterpillar (“CAT”) 772G haul truck (44 t capacity). Single lane ramps have been designed in the lower benches in order to access additional mineralisation at the pit bottom. No topographical or environmental constraints have been considered when engineering the final pit design.

The RoM quantities within the final pit design are shown in Table 15-39. RoM has been separated into high grade (“HG”) and low grade (“LG”). The mining modification factors applied are 2% mining loss and 5% dilution at 0 kg/t grade. The final Miriam pit design (Figure 15-38) is approximately 1,400 km long, 550 m wide and reaches a maximum depth of 120 m.

Table 15-39: Miriam Final Pit Design Quantities

Material	Tonnage (Mt)	Grade (kg/t eU)
High Grade (>0.6 kg/t U)	5.2	0.98
Low Grade (0.4 ≥ kg/t U ≤ 0.6)	2.3	0.49
Total RoM	**7.5**	**0.82**
Waste	91.8	
Total Movement	99.4	
Strip Ratio (t_{waste}/t_{ore})	12.2	

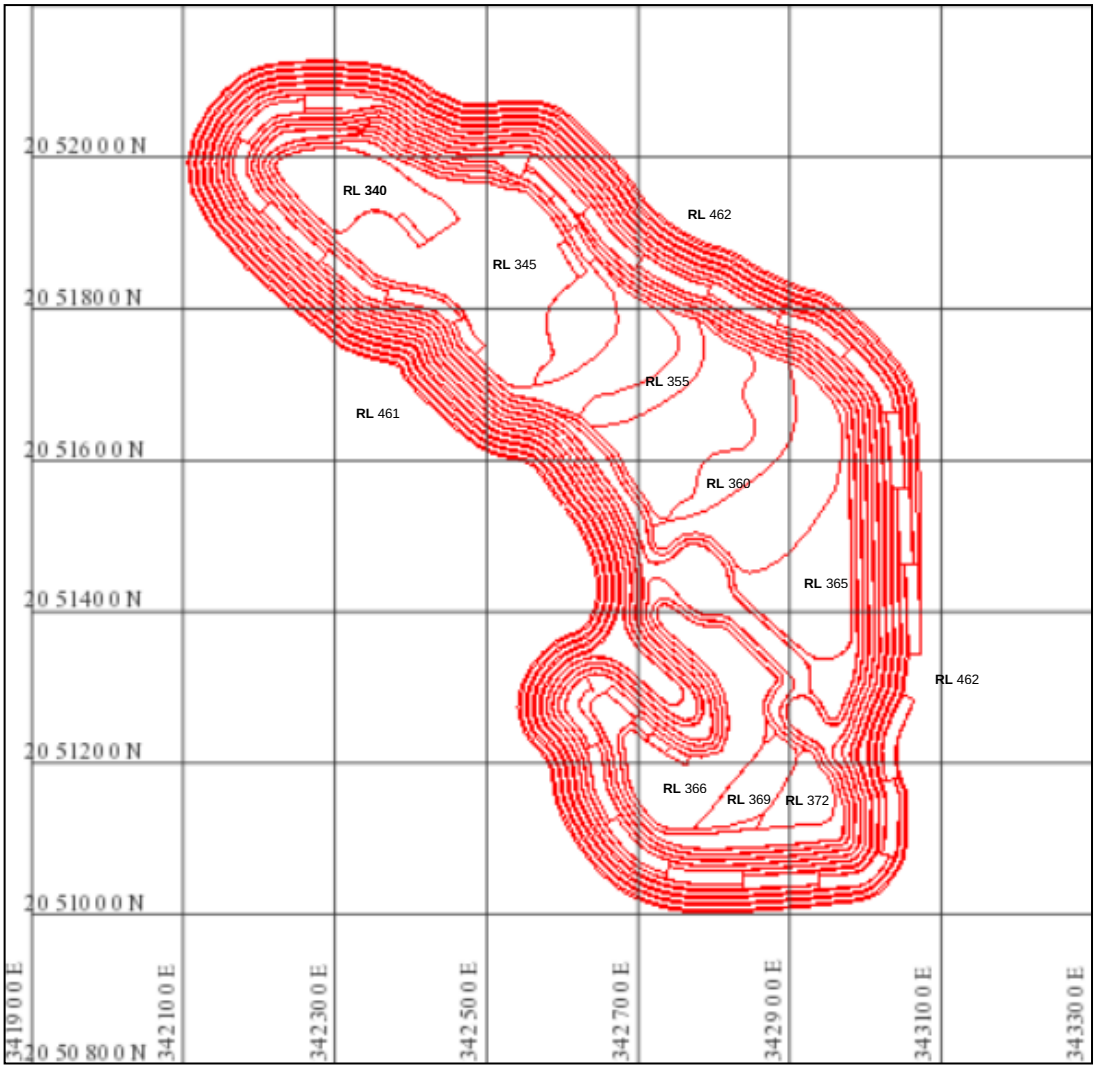


Figure 15-38: Miriam Final Pit Design

A comparison between the selected optimised shell and the final pit design is shown in Table 15-40. SRK considers that the differences are within acceptable limits. SRK would recommend that future work take into consideration the location of the final and pushback ramps when evaluating overall slope angles for pit optimisation work. This may decrease the differences in quantity between the optimised shell and final pit design.

Table 15-40: Comparison between Selected Pit Shell and Final Pit Design

Parameter	Units	Optimised Shell	Final Pit Design	Difference	Difference (%)
Total Rock	(Mt)	90.8	99.3	8.5	9.4
Waste	(Mt)	82.9	91.8	8.9	10.8
Ore	(Mt)	7.9	7.5	-0.4	-5.0
	(kg/t)	0.82	0.82	0.00	0.4
Strip Ratio	(t:t)	10.5	12.2	1.7	16.5

The final designed pit has been divided into three pushbacks each to be developed with independent ramp access and with similar bench-berm configurations (Figure 15-39). The third pushback is the final pit design. Three pushbacks were created based on the following criteria:

- Maintaining minimum surface area for ramp development for each pushback;
- Maintain consistent RoM production as targeted; and
- Minimum mining width of 25 m between pushbacks.

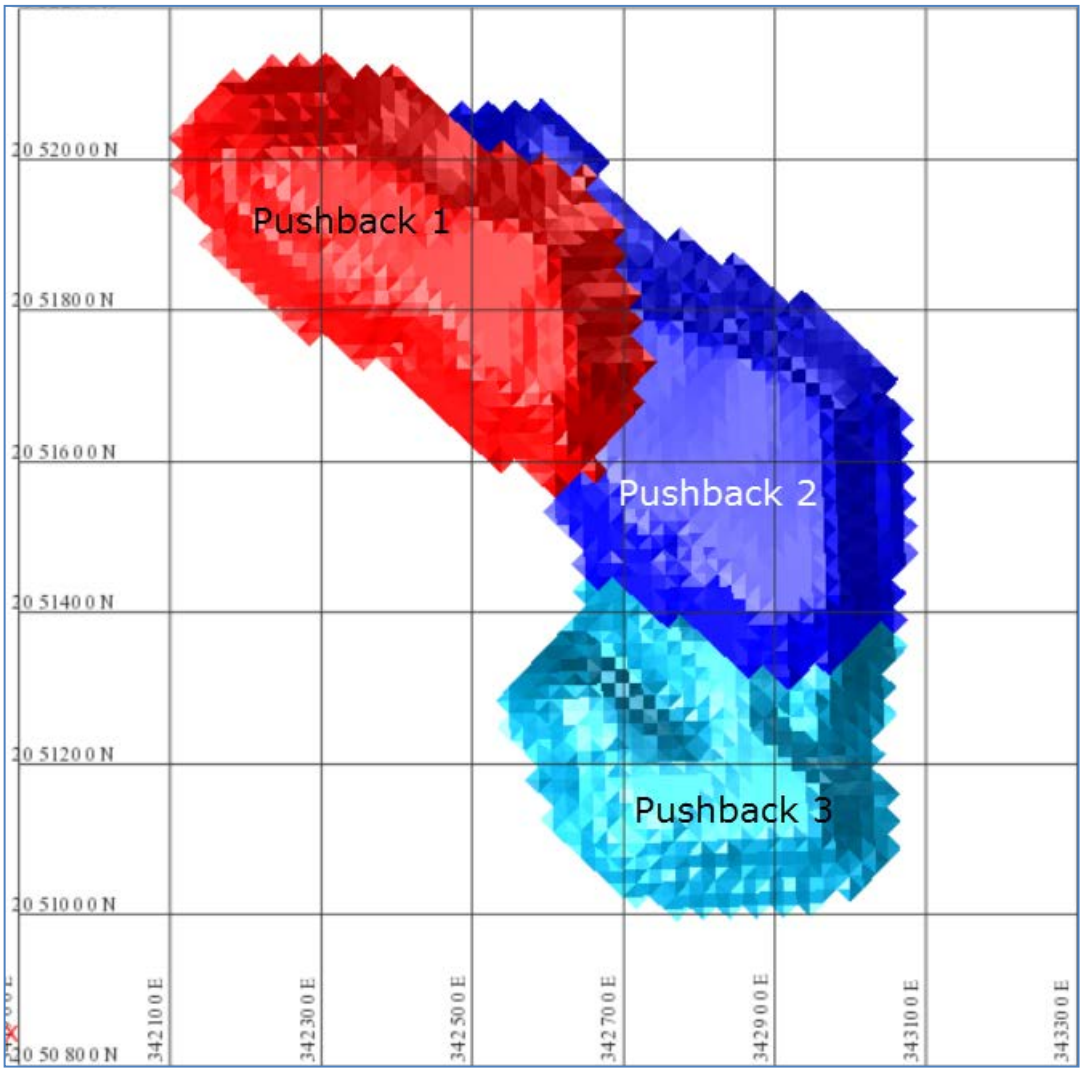


Figure 15-39: Miriam Open Pit Pushbacks

Waste Rock Dump Design

Three waste rock dumps ("WRD") have been planned with a combined designed capacity of 57.5 M loose cubic meters ("lcm") assuming a swell factor of 30%. The design criteria used to create the waste dumps are provided in Table 15-41.

Table 15-41: Waste Rock Dump Design Parameters

Parameter	Units	Value
Maximum Rehabilitation Angle	(°)	26
Lift Height	(m)	10
Maximum Height	(m)	60
Rill Angle	(°)	35
Berm Width	(m)	10
Ramp Width	(m)	17
Ramp Grade	(m_{vert}/m_{hor})	1:10

To prevent sterilisation of potential mineralisation, for potential increases in uranium price, an exclusion zone for waste storage was established surrounding the final pit design. An optimised shell corresponding to the optimised pit shell based on a revenue factor of 2.0 (equivalent to a selling price of USD 364/kg U or USD 140/lb U_3O_8) has been used for the exclusion zone. Figure 15-40 shows the locations of the waste rock dumps along with the final pit design and the exclusion zone.

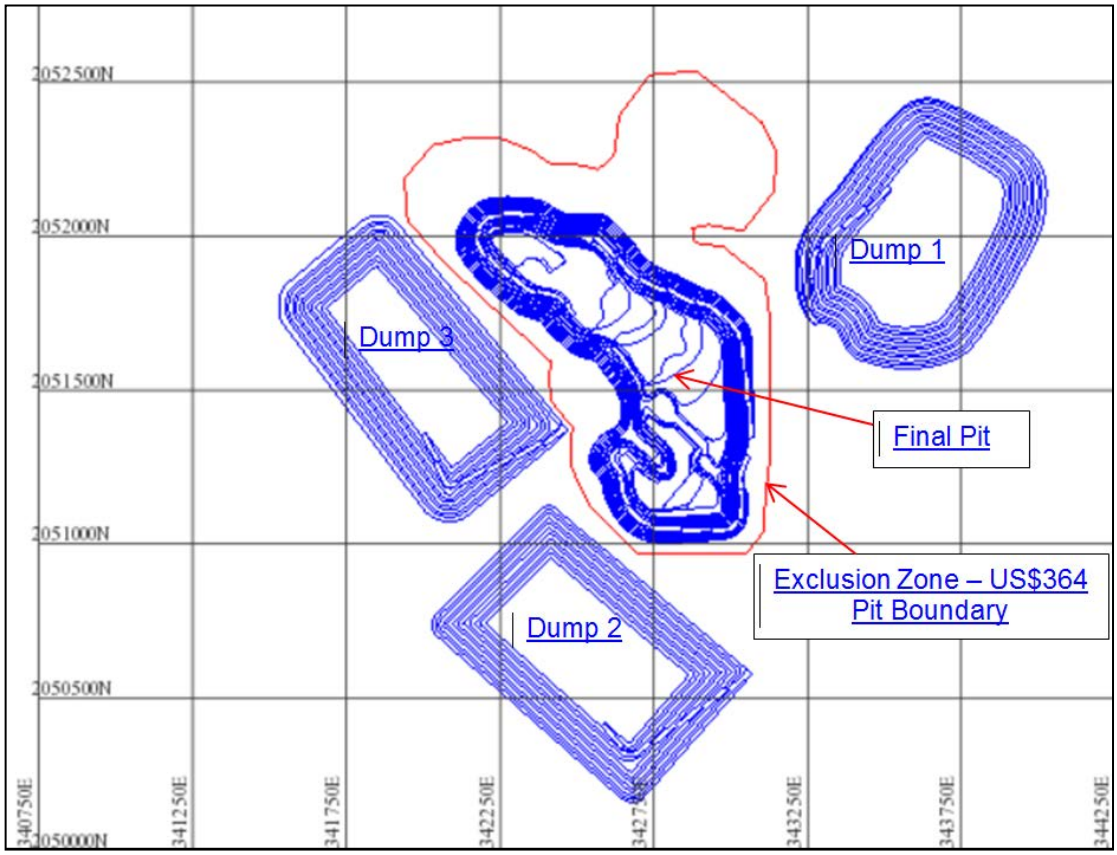


Figure 15-40: Miriam Waste Dump Locations and Designs

The designed capacities of the waste dumps are provided in Table 15-42.

Table 15-42: Miriam Waste Rock Dump Capacities

Waste Dump	Units	Value
Dump 1	(M lcm)	19.2
Dump 2	(M lcm)	19.2
Dump 3	(M lcm)	19.1
Total	**(M lcm)**	**57.5**

15.10.6 Life of Mine Schedule

Approach

The open pit life of mine ("LoM") plan is based on the final pit design and the three pushback pit shells. The LoM schedules were created in CAE's NPVS.

Scheduling Parameters

The key scheduling parameters were as follows:

- Plant feed target of 1,400 ktpa;
- Limit total material movement ex-pit to 19.5 Mtpa;
- Two years of pre-production waste stripping;
- Decrease material movement in year -2 for mine start-up; and
- Limit vertical advancement to 60 m per year.

Schedule Results

The HG is classified as mineralisation above 0.6 kg/t U, while LG is between 0.4 kg/t U and 0.6 kg/t U. Figure 15-41 and Figure 15-42 shows total material movement and plant feed schedule respectively.

The main features of the mine schedule are summarised as:

- There are two years of pre-production required prior to mining RoM and hauling it to the processing facility;
- HG material is mined and processed as a priority, with most LG material stockpiled and fed at the end of the mine life;
- The remaining ore will be mined in year 4, in excess to the requirement for plant feed. This material, both HG and LG will be stockpiled and then fed to the plant in the next couple of years; and
- Years 5 to 6 will only consist of rehandle off the RoM stockpile to the processing facility.

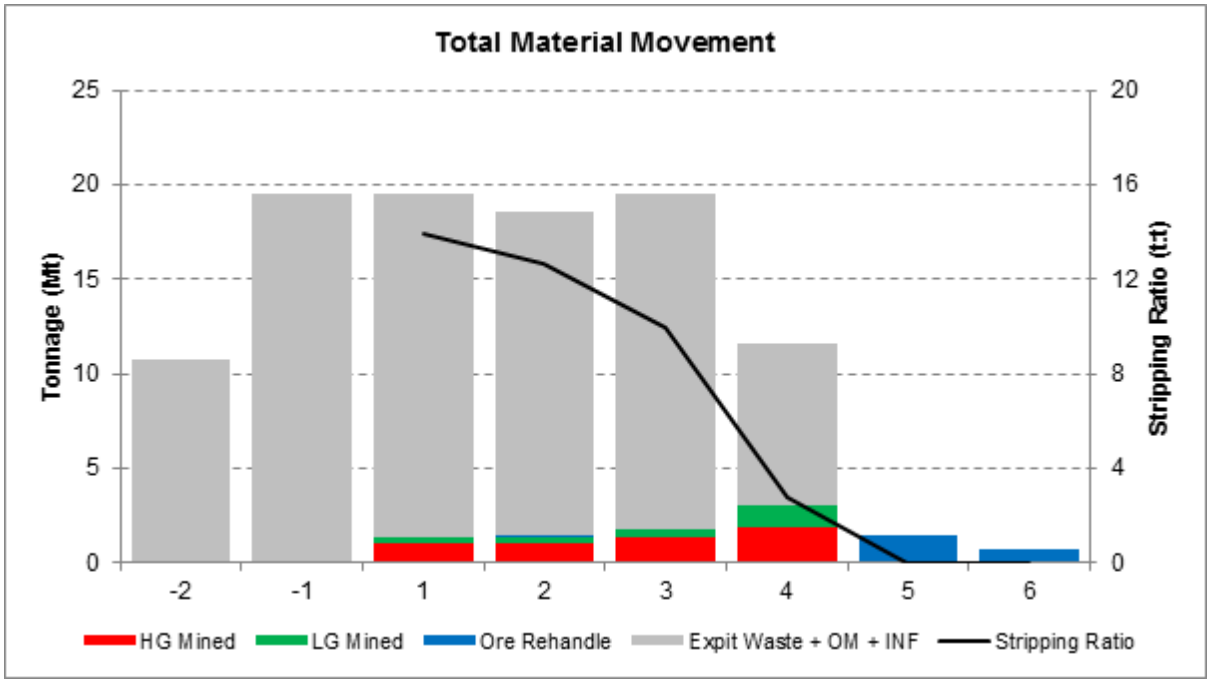


Figure 15-41: Total Material Movement for Miriam Open Pit

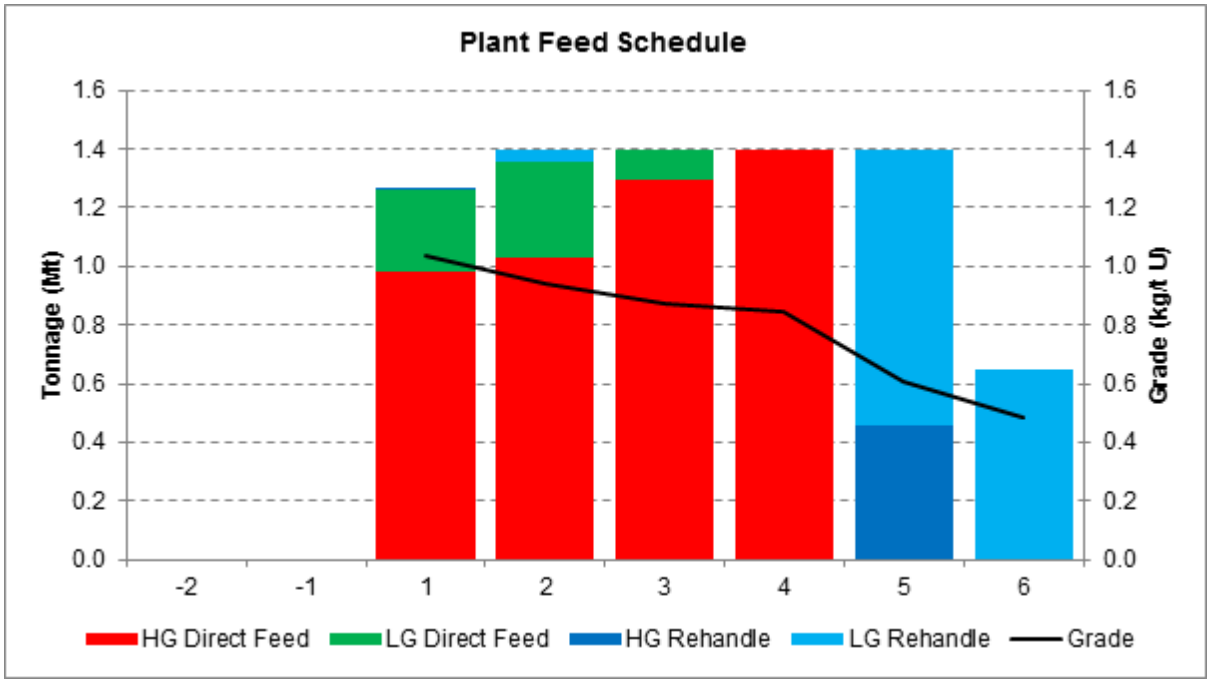


Figure 15-42: Plant Feed Schedule for Miriam Open Pit

The detailed production schedule for the Miriam open pit operation is presented in Table 15-43. The HG and LG stockpile balance is shown in Table 15-39.

Table 15-43: Miriam Open Pit Production Schedule

	Units	Project Year Total	-2	-1	1	2	3	4	5	6
Physicals										
Total Rock	(kt)	**99,354**	10,725	19,500	19,500	18,525	19,500	11,604		
Waste	(kt)	**91,844**	10,725	19,500	18,196	17,170	17,718	8,536		
Stripping Ratio	(t:t)	**12.2**			14.0	12.7	9.9	2.8		
Ore Ex-Pit	(kt)	**7,510**			1,304	1,355	1,782	3,068		
Ore Rehandle	(kt)	**2,095**				45			1,400	650
Ex-pit Ore										
Total Ore Mined	(kt)	**7,510**			1,304	1,355	1,782	3,068		
	(kg/t U)	**0.82**			1.02	0.96	0.79	0.70		
	(t U)	**6,192**			1,330	1,296	1,410	2,156		
HG Ore Mined	(kt)	**5,162**			981	1,031	1,292	1,859		
	(kg/t U)	**0.98**			1.20	1.10	0.90	0.84		
	(t U)	**5,047**			1,172	1,136	1,169	1,570		
LG Ore Mined	(kt)	**2,347**			323	325	490	1,209		
	(kg/t U)	**0.49**			0.49	0.49	0.49	0.49		
	(t U)	**1,146**			158	160	241	587		
Total Process Feed										
	(kt)	**7,510**			1,260	1,400	1,400	1,400	1,400	650
	(kg/t U)	**0.82**			1.04	0.94	0.87	0.84	0.60	0.49
	(t U)	**6,192**			1,308	1,318	1,222	1,182	845	316
Ore Rehandle										
	(kt)	**2,095**				45			1,400	650
	(kg/t U)	**0.57**				0.49			0.60	0.49
	(t U)	**1,184**				22			845	316

Table 15-44: Miriam Open Pit HG and LG Stockpile Balance

Case	Units	Project Year Total	-2	-1	1	2	3	4	5	6
HG Movement										
Mine to RoM	(kt)	**4,703**			981	1,031	1,292	1,400		
	(kg/t U)	**0.99**			1.20	1.10	0.90	0.84		
	(t U)	**4,659**			1,172	1,136	1,169	1,182		
Mine to HG Stockpile	(kt)	**459**						459		
	(kg/t U)	**0.84**						0.84		
	(t U)	**388**						388		
HG Stockpile to Plant	(kt)	**459**							459	
	(kg/t U)	**0.84**							0.84	
	(t U)	**388**							388	
HG Stockpile Balance	(kt)							459		
	(kg/t U)							0.84		
	(t U)							388		
HG Processed Total	(kt)	**5,162**			981	1,031	1,292	1,400	459	
	(kg/t U)	**0.98**			1.19	1.10	0.90	0.84	0.84	
	(t U)	**5,047**			1,172	1,136	1,169	1,182	388	
LG Movement										
Mine to RoM	(kt)	**711**			279	325	108			
	(kg/t U)	**0.49**			0.49	0.49	0.49			
	(t U)	**349**			136	160	53			
Mine to LG Stockpile	(kt)	**1,636**			45		382	1,209		
	(kg/t U)	**0.49**			0.49		0.49	0.49		
	(t U)	**796**			22		188	587		
LG Stockpile to Plant	(kt)	**1,636**				45			941	650
	(kg/t U)	**0.49**				0.49			0.49	0.49
	(t U)	**796**				22			458	316
LG Stockpile Balance	(kt)				45		382	1,591	650	
	(kg/t U)				0.49		0.49	0.49	0.49	
	(kt U)				22		188	774	316	
LG Processed Total	(kt)	**2,347**			279	369	108		941	650
	(kg/t U)	**0.49**			0.49	0.49	0.49		0.49	0.49
	(t U)	**1,146**			136	182	53		458	316

15.10.7 Operating Strategy

Approach

A general operating strategy is required for the equipment selection and estimated productivities. The objectives of the operating strategy are summarised below:

- Equipment fleet selection;
- Development of loading productivities;
- Development of haulage cycle times;
- Development of equipment operating times; and
- Development of drill and blast parameters.

Loading

The equipment fleet has been selected based on the outcomes of the LoM schedule, material movement profile, required selectivity of the orebody and mining space within the pit and designs and pushback shells.

5.7 m^3 bucket capacity excavators have been selected as the primary shovel for mining ore and waste. A 44 t off-highway truck will be used as the primary truck.

All plant feed will be stockpiled at Miriam until scheduled as feed for the processing plant. A 3.5 m^3 front-end loader will be used as the primary loader to load RoM from the stockpile onto 24 t articulated trucks (secondary trucks). They will haul the plant feed to the processing facility.

The loading productivities have been estimated from first principles applying both SRK's experience, information provided from the equipment suppliers and CAT's Fleet Production and Cost ("FPC") software. The loading productivities are shown in Table 15-45.

Table 15-45: Loading Productivities

	Units	Ore	Waste	RoM Rehandle
Loading				
Loading Unit		Primary Shovel	Primary Shovel	Primary Loader
Bucket Size	(m^3)	5.7	5.7	3.5
Loading Spot Time	(min.)	0.70	0.70	0.70
Loading Cycle Time	(min.)	0.35	0.35	0.50
First Bucket Dump	(min.)	0.05	0.05	0.10
Haulage				
Truck		Primary Truck	Primary Truck	Secondary Truck
Capacity	(t)	44.2	44.2	23.6
Capacity	(m^3)	31.3	31.3	14.4
Dump & Spot Time	(min.)	1.20	1.20	1.20
Loading Parameters				
Bucket Fill Factor	(%)	90	90	95
In-Situ Density	(t/bcm)	2.30	2.30	2.30
Swell Factor	(lcm/bcm)	1.30	1.30	1.30
Loose Density	(t/lcm)	1.77	1.77	1.77
Moisture Factor	(%)	5.0	5.0	5.0
Passes	(#)	5	5	4
Loading Productivity				
Total Loading Cycle Time	(min.)	2.15	2.15	2.30
Operator Efficiency	(%)	80	80	80
Loader Productivity	(t/hr)	940	940	469
Loading Unit Utilisation	(%)	59.9	59.9	58.9
Loading Productivity	**(Mtpa)**	**4.7**	**4.7**	**2.3**

The annual loading requirements for the LoM are shown in Figure 15-43.

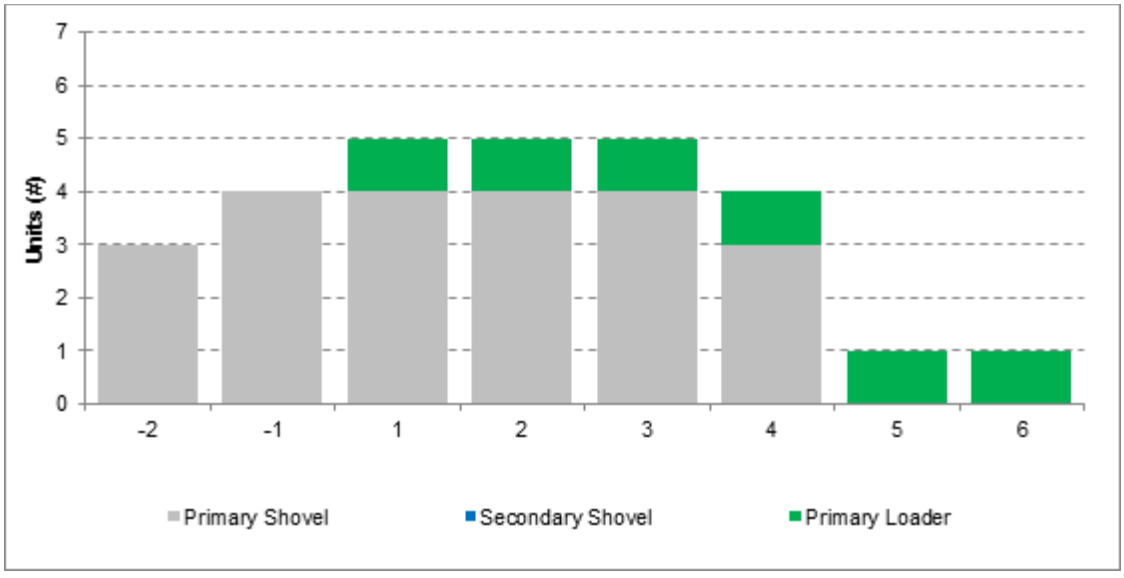


Figure 15-43: Annual Loading Requirements

Haulage

CAT's FPC software has been used to estimate the haulage cycle times. The cycle times have been estimated based on:

- Average haulage distances along the bench for each pushback;
- In-pit ramp distances based on 10% ramp gradient;
- Average speeds of 12 km/hr uphill, 20 km/hr downhill and 25 km/hr on flat;
- Travel time calculations from the pit edge to the RoM stockpile and WRDs for each pushback;
- Loading cycle time and truck dump and spot time from FPC based on the selected equipment (Table 15-45); and
- A haulage distance of 13.25 km from the RoM stockpile to the processing facility.

The annual average cycle times by material type (O re Ex-Pit, Waste and RoM Rehandle) and haulage productivities by truck type (Primary and Secondary Truck) are shown in Figure 15-44.

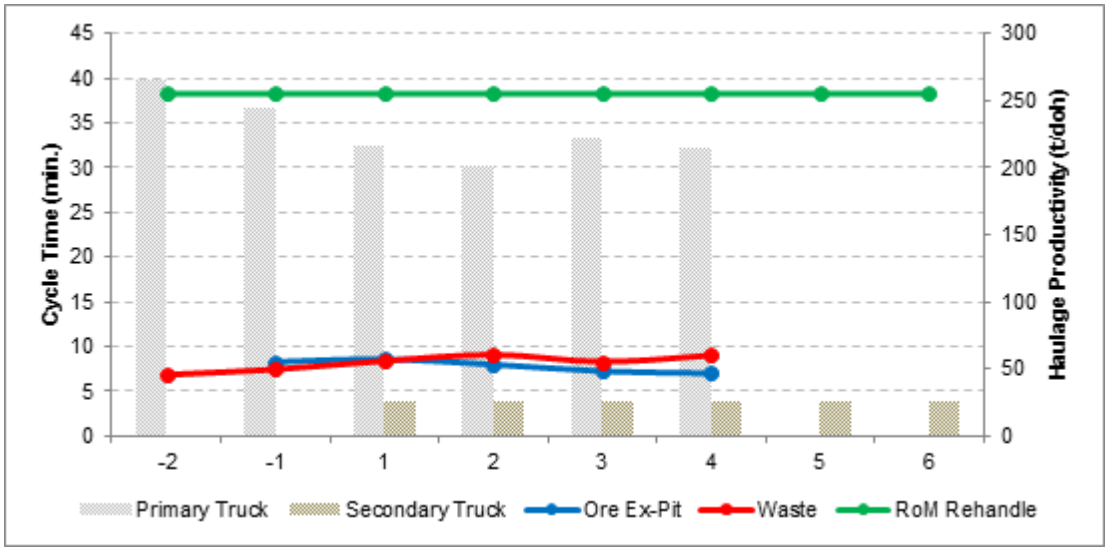


Figure 15-44: Haulage Cycle Time and Productivities

The annual haulage requirements for the LoM are shown in Figure 15-43.

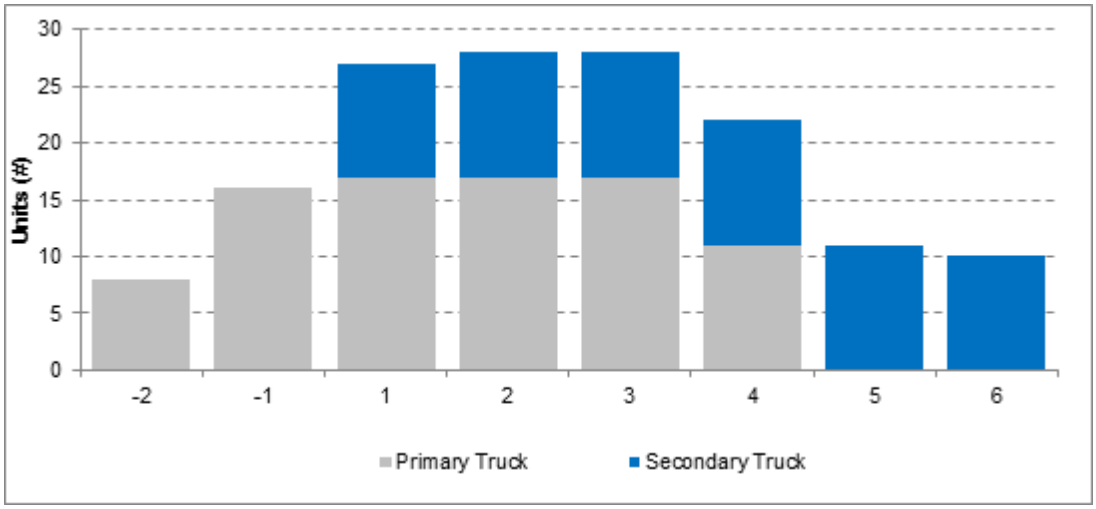


Figure 15-45: Annual Haulage Requirements

Drilling

The drill and blast estimate is based on the production schedule ex-pit material movement and material types. Drill and blast activities are anticipated to be required for all material types. The drilling requirements have been split into three categories: production, pre-split and in-fill.

All material bbove the 400 mRL production drilling will be undertaken on a 5.7 m by 4.9 m spacing on 10 m benches with a 171.5 mm blasthole diameter. All material below the 400 mRL 2.9 m by 2.5 m drill patterns will be used on 5 m benches with an 88.9 mm blasthole drill. Pre-split drilling will be undertaken with an 88.9 mm blasthole drill rig. The 171.5 mm drill rig will also be responsible for the infill drilling on a 25 m by 25 m spacing for 10 m bench heights. All ore material will be in-fill drilled along with 30% of the waste to account for the ore-waste boundary.

SRK recommends that estimated penetration rates are benchmarked with operating information and validated by the equipment suppliers.

The annual drilling requirements for the LoM are shown in Figure 15-43.

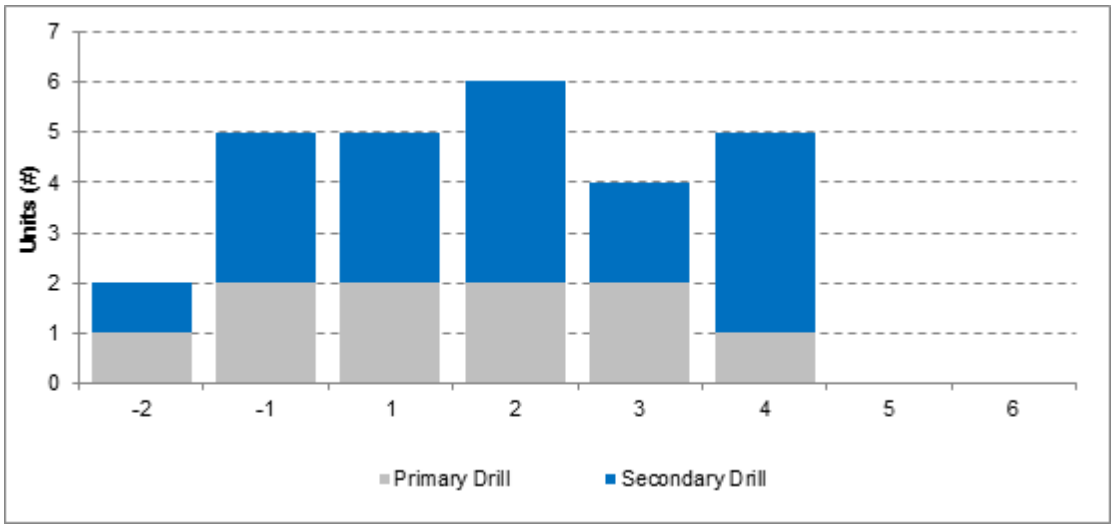


Figure 15-46: Annual Drilling Requirements

Blasting

The blasting parameters have been based on average rock densities and pattern type. The powder factors have been assumed based on operational experience for the ore and waste given the geotechnical parameters. Bulk emulsion has been used for 25% of the holes below 400 mRL to account for wet holes. The drill and blast parameters for production and pre-split are shown in Table 15-46 and Table 15-47, respectively.

SRK notes that changes in pattern design and powder factors will be required in certain areas. SRK recommends the blasting requirements are benchmarked with an explosives supplier.

Table 15-46: Production Drill and Blast Parameters

		Ore	Ore	Ore	Waste	Waste	Waste
	Units	**Dry**	**Dry**	**Wet**	**Dry**	**Dry**	**Wet**
Drill Pattern							
Density	(t/m^3)	2.3	2.3	2.3	2.3	2.3	2.3
Bench Height	(m)	10.0	5.0	5.0	10.0	5.0	5.0
Hole Diameter	(mm)	171.5	88.9	88.9	171.5	88.9	88.9
Subdrill	(m)	1	0.5	0.5	1	0.5	0.5
Hole Depth	(m)	11.0	5.5	5.5	11.0	5.5	5.5
Stemming Height	(m)	3.5	2.0	2.5	3.5	2.0	2.5
Burden	(m)	4.9	2.5	2.5	4.9	4.9	2.5
Spacing	(m)	5.7	2.9	2.9	5.7	2.9	2.9
Volume Rock per Hole	(m^3)	280	36	36	280	36	36
Quantity Rock per Hole	(t)	644	83	83	644	83	83
Re-drill Factor	(%)	10%	10%	10%	10%	10%	10%
Blasting							
Charge Height	M()	7.5	3.5	3.0	7.5	3.5	3.0
Charge per Hole	(kg)	140	18	18	140	18	18
Powder Factor	(kg/t)	0.22	0.21	0.21	0.22	0.21	0.21
Penetration Rate							
Penetration Rate	(m/hr)	0.92	0.92	0.92	0.92	0.92	0.92
Productivity	(m/doh)	36.4	35.0	35.0	36.4	35.0	35.0
Productivity	(t_{rock}/doh)	2,134	530	530	2,134	530	530

Table 15-47: Pre-split drill and blast parameters

Parameter	Units	Value
Bench Height	(m)	10.0
Hole Diameter	(mm)	88.9
Spacing	(m)	1.25
Charge Height	(m)	9.0

Operating Hours

The equipment operating hours have been developed from first principles for all equipment. The key assumptions are that the operation will operate 350 days per year with three 8-hr shifts per day. The estimates for availability, use of availability, effective utilisation and operating hours are shown in Table 15-48.

Table 15-48: Open Pit Equipment Operating Time

	Units	Primary Shovel	Primary Loader	Primary Truck	Secondary Truck	Primary Drill	Secondary Drill	Track Dozer	Grader	Rockbreaker	Water Truck	Fuel Truck	Explosives Truck	Tire Handler	Lighting Plant	Light Vehicle (In-Pit)
Calendar Time	(hr/yr)	8,400	8,400	8,400	8,400	8,400	8,400	8,400	8,400	8,400	8,400	8,400	8,400	8,400	8,400	8,400
Mechanical Loss																
Total Mechanical Loss	(hr/yr)	1,093	1,093	1,093	1,093	1,093	1,093	1,093	1,093	1,093	1,093	680	680	561	531	500
Available Hours	(hr/yr)	7,307	7,307	7,307	7,307	7,307	7,307	7,307	7,307	7,307	7,307	7,720	7,720	7,839	7,869	7,900
Availability	**(%)**	**87.0**	**87.0**	**87.0**	**87.0**	**87.0**	**87.0**	**87.0**	**87.0**	**87.0**	**87.0**	**91.9**	**91.9**	**93.3**	**93.7**	**94.0**
Operating Standby																
Total Standby Hours	(hr/yr)	1,134	1,134	1,134	1,134	1,397	1,397	1,659	2,184	5,334	2,184	3,759	5,334	5,334	4,272	5,334
Operating Hours	(hr/yr)	6,173	6,173	6,173	6,173	5,911	5,911	5,648	5,123	1,973	5,123	3,961	2,386	2,505	3,597	2,566
Use of Availability	**(%)**	**84.5**	**84.5**	**84.5**	**84.5**	**80.9**	**80.9**	**77.3**	**70.1**	**27.0**	**70.1**	**51.3**	**30.9**	**32.0**	**45.7**	**32.5**
Operating Delays																
Total Standby Hours	(hr/yr)	1,138	1,225	1,084	1,084	795	795	1,320	707	1,057	896	308	308	308	571	308
Direct Operating Time	(hr/yr)	5,036	4,948	5,089	5,089	5,116	5,116	4,329	4,416	916	4,227	3,653	2,078	2,197	3,027	2,258
Operating Efficiency	**(%)**	**81.6**	**80.2**	**82.4**	**82.4**	**86.6**	**86.6**	**76.6**	**86.2**	**46.4**	**82.5**	**92.2**	**87.1**	**87.7**	**84.1**	**88.0**
Effective Utilisation	**(%)**	**59.9**	**58.9**	**60.6**	**60.6**	**60.9**	**60.9**	**51.5**	**52.6**	**10.9**	**50.3**	**43.5**	**24.7**	**26.2**	**36.0**	**26.9**

Equipment Requirements

The annual equipment requirements are shown in Table 15-49. The ancillary equipment requirements have been based on the material movement profile, the primary equipment size and requirements, labour requirements and the number of working areas.

Table 15-49: Annual Open Pit Equipment Requirements

Equipment Requirements	Units	Maximum	-2	-1	1	2	3	4	5	6
Primary Shovel	(#)	**4**	3	4	4	4	4	3		
Primary Loader	(#)	**1**			1	1	1	1	1	1
Primary Truck	(#)	**17**	8	16	17	17	17	11		
Secondary Truck	(#)	**11**			10	11	11	11	11	10
Primary Drill	(#)	**2**	1	2	2	2	2	1		
Secondary Drill	(#)	**4**	1	3	3	4	2	4		
Track Dozer	(#)	**2**	1	2	2	2	2	2		
Grader	(#)	**2**	1	1	2	2	2	2	1	1
Rockbreaker	(#)	**1**	1	1	1	1	1	1		
Water Truck	(#)	**1**	1	1	1	1	1	1	1	1
Fuel Truck	(#)	**1**	1	1	1	1	1	1		
Explosives Truck	(#)	**1**	1	1	1	1	1	1		
Tire Handler	(#)	**1**	1	1	1	1	1	1	1	1
Lighting Plant	(#)	**19**	10	16	17	19	16	16	2	2
Light Vehicle (In-Pit)	(#)	**33**	26	29	32	33	32	31	4	4

Labour Requirements

The Arlit region has a history of mining and provides a workforce familiar with the techniques and skills required for the proposed operations. Niger has its own mining school and the experience of the existing mines is that it provides competent professionals ready for the industry. SRK has assumed that the Madaouela project can be staffed entirely with Nigerien labour for both technical and labour functions. Any requirement for expatriate assistance is assumed to be covered by the general and administrative costs.

The labour requirements have been estimated from the outcomes of the production schedule and the equipment levels. The labour estimates have been based on the following criteria according to the position: material movement profile, equipment fleet requirements and number of shifts.

The project schedule is based on four work crews working three 8-hr shifts per day. This shift structure complies with Nigerien legislation and removes any requirement for overtime payments. Office staff are assumed to work a standard 40-hr work week.

It has been assumed that the management and supervisory roles in the last three years of the open pit operation, when only RoM rehandle to the processing facility is occurring, will be handled by the underground staff. Therefore only equipment operators and maintenance personnel have been assumed for those years.

A summary of the estimated annual labour requirements is provided in Table 15-50.

Table 15-50: Annual Open Pit Labour Requirements

Labour Requirements	Units	Maximum	-2	-1	1	2	3	4	5	6
Mine Operations	**(#)**	**187**	**91**	**134**	**182**	**187**	**185**	**158**	**48**	**44**
Mine Manager	(#)	**1**	1	1	1	1	1	1		
Mine Operations Superintendent	(#)	**1**	1	1	1	1	1	1		
Mine Operations Supervisor	(#)	**4**	4	4	4	4	4	4		
Production Engineer	(#)	**2**	2	2	2	2	2	2		
Drill and Blast Superintendent	(#)	**1**	1	1	1	1	1	1		
Drill and Blast Supervisor	(#)	**4**	4	4	4	4	4	4		
Drill and Blast Engineer	(#)	**2**	2	2	2	2	2	2		
Trainer	(#)	**1**	1	1	1	1	1	1		
Truck Operators	(#)	**100**	28	56	96	100	100	80	40	36
Shovel Operators	(#)	**16**	12	16	16	16	16	12		
Track Dozer Operators	(#)	**8**	4	8	8	8	8	8		
Grader Operators	(#)	**8**	4	4	8	8	8	8	4	4
Ancillary Operators	(#)	**16**	16	16	16	16	16	16	4	4
Dispatch Operators	(#)	**4**	4	4	4	4	4	4		
Drillers	(#)	**6**	2	5	5	6	4	5		
Dewatering Crew	(#)	**1**	1	1	1	1	1	1		
Mine Ops Coverage	(#)	**12**	4	8	12	12	12	8		
Mine Maintenance	**(#)**	**62**	**38**	**46**	**58**	**62**	**54**	**58**	**21**	**21**
Maintenance Superintendent	(#)	**1**	1	1	1	1	1	1		
Maintenance Supervisor	(#)	**4**	4	4	4	4	4	4		
Maintenance Planner	(#)	**1**	1	1	1	1	1	1	1	1
Mechanic	(#)	**24**	12	16	20	24	20	20	8	8
Electrician	(#)	**8**	4	4	8	8	8	8	4	4
Welder	(#)	**8**	4	4	8	8	8	8	4	4
Drill Maintainer	(#)	**8**	4	8	8	8	4	8		
Maintenance Assistants	(#)	**4**	4	4	4	4	4	4	4	4
Mine Maintenance Coverage	(#)	**4**	4	4	4	4	4	4		

Labour Requirements	Units	Maximum	-2	-1	1	2	3	4	5	6
Technical Services	**(#)**	**20**	**20**	**20**	**20**	**20**	**20**	**20**		
Chief Mine Engineer	(#)	**1**	1	1	1	1	1	1		
Senior Mine Planning Engineer	(#)	**1**	1	1	1	1	1	1		
Planning Engineer	(#)	**1**	1	1	1	1	1	1		
Mine Surveyor	(#)	**2**	2	2	2	2	2	2		
Chief Mine Geologist	(#)	**1**	1	1	1	1	1	1		
Senior Mine Geologist	(#)	**1**	1	1	1	1	1	1		
Mine Geologist	(#)	**1**	1	1	1	1	1	1		
Samplers	(#)	**8**	8	8	8	8	8	8		
Grade Control Technicians	(#)	**2**	2	2	2	2	2	2		
Specialist Consultant	(#)	**1**	1	1	1	1	1	1		
Administrative Assistant	(#)	**1**	1	1	1	1	1	1		

15.10.8 Capital Costs and Operating Costs

Equipment

The majority of the mining equipment capital costs for the Madaouela project have been sourced from the manufacturers. Table 15-51 shows the manufacturer source and model for the equipment capital costs. The remaining equipment has been sourced from the Infomine cost database[2].

Table 15-51: Open Pit Manufacturer Quote Sources and Descriptions

Equipment	Manufacturer	Model/Type	Description
Primary Shovel	CAT	390F	Backhoe excavator 5.7 m^3
Primary Loader	CAT	962H	Front end loader 3.5 m^3
Primary Truck	CAT	772G	Off-highway haul truck 44 t
Secondary Truck	CAT	725B	Articulated truck 24 t
Primary Drill	CAT	MD6240	Drill 88.9 – 152.4 mm (3.5-6 in.)
Secondary Drill	CAT	MD5075	Drill up to 127mm (5 in.)
Track Dozer	CAT	D8R	Track dozer 328 hp
Grader	CAT	14M	Grader 250 hp
Rockbreaker	CAT	H160E	Hammer attachment on excavator
Water Truck	Infomine		19,000 L water truck
Fuel/Lube Truck	Infomine		Mobile fuel and lubricant truck
Explosives Truck	Infomine		Mobile truck for loading blastholes
Tyre Handler	CAT	966H	Loader with tyre mount
Lighting Plant	Infomine		Mobile lighting plant
Light Vehicle	Infomine		Four wheel drive light vehicle

Capital Costs

The equipment capital costs used in the cost estimate are shown in Table 15-52.

[2] Infomine, 2014-2015. Equipment Cost Calculator.[online] Available at: <http://costs.infomine.com/> [Accessed September, 2014].

Table 15-52: Open Pit Equipment Capital Costs

Equipment	Units	Purchase Cost
Primary Shovel	USD	1,025,500
Primary Loader	USD	294,000
Primary Truck	USD	615,000
Secondary Truck	USD	388,000
Primary Drill	USD	1,620,000
Secondary Drill	USD	620,000
Track Dozer	USD	620,500
Grader	USD	488,865
Rockbreaker	USD	88,000
Water Truck	USD	278,300
Fuel/Lube Truck	USD	93,170
Explosive Truck	USD	91,795
Tyre Handler	USD	488,000
Lighting Plant	USD	22,000
Light Vehicle	USD	42,900

Replacement costs throughout the mine life are based on the estimated equipment life for each unit. All equipment assumed to be replaced at full purchase cost. No allowance for salvage values of the mining equipment has been considered in this study.

Miscellaneous costs including mining software and survey equipment have been estimated based on labour requirements for the operation.

A summary of the estimated capital costs for open pit mining is provided in Table 15-53. Contingency and engineering, procurement and contract management have not been included in this estimate.

Table 15-53: Open Pit Capital Cost Estimate

Capital Costs	Units	Total	-2	-1	1	2	3	4	5	6	7
Equipment Capital	**(USDm)**	**29.9**	13.7	9.7	5.4	1.1					
Equipment Replacements	**(USDm)**	**0.3**					0.2	0.1			
Miscellaneous	**(USDm)**	**0.8**	0.8								
Total	**(USDm)**	**31.1**	**14.6**	**9.7**	**5.4**	**1.1**	**0.2**	**0.1**			

Operating Costs

Labour figures are based on a simplification of GoviEx's existing pay structure (Table 15-54), which uses a sliding scale to account for years of experience and years with the company. The labour costs include a burden of 30% to account for the costs of recruitment, insurance and other employment related costs.

Table 15-54: Labour Costs

Roles	Units	Value
Managers	(USDpa)	218,400
Superintendents	(USDpa)	171,600
Supervisors	(USDpa)	85,800
Senior Engineers	(USDpa)	156,000
Engineers/Senior Technical	(USDpa)	85,800
Technical	(USDpa)	46,800
Skilled	(USDpa)	15,600
Unskilled	(USDpa)	9,360

Operating costs have been developed from first principles. Where possible, equipment and consumable costs are sourced from suppliers; remaining data was sourced from GoviEx and SRK's own databases of similar operations. The consumption rates and mining equipment operating costs are based on the Infomine cost database. The fuel cost for the cost estimate has been provided by GoviEx at 1.10 USD/L. The equipment operating costs per operating hour are shown in Table 15-55.

Table 15-55: Open Pit Operating Costs per Operating Hour

Equipment	Total (USD/hr)	Fuel (USD/hr)	Lubricants (USD/hr)	Maint. (USD/hr)	Tyres (USD/hr)	Wear Parts (USD/hr)
Primary Shovel	**153.31**	58.19	14.12	53.05		27.95
Primary Loader	**52.85**	16.61	6.12	15.36	7.53	7.23
Primary Truck	**123.50**	48.29	13.20	34.59	27.42	
Secondary Truck	**72.60**	24.86	6.80	25.19	15.75	
Primary Drill	**97.57**	67.74	12.72	5.87		11.24
Secondary Drill	**91.11**	54.84	12.42	6.63		17.23
Track Dozer	**90.70**	45.65	8.20	28.88		7.97
Grader	**62.75**	25.08	7.60	21.18	2.77	6.12
Rockbreaker	**56.81**	42.62	6.76	7.43		0.00
Water Truck	**35.34**	24.84	2.57	3.81	4.12	
Fuel/Lube Truck	**13.29**	10.58	0.98	1.11	0.62	
Explosive Truck	**24.47**	20.26	1.40	1.68	1.13	
Tyre Handler	**57.38**	18.70	6.76	16.34	7.89	7.69
Lighting Plant	**2.38**	1.76	0.20	0.40	0.02	
Light Vehicle	**18.57**	15.78	1.76	0.79	0.24	

It has been assumed that all charging and blasting activities will be conducted by the owner. The unit costs shown in Table 15-56 have been obtained from Orica's National Price List[3]. Orica's prices have been obtained in AUD and converted to USD using an exchange rate of AUD 0.80/USD. The ammonium nitrate cost was obtained from the Client as USD 1,300/t delivered to site. An emulsion price of USD 2,000/t was used in the cost estimate.

Table 15-56: Blasting Unit Costs

Cost Parameter	Description	Units	Cost
Primer	400 g	(AUD/unit)	12.27
Detonator	6 m	(AUD/unit)	6.88
Detonator	12 m	(AUD/unit)	11.78
Surface Delay	9 m	(AUD/unit)	11.67

The results from the cost estimate are presented in Table 15-57 and Table 15-58 split by category and equipment, respectively. The costs are inclusive of the RoM rehandle cost.

Table 15-57: Open Pit Operating Costs - Category

Category	Units	Unit Costs
Labour	(USD/t ex-pit)	0.41
Maintenance	(USD/t ex-pit)	0.40
Diesel	(USD/t ex-pit)	0.67
Oil/Lubricants	(USD/t ex-pit)	0.16
Tyres	(USD/t ex-pit)	0.21
Wear Parts	(USD/t ex-pit)	0.07
Explosives	(USD/t ex-pit)	0.56
Miscellaneous	(USD/t ex-pit)	0.00
Total	**(USD/t ex-pit)**	**2.47**

Table 15-58: Open Pit Operating Costs - Equipment

Category	Units	Unit Costs
Mine Staff	(USD/t ex-pit)	0.22
Loading	(USD/t ex-pit)	0.23
Hauling	(USD/t ex-pit)	1.04
Drilling	(USD/t ex-pit)	0.12
Ancillary	(USD/t ex-pit)	0.29
Blasting	(USD/t ex-pit)	0.56
Miscellaneous	(USD/t ex-pit)	0.00
Total	**(USD/t ex-pit)**	**2.47**

A summary of the estimated operating costs for open pit mining is provided in Table 15-59 and Table 15-60, by category and equipment, respectively. A contingency has not been included in this estimate.

[3] Orica, 2015. Orica National Price List.[online] Available at: <http://www.oricaminingservices.com/au/en/section/products_and_services/price_list> [Accessed February, 2015].

Table 15-59: Open Pit Operating Cost Estimate – Category

Operating Costs	Units	Total	-2	-1	1	2	3	4	5	6
Labour	**(USDm)**	**40.3**	5.2	6.0	6.9	7.1	6.9	6.6	1.1	0.5
Maintenance	**(USDm)**	**40.0**	2.9	5.5	7.6	7.8	7.7	5.7	1.9	0.9
Diesel	**(USDm)**	**66.1**	5.6	9.9	12.4	12.8	12.3	9.9	2.2	1.1
Lubricants	**(USDm)**	**15.4**	1.2	2.2	2.9	3.0	2.9	2.3	0.6	0.3
Tires	**(USDm)**	**21.2**	1.4	2.7	4.1	4.2	4.1	3.0	1.1	0.5
Wear Parts	**(USDm)**	**6.7**	0.6	1.2	1.3	1.3	1.2	1.1	0.1	0.0
Explosives	**(USDm)**	**55.2**	4.2	10.1	10.5	11.1	9.8	9.4		
Miscellaneous	**(USDm)**	**0.1**	0.0	0.0	0.0	0.0	0.0	0.0		
Total	**(USDm)**	**245.0**	**21.2**	**37.6**	**45.7**	**47.3**	**44.9**	**37.9**	**7.1**	**3.3**

Table 15-60: Open Pit Operating Cost Estimate – Equipment

Operating Costs	Units	Total	-2	-1	1	2	3	4	5	6
Mine Staff	**(USDm)**	**21.9**	3.6	3.6	3.6	3.6	3.6	3.6	0.1	0.0
Loading	**(USDm)**	**22.9**	2.4	4.3	4.4	4.3	4.5	2.8	0.2	0.1
Hauling	**(USDm)**	**103.5**	6.7	13.1	19.9	20.7	20.0	14.7	5.7	2.7
Drilling	**(USDm)**	**12.4**	0.7	2.2	2.3	2.6	2.0	2.5		
Ancillary	**(USDm)**	**28.2**	3.3	4.2	4.8	4.9	4.8	4.7	1.1	0.5
Blasting	**(USDm)**	**56.0**	4.3	10.3	10.7	11.3	9.9	9.6		
Miscellaneous	**(USDm)**	**0.1**	0.0	0.0	0.0	0.0	0.0	0.0		
Total	**(USDm)**	**245.0**	**21.2**	**37.6**	**45.7**	**47.3**	**44.9**	**37.9**	**7.1**	**3.3**

Sensitivity Analysis

A cost sensitivity analysis was conducted on the operating efficiencies, fuel cost, labour rates, ammonium nitrate (“AN”) cost, and blasting consumable costs. The results are shown in Figure 15-47. The capital and operating cost estimates are most sensitive to changes in the operating efficiency and fuel cost.

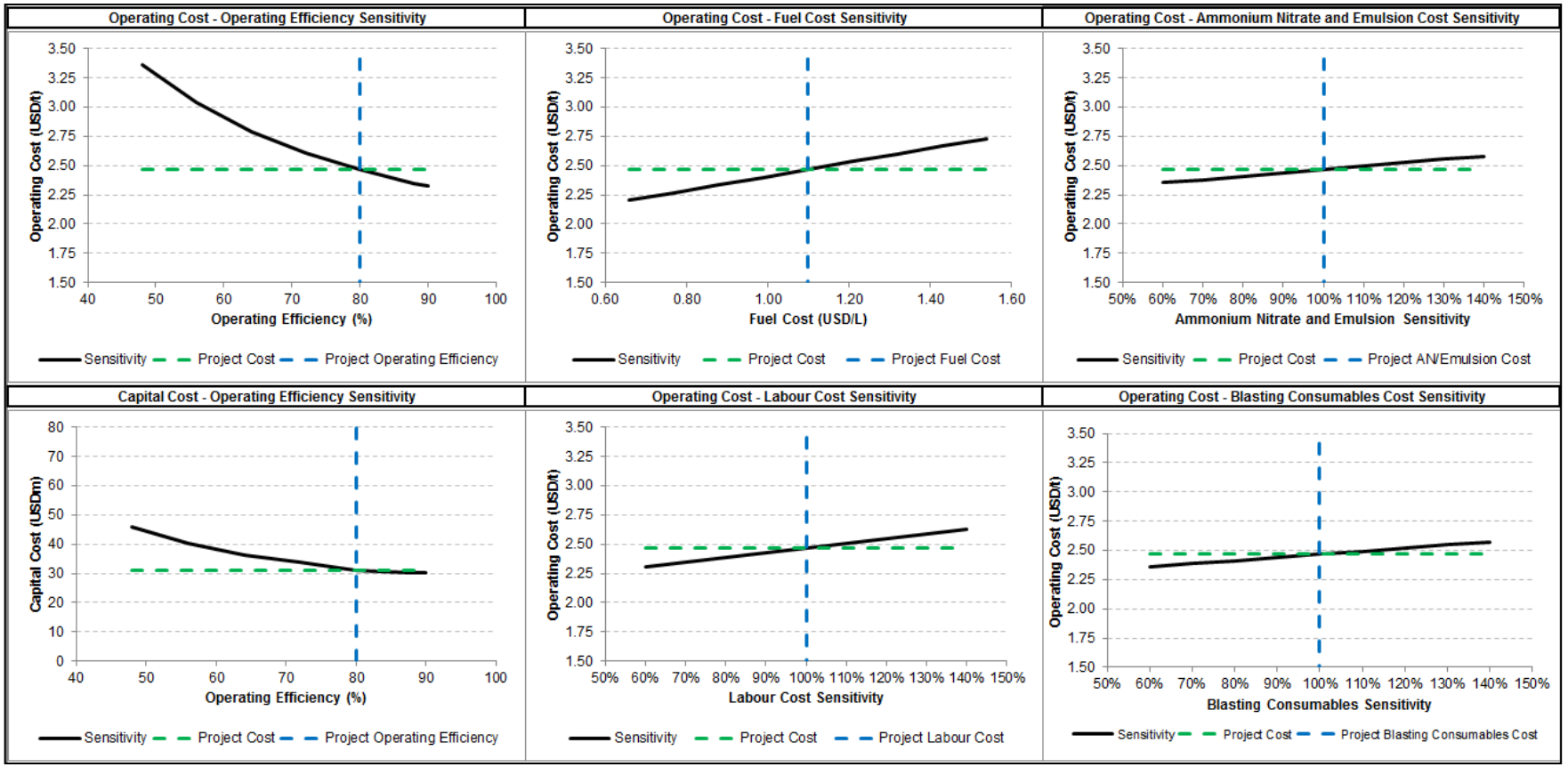


Figure 15-47: Open Pit Cost Sensitivity Analysis

15.10.9 Conclusions

The results of the open pit mining study are summarised below:

- The open pit mining of the Miriam deposit has demonstrated to be technically viable using selective open pit mining methods;
- The final pit design includes 7.5 Mt of plant feed at 0.82 kg/t U with 91.8 Mt of waste at a strip ratio of 12.2;
- The mining modifying factors have been estimated at 2% ore loss and 5% dilution at a 0 kg/t U grade;
- The mine schedule has been optimised by deferring the processing of low grade material below 0.6 kg/t U which is stockpiled and processed at the end of the mine life;
- The open pit mine life is six years with four years of ex-pit plant feed and two years of plant feed supplied from the stockpile;
- The total life of mine capital costs are 31.1 USDm;
- The average mine operating costs are USD 2.47/t; and
- The cost sensitivity analysis shows that the operating costs are most sensitive to changes in the operating efficiency and fuel cost.

15.10.10 Risks and Opportunities

The Miriam open pit recovers just over 50% of the Measured and Indicated Mineral Resource. Investigation of the technical and economic feasibility of extracting higher grade ore remaining from the open pit using underground methods and accessed from the open pit may provide opportunity to increase mineralisation recovery of the Miriam resource.

Pit extraction occurs at the start of project life and additional uranium metal or higher grade maximises the opportunity to increase total metal production together with an extended pit life which increases project value by lower mine operating cost in the first years of operation and delays expenditure on underground access and infrastructure.

The use of only local employees creates a risk that production targets will not be met if the operational efficiency of 80% is not achieved. An appropriate management and training strategy will need to be implemented in order to meet these requirements.

16 RECOVERY METHODS (ITEM 17)

16.1 Introduction

A preliminary economic assessment ("PEA") of the Madaouela project was undertaken in 2010. In the PEA two alternative processing flowsheets were considered; milling followed by two stage tank leach and heap leaching of crushed ore, both at a mining rate of 1.5 Mtpa. Both flowsheet options incorporated the use of radiometric ore sorting ("ROS"), based on hand sorting tests, to upgrade the feed ore after initial crushing. Design parameters were based on preliminary metallurgical testwork undertaken by SGS and others although no column tests had been undertaken for the heap leach option.

Further investigations and testwork were undertaken for the IDP studies mainly by SGS and Mintek. This included; ore mineralogy (including investigations of the minerals in the leach tails), leaching testwork (rolling bottle, agitated and in columns), ore sorting, scrubbing and flotation. Samples used in the testwork investigations were sources from the stockpiles adjacent to the Marianne and Marilyn deposits near the army camp, which is understood to have been mined in the 1960s from Marianne and these samples, were differentiated in open piles as high grade, low grade and waste. It is suspected that the samples had partially deteriorated physically both on the stockpiles and in transport with the generation of fines noted however extensive mineralogical characterization indicated limited oxidation of the ore. Based on this, the samples were considered suitable for study.

Based on the testwork results as part of the IDP studies, Tenova Bateman, a mineral project engineering company based in RSA, was appointed to develop designs, capital and operating costs to a PFS level (nominally +/- 25 – 30 % accuracy) for three alternative treatment flowsheets including the 2 stage and heap leach process routes considered in the PEA and including a third process route incorporating dry milling and "pugging", a technique used at the nearby Somair and Cominak operations.

This flowsheet had been considered for the PEA but had been rejected due to economic and operational considerations. Initial evaluations of the three initial case flowsheets indicated that the operating costs were driven mainly by acid consumption and the cost of sulfur for acid production required for leaching and the cost of lime used for pH adjustment and minor changes to the uranium recovery flowsheets were incorporated to increase solution bleed from the circuit thus reducing lime requirements. It was also indicated that dry milling and pugging would be the least economically attractive route due to higher capital and operating costs mainly related to higher acid consumption compared to the two alternative flowsheets and was not considered further.

The selected process flowsheets developed during the IDP studies and on which the costs were initially developed included the use of the tank leach after initial ore upgrade using ROS and flotation alternatives. The ROS option included the treatment of ore after initial crushing to remove sterile and low grade gangue while the flotation option incorporates milling and desliming / flotation with the flotation concentrate containing a higher level of acid consuming carbonate minerals being discarded. The ROS flowsheet are based on flotation and testwork undertaken in the feasibility investigations by Mintek.

The ROS option was based on a design throughput of 4,020 tpd, to reflect a revised mining option and to maintain a similar uranium production while the flotation option is based on 3,350 tpd. A third option incorporating ROS sorting agglomeration and heap leach at a

throughput of 4,020 tpd is also included.

An initial IDP study base case process option, defined by economics, was ROS, acid leach and RIP/IX to produce yellowcake. The study noted that further optimisation of the process flowsheets was possible and that further testwork was required to validate their applicability. This testwork was completed, described in Section 12, and included Ablation, Cyanex 600, and Nanofiltration.

The selected process flowsheet presented herein and on which the revised costs have been developed is: Crushing; ROS; Milling; Ablation; two stage leach; and Cyanex 600 SX. The selected process includes the use of the tank acid leach after initial ore upgrade using ROS & Ablation followed by enhanced SX using Cytec solvent Cyanex 600.

16.1.1 Further Investigations Undertaken and Planned.

The metallurgical testwork undertaken on the Madaouela project have been reviewed in detail in Section 12 of this report. The majority of work was undertaken and completed before the start of the IDP studies although some further ongoing testwork was undertaken in parallel with the studies.

16.1.2 Flowsheet Development and Evaluation

Flowsheet Alternatives and Similar Operations

The selection of flowsheets for consideration in the IDP was based on the previous studies and PEA investigations, industry practices on similar ore types and the nearby Areva operations in Niger; Somair and Cominak.

At the two above named properties acid pugging is utilised for preparing the feed prior to both tank and heap leaching and it is understood that high uranium recoveries into product are achieved. The two operations initially started with the pugging and tank leach route on high grade ores while marginal and lower grade ores were initially stockpiles. These are now understood to be being processed by crushing, pugging / agglomeration and heap leaching.

It is also understood that there is significant corrosion at the two Areva operating and this is thought to result from the practice of acid pugging, the generation of gases during acid addition and during the extended curing time and is possibly also related to the use of chlorides in the uranium recovery circuit for the stripping of uranium and molybdenum.

Two stage countercurrent acid leaching is understood to have been practiced at several uranium operations including the White Mesa and Key Lake operations in North America. The advantage of two stage leaching is that it would appear to be easier to design and operate compared to acid pugging as the acid is contained with vessels compared to pugging where concentrated acid is added to milled dry solids and left to cure. The heat of reaction in pugging causes a rise in temperature and results in the generation of corrosive and noxious fumes. This can be exaggerated by the use of additional oxidants such as ferrous sulfate, sodium nitrite or oxygen.

Initial Case Flowsheet

The flowsheets investigated by Tenova Bateman were developed in conjunction with SRK and Mintek who undertook the majority of the testwork. Further testwork carried out by Cytec, Black Range Minerals and ANSTO refined the base case to include: ablation; enhanced solvent extraction; and acid recovery options. The flowsheet is illustrated graphically in Figure 16-4.

Selected Flowsheet

The flowsheet developed was as follows; Crushing, ROS, ablation and two stage acid tank leaching, Cyanex 600 SX.

The selected case throughput on which the design and initial capital and operating costs, based on quotes, was 4,020 tpd or approximately 1.5 Mtpa. The investigations were based on treatment of ore from the Marianne/ Marilyn and Miriam deposits at a feed grade of around 0.1% U_3O_8 and 1,200 ppm Mo.

Ore Preconcentration and Gangue Removal

Due to the high indicated consumption of acid shown from the testwork process options were considered for the removal of acid consuming gangue minerals from the feed ore and upgrade of the uranium grade were considered to be of significant benefit in treating the Madaouela ore. These included flotation and radiometric ore sorting ("ROS"), ROS was chosen for the selected flowsheet.

Ore sorting for uranium uses ROS where individual rock particles are presented to a scintillometer and the particles are accepted or rejected based on the level of radiation. A slightly different approach utilises ore sorting based on X ray fluorescence ("XRF") where each rock is analysed for its uranium contents and accepted or rejected (RADOS). This latter approach was used in the testwork evaluation of the Madaouela ores due mainly to the equipment being available.

The RADOS ore sorting testwork indicated that based on a synthetic blend of feed material from the available stockpiled samples, around 45 % of the material fed to an ore sorter could be rejected with the losses of < 10 % of the uranium.

Concerns were highlighted related to ore sorting testwork and the representativeness of the samples tested. Firstly it was noted that the samples tested appeared to be physically weathered with the generation of a significant levels of fines in the feed material and generated during feed preparation for the ROS tests. These were removed prior to ore sorting of the coarser fractions and affected the overall yield and recovery from the sample prepared with the fines along with any further fines generated during the testwork bypassing the ROS circuit and being combined with the accept product.

The RADOS ore sorting tests had been designed to upgrade the mineralised ore material, by the removal on waste material within the ore zone. In process design RADOS has been included to remove the waste material that will occur from the barren hangingwall. Hence the RADOS sensitivity will be reduced to focus on removing material with very low or no concentrations of uranium rather than removing uranium material below a defined cutoff.

It should also be highlighted that leach testwork on a ROS accept product resulted in a higher acid consumption compared to treating a whole ore feed although this is suspected to have been possibly caused by a higher level of carbonates in the feed to leach having bypassed the ROS sorting in the fines fraction not processed. These limitations are addressed through the use of Ablation.

Ablation testwork indicated that approximately 95-99 % uranium recovery into the fine fraction was likely to be achieved on a full scale plant. The tests proved that a greater than 64% mass reduction in the material requiring to be leached was feasible. The extraction of uranium during the leach phase and subsequent uranium PLS recovery may also be significantly improved due to the ablation process. These aspects will require further refinement during the

detailed design stage.

Ablation additionally substantially reduces the risks related to ROS described above; firstly, as fine material reporting through to the accept product would be processed by the ablation stage before the acid leach stage; and secondly all ore material post ROS must be secondary crushed prior to ablation reducing the impact of mined material size on feed material to the acid leach.

Uranium and Molybdenum Recovery

In developing the circuit for the downstream recovery of uranium and molybdenum various options were considered. These needed to consider the uranium deportment and optimum size for leaching based on the testwork undertaken on milled ore and the high levels of molybdenum anticipated in the Madaouela ore and reporting into solution after leaching.

Due to the initial proposed coarse grind acceptable for uranium leaching, resin in pulp ("RIP") where the uranium is recovered from the slurry after uranium leaching was not considered due to potential damage to the resin by the coarse quartz particles in the slurry from the sandstone host rock and it was decided that uranium recovery would be undertaken from solution.

The chosen processing flowsheet, following solid-liquid separation, would produce a pregnant leach solution from which the uranium and molybdenum would be recovered and based on the feed ore analysis and testwork it was indicated that the molybdenum tenor in the ore and pregnant leach solution could be as high as 1,200 ppm Mo, from which the uranium at a tenor of 2,500 ppm would need to be selectively separated.

Several separation and recovery techniques were considered and tested, however using solvent extraction on the PLS to extract the molybdenum and uranium then stripping the molybdenum and uranium independently to produce a molybdenum oxide product and yellowcake was selected.

A flowsheet utilising Cyanex 600 in an enhanced SX process and no RIP route was developed, with the advantage that the Cyanex 600 approach allowed for the precipitation of molybdenum and subsequent production of molybdenum oxide.

Following uranium purification in SX the uranium would be initially precipitated from the SX strip solution as sodium diurinate ("SDU"), redissolved with sulfuric acid, reprecipitated using hydrogen peroxide, filtered, dried and packaged into drums as uranium oxide.

16.1.3 Work undertaken by Tenova Bateman

Tenova Bateman were effectively responsible for the process development of the alternative flowsheets considered and agreed, and the detailed design work required to generate capital and operating costs to the required accuracy. The flowsheet concepts initially considered were based on directives for SRK while the uranium recovery process alternatives were developed in discussions between Tenova Bateman, Mintek who undertook the majority of the testwork, SRK and GoviEx.

The detailed flowsheets and mass balances were developed using Aspen Plus, a thermodynamic chemical / mineral simulation program. The following design work was undertaken by Tenova Bateman for each option to provide details from which the capital and operating costs could be generated.

- Block flow diagrams (“BFDs”);
- Detailed process design criteria;
- Process flow diagrams (“PFDs”);
- Mechanical equipment list with motor sizes;
- General arrangement and layout drawings;
- Electrical Load List;
- Preliminary quantities for concrete and steelwork taken from the GA drawings.

Further details of the philosophy for generating the capital and operating costs are presented in Section 16.1.7.

16.1.4 Design Parameters

Key Design Parameters

The main overall design parameters related to throughput, uranium recovery and production are presented in Table 16-1.

Table 16-1: Key design parameters

Flowsheet	Selected Flowsheet
Total Feed (tpd)	4,020
U Head Grade (ppm)	1,000
Total U Feed (kg/hr)	142
U Product (kg/hr)	133
U Product (lb/a)	2,569,094
ROS Recovery (%)	98.0%
Ablation Recovery (%)	99.2%
Leach Recovery (%)	98.5%
Solids / Liquid Separation Recovery (%)	99.0%
Recovery SX (%)	99.0%
SDU Precipitation (%)	99.8%
Overall U Recovery (%)	**93.7%**

General

Plant Overall Design Availability	%	86
Feed size from mining	mm (80 % passing)	250
Feed size after crushing	mm (80 % passing)	20
ROM Stockpile	t	12,000 – 15, 000 (approx 3.5 days)

Crushing, ROS, Milling, Ablation and 2 Stage Tank Leach

Feed size to ROS	mm	20-300
Number of ROS circuits		2
ROS sorting size ranges	mm	20 – 90
	mm	90 – 300
Uranium Loss in ROS Rejects	%	2
ROS Yield	%	55
Mill product size	micron (80 % passing)	500
Ablation mass pull to leach	%	36
Ablation product size	micron (150% passing)	149
Uranium loss in ablation rejects	%	0.8
Acid Consumptions in Leach	kg/t ore	65

Uranium Recovery and Precipitation

PLS Flow to Uranium recovery	m^3/h	50 (approx.)
Solution Tenor to Uranium recovery		
Uranium	g/L	2.5
Molybdenum	mg/L	800 (approx)
SX Solution Feed	m^3/h	12 - 50
Extract Stages		2-3
Stripping Stages		3-4

16.1.5 Reagents, Utilities and Services

Acid Plant Capacity	tpd	80
Water consumption	m^3/t feed	2.3

16.1.6 Brief Process and Plant Descriptions

The following section provides a brief description of the treatment process. A refined block flow diagram (“BFD”) is provided at the end of this section Figure 16-4. The Selected Flowsheet Option: Crushing, ROS, Milling, Ablation, Two stage leach and Cyanex 600 SX.

Ore Receipt and Feed Preparation

Mined ore from Marianne and Marilyn is crushed underground (80 % < 306 mm) prior to transport and is received on surface at the transfer point at the discharge of the inclined conveyor from the mining operation. From here material is transferred to the ROM stockpile via the stockpile feed conveyor. Facility is also provided to receive primary crushed ore from other mining operations or from static stockpiles around the plant area into a feed receipt bin by road truck or front end loader (“FEL”).

Feed preparation effectively consists of two stage crushing with intermediate screening. Ore from the stockpile is reclaimed by 3 apron feeders and routed to a secondary jaw crusher operating in open circuit with the product screened and the oversize routed to a tertiary cone

crusher with the product returned to the sizing screen. Product from the crushing circuit is nominally < 20 mm (80 % passing).

The primary crushed ore from the stockpile is routed to ROS where fines are screened out and routed directly to fine crushing with the coarse fractions (+20 mm) subject to preconcentration by ROS. The accept product only is routed to fine crushing while the rejects are routed to a waste stockpile.

Accept product from the ROS is secondary crushed and mixed with water to create 20% slurry, which is fed into the Ablation stage, in which one or more set of two jets of slurry site opposite each other to create a high energy impact zone. The high energy impact zone causes a break down of the ore material into coarse and fine particles, which are separated by screening and gravity. The uranium is predominately in the fine particles and this fine material slurry is thickened prior to feeding into the acid leach tanks. The coarse material, which is predominately sand, is thickened and sent to the tailings dam. An additional recovery step involves gravity separation of residual uranium and heavy mineral particles from the coarse material that is then fed back into the Ablation product. This improves overall recovery to 99% and also results in carbonate reduction and thus reduces acid consumption for the two stage acid leach step.

The two stage leaching circuit consists of primary and secondary agitated leach tanks in recirculation with a leach thickener after the first stage and a belt filter after the second stage leach. In the first stage solution from the second stage belt filters is recirculated to leach the fresh feed after which the slurry is thickened with the thickener overflow routed to the uranium recovery circuit as pregnant leach solution (“PLS”). The thickener underflow is routed to the second stage leach, which uses fresh acid to further leach the milled solids. After the second stage the slurry is filtered and the filtered solids residue is washed with fresh solution and discarded to the tailings disposal system.

Leach tanks are agitated and aerated to allow the milled ore to react with sulfuric acid allowing dissolution of the contained uranium. The acid consumption (per ton of feed to the leach) in this case is forecast to be lower because of the rejection of the acid consuming constituents of the feed during flotation. The solution from leach (thickener overflow) is routed to uranium recovery.

Uranium Recovery

Selected Cyanex 600 Option

Cytec has developed a new solvent extraction (SX) process to recover molybdenum from acidic solutions using CYANEX® 600 extractant. CYANEX 600 is a highly selective, stable phosphinic acid based reagent and allows molybdenum and uranium recovery from acidic solutions by the same solvent. The principle advantages are;

- Highly selective for uranium;
- Highly selective for molybdenum;
- High selectivity against anionic species, including sulfate, chloride, arsenic and metals like iron, aluminum, manganese, copper, cobalt, nickel, chromium, and rhenium;
- Fast kinetics;
- Excellent stability;
- Good phase disengagement properties;
- High solubility;
- No rich strip liquor purification required.

Based on the test results, the use of 10 Vol% CYANEX 600 could be used to selectively extract and separate Molybdenum and Uranium while achieving recoveries greater than 98% for Mo and 99% for U.

The overall circuit configuration would consist of two extraction steps, one stripping step for iron, two stripping steps for molybdenum, an ammonia washing step and two uranium stripping steps. A wash stage swill be used after the Uranium strip to prevent phosphoric acid transfer. Phosphoric acid will be recovered through an evaporation step in order to reduce reagent requirements. Additional uranium recovery will come from this step through a bleed stream that would feed back into the solvent extraction process.

One iron scrub stage operating at an organic: acid ratio (O/A ratio) of 10:1 would reduce the organic iron from 204 ppm to less than 95 ppm using 50 gpl sulfuric acid and less than 55 ppm using 100 gpl sulfuric acid. Any iron present in the organic phase after the scrub stage will transfer to the rich U liquor.

Two stages of molybdenum strip with an O/A ratio of 1:1. The Strip solution will be 0.2M ammonium carbonate, ammonium hydroxide is used as make-up at a 3:1 molar ratio of Ammonia/Organic Moly when the Mo concentration in the rich solution reaches 50 gpl. Ammonium Tetramolybdate product can be precipitated from the rich liquor using sulfuric acid. During Mo stripping the organic phase will load ammonia. Approximately 10-20% neutralization of the ligand is expected. Molybdenum oxide will be precipitated by conversion of the tetramolybdate by oxygen sparging.

This will be followed by an Ammonia Wash with an O/A of 10 in 20 gpl sulfuric acid.

The Uranium Strip will occur in two steps utilizing an O/A of 10 in 600 gpl Phosphoric Acid. The Uranium transfer is predicted to be 7.66 gpl and iron transfer to be approximately 0.55 gpl (using 100 gpl acid wash) resulting in U/Fe selectivity of 13.9. If silica is present in the PLS it will be treated with Polyethylene Oxide to improve phase disengagement times and physical properties. Settler design with a low specific settler flow (i.e. 3 $m^3/hr*m^2$) should be considered if silica is present in the PLS. Estimation of organic losses through entrainment would be 40 ppm.

Utilities and Services

Acid Plant

Acid for the process is supplied by a dedicated onsite acid plant from the burning of sulfur. The main process steps and plant areas are;

- Covered sulfur storage;
- Sulfur melting, filtration and combustion;
- Gas cooling;
- Double conversion / double adsorption (“DC/DA”) acid plant including gas / gas heat exchangers, 4 stage catalytic converter, drying and adsorption towers;
- Steam and power generation from waste heat;
- Cooling water system;
- Acid storage and distribution.

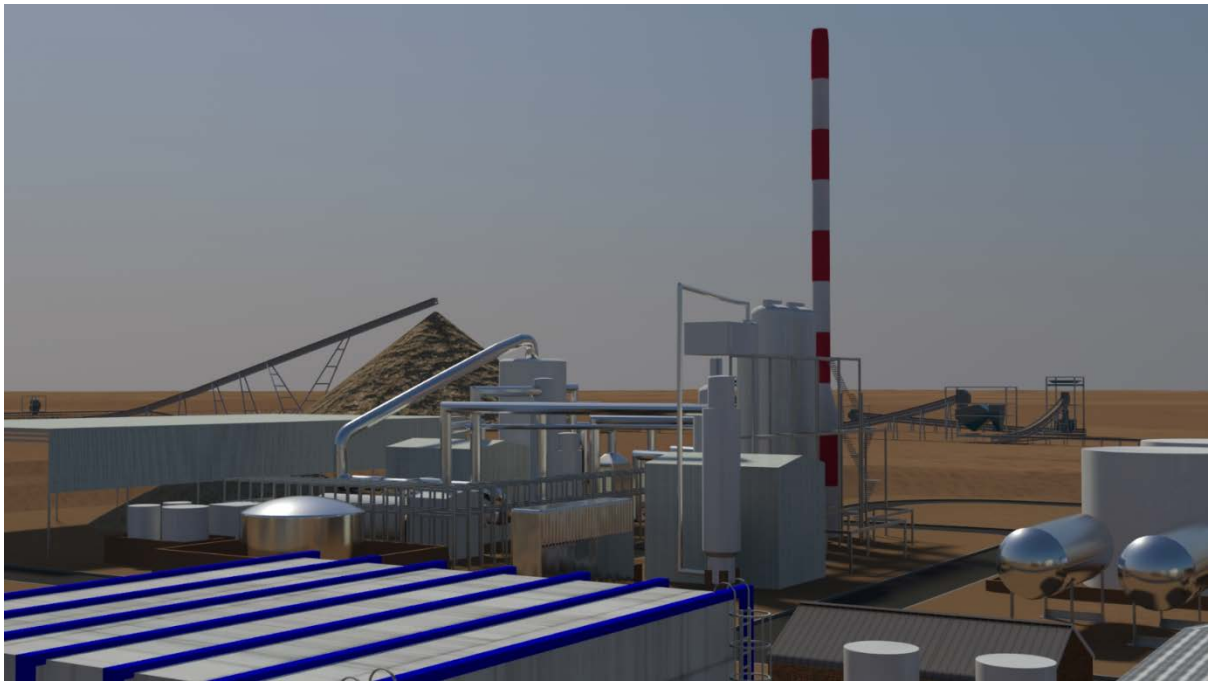


Figure 16-1: Visualisation of proposed acid plant

Reagent Storage, Handling, make up and distribution

The majority of reagents are supplied either in bagged form or in isotainers with the reagents made up by dilution with treated water prior to being dosed into the process. Required reagents include;

- Caustic (sodium hydroxide) – received as solution in isotainers for dilution and distribution;
- Flocculant – supplied in bags and made up in a packaged wetting, storage and distribution system;
- Hydrogen Peroxide – supplied in isotainers and dosed directly into the process as an oxidant in leach and for final uranium precipitation;
- Sodium carbonate (soda ash) – supplied in bulk bags for make-up, solution storage and distribution.

Water Services

Water circuits on site include those for;

- Raw water receipt, storage and transfer;
- Potable water treatment, storage and reticulation;
- Fire water storage, HP pumps and fire water ring mains;
- Process water storage, collection and distribution;
- Gland seal water pumping and distribution;
- Demineralised water storage and distribution.

Other Services

- Process (compressed) air supply;
- Instrument air drying, filtration and distribution;
- Start up electric boilers;
- Diesel storage and distribution. Used for acid plant start up etc.

Site Facilities

Site facilities in addition to those detailed above include the following

- Plant offices / administration building, electrical substations and plant MCC and control rooms, plant maintenance workshops, medical centre, canteen, change house, laboratory, warehouse, etc including water and sewerage reticulation and treatment;
- Gatehouse, security building, access control and perimeter security fencing;
- Construction camp including accommodation (550 personnel), messing and recreation facilities;
- Site gravel roads (exc stormwater provisions).

Off-site Facilities

Off-site facilities include the following;

- Water supply for the borehole field including HT power supply and transformers, water pipeline to the plant area and access road from the plant site to the borehole field
- Plant area access road (sealed 12 m wide) from the main Airlit to Agadir road.

Comments on the Selected Plant Design

The optimum arrangement for ROS is still to be resolved from further planned testwork. This is related to the size fractions to the processed and size of material to be screened out which bypasses the ROS circuit, as part of the next detail design engineering stage.

The overall plant availability is still to be investigated in detail. Currently limited buffer storage is included between crushing and milling and this need to be reviewed related to the forecast overall achievable plant availability.

Plant Description

General arrangement drawings for each plant sections were developed for the 5000 tpd original initial case flowsheet options. These were assembled into an overall plant layout for each of the processing options.

The general arrangement drawing have not been updated for the current revised throughputs although have been modified to reflect the revised flowsheet option incorporating ROS. The general arrangement drawings for the overall plant site and the plant sections are included in the Tenova Bateman report.

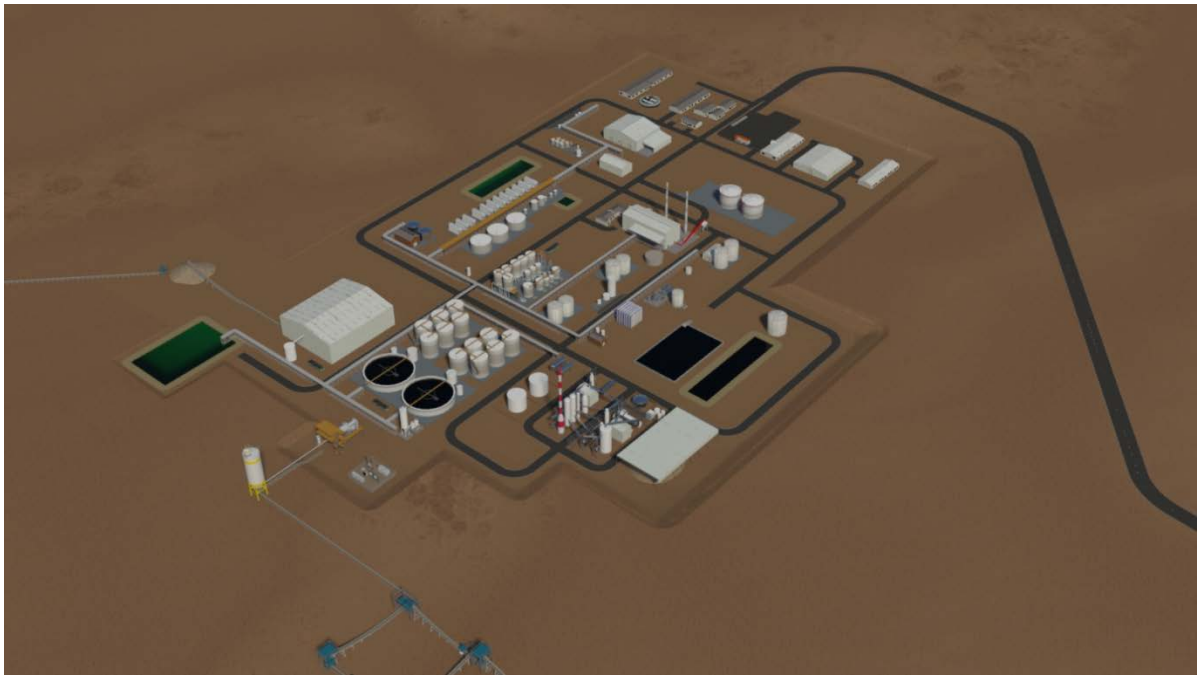


Figure 16-2: Visualisation of the proposed processing plant

16.1.7 Forecast Operating and Capital Costs

General

Tenova Bateman have calculated forecast operating and capital costs for the processing option presented. These are summarised in Table 16-2. Further details are provided in the individual capital and operating cost sections.

Table 16-2: Overall summary of forecast overall operating and capital costs

		Selected Option
		Enhanced SX
Capital cost	USD m	234
Operating costs	USD/t ore to plant	23

It is important to note that the selected enhanced SX option will produce a molybdenum oxide salable by-product. The MoO product has been valued at USD 11 /lb for the purpose of this project and an overall calculated recovery rate of 64% has been achieved based on the Ablation and Cytec studies. Modelling carried out at Cytec indicated a likely PLS Mo concentration of 0.875 g/L.

This revenue stream needs to be studied further, including, market research on the likely longterm estimate for molybdenum oxide. Taking the molybdenum oxide revenue stream into account will allow the selection of the flowsheet option that provides the optimum return on investment.

16.1.8 Forecast Operating Costs and Costing Philosophy

General

Operating costs were generated from first principles by calculating consumptions of reagents, power etc and labour complements and applying appropriate unit rates supplied by GoviEx

relevant to local conditions or by incorporating quoted prices for international supply and delivery to site.

Summaries of the operating costs for the two options are presented in Table 16-3.

Table 16-3: Forecast operating costs

Throughput	Values in USD pa
	Selected Flowsheet
	4020
	Enhanced SX
Labour	10,934,040
Maintenance Materials	3,917,308
Electric Power	3,348,538
Fuel	1,429,632
Reagents and Consumables	11,079,973
Waste handling	335,566
ROS	733,650
Ablation	2,241,045
TOTAL	34,199,750
USD/t Ore	23

Operating and Maintenance Labour

Tenova Bateman generated labour complements for the different flowsheets. In summary these included the following total personnel presented in Table 16-4.

Table 16-4: Summary labour complements

	Selected Processing Option
Management / Admin	72
Supervisors	42
Operators	212
Labour	114
TOTAL	438

Labour rates were provided by GoviEx based on in-country experience and investigations undertaken and were generally as presented in Table 16-5.

Table 16-5: Summary labour unit costs

Personnel Description	Grade	Monthly Salary (USD)
Unskilled Labour	A4	600
Operators / Technicians	B2 – B4	1,000
Supervisors	B7	3,000
Engineers	C3 – C5	5,500
Senior Personnel	D2 – D4	10,000

A 30 % loading has been added to the cost of salaries to cover other costs of labour such as food, overtime, transport, etc. and other “on” costs.

Unit Reagent and Consumable Costs

Delivered costs of reagents are generally based on quoted priced CIF Cotonou with a transport cost of USD 190 /t Cotonou to site for bulk or container transport. The main reagent and consumable costs incorporated in the IDP investigations are presented in Table 16-6.

Table 16-6: Unit consumable costs (USD)

		Supply Price	Transport Cost	Total Delivered Cost
Sulfur	USD/t	150	190	3400
Phosphoric Acid	USD/t	950	190	1140
Sodium Carbonate	USD/t	200	190	390
Ammonium Hydroxide	USD/t	530	190	720
Ammonium Carbonate	USD/t	1505	190	1695
Caustic Soda	USD/t	520	190	710
Hydrogen Peroxide	USD/t	500	190	690
Flocculant	USD/t	3,300	190	3,490

Electric power costs were based on a unit rate of USD 0.204 / kWh.

A nominal allowance was made for other consumable costs not indentified and quantified such as minor reagents, product drums, laboratory consumables etc.

Other Operating Costs

The cost of maintenance spares and consumables was factored as 10 % of the installed cost of the mechanical equipment.

A cost for contaminated waste handling has been included based on Tenova Bateman’s experience on other uranium projects.

The cost of operating the ROS section excluding labour and power was estimated at USD 0.5 /t based on other similar operations as advised by the Ukrainian equipment supplier.

The cost of operating the ablation unit was estimated at USD 3/t treated as supplied by Black Range Minerals including power load.

Key Operating Costs Parameters

As shown in Table 16-6 the cost of sulfur is a major driver in the operating costs. The cost of sulfur was based on the latest long term sulfur price forecast which indicates a likely oversupply situation, the forecast supply cost FOB Europe / Middle East has been defined at USD 150 /t. with a delivery cost to site of USD 190 /t.

Forecast Accuracy

Tenova Bateman forecast accuracy for both the operating and capital costs of -15 % + 40 % with a probability of not more than 50 % and has not included a contingency allowance on the operating costs.

Forecast Capital Costs and Costing Philosophy

Capital Cost Summary

A summary of the forecast capital costs for the different process options being considered is presented in Table 16-7.

Table 16-7: Summary of forecast capital costs

	Chosen Option
	Enhanced SX
	USD M
Direct Costs	
Infrastructure and Earthworks	32.6
Civils	6.1
Structural	12.7
Mechanical Equipment	71.4
Piping and valves	15.3
Electrical	11.3
Instrumentation	7.0
Transport	9.4
Spares, First fill, commissioning	7.4
Acid Plant Package	8.2
Total Direct Costs	181.6
Indirects	
EPCM	27.2
Other	3.9
Contingency	21.3
TOTAL	**234.0**

Capital Costing Philosophy

General

For the developed flowsheet, both with a mined ore feed of 4,020 tpd and a feed to leach of around 810 tpd after ablation and ROS the costs for the mechanical equipment were generally factored with only around 10 % of the mechanical equipment costs based on quotes with the remaining 90% based on factored costs (80 %), database numbers and allowances.

Mechanical Equipment – Budget prices were initially obtained for all major mechanical equipment items for the initial cases of 3350 and 5000 tpd throughput based on emailed quotations in response to duty data sheets to a single vendor. Minor equipment was prices from data base information. Installation costs were estimated from a percentage of the supply cost with contractors' indirect field costs based on a percentage of the erection costs.

The cost of the acid plant was based on a design and supply package with erection costs factored. The cost of the ROS package was based on limited information provided by equipment manufacturers based in the Ukraine and in Germany although detailed costs and

scope of work were not provided. The OPEX and CAPEX for the ablation circuit were provided by the equipment manufacturer although these will be revised after further process optimisation. Infrastructure and Earthworks – Preliminary quantities were generated from the general arrangement drawings. Supply and erection rates were based on similar projects and in-country data provided by GoviEx.

Civils – Preliminary quantities were generated from the general arrangement drawings. Supply and erection costs were based on in-country data provided by GoviEx. Preliminary and General ("P&G") costs were based on a percentage of the supply and erection costs

Structural Steelwork – Preliminary quantities were generated from the general arrangement drawings. Supply and erection rates were based on similar projects. P&G costs were based on a percentage of the supply and erection costs and adjusted by GoviEx based on local conditions.

Plateworks and Tanks – Quantities were generated from the equipment list and supply rate are based on in-house database numbers for similar project in a similar location. Erection costs have been factored as a percentage of the supply costs.

Piping and Valves – All supply costs have been factored from the priced mechanical equipment lists. The erection cost has been calculated by using a factored percentage of the piping and valves supply cost.

Electrical, control and Instrumentation – All supply costs for inter-plant electrical equipments complete with bulk supplies (cables, etc) along with instruments have been factored from the priced mechanical equipment lists. The erection cost has been calculated by using a factored percentage supply costs.

Freight, Duties and Clearing – All ocean freight and overland transport costs from the point of origin to site has been included in the cost estimate. A percentage allowance has been made on the supply "ex-work" cost.

Commissioning Spares and Start-up Costs – Provision has been made on a factorised basis for the following Commissioning and Start-up costs including:

- Supply of commissioning spares;
- Initial first fills of oil and lubricants;
- Vendors assistance on site during construction and commissioning.

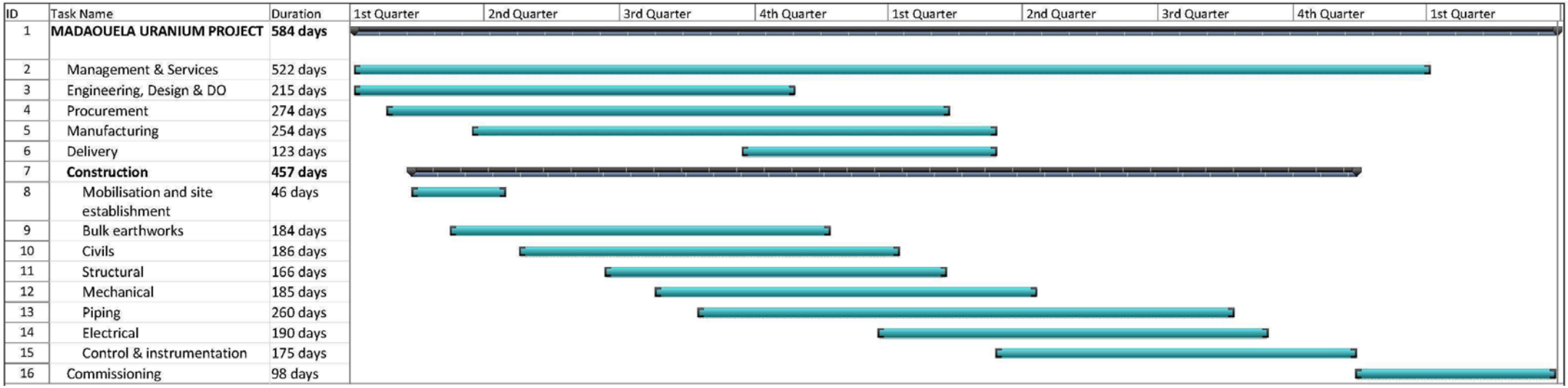
EPCM Costs – factored from total direct field costs

A 10 % contingency has been included on the calculated capital costs.

16.1.9 Project Execution and Indicative Timescale

A detailed project execution philosophy has not yet been prepared for the project. However an indicative high level project programme has been prepared and is presented in the Tenova Bateman report.

The programme indicates project duration of 20 months from project award to start of commissioning with a further 4 month commissioning and ramp up. An 18 month site construction period is indicated.



ID	Task Name	Duration
1	**MADAOUELA URANIUM PROJECT**	**584 days**
2	Management & Services	522 days
3	Engineering, Design & DO	215 days
4	Procurement	274 days
5	Manufacturing	254 days
6	Delivery	123 days
7	**Construction**	**457 days**
8	Mobilisation and site establishment	46 days
9	Bulk earthworks	184 days
10	Civils	186 days
11	Structural	166 days
12	Mechanical	185 days
13	Piping	260 days
14	Electrical	190 days
15	Control & instrumentation	175 days
16	Commissioning	98 days

Figure: 16-3: Processing Plant Execution Schedule

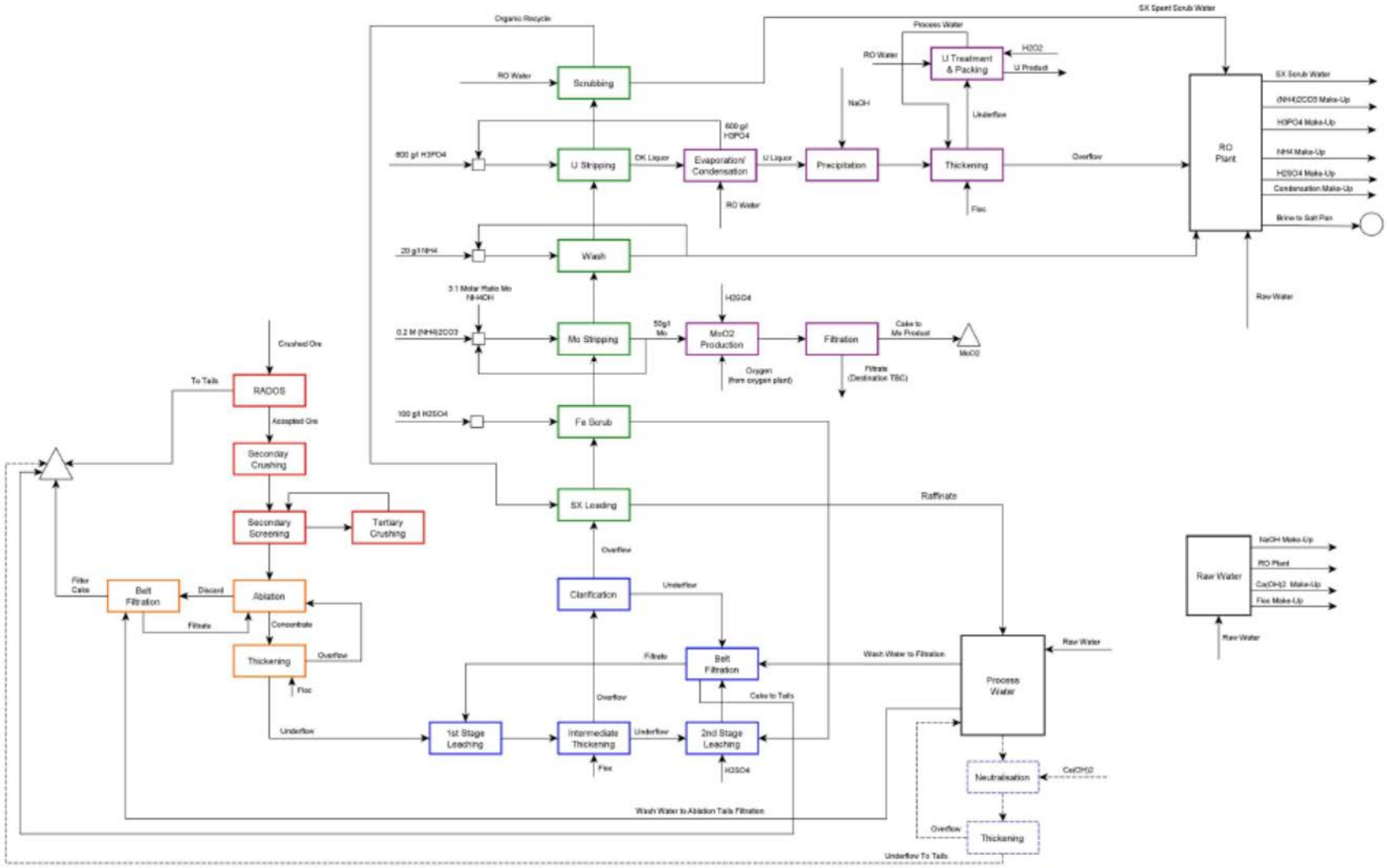


Figure 16-4: Selected Flowsheet – crushing, ROS, milling, ablation, 2 stage leach and enhanced SX

16.2 Tailings

Various process options are currently being considered which will define the waste disposal strategy selected.

SRK has developed preliminary designs for the options being considered for the disposal and storage of tailings and has provided high-level cost estimates for each for use in the project economic model. The designs have been completed based on the design criteria for the current project as developed by others, applicable national and international standards and, where appropriate, based on SRK's experience.

For the TSF option, SRK has selected a dry stacked TSF as the most appropriate concept. This concept is similar to the disposal methodology used at the nearby Somaïr and Cominak operations. The TSF with a production rate of 2,250 tpd has been designed to store a total of 16.5 Mt, or 10. Mm^3 of tailings.

SRK has provided risks and opportunities related to the current design and a forward works plan to progress the designs to the next stage. The designs presented are technically feasible and will provide tailored solutions to the project requirements.

17 PROJECT INFRASTRUCTURE (ITEM 18)

This section presents the proposed infrastructure assets for the Madaouela project as well as an evaluation of the transport options to site for consumables and equipment.

17.1 Surface Infrastructure

This section presents the surface infrastructure assets proposed at four separate operational areas:

- Marianne-Marilyn underground operation (“M&M”);
- MSNE-Maryvonne underground operation (“MNSE”);
- Miriam open pit operation (“Miriam”), and;
- Explosives Magazine (“Magazine”).

17.1.1 Site Layout

Surface infrastructure compounds will be located adjacent to the Marianne-Marilyn and MSNE underground portals that are linked by a 3 km long heavy vehicle road. This road runs parallel to a surface conveyor that transports ROM material from the mining operations to the process plant.

It is anticipated that the mining and support workforce will all change at the change facility located at the Marianne-Marilyn compound before bus transportation to their place of work, including underground operations.

The proposed Marianne-Marilyn compound has been located approximately 100 m south of the Marianne-Marilyn portal and adjacent to the main site access road which links the process plant site to the existing Arlit-Tchirozine (N25) road.

The proposed MSNE compound has been located approximately 100 m north of the MSNE portal and adjacent to the road linking the Marianne-Marilyn and MSNE portals.

The proposed Miriam compound has been located approximately 100 m northeast of the Miriam pit, and adjacent to the road to MNSE.

The explosives magazine area has been located just North of MNSE with a minimum 500 m clearance from all other assets.

All surface infrastructure assets have been located within the Mad I and Agal licenses. The layout plan is shown in Figure 17-1.

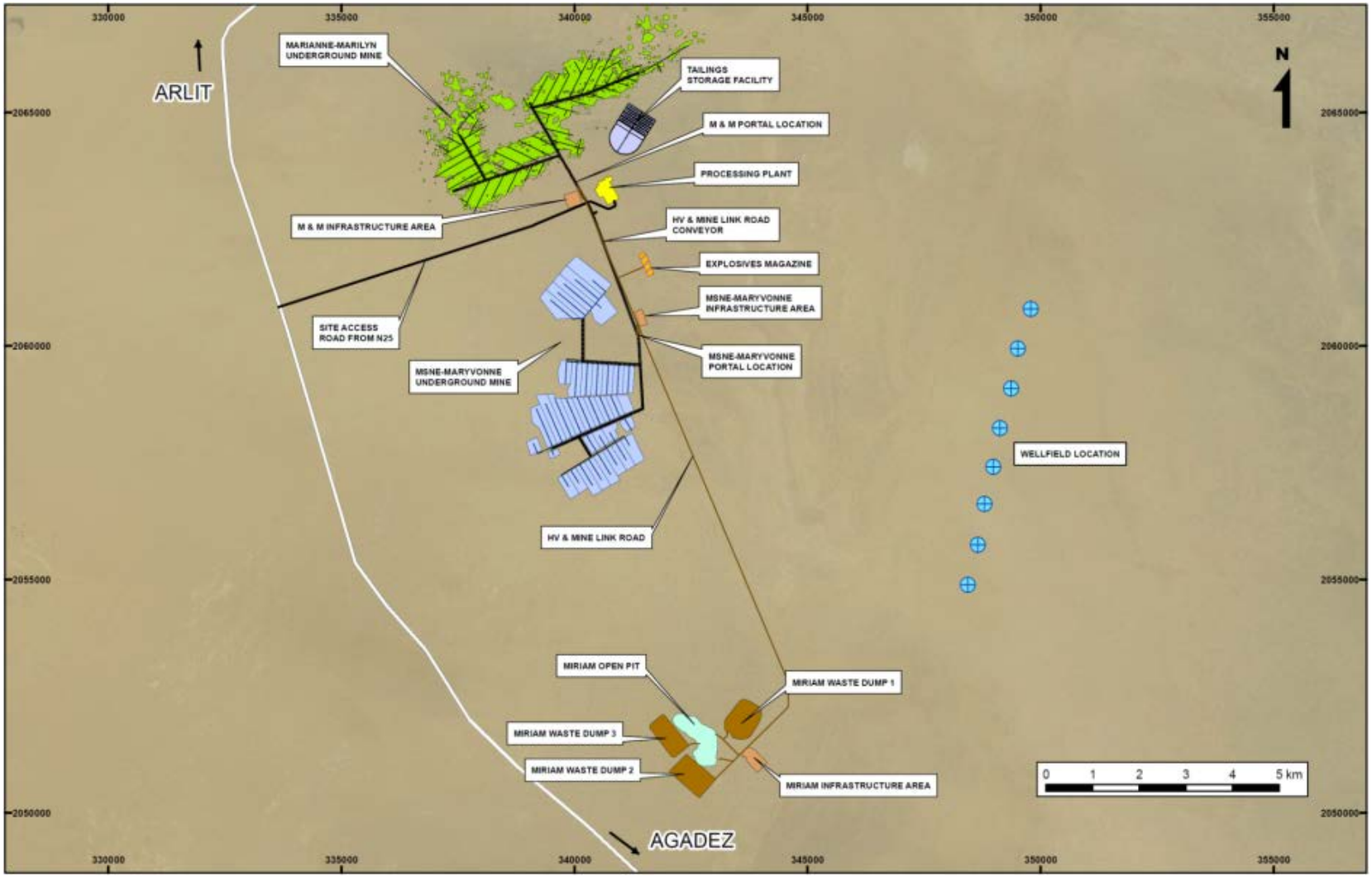


Figure 17-1: Madaouela surface infrastructure layout

GoviEx has 100% ownership of Mad I, II, III and IV, and applications filed November 22, 2012 for Agal and Eral are pending. GoviEx has been advised by senior government officials that Agal and Eral can expect the applications to be approved.

17.1.2 Light and Heavy Vehicle Maintenance

A total of 36 underground and 28 open pit main production units will have to be maintained at the Miriam and Marianne-Marilyn workshops.

The main light and heavy vehicle maintenance workshops will be located at the Marianne-Marilyn compound. At the MSNE and Miriam compounds only a limited service workshop will be set up. SRK proposes the use of a truck with low-loader to move especially the underground equipment between compounds, due to their low speed and poor visibility. The workshops are constructed from light-weight pre-fabricated fabric covered structures with roller doors at one end. A vehicle wash bay has been provided near the workshops at the Marianne-Marilyn, MSNE and Miriam compounds.

17.1.3 Light and Heavy Vehicle Roads

Two classes of road are proposed to support the movement of light vehicles (“LV”) and heavy vehicles (“HV”) so that the interaction between LV and HV is minimised.

- LV roads of 6 m wide will enable traffic movement within the Marianne-Marilyn and MNSE compounds and to the Explosive Storage Area from the Marianne-Marilyn to MSNE road.
- HV roads will enable the movement of the mining fleet between Marianne-Marilyn and MNSE portals and the compounds. An HV road width of 11 m required to permit two-way traffic movement near the underground operations and 14 m near the open pit operation.

SRK is of the opinion that due to the low volume of traffic the pavement construction can be limited to grading and a surface layer of 200 mm of graded lateritic material. Regular grading will be required, but limited to the low rainfall. This is in line with the pavement construction at Somaïr and Cominak operations nearby. It is anticipated that waste rock will be used to maintain the roads for the remainder of the mine life.

17.1.4 Change House

The change house will be located at the Marianne-Marilyn compound and will be capable of accommodating 118 personnel per shift. The anticipated workforce excludes General and Administrative (“G&A”) and process plant personnel as it is anticipated that this personnel will either not require change facilities or their change facilities will be located elsewhere.

17.1.5 Waste Management

Separate waste management areas are proposed at the Marianne-Marilyn, MSNE, and Miriam compounds with each compound comprising an open area for separation of waste streams, non-incinerated waste storage and a hydrocarbon storage tank.

17.1.6 Administration Assets

It has been assumed that the G&A workforce (excluding process plant) will comprise 50 persons per shift. Based on this assumption prefabricated administration buildings are proposed at the Marianne-Marilyn compound which will support mining, maintenance, logistics and services operations and comprise offices, work stations, reception area, IT rooms, training rooms, ablutions and a storeroom. A perimeter road, common parking areas and walkways between the buildings will be provided.

Limited office and welfare facilities are planned at the MNSE and Miriam sites to limit the need

for driving back and forth between these operations and the Marianne-Marilyn compound.

A pre-fabricated medical building to support basic triage will be provided at the Marianne-Marilyn compound. The office building at the MNSE and Miriam sites will include provision for emergency first aid.

A prefabricated laboratory will perform daily analysis of mining samples.

17.1.7 Logistics Warehousing

The logistics warehousing has been designed based on the anticipated demands from the mining operations and will be located within the Marianne-Marilyn compound and adjacent to the main site entrance limiting the risk of errant drivers entering secure and high risk areas. Security and access controls will monitor the movement of goods through the facility.

It is assumed that all consumables will be palletised and transported to site by 40 t articulated vehicles. Any larger item will be delivered by low-loader direct to the appropriate store.

Containerised transportation of goods for commodities that can be palletised is the preferred option and would be handled within designated goods warehouses where full containers can be directly unloaded, offloaded and either set aside whilst awaiting unloading into designated warehousing and empty containers or containers containing waste materials can be loaded.

The warehouses are located adjacent to the proposed vehicle maintenance workshops to permit ease of access. All vehicle movements are controlled through a one way system and a series of security points with separate 20 m wide goods in and 10 m wide goods out reinforced concrete aprons.

It is assumed that approximately 240 Twenty Foot European Unit (“TEU”) per annum will be required for general spares and consumables for mining operations. A single 25 m wide, 65 m long light-weight pre-fabricated fabric covered structure is proposed to accommodate 1,740 pallets equivalent to 4 months consumption.

It is assumed that 20 container flats per 28 day period will be required for non-palletised goods and that 50 % of the contained goods/commodities will require covered storage within a 25 m wide, 65 m long light-weight pre-fabricated fabric covered structure equivalent to 4 months consumption.

An uncovered storage area has been provided for 6 months storage of tyres (to enable curing). Storage for 68 tyres has been provided.

A 10 m wide, 15 m long light-weight pre-fabricated fabric covered structure within a bunded area is proposed to accommodate 50 pallet drums of lubricants and greases equivalent to 1.2 l/operating hour and 4 months consumption.

Given the mode of delivery, it is considered appropriate to utilise a container reach stacker to offload containers and forklift trucks to offload and stack pallets. Goods will be initially stored in a holding area for registering in the goods-in log. It is assumed there will be no requirement for Customs examination, as this will happen in Arlit before they can make delivery.

From the holding area the palleted goods will be transferred to their appropriate storage area within the warehouse by high reach forklifts capable of unloading into a rack storage system consisting of eight banks, 5 pallets high and each 44 pallets long. Alleyways between the banks and warehouse structure provide access for goods movement.

Design of the warehouse facility has taken into account the need to minimise demurrage on

standing containers and assumes that there will be a steady state flow of containers, with empty containers replacing those offloaded for the return journey.

The warehouse will distribute the goods to the MNSE and Miriam compounds or other point of end use by curtain sided trucks.

17.1.8 Fuel Storage and Vehicle Refuelling

It is anticipated that the primary fuel storage will be located within the process plant site and that only a five day supply will be required to support underground operations.

Fuel will be delivered by a bowser from the processing plant to 60 m^3 bunded fuel tanks located at the Marianne-Marilyn, MNSE and Miriam sites based on a daily fuel consumption of 10,500 litres. The tanks will be located within bunded areas and fuel pumps will refuel light and heavy vehicles and also feed underground fuel tanks to support the refuelling of mobile equipment underground.

17.1.9 Security

A prefabricated security building is proposed which will contain basic welfare provision, ablutions and standard communications and security systems. Security buildings will be located at the main site entrance, entrances to secure facilities and at interfaces between the LV and HV road networks.

A 2.4 m high palisade fence line with concertina razor wire has been provided around the Marianne-Marilyn and MSNE compounds. An increased level of security is provided around the magazine perimeter with CCTV, guard posts and regular security patrols. Other areas of the site will be patrolled at regular intervals by security personnel using 4x4 vehicles.

17.1.10 Power Supply

Power is supplied to the project site via an overland transmission line connecting the existing Arlit-Tchirozine infrastructure with the proposed sub-station at the process plant site.

Further details on power supply infrastructure are contained in Section 17.4.

17.1.11 Power Distribution

It is assumed that power will be distributed from the process plant substation via a buried 11 kV buried cable to local substations. At this stage in the design process, it is envisaged that the distribution to the mining plant would be provided via a series of dedicated ring and radial circuits at the primary distribution voltage of 11 kV. Where the equipment requires an intermediate operating voltage, for example 3.3 or 6.6 kV, a local transformer would be provided.

It is proposed that the underground HV/LV distribution network cables will be located at the backs of the underground development, in the central conveyor drive.

Further refinement of power generation costs will be undertaken at the next design phase by optimising the load growth profile in accordance with the overall development programme.

17.1.12 Distribution Control and Monitoring (SCADA)

The electrical network would be operated and monitored by a Supervisory Control and Data Acquisition (“SCADA”) system which would control and monitor all the relevant equipment within the electrical network so the operators would have fast and accurate information of the

network and have the ability to operate the system remotely.

17.1.13 Backup Power Generation

Diesel powered backup generators are proposed to support all critical assets comprising underground ventilation and water pumps at the Marianne-Marilyn and MSNE compounds and lighting and security at the Magazine.

Capacity of the required Backup power is estimated at 2.5 MW at Marianne-Marilyn, 2.5 MW at MNSE, 0.5 MW at Miriam and 0.3 MW at the Magazine.

17.1.14 Lighting

External LED lighting has been proposed at the following main areas:

- flood lighting to vehicle maintenance areas;
- road and access lighting at junctions; and
- critical and high security facilities.

17.1.15 Communications

An allowance has been made for the following site and project wide communications infrastructure:

- Business data network and server/computer systems;
- Voice over IP (“VoIP”) data telecommunications system within the mine site;
- Radio over IP (“RoIP”) for 2-way UHF radio services within the mine site;
- WIFI throughout mine site primarily for data network access for mobile plant;
- Satellite voice and data communications links;
- Access control system to establish access permissions and a live record of personnel in restricted areas;
- Security and surveillance system to provide area surveillance, detection and assessment;
- Intrusion and detection alarms for key locations;
- SCADA field units, central control, masts, tower and RoIP system; and
- Control room requirements and systems.

17.1.16 Potable Water

It is assumed that potable water pipeline will supply Marianne-Marilyn and MSNE compounds from the main processing plant. Both compounds will need to provide potable water for 350 operational staff per day with 50 G&A staff giving a total of 400 people per day.

A consumption of 110 L/person/day has been assumed which will require 44,000 litres/day in order to provide a domestic flow rate of approximately 1 L/s. An 80 mm diameter HDPE rising main is proposed to accommodate a peak flow up to 5 L/s within each compound.

The potable water for the Miriam and Magazine operations will be supplied by a water bowser as construction of a potable water line over the additional 8 km distance is not deemed warranted by SRK.

17.1.17 Foul Water

The foul water has been designed to accept 110 litres/person/day from 400 people.

The Marianne-Marilyn site foul network collects the foul water from the change house and administration block. A 150 mm diameter pipe was sized as appropriate to convey the waste to a package sewerage treatment plant. The tank will discharge treated water to the ground via a reversed land drain arrangement. The sludge within tank will periodically require emptying and disposal at an appropriate waste location.

The MSNE, Miriam and Magazine sites will require a foul network that will accept only waste from the welfare building. The foul network will discharge to a septic tank which will discharge treated water to the ground via a reversed land drain arrangement and will also require periodical emptying of the sludge.

17.1.18 Fire and Dust Suppression

It is assumed that fire and dust suppression will be undertaken by a water bowser, which will be filled at the Marinne-Marilyn, MSNE and Miriam compounds. Building fire suppression sprinkler systems are proposed within each surface infrastructure building.

17.2 Explosives Magazine

The Magazine is required to store all components of the explosives mixture. Due to the type and quantity of material to be stored, the Magazine area is located 500 m from all other infrastructure. It is proposed to locate the Magazine to the east of the main LV road connecting Marianne-Marilyn and MNSE mines.

It is assumed that material delivery will occur once every 6 months, therefore the explosive storage area has been designed accommodate 8 months consumption of prill, detonators, detonator cords and primers. A batching plant and diesel storage area are also proposed.

17.2.1 Infrastructure Objects Magazine

It is anticipated that prill will be stored in bulk bags and a 25 m wide, 50 m long light-weight pre-fabricated fabric covered structure will accommodate 1,470 t of prill based on a daily consumption of 6,050 kg.

Primers will be stored in palletized boxes within five 20’ ISO containers, equivalent to 40 t of primers. Detonators will be stored in palletized boxes within five 40’ ISO containers, equivalent to 720,000 detonators. Detonator cords will be stored in palletized boxes within two 20’ ISO containers, equivalent to 500,000 m of detonator cords.

A bunded fuel tank within a bunded area will contain 2,500 l diesel based on a daily consumption of 440 L (6 % by mass of prill), equivalent to 5 days consumption.

A light-weight pre-fabricated fabric covered structure will house the batching plant.

Each storage area and the batching plant will be surrounded by a 3 m high bund with 50 m between each storage area. A triple fenceline is proposed around the perimeter of the Magazine. Four standard lighting columns and elevated guard towers are proposed on each corner of the Magazine to provide increased visibility whilst security bunkers are proposed at regular intervals around the Magazine perimeter.

17.3 Conceptual Layouts

The following sections present the conceptual layouts for the Marianne-Marilyn, MSNE, Miriam and Magazine compounds and the visualisation based on these conceptual layouts.

17.3.1 Marianne-Marilyn Compound

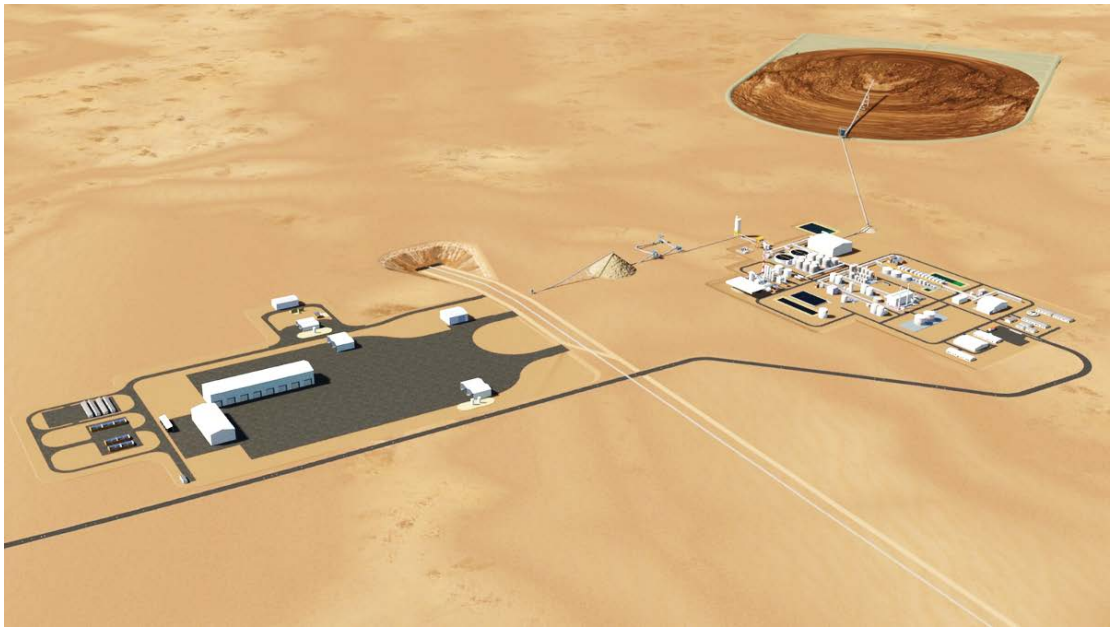


Figure 17-2: Visualisation of the Marianne-Marilyn conceptual compound layout, plant and portal

17.3.2 MNSE/Miriam Compounds

The MNSE and Miriam compounds have been designed and costed in the same manner and the following figures present the MSNE conceptual layout and the associated visualisation.

Figure 17-3: Visualisation of the MSNE conceptual compound layout and portal

17.3.3 Magazine Compound

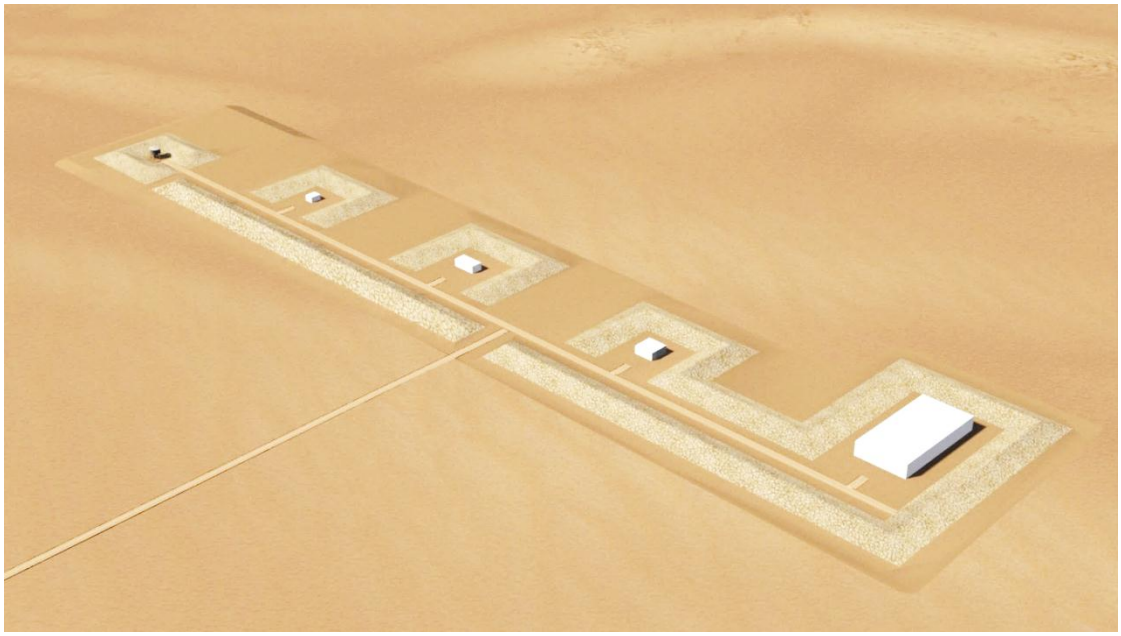


Figure 17-4: Visualisation of the conceptual Magazine Layout

17.4 Power Supply

An existing 132 kV overhead transmission line connects the SONICHAR power station near Tchirozérine with the town of Arlit. The transmission line passes approximately 9.5 km southwest of the proposed Madaouela development. SONICHAR confirmed in November 2012 that the transmission line has an available capacity of 22 MW.

SRK and Tenova have estimated the maximum power demand as 17 MW, comprising 5 MW for the mine and up to 12 MW for the process plant pending the process option chosen.

SRK propose the construction of the following infrastructure:

- 9.5 km long 110 kV transmission line; and
- Substation (110/33 kV, 12-15 MVA) on the Madaouela site boundary.

17.5 Capital Costs

Table 17-1 presents the anticipated CAPEX for surface infrastructure, which have been derived to an accuracy of ± 25 %.

Table 17-1: Surface infrastructure CAPEX

M&M		**USD 12,630,000**
Operations		USD 3,382,000
Support		USD 1,208,000
Services		USD 8,041,000
MSNE		**USD 2,771,000**
Operations		USD 913,000
Support		USD 207,000
Services		USD 1,652,000
Miriam		**USD 2,347,000**
Operations		USD 1,438,000
Support		USD 207000
Services		USD 702,000
Explosives Store		**USD 2,950,000**
Operations		USD 1,059,000
Support		USD 802,000
Services		USD 1,091,000
Power Transmission to site		**USD 3,860,000**
Mobile equipment		**USD 1,052,000**
Total Direct CAPEX		**USD 25,610,000**
Indirect CAPEX		
EPCM Cost	18 %	USD 4,420,000
Engineering Cost	5 %	USD 1,228,000
Total Indirect CAPEX		*USD 5,648,000*
NET CAPEX		**USD 31,258,000**

17.6 Operating Costs

Table 17-2 presents the agreed labour rates used to define the surface infrastructure OPEX as presented in Table 17-3.

Table 17-2: Labour rates

Category	Hourly Rate (USD)
Labour Unskilled – Local	2.31
Labour Semi Skilled – Local	3.85
Security	4.25
Supervisor	21.15
Manager	38.46

Table 17-3: Surface infrastructure OPEX

Area		Annual OPEX (USD)
M&M		**USD 473,000**
Road Maintenance		USD 69,000
Communications		USD 45,000
Building Maintenance		USD 47,000
Warehousing		USD 86,000
Power Consumption		USD 227,000
MSNE		**USD 303,000**
Road Maintenance		USD 75,000
Power Consumption		USD 227,000
Miriam		**USD 236,000**
Road Maintenance		USD 8,000
Power Consumption		USD 227,000
Magazine		**USD 317,000**
Magazine management		USD 66,000
Security		USD 219,000
Power Consumption		USD 31,000
Madaouela Surface Infrastructure – Total		**USD 1,328,000**

17.7 TRANSPORT AND LOGISTICS

17.7.1 Introduction

This section discusses potential logistics scenarios to transport the anticipated quantities of reagents and consumables required to support the proposed Madaouela mine and process plant.

This assessment will assess existing regional infrastructure to establish multi-modal solutions to transport reagents and consumables to Arlit, approximately 12 km north-west of the proposed development.

This assessment will not consider the construction of new materials handling and/or transport infrastructure. It is assumed that the provision of storage facilities at ports and/or transitions between transport modes will be the responsibility of the transport suppliers and any associated costs will be included in transportation rates.

The data presented in this section is based on a desktop review and as such the condition or capacity of the infrastructure considered has not been verified. However, this assessment should be used to inform discussions with suppliers and haulage operators so that viable transportation costs can be established. Logistics costs used in the study are based on quotation.

17.7.2 Operational Requirements

It is anticipated that the proposed Madaouela process plant will consume:

- 320 t of caustic soda per month;
- 260 t of sulphur per month; and
- 1000 t of other reagent and consumables per month.

It is anticipated that sulphur will be transported in covered bulk material containers and all other reagents/consumables will be transported in ISO 25T shipping containers.

17.7.3 Reagent/Consumable Sources

The Project initially considered sea ports in Algeria, Tunisia, Libya, Nigeria, Benin, Togo, Ghana and Cote d'Ivoire based on their regional proximity to Niger.

It is proposed that only sea port facilities with existing container and bulk material handling infrastructure will be considered so that logistics operations and associated security are centralised. The ports identified that satisfy this criteria presented in Figure 17-5.

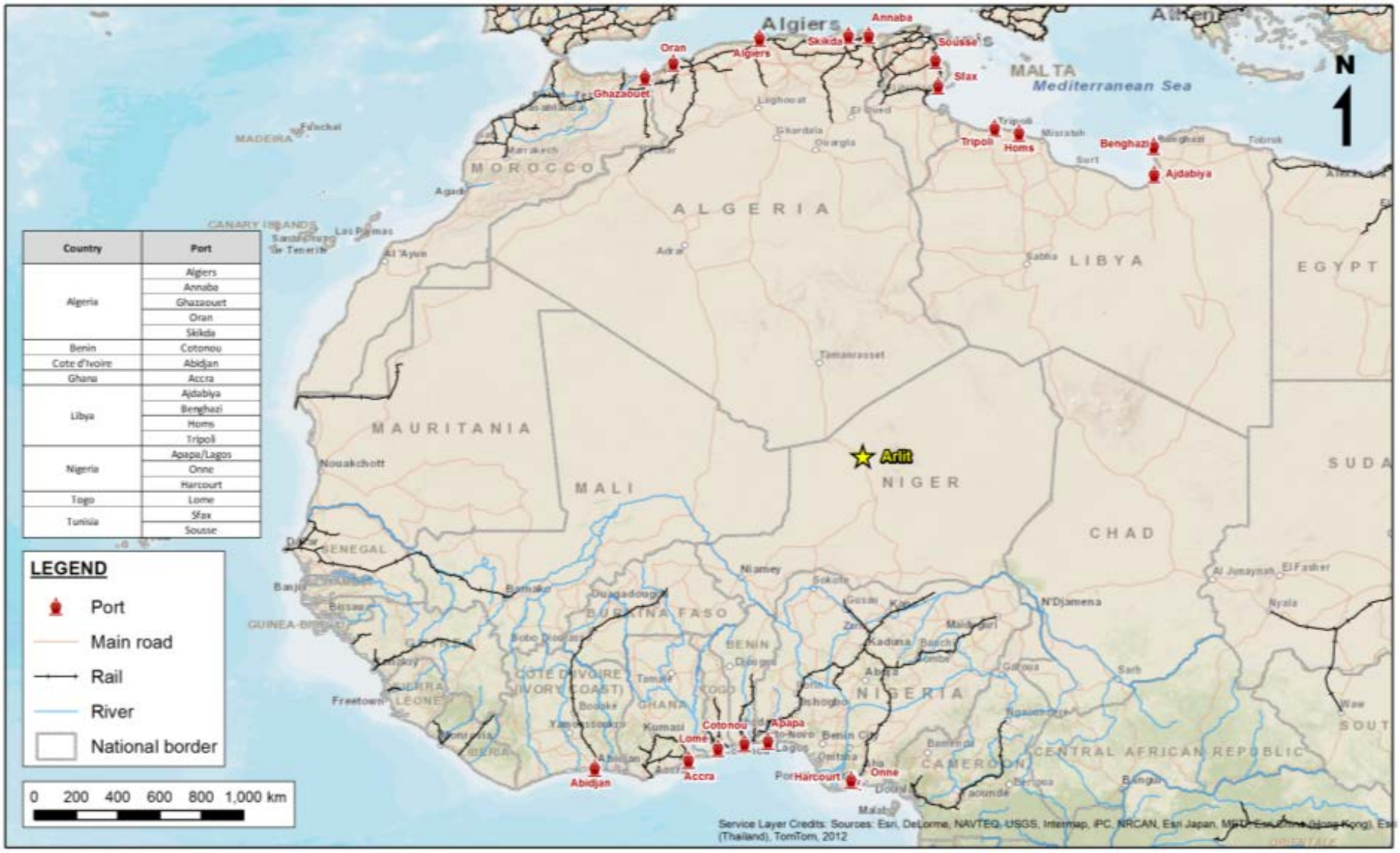


Figure 17-5: Existing regional infrastructure

17.7.4 Existing Regional Infrastructure and feasible logistical options

Introduction

The following sections assess existing infrastructure within the countries considered and potential logistics options to Arlit.

Niger Infrastructure

Niger does not currently contain rail infrastructure. There have been numerous proposals to extend neighbouring railway lines into Niger. The closest railway network to Niger is located in the north of Nigeria.

In May 2015 the governments of Niger and Benin signed a deal to finalize the terms of the construction and operation of a railway linking Niamey with the port of Cotonou, expected to be finished in the middle of 2016.

French firm Bollore will cover the entire 1.07 billion euro cost of building the rail link, according to the reported terms of the deal. Bollore will own a 40 percent stake in the operator BENI rail, in which the two countries will each own 10 percent. The remaining stake will be owned by private investors from Niger and Benin.

Work began in April 2014 on a 574 km railway to join Niamey to the eastern Benin town of Parakou, which has an existing rail link to Cotonou. The line is expected to carry mostly freight.

The link would form part of a planned 2,800 km network joining up Ivory Coast, Benin, Burkina Faso, and Niger.

There are good north-south road links from Arlit and the road network within Niger comprises of a mixture of bituminous and laterite surfaced roads.

Algeria

Algeria shares its southern border with Niger and satellite imagery suggests there are five ports capable of receiving and handling containers and bulk material. Utilising a port in Algeria would only require crossing one international border.

Existing Rail Infrastructure

All ports have connections to the national standard and narrow gauge rail infrastructure which extends approximately 330 km south terminating at Djelfa. Initial observations from satellite imagery indicate that Djelfa station contains a road truck parking area and formal infrastructure to permit the transfer of containers from trains to road trucks.

Rail connections from Algiers to Djelfa are shorter than those from Bejaia and so only a rail option from Algiers has been considered.

Existing Road Infrastructure

Algeria contains approximately 110,000 km of roadways of which approximately 75% is of bituminous construction. It is anticipated that the N1, which starts to the south west of Algiers, would be the main road utilised to travel down to Niger.

Potential Logistics Options

Seven potential logistics options identified (Table 17-4) for commercial assessment.

Table 17-4: Algeria Logistics options

Option	Port	Rail transportation to Djelfa (km)	Road Transportation to Arlit (km)	Total Distance (km)
1a	Algiers	329	2260	2589
1b	Algiers	n/a	2553	2553
1c	Annaba	n/a	2791	2791
1d	Ghazaouet	n/a	2595	2595
1e	Oran	n/a	2647	2647
1f	Skikda	n/a	2735	2735
1g	Bejaia	n/a	2627	2627

Tunisia

Tunisia shares its western border with Algeria and satellite imagery suggests there are three ports are capable of receiving and handling containers and bulk material. Utilising a port in Tunisia would require crossing two international borders.

Existing Rail Infrastructure

All ports have connections to the national standard and metre gauge rail infrastructure that crosses into Algeria near Ouled Moumen. Rail infrastructure extends to Tozeur which is the closest station to the Niger border, however initial observations from satellite imagery indicate that the station does not contain formal road/rail transfer infrastructure. Therefore no rail logistics options have been considered.

Existing Road Infrastructure

Tunisia has a 19,000 km long road network of which approximately 60% is of bituminous construction.

Potential Logistics Options

Three potential logistics options identified (Table 17-5) for economic assessment.

Table 17-5: Tunisia Logistics Options

Option	Port	Road Transportation to Arlit (km)
2a	La Goulette	2987
2b	Sfax	2819
2c	Sousse	2878

Libya

Libya shares its western border with Algeria and is currently re-building its infrastructure following the recent civil war. Satellite imagery suggests there are four ports are capable of receiving and handling containers and bulk material. Utilising a port in Libya would require crossing two international borders.

Existing Rail Infrastructure

There are no operational railways in Libya and plans to construct a 3,170 km long national rail network were suspended in 2012 due to the civil war.

Existing Road Infrastructure

Approximately 50% of Libya's 100,000 km road network are of bituminous construction.

Potential Logistics Options

Four potential logistics options identified (Table 17-6) for economic assessment.

Table 17-6: Libya Logistics Options

Option	Port	Road Transportation to Arlit (km)
3a	Ajdabiya	3155
3b	Benghazi	3316
3c	Homs	2650
3d	Tripoli	2553

Nigeria

Nigeria shares its northern border with Niger and satellite imagery suggests there are three ports are capable of receiving and handling containers and bulk material at Onne, Port Hardcort and Appapa.

The harbours at Onne and Port Hardocort are located within the same estuary to the south east of the country and are currently heavily utilised by the oil industry.

High risk of piracy and armed robbery faced by commercial shipping recognised within the gulf of Guinea around Nigeria[4].

Existing Rail Infrastructure

Nigeria has a 3,505 km long, predominantly narrow gauge track rail network with a vast number of stations that vary in their capabilities to handle rail cargo. The Nigerian Railway Corporation's (NRC) is engaged in freight train operations throughout the country.

The ports of Hardcourt and Appapa have connections to national rail infrastructure. Export from Onne could potentially utilise rail infrastructure but would require road haulage and additional handling to transport the material to the rail network.

The rail infrastructure from Appapa and Port Harcourt merge at Kaduna and branch lines terminate in Kaura Namoda and Nguru, however neither of these stations have sufficient infrastructure to receive and handle containers and bulk materials.

Initial observations of satellite imagery indicate that stations at Kano (220 km south-west of Nguru) and Gusau (50 km south of Kaura Namoda) potentially have sufficient rail/road interface infrastructure to support the transfer of containers to trucks. However, further investigation will be required to confirm the validity of the above observations.

[4] https://www.cia.gov/library/publications/the-world-factbook/geos/ni.html

Existing Road Infrastructure

Nigeria has an approximate 190,000 km long road network of which GoviEx and SRK understands only 30,000 km is of bituminous construction. There are numerous routes between Nigeria and Niger utilising the A1/N29 from Appapa/Guasau or the A2/N10 from Port Harcourt/Kano.

River Transportation

SRK has considered the potential for barge transportation along the Benue or Niger rivers, however initial observations of satellite imagery suggest the viability of these options would be limited without significant capital investment.

Potential Logistics Options

Seven potential logistics options identified (Table 17-7) for economic assessment.

Table 17-7: Nigeria Logistics Options

Option	Port	Rail transportation (km)	Road Transportation to Arlit (km)	Total Distance (km)
4a	Appapa, Lagos	1131 (to Gusau)	1081	2212
4b	Appapa, Lagos	1107 (to Kano)	960	2067
4c	Appapa, Lagos	n/a	1839	1839
4d	Onne	n/a	2001	2001
4e	Port Harcourt	1149 (to Guasau)	1081	2230
4f	Port Harcourt	1125 (to Kano)	960	2085
4g	Port Harcourt	n/a	1990	1990

Benin

Benin shares its eastern border with Nigeria, northern border with Niger and satellite imagery suggests there is one port at Cotonou capable of receiving and handling containers and bulk material.

Existing Rail Infrastructure

A 400 km long meter gauge rail line links Cotonou with Parakou, located approximately 320 km south of the Niger border. GoviEx and SRK understands that Parakou station has sufficient rail/road interface infrastructure to receive and transfer shipping containers to trucks.

Existing Road Infrastructure

The main road route through Benin is the RNIE2 which travels parallel with the rail line from Cotonou towards Niger.

Potential Logistics Options

Two potential logistics options identified (Table 17-8) for economic assessment.

Table 17-8: Benin Logistics Options

Option	Port	Rail transportation (km)	Road Transportation to Arlit (km)	Total Distance (km)
5a	Cotonou	432	1502	1934
5b	Cotonou	n/a	1928	1928

Togo

Togo does not border Niger but shares its eastern border with Benin and satellite imagery suggests there is one port at Lome capable of receiving and handling containers and bulk material.

Existing Rail Infrastructure

A 298 km long meter gauge rail line links Lome with Blitta, however it is unclear from satellite imagery whether Blitta has sufficient rail/road interface infrastructure to receive and transfer shipping containers to trucks. Therefore only road transportation has been considered.

Existing Road Infrastructure

Togo has a 7,000 km long road network of which GoviEx and SRK understands only 30% is of bituminous construction. A direct route from Lome to Arlit would cross into Benin and then principally follow the route from Cotonou to Niger.

Potential Logistics Options

One potential logistics option identified (Table 17-9) for economic assessment from Togo. However, given the similar routes but longer distance, transportation from Lome may only be economically viable should the capacity and availability of port facilities at Lome be considered more favourable than those at Cotonou.

Table 17-9: Togo Logistics Option

Option	Port	Road Transportation to Arlit (km)
6a	Lome	2010

Ghana

Ghana does not border Niger but shares its eastern border with Togo and satellite imagery suggests there is one port at Tema (25 km east of Accra) capable of receiving and handling containers and bulk material.

Existing Rail Infrastructure

Although the Tema port is connected to rail infrastructure, utilising Ghana's rail is not considered viable as it generally travels east to west and does not travel in the direction of Niger.

Existing Road Infrastructure

Road transportation from Ghana is considered viable as the direct route, but does involve crossing into Togo and Benin prior to crossing into Niger. The total distance to Niamey is just less than 1,200 km and is connected by roads that are in good conditions and are, for the majority, made up of asphalt.

Potential Logistics Options

No potentially viable logistics options in Ghana identified.

Table 17-10: Ghana Logistics Options

Option	Port	Road Transportation to Arlit (km)
7a	Tema	2210

Côte d'Ivoire

Côte d'Ivoire does not border Niger but shares its eastern border with Ghana and northern border with Burkina Faso. GoviEx understands the port of Abijan is capable of receiving and handling containers and bulk material.

Existing Rail Infrastructure

A 1,300 km long meter gauge rail line extends from the Abijan port facility to Ouagadougou in Burkina Faso and is the main route used to import material into Burkina Faso's capital. Ouagadougou station has sufficient rail/road interface infrastructure to receive and transfer shipping containers to trucks for onward transport to Arlit.

Existing Road Infrastructure

The roads within Cote d'Ivoire are a mixture of bituminous and non-bituminous construction.

Potential Logistics Options

One potential logistics options identified (Table 17-11) for economic assessment from Côte d'Ivoire.

Table 17-11: Côte d'Ivoire Logistics Options

Option	Port	Rail transportation (km)	Road Transportation to Arlit (km)	Total Distance (km)
8a	Abijan	1260	1705	2965

17.7.5 Economic Assessment

Introduction

Section 17.7.4 identified 25 potentially technically viable logistics options, however based on current security risks GoviEx considers that importing through Nigeria and Libya present

increased operational risk and so will not be considered further. Therefore, a total of 14 logistics options are considered for economic assessment.

Assumptions and Exclusions

Based on a database of similar transportation activities from SRK standard transportation unit costs (Table 17-12) to facilitate an initial economic assessment.

Table 17-12: Assumed Transportation Costs

Mode of Transport	Unit	Rate
Rail	USD/t/km	0.10
Road	USD/t/km	0.08

The above transport rates have been used to assess all options assuming similar costs for road and rail infrastructure in the various countries. This approach provides suitable means to assess and compare the various options, however it should be noted that the above rates do not include:

- Stowage and port scanning costs;
- port customs import duty and handling costs;
- down time costs;
- no container purchase or rental costs;
- international border customs import/export duty costs;
- country specific road and rail maintenance charges;
- security;
- dangerous merchandise fee;
- cargo insurance; and,
- Construction of new infrastructure to facilitate the handling, storage and transportation of containers and bulk materials.

Economic Assessment

Table 17-13 presents the indicative transportation unit costs for the various logistics options.

Table 17-13: Indicative Transport Costs to Arlit

Option	Incoming Country	Incoming Port	Logistics to Arlit	Logistics Unit Cost / Tonne (USD)
5b	Benin	Cotonou	Road	154
5a	Benin	Cotonou	Rail and Road	163
6a	Togo	Lome	Rail and Road	172
7a	Ghana	Tema	Road	176
1b	Algeria	Algiers	Road	204
1d	Algeria	Ghazaouet	Road	208
1g	Algeria	Bejaia	Road	210
1e	Algeria	Oran	Road	212
1a	Algeria	Algiers	Rail and Road	214
1f	Algeria	Skikda	Road	219
1c	Algeria	Annaba	Road	223
2b	Tunisia	Sfax	Road	226
2c	Tunisia	Sousse	Road	230
2a	Tunisia	La Goulette	Road	239
8a	Cote d'Ivoire	Abijan	Rail and Road	262

The indicative unit costs presented in Table 17-13 show that importing from Cotonou, Benin and Lome, Togo are the most cost effective logistics options for transporting processing reagents and consumables to Arlit.

The distances associated with crossing the Sahara desert render ports in North Africa less economically favourable.

Although the indicative unit cost used for rail transportation is greater than that for road transportation, there are benefits in utilising rail network compared to road haul. The main benefits are that large quantities of material can be conveyed per train consist and so the reduced traffic creates less potential disruption to local populations and existing transport infrastructure.

It should be noted that the above analysis is based on indicative unit rates and are based on the assumptions and exclusions discussed within this Section.

Inland Transportation

A more detailed analysis of Cotonou, Lome and Tema was undertaken as these ports are the forecast lowest cost options. The analysis was completed with the assistance of SDV/Bollore, who provide similar services to other uranium mining companies on Niger.

Bolloré provides land transportation services including collection, delivery and, international transportation of cargo. Bolloré has extensive fleets of vehicles worldwide and offers a full range of vehicles, trailers, vans, modular equipment and specialist trailers. Due to the range

of their fleet, SDV is able to transport a wide variety of cargo, ranging from the smallest parcel to bulk aggregate deliveries in developing nations.

Bolloré has an extensive range of equipment in country allowing transport of a large range of cargo and materials, and has a fleet consisting of mechanical horses and trailer units comprised of inter-links, semi-trailers, lowbeds and tipper trailers, and has the ability to transport containerised, break-bulk and loose form cargo.

- Fleet of 6,000 vehicles supported by preferential agreements with reliable local hauliers;
- Own workshops where for fleet maintenance technical quality checks of the equipment used by our sub-contractors;
- Long-distance fleet with GPS tracking.

The Madaouela Project is considering a service that requires Corridor Management based on the following routes by truck from Niamey to:

- Arlit: 950 km (average transit time including customs clearance = 2 days);
- Cotonou: 980 km (average transit time not including customs clearance = 14 days);
- Lome (via Ouagadougou): 1,120 km (average transit time not including customs clearance = 18 days);
- Tema (via Ouagadougou): 1,260 km (average transit time not including customs clearance = 18 days).

Routes to Niger

There are currently four (4) established routes (corridors) to Niger; via Cotonou to Niamey, via Lomé to Niamey, via Abidjan to Niamey, and via Tema to Niamey.

Bollore is very familiar with all existing routes and, where required, and maintains existing route surveys for each corridor. Proven expertise in the management of transit corridors and customs clearance, allows optimisation of long-distance transport times.

Table 17-14: Transport Corridor Options

CORRIDOR	ESTIMATE TRANSIT TIME	STRENGTHS	WEAKNESSES
TEMA	22 days		▪ Port congestion up to 3 days
ABIDJAN (by Road)	22 days	▪ No Port congestion ▪ Equipment availability and expertise ▪ Heavy Lifts and Out of Gauge cargo	
ABIDJAN (by Rail)	24 days	▪ Regular Transit time ▪ No delay at the border ▪ Risk of accident=zero ▪ No Road Bond Guarantee	▪ Transhipment in Ouagadougou from rail onto truck bound to Niger
LOME	22 days	▪ No Port congestion	▪ Road conditions
COTONOU	21 days	▪ No Port congestion	▪ Road conditions

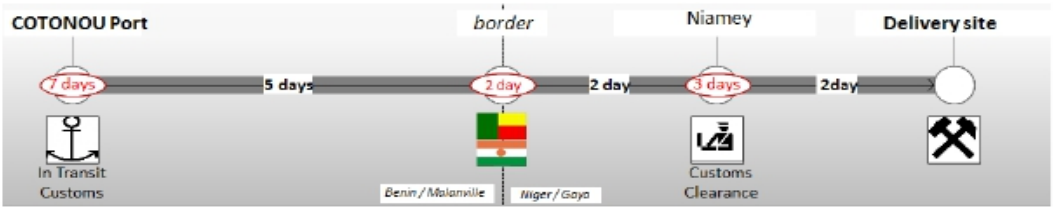
The decision of the optimal transportation route is a complex decision and one which requires the consideration of multiple variables, and may change over time. It is necessary that the

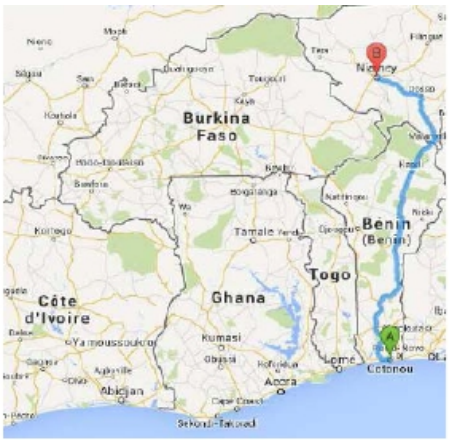
chosen option will allow for the secure transport of cargo whilst minimizing transit times and costs. As such, a non-exhaustive list of key factors which must be considered include:

- Nature of the Cargo: Containerised, Out of Gauge or Break Bulk.
- Port of Entry: Distance from Final Destination
- Customs Regimes: Processing Times,
- Onward Transportation: Available of Trucking Fleet, Road Conditions, Security.

The forthcoming sections analyse each of the options in the context of these variables.

Cotonou to Niamey





The corridor through Benin and up to Niamey takes approximately 21 working days, including customs clearance and regular formalities.

The total distance to Niamey is approximately 1,000 km and is connected by roads that are in average conditions and are, for the majority, made up of asphalt.

Following an international call for bids supervised by the World Bank (IFC), the Port Authority awarded the 25-year concession for the new container terminal at the port of Cotonou to Benin Terminal, a Bolloré Africa Logistics company through a public- private partnership

Port of Entry – Cotonou

The Port of Cotonou is one of the largest ports in West Africa and acts as a gateway not just to Togo itself but also to the region's landlocked countries.

Bolloré Africa Logistics has a strong and established presence at the port. As mentioned the Port Authority awarded the 25-year concession for the new container terminal at the port of Cotonou to Benin Terminal, a Bolloré Africa Logistics company through a public-private partnership.

Access Specifications

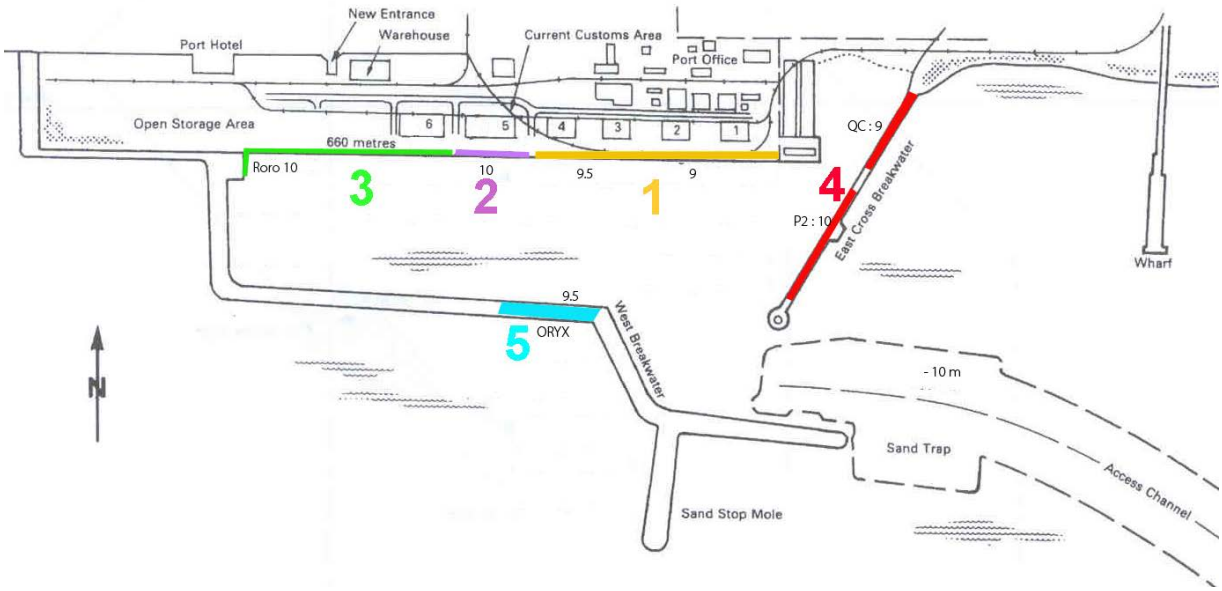


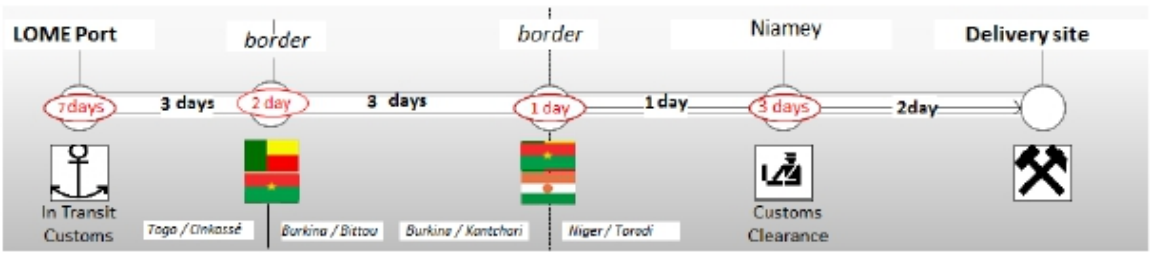
Table 17-15: Key to Cotonou Port map

REF	MAP REFERENCE NUMBER AND DESCRIPTION	LENGTH (M)	DRAUGHT (M)
1	Berths1,2,3: General Cargo	620	9.25
	Berth 4: General Cargo	620	9.5
2	Berth 5: Car carriers/ Heavy general cargo	200	9.5-10
3	Berths 6,7,8: Containers & roro vessels	500	10
4	Berths 9,10: Container	500	13
5	East breakwater/jetty QC: oil, cereals, palm oil	400	10
	East breakwater/jetty P2: Petroltanker, clinker, gypsum, slag		
6	Oryx berth/Puma energy: private berth for tanker Vessels 200 or bunkering operations by Addax/Oryx		9.5-10

Port Congestion

Waiting times in Cotonou are contingent on the type of vessel. Container ships are usually allowed to berth within a number of hours however for other vessels it can take up to 7 days.

Lomé to Niamey



The Port of Lomé is the main Togolese port with more than 80% of the country's trade passing through each year. The port acts as a gateway for its neighbours of Burkina Faso, Mali and Niger by providing a port of entry for cargo in transit regionally.

The group has a strong presence in Togo owing to its award of the concession of the container terminal at the Port of Lomé. As such, Bolloré Africa Logistics through its subsidiary – Togo Terminal – has implemented a programme to modernize and extend the container terminal at the port of Lomé including the construction of a third 450 m wharf and increasing the draught in the port to 15 m.

Access Specifications

The map below provides an overview of the Port's Quays and Terminals.

Figure 17-6: A Map of the Port of Lomé

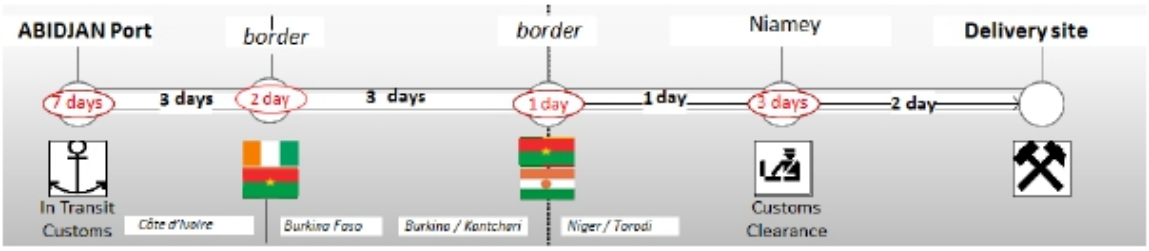
Table 17-16: Key to Lome Port map

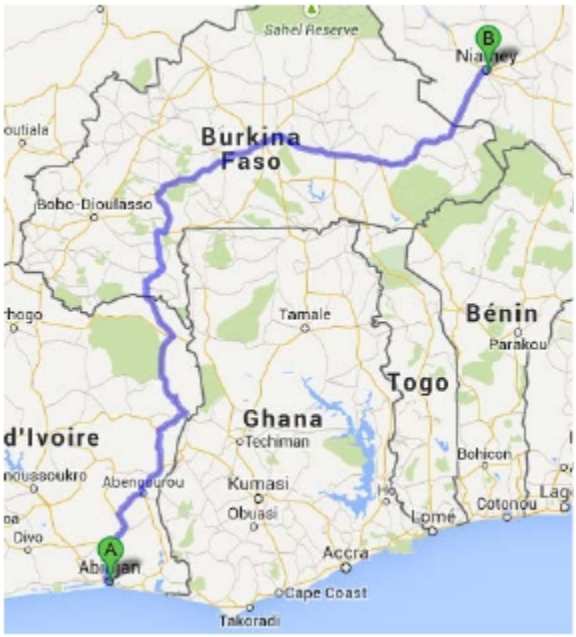
REF	MAP REFERENCE NUMBER AND DESCRIPTION	LENGTH (M)	DRAUGHT (M)
1	General cargo (1)	150	6
2	General cargo (2)	150	10
3	General cargo (3)	150	10
3	General cargo (4)	150	8.5
2	Containers (5)	230	11
2	Containers (6)	210	12
7	Terminal Container	450	15
3	Mineral berth	190	12
	Tanker berth	187	14

Port Congestion

Average waiting times are contingent on the vessel type. For liner services, the waiting time is on average 4 hours. For break bulk and other vessels, waiting times extend to 4 days on average.

Abidjan to Niamey





The route between Abidjan and Niamey is approximately 1,600 km and stretches across both Ivory Coast and Burkina Faso – the current route is considered good in condition.

Including formalities and customs procedures the current time frame for delivery is 22 working days. In 2004 Bolloré Africa Logistics was awarded the 15-year concession for the container terminal at the port of Abidjan

The group also operates a rail concession that provides a link between Abidjan and Ouagadougou in Burkina Faso.

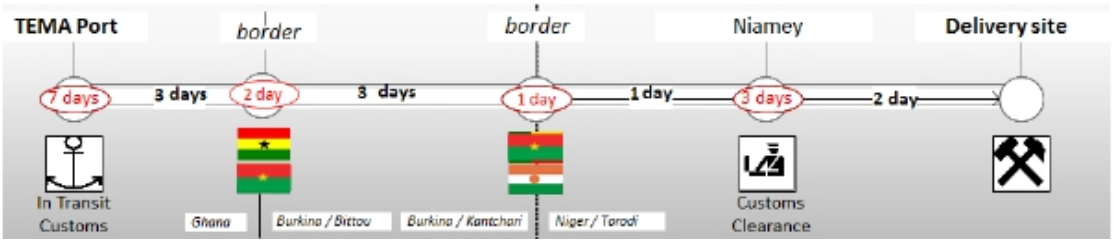
Rail Concession (Citirail)

SITARAIL is a subsidiary of Bolloré Africa Logistics based in Abidjan, operating railways in Côte d'Ivoire and Burkina Faso.

There are daily departures to Ouagadougou whereby cargo will be transferred to trucks and transported onto Niamey. The transit time from Abidjan through to Ouagadougou is 3-4 days.

Bolloré Africa Logistics also offers a storage/marshalling yard in Ouagadougou for offloading/Loading operations.

Tema to Niamey



As a result of an effective investment policy supported by a qualified workforce, productivity at the terminal has risen constantly every year, making MPS one of the most efficient terminals in Africa.

In 2007 Bolloré Africa Logistics was awarded the 20-year concession for the container terminal at the port of Tema.

The group currently provides a complete corridor solution through Tema to Niamey via Burkina Faso.

The route takes 22 days, which includes customs clearance formalities included.

The total distance to Niamey is just less than 1,200 km and is connected by roads that are in good conditions and are, for the majority, made up of asphalt.

Figure 17-7: Tema Port Layout

The port only has two deep-water berths of 11.5m for which there is frequent congestion. The port often lightens vessels and shifts them to the shallow draft berths. The port has 12 berths located on two quays with a total paved quay apron of 2,196 m².

- Berths 1 & 2: On quay 2 is a to containers ships.
- Berths 3-11: Multi-purposed berths and berth 12 is a clinker berth

Transport Cost Validation

As part of the logistics study GoviEx have obtained transportation costs from Bolloré to transport material to Arlit from Cotonou, inclusive of administrative cost. Based on the quotation SRK has been instructed by GoviEx to use a unit rate of USD190/t to transport process reagents/consumables from Cotonou to Arlit.

17.7.6 Conclusion

The indicative assessment presented within this memorandum and those provided by Bolloré both recommend that road transportation from Cotonou in Benin represents the most cost effective option to transport process reagents/consumables to Arlit.

Bollore have provided In accordance with the clients' instruction a transportation unit rate of USD190/t will be incorporated into the Technical Economic Model.

18 MARKET STUDIES AND CONTRACTS (ITEM 19)

This section aims to provide an overview of the fundamental principles of the uranium market and how the derived U_3O_8 is sold into the market; transported; and transformed for use in nuclear reactors. As such the following elements will be described in order to:

- Understand the position and role of uranium within the nuclear fuel cycle;
- Analyse U_3O_8 demand with particular reference to the U_3O_8 requirements of the world's Reactors;
- Explain the transformation of U_3O_8 into UF_6 and the role of the Conversion Facilities who provide such a service;
- Summarise the requirements for transportation of U_3O_8 from GoviEx's Madaouela Uranium Project to the Conversion Facilities;
- Examine the contractual relationship between GoviEx as the Uranium Producer and the Conversion Facilities;
- Analyse the uranium market prices and pricing mechanisms;

18.1 Nuclear Fuel Cycle

The "nuclear fuel cycle" includes all nuclear operations ranging from the mining of uranium ore to the reprocessing of spent fuel and the ultimate radioactive waste disposal.

The fuel cycle is made up of a series of processes that manufacture reactor fuel, burn the fuel in a reactor to generate electricity, and manage the spent reactor fuel (Figure 18-1). These processes are grouped into three components, the front end, which includes all activities prior to placement of the fuel in the reactor, the service period, when the fuel is converted into energy in the reactor, and the back end, which covers all activities dealing with spent fuel from the reactor. If the spent fuel is sent to storage, the cycle is referred to as open. If it is reprocessed to recover useful components, it is known as closed. The United States employs an open fuel cycle, while France, Russia, China, and Japan reprocess their spent fuel.

The components of the cycle are organized as follows within this case study.

The front end

- Uranium Metallurgy, Conversion to Uranium Hexafluoride, and Fabrication of Fuel Rods
- Uranium Enrichment

The service period

- Nuclear Reactors

The back end

- Reprocessing Spent Fuel
- Nuclear Waste

Figure 18-1 below, identifies the key process steps required for both the Front End and Back End activities within the nuclear fuel cycle. (European Nuclear Society, 2003)

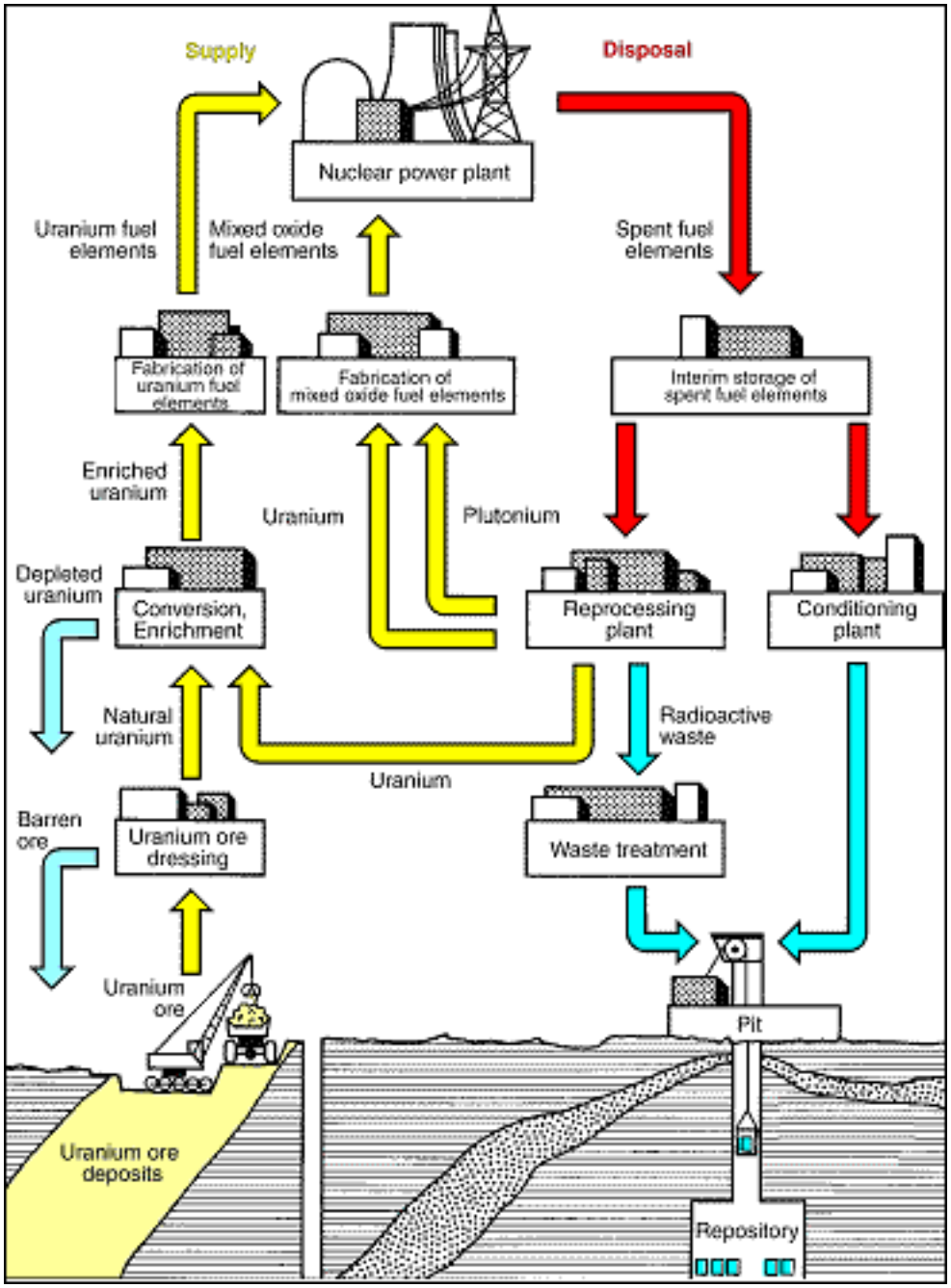


Figure 18-1: Key process steps required for Front End and Back End activities within the nuclear fuel cycle

The ' front-end' of the cycle begins with the extraction of the uranium ore by mining, and then,

- It is milled to refine the material to U_3O_8. The U_3O_8 is packed in a 55 US gallon drum (UN 1A2W) which are then transported from the mill to a UF_6 Conversion facility.
- Once the U_3O_8 has been converted to UF_6 at the Conversion Facility it this then transported to an Enrichment facility.
- At the Enrichment facility the U-235 isotope is concentrated from 0.711% up to a maximum of 5%.
- Following Enrichment, the EUP is produced which is transported to a Fuel Fabrication facility where the fuel that powers the Reactor is manufactured

- The nuclear fuel is transported from the Fuel Fabrication facility to the utility site where it is loaded into the Reactor to generate electricity

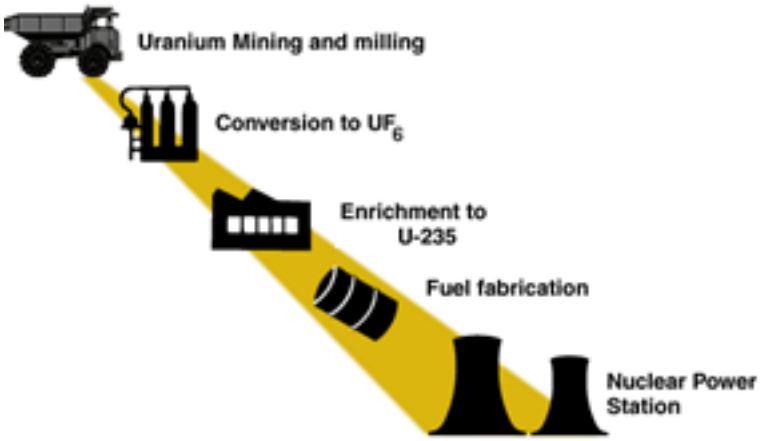


Figure 18-2: Front End Fuel Cycle

After three (3) years of electricity production, nuclear fuel is removed from the Reactor and undergoes further steps including temporary storage, reprocessing, recycling and eventually disposal. This is commonly known as the Back End of the nuclear fuel cycle.

18.2 Uranium Market

Research written on the uranium market by the various analysts and the industry consultants reiterates the overall theme. As the industry moves through 2015 the uranium market remains in surplus from 2013 and 2014, due to a number of factors, but predominately as a result of continued the absence of Japanese nuclear power generation, and explains the prevailing low uranium price. The extent of the surplus has been declining, and an increasing number of future projects have been postponed reducing the future supply growth.

However, in the medium to longer term Japanese nuclear power generation is forecast to increase with reactor restarts. In addition, China's strong build program in nuclear power generation will result in an increase in demand with a major source of supply no longer available or delayed. Additionally the uranium mining industry, without a marked increase in metal price, will not have sufficient projects whose economic hurdle rates are sufficient to fill the expanding supply gap.

Therefore from 2017 the market watchers believe that the uranium market will move into a deficit as existing supply cannot meet the growing demand requirements and the uranium price will need to be substantially higher to provide the motivation for producers to solve the impending supply gap. These two major factors are therefore expected provide the impetus to improved uranium prices from 2017 and beyond, if not before. Analysts are predicting long term spot prices of USD 60/lb to USD 70/lb with term contract prices extended to be at least USD 5/lb to USD 10/lb higher than spot.

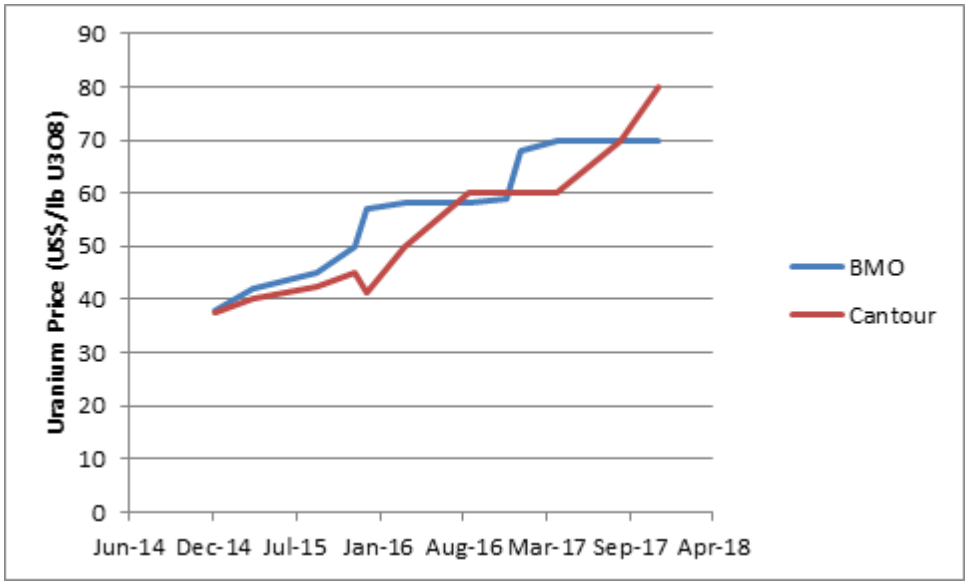


Figure 18-3: BMO and Cantor Fitzgerald forecast, Uranium spot price forecast

Source: BMO and Cantor Fitzgerald Capital Forecast (2015)

18.2.1 Demand

At the end of 2010 there were 30 countries in the world with 441 operating nuclear reactors and roughly 375 gigawatts-electric (GWe) production. However, following the Fukushima Daiichi accident, this figure has reduced to 432 units with roughly 370 GWe operating as of March 2013. Global reaction to Fukushima Daiichi accident was initially negative with Germany announcing that it would phase out nuclear power, and the then Japanese parliament committing to its continued nuclear phase out program.

In December 2012 the Japanese parliament changed following elections, with the Liberal Democratic Party's (LDP) achieving a landslide victory with a pro-business and pro-nuclear platform. The LDP garnered a two-thirds majority in Japan's lower house of parliament after campaigning on the "need to restart the reactors to cut the cost of importing natural gas and coal to fire conventional power stations". The LDP control of the Japanese landscape was increased in July 2013 when the party also gained control of the upper house, House of Councillors.

Pre-Fukushima, nuclear power accounted for nearly 30 percent of Japan's energy supply. Alternatives like gas, oil and coal resources are limited on the island nation, and the high cost of importing energy has placed a significant burden on the country, which is trying desperately to rebuild its economy. To alleviate the problem, the LDP believes Japan must restart the remaining idled reactors quickly and has proposed a three-year review to determine which are safe enough to come back online. In September 2013 Japans last remaining operating reactor was shut down. The process of regulatory clearance for restarting 48 reactors is under way, but slow and expected to take some years, with a target of 20% of the country's electricity being from nuclear energy. While lower than the previous 30% it will require the majority of the idle reactors to be restarted.

In June 2015 the government's draft plan for electricity generation to 2030 was approved. This has nuclear at 20-22%, renewables 22-24%, LNG 27% and coal 26% (WNA). While lower than the previous 30% it will require the majority of the idle reactors to be restarted.

But while Germany may have changed its perspective on the nuclear renaissance, nations like China, France, India Argentina, Bangladesh and oil-rich Saudi Arabia are continuing to develop a nuclear power program. The Middle Eastern nation has no nuclear power program currently, but is pondering constructing a facility with a capacity of 17 gigawatts by 2032 (Reuters).

In addition to the restart of the reactors in Japan the other key factor effecting uranium demand will be the rate of development in China. In 2012, China had 15 nuclear power units with a total electric capacity of 11 gigawatts, accounting for just 1.9% of the total electricity output. In 2013, this rose to 2% with 17 reactors, and 28 under construction. By comparison, the USA is currently 20% and France 74%.

China has the world's most ambitious nuclear program with plans to provide 6% or 58 GW of capacity by 2020, and a further increase to 16% or 200 GW by 2030. By 2030, China is expected to consume nearly half of global uranium production.

The impetus for increasing nuclear power share in China is increasingly due to air pollution from coal-fired plants. China's policy is for closed fuel cycle. China has become largely self-sufficient in reactor design and construction, as well as other aspects of the fuel cycle, but is making full use of western technology while adapting and improving it.

India's commitment to nuclear power generation is following a similar path to that of China, but a much earlier stage. The impetus for India is the same as China with a WHO 2014 survey finding that 13 of 20 most polluted cities in the world were in India.

In 2013, India had 21 reactor with installed capacity of 5,780 MW consuming approx. 1580 TU per annum. The country is now targeting installed capacity of 63,000 MW (equivalent to 25% total energy demand) by 2032.

Ux Consulting (UxC) forecasts that under their 2015 Base Case forecast 442 reactors (~384 GWe net), and UxC anticipates that this will grow to 478 reactors by 2020 (~425 GWe net) and 528 reactors (~485 GWe) by 2025. By 2030, UxC forecasts an increase to 591 reactors (~575 GWe net). As the figures show, the vast majority of the growth over the next two decades is anticipated to come from Asia (especially China); however, some nuclear gains are also envisioned in Eastern Europe, North America, and Africa & the Middle East.

Table 18-1: Reactor Units & Nuclear Capacities Anticipated by Country

Units	2010	2015e	2020e	2025e	2030e
North America	**124**	**120**	**123**	**120**	**123**
Canada	18	19	18	13	15
USA	104	99	103	105	106
Mexico	2	2	2	2	2
Western Europe	**129**	**117**	**112**	**104**	**94**
France	58	58	57	57	57
UK	19	16	15	14	9
Germany	17	9	6	0	0
Sweden	10	10	10	10	10
Finland	4	4	5	6	6
Spain	8	7	7	7	7
Switzerland	5	5	4	4	3

Units	2010	2015e	2020e	2025e	2030e
Beliguim	7	7	7	5	0
Netherlands	1	1	1	1	2
Eastern Europe	**67**	**71**	**75**	**73**	**79**
Russia	32	36	37	33	33
Ukraine	15	15	15	15	15
Czech Republic	6	6	6	6	6
Other	14	14	17	19	25
Asia & Oceania	**115**	**126**	**154**	**213**	**262**
Japan	54	37	38	36	30
South Korea	20	25	28	32	35
China	13	32	50	96	134
India	20	23	27	34	44
Other	8	9	11	15	19
Africa & Middle East	**2**	**3**	**7**	**9**	**21**
South Africa	2	2	2	3	5
Saudi Arabia	0	0	0	0	2
UAE	0	0	4	4	7
Other	0	1	1	2	7
South America	**4**	**5**	**7**	**9**	**12**
Brazil	2	2	3	5	6
Argentina	2	3	4	4	6
Other	0	0	0	0	0
Total	**441**	**442**	**478**	**528**	**591**
Gwe	**375**	**384**	**425**	**485**	**575**

Source: UxC

18.2.2 Supply

As described above, analysts and market watchers are forecasting increasing U_3O_8 demand in the coming years as more countries look to nuclear power to resolve their forecast power gaps and deal with the issue of reducing carbon emissions.

In 2013 the market saw a significant drop in available supply. Near the end of 2013, the US-Russian Highly Enriched Uranium (HEU) Agreement — officially dubbed the Megatons to Megawatts Program — came to an end. The HEU agreement has accounted for nearly 15 percent of global uranium supply, and its termination helps support a looming uranium supply shortage, especially if the industry keeps experiencing supply setbacks.

The supply of uranium is provided from two sources, the first and primary source is mined production and the second is sales from inventory predominantly by governments, especially the US. Historically these other supplies have ranged between 25% and 30%, but with end of HEU and declining government inventories the expectation is that this figure will reduce to approximately 15%. The supply side therefore is going to rely more heavily on primary mined material and hence will an increased exposure to the risks of operational issues and project

slippage at the major mines.

In 2013 it is forecast that approximately 50% of global uranium supply came from 9 operations, while looking forward to 2020 the figure is 59% of world supply dependant on 13 mines.

Table 18-2: Forecast Mined Production 2014 to 2022 (Mlb)

Existing Supply	**2014**	**2015e**	**2016e**	**2017e**	**2018e**	**2019e**	**2020e**	**2021e**	**2022e**
Africa	22.5	21.5	22.0	22.5	23.0	23.5	26.0	27.1	27.5
Australia	12.8	12.9	14.9	14.6	11.6	11.6	11.6	11.6	11.6
Canada	23.6	30.4	34.9	38.4	39.5	40.2	40.9	40.9	39.6
Kazakhstan	59.4	60.5	60.7	60.7	60.7	61.6	63.1	64.1	55.9
Russia	8.2	9.0	9.2	9.5	9.5	9.5	9.5	9.5	9.5
USA	4.9	3.1	3.3	3.7	4.0	4.1	4.1	4.1	4.1
Uzbekistan	6.3	6.3	6.3	6.3	6.3	6.3	6.3	6.3	6.3
Other	8.7	8.3	8.7	8.3	9.4	9.2	9.5	9.9	9.9
Sub Total	**146.5**	**152.0**	**159.9**	**164.0**	**164.0**	**166.0**	**171.0**	**173.5**	**164.4**
New Mine Supply									
Australia									
Canada									
Africa			3.0	6.0	9.0	10.0	10.0	10.0	10.0
USA		-	0.3	0.6	0.6	0.6	0.6	0.6	0.6
Sub Total	**-**	**-**	**3.3**	**6.6**	**9.6**	**10.6**	**10.6**	**10.6**	**10.6**
Total Supply	**146.5**	**152.0**	**163.2**	**170.6**	**173.6**	**176.6**	**181.6**	**184.1**	**175.0**

Source: Ux Consulting, GoviEx

While there are a considerable number of uranium projects at various stages from exploration to development that would be sufficient to fill the required increasing demand there are two major issues that cause concern and point to considerably higher prices for uranium in the medium to long term. These issues are incentive pricing and project slippage.

The low prices had an impact on uranium producer and projects, with some producers either cutting production for existing unprofitable mines or choosing to delay the development of new mines. These actions could have a significant impact on the market several years from now when more production will be required to meeting increasing demand.

Market commentators and producers also comment on the guarantee of future production and the incentive price necessary for most new projects being several years out. The incentive price must be more reflective of a long-term contract price, and given the cost of inflation, UxC expects this would be in a range of USD 65–75 per pound U_3O_8. This incentive price would include cash costs, capital costs, overhead, and a minimum 15% rate of return. Recently, banks have shown little interest in loaning money for the development of planned uranium projects since the rates of returns are so small or negative in some cases at today's prices.

The IAEA Red Book highlights the decline in the quality of Reasonably Assured Resources (RAR) since 2011. The RAR with a forecast cash cost of less that USD 36/lb U_3O_8 has

declined between 2011 and 2013, and the majority of RAR are not profitable at current spot price. The figures presented by IAEA imply that the incentive price for the majority of new production will be greater than >USD 70/lb

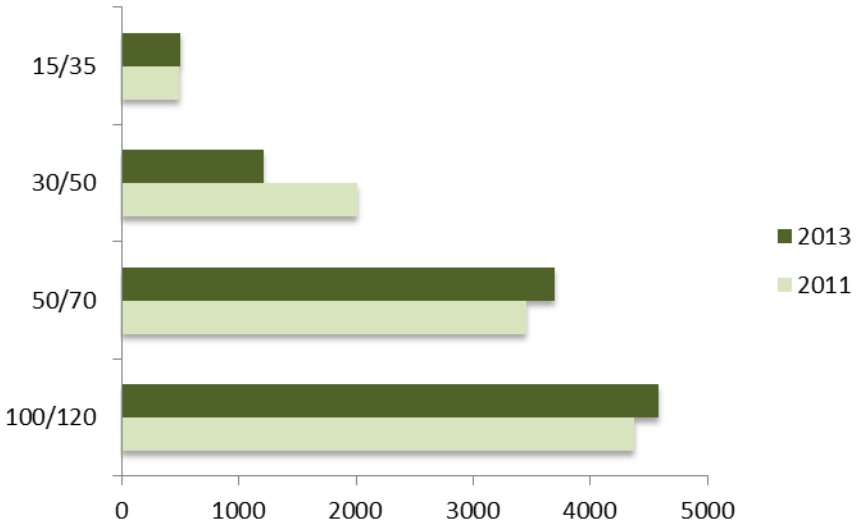


Figure 18-4: RAR (tonne Uranium (x-axis)) reported by Cash Cost / Incentive Cost (USD/lb U_3O_8)

For the supply to keep up with demand growth, uranium prices need to increase to incentivise new production. Importantly, the lead time to put new mines in to production means that the incentive price needs to be reflected sooner rather than later, otherwise production may well fall short of requirements.

18.2.3 Secondary Supplies

There are several sellers in the uranium market that are supplied by various sources of inventory. This inventory is in the form of U_3O_8, UF_6, and enriched uranium product (EUP). These sellers will continue to be present in the market for the next several years. The primary inventory seller post 2013 is the U.S. Department of Energy (DOE), which holds a large amount of inventories.

Table 18-3: Forecast Inventory Supplies

	2012e	2013e	2014e	2015e	2016e	2017e	2018e	2019e	2020e
FSU	17.0	14.1	14.1	14.1	14.1	12.6	12.6	12.6	12.6
HEU	12.0	12.0							
Enricher Sales	3.9	4.5	4.3	4.3	4.0	4.0	4.0	4.0	4.0
Other (Include USA)	14.4	15.3	16.5	16.0	16.1	16.2	13.6	14.7	12.5
Total Non –Trad.	47.3	45.9	34.9	34.4	34.2	32.8	30.2	31.3	29.1

Source: UxC, BMO Capital Markets

The secondary sources are varied and comprise the following:

- MOX and RepU – Mixed oxide fuel (MOX) is a combination of plutonium oxide recovered from spent fuel and new uranium oxide from DU (a “waste” product of the fuel enrichment), while reprocessed uranium (RepU) involves the removal of uranium and plutonium from spent fuel to fabricate new fuel. Although these two fuel sources have

been used for many years, the contribution has been quite low (approximately 5% of total uranium supply).

- US/USEC Government Stockpile Sales – Government strategic stocks that have been deemed surplus. Some of this material is sold through the United States Enrichment Corporation (USEC), a public company that was previously a government organization. In 2008, the US Department of Energy (DOE) released its Excess Uranium Inventory Management Plan that provided details regarding the US government's plans to dispose of its excess uranium inventories. These inventories, totaling 153 million pounds, were built up over decades primarily through enrichment activities, weapons programs, and the US-Russian HEU (highly enriched uranium) program. The uranium is in various forms, some of which are readily saleable, whereas others require substantial processing to bring to commercial reactor standards. The DOE plan calls for the selling of uranium into the US market over a period of at least 25 years and not exceeding 10% of US demand, except in the case of special needs such as initial cores. It is important to note that the United States government said that the disposition would be carried out in a "manner that minimizes any material adverse impacts on domestic uranium mining." This is interpreted to mean the government will look to sell the uranium in such a way that it does not have an overwhelming negative effect on the prevailing uranium price, thereby hurting domestic uranium producers. Re-Enriched Tails – Derived from the waste streams of previous enrichment, called depleted uranium (DU), that are reprocessed to extract additional usable uranium.
- HEU – In 1993, the governments of the United States and Russia agreed to allow 500 tonnes of Russian surplus HEU containing about 90% ^{235}U to be sold into the market as blended-down LEU containing about 4% ^{235}U; this equates to approximately 395 million pounds of U_3O_8. The agreement called for the material to be sold into the market over a 20-year period, which finished in 2013.

18.2.4 Outlook 2016 onwards

The near-term and medium-term outlooks continue to have some residual inventories available to the market at a time when uranium demand seems to be static. However, demand improves in the long-term outlook as the scale and pace of Japanese reactor restarts should improve, with Chinese reactor growth as well. This could set up for increased price volatility beyond 2016, as a number of operating and planned projects could fall victim to low prices over the next couple of years. Based on history alone, uranium prices can make huge swings when future production levels are uncertain due to the long lead times required to bring new projects online.

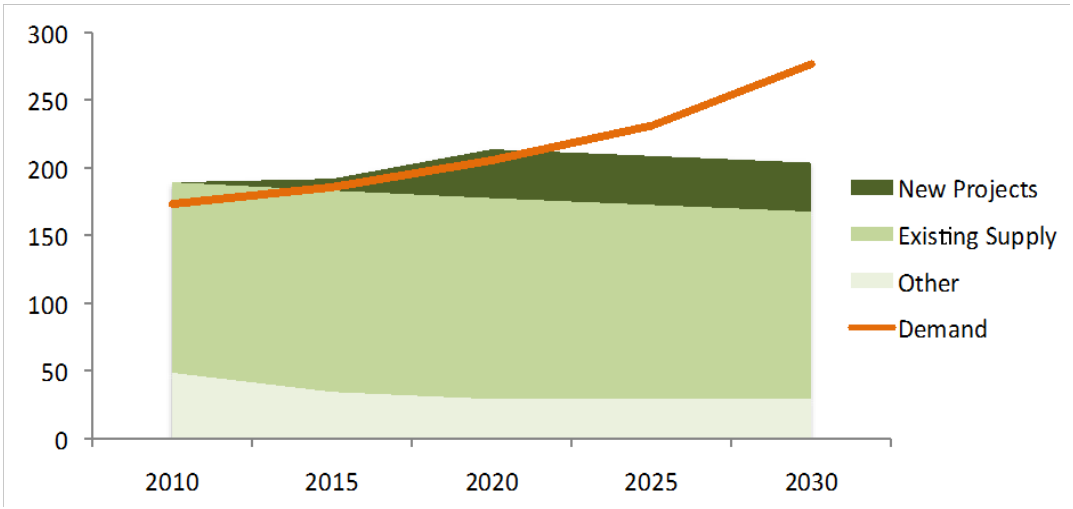


Figure 18-5: Uranium Supply and Demand (Mlb U_3O_8)

Source: GoviEx

18.3 Conversion Facilities

Globally there are seven (7) Conversion Facilities that have the capability to convert U_3O_8 into UF_6. Table 18-4 below, shows the capacity and estimated annual production of each of the Conversion Facilities.

Table 18-4: Conversion Facilities

Conversion Facility	Nameplate Capacity tonnes U as UF6	Estimated Production 2013
Cameco, Port Hope, Canada	12,000	8,750
Atomenergoprom, Russia	25,000	13,750
Comurhex, France	14,000	9,800
ConverDyn, USA (Honeywell)	15,000	10,500
Springfields, UK *	6,000	5,000
IPEN Brazil	40	28
CNNC China	Not known	3,650
Total	**72,540**	**51,478**

Source Data: WNA – The Global Nuclear Fuel Market – Supply and Demand 2013 - 2020

*The Springfields Conversion Facility in the United Kingdom is no longer in operation.

Atomenergoprom: The largest Conversion Facility is Atomenergoprom in Russia, however it is unusual for U_3O_8 that is sourced from outside of Russia to be delivered and converted to UF_6 by Atomenergoprom, as this facility is primarily used for internal Russian Conversion requirements.

ConverDyn (Honeywell Metropolis Works, USA): The Honeywell Metroplis works is situated in Metropolis, Illinois, USA. The Conversion Facility is owned by Honeywell but it is operated by ConverDyn and is the only U_3O_8 to UF_6 Conversion Facility in the USA. This facility and has been operating since 1958. ConverDyn's ownership stucture is fifty percent

(50%) Honeywell and fifty percent (50%) General Atomics based in Denver, Colorado, USA. ConverDyn manage all aspects of the Conversion process including U_3O_8 deliveries, sampling and storage.

Cameco (Canada): Cameco's Conversion method is split between two (2) processing plants. U_3O_8 is delivered to Cameco's Blind River facility in Ontario, Canada where it is refined to UO_3. The $U_3O_8$is then transported to Cameco's Port Hope facility, where it is converted to UF_6.

GoviEx will require a weighing, sampling, analysis and storage contracts with both Converdyn and Cameco Cororation, so that they can make deliveries by Book Transfer to their customers who choose to have their Conversion service carried out by either Converdyn or Cameco.

Comurhex (France): Comurhex's Conversion method is also divided between two (2) processing plants. U_3O_8 is delivered to the Comurhex Malvesi site where it is converted to UF_4. The UF_4 is then transported to Comurhex Pierrelatte where it is converted from UF_4 to UF_6. The Comurhex plant at Pierrelatte has been operating since 1959. Comurhex became a full subsidiary of Areva in 1992.

18.3.1 Contracts with the Conversion Facilities

Deliveries to a Conversion Facility can be made by physical delivery or by Book Transfer. In order to maximise the sales opportunities relating to the U_3O_8 mined from the Madaouela Uranium Project, it will be essential for GoviEx to have the ability to physically deliver its U_3O_8 to the Conversion Facilities of ConverDyn, Cameco and Comurhex.

As such, GoviEx will be need to enter into a weighing, sampling, analysis and storage contract, in order for a U_3O_8 holding account to be established in the name of GoviEx ("U_3O_8 Holding Account") with each of the aforementioned Conversion Facilities.

The Conversion Facility is responsible for the management of its customers U_3O_8 Holding Accounts, which includes the administration process for crediting and debiting U_3O_8 transactions. Typically, U_3O_8 Holding Account statements are issued by the Conversion Facility on a quarterly basis.

In general, the Conversion Facilities do not offer their customers the ability to physically withdraw U_3O_8 and UF_6 from their U_3O_8 Holding Accounts; it can only be Book Transferred between customers who have U_3O_8 Holding Accounts at the same Conversion Facility.

18.4 Weighing, Sampling, Analysis and Storage of U_3O_8

Prior to any physical delivery of U_3O_8, a Producer will be required to enter into a contract with a Conversion Facility for the weighing, sampling, analysis and storage of U_3O_8. Dependant on the particular Conversion Facility, the terms and conditions within such contract will include but not be limited to:

- The Producers obligation to provide a U_3O_8 delivery schedule each year;
- The conditions of delivery for example, packaging, loading, marking, labelling, emergency response, shipping documentation;
- The Conversion Facilities weighing and sampling process;
- The specification of the U_3O_8 that will be accepted at the Conversion Facility for Conversion without surcharges. The specification for each Conversion Facility may vary slightly but will broadly be in line with ASTM specification C967-13 Standard

Specification for Uranium Ore Concentrates;

- The terms for the creation of a U_3O_8 Holding Account and the subsequent storage thereof;
- Fees and surcharges including but not limited to:
- Weighing and sampling,
- Surcharges for U_3O_8 relating to impurity concentration levels,
- Storage fees; and
- Book Transfer fees;
- U_3O_8 Title and ownership arrangements;
- Physical withdrawal conditions if allowed; and
- Loss and damage insurance provisions.

18.5 Physical Delivery of U_3O_8

It is the responsibility of the party physically delivering the U_3O_8 (usually the Producer) to pay for the transportation to the Conversion Facility.

A percentage quantity of the total U_3O_8 delivery quantity will be credited to the U_3O_8 Holding Account on the date of delivery. The U_3O_8 is then weighed and analysed by the Conversion Facility to confirm its acceptance. This analysis can take up to one hundred (100) days. Once completed the balance of U_3O_8 will be credited to the U_3O_8 Holding Account. A fee may be charged for this service, particularly if there are any surcharges imposed by the Conversion Facility relating to impurity concentration levels. These charges vary depending upon the terms and conditions negotiated in the contract.

Title to the U_3O_8 will remain with the U_3O_8 owner (usually the Producer) until it is sold. Risk of loss and damage will transfer to the Conversion Facility upon physical delivery of the U_3O_8 at the Conversion Facility.

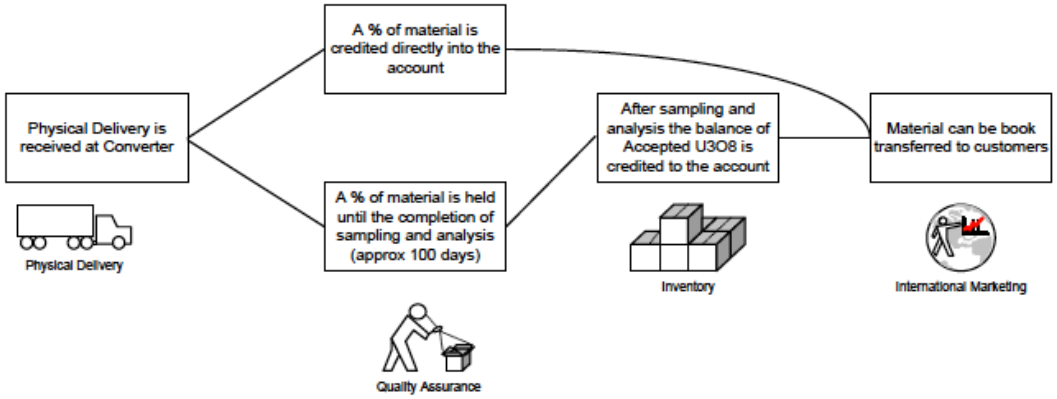


Figure 18-6: Steps involving a physical delivery of U_3O_8

18.6 Book Transfer Delivery

When a customer agrees to purchase U_3O_8 from a Producer (or other supplier), the customer will expect to receive delivery by means of Book Transfer. This transaction will appear as a credit in the customers U_3O_8 Holding Account and a debit in the Producers account at the

Conversion Facility.

Title passes from the seller to the buyer on the date the U_3O_8 is Book Transferred. Risk of loss and damage remains with the Conversion Facility.

It is standard practice for a Conversion Facility to impose a charge for Book Transferring U_3O_8. The only exception is if the buyer is a Conversion customer of the Conversion Facility.

Normally the seller will provide a Book Transfer notice document around ten (10) working days prior to the required delivery date to instruct the Conversion Facility to effect the Book Transfer from one U_3O_8 Holding Account to another.

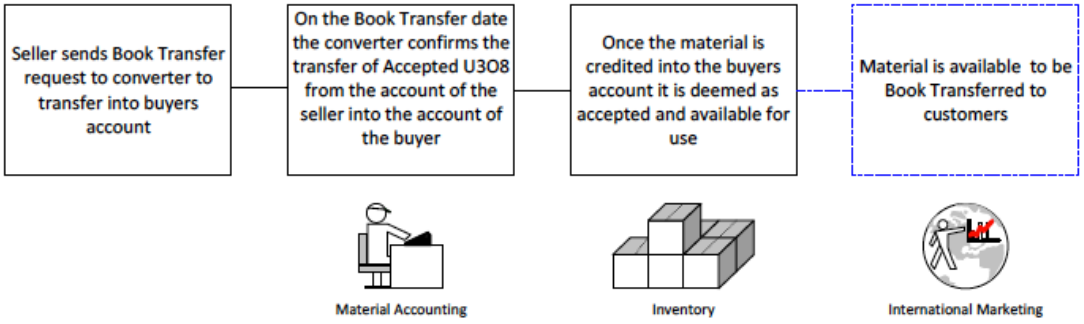


Figure 18-7: Process flow for a U_3O_8 delivery by Book Transfer.

18.7 Transport to Market

It is proposed that deliveries will be made by road from the operating facilities south through Niger and across the border into Benin, and shipped out of the Port of Cotonou using the shipping line CMA CGM who is the world's third largest container shipping line.

A Nigerian trucking company familiar with the regulations for radioactive transport will be used for the road transportation through Niger into Benin to the Port of Cotonou.

The current proposed transport scheme is one that has been established for several years and improved upon year on year. When GoviEx's shipments commence, it will be prudent to ensure the route is well managed to main the ability to transport the U_3O_8 to Conversion Facilities in order to support sales to customer.

Great emphasis is placed on security to ensure the requirements of the Nigerian Government are met. Military escorts will be present will every convoy. Typically there will be two (2) different escorts present with each convoy.

Presently, U_3O_8 transports in Niger are carried out in convoys of less than fifteen (15) trucks for security purposes. If there is a requirement to transport a larger number of convoys, different security provisions will apply.

There are three (3) escort trucks per consignment of ten (10) twenty foot (20ft) isofreight containers ("ISO's"). The road transport from the Arlit area to the Port of Cotonou takes between five (5) to six (6) days. The road transport will only take place during daylight hours, therefore trucks will stay overnight at five (5) specific and pre- agreed secure locations en route to Cotonou.

The roads in this area are in poor condition therefore specialist equipment will be made available such as cranes and tow trucks to get the trucks back on the road if they experience problems en route.

18.7.1 Sea Freight from Cotonou to Europe and USA

The sea freight of the ISO's from Cotonou and Europe or the USA will be carried out by a scheduled liner service. The shipping of radioactive material by sea must be carried out in accordance with the rules and regulations of the International Maritime Dangerous Goods Code 2012 edition or latest version thereof.

The shipping line currently used for sea freight of U_3O_8 from Cotonou is CMA CGM. This company operates two (2) routes into European ports including Le Havre in France. From Le Havre, the ISO's can be discharged for onward delivery to the Comurhex Conversion Facility or alternatively the ISO's could be routed from Le Havre to Antwerp when they can be shipped to North America for delivery to the Conversion Facilities of ConverDyn and Cameco.

18.7.2 Radiation Protection

It is a requirement of the Transport Safety Regulation TD-R-1, 2005 Edition that a Radiation Protection Programme is put in place for all transport of radioactive material. GoviEx will use these WNTI publish documents in conjunction with its own procedures to ensure that all regulatory requirements are met or exceeded with regard to radiation protection.

18.8 Uranium Markets

When selling a commodity dependent upon strategy and available inventory, a supplier may look to sell into the spot market or the long term market. Spot sales are those where terms and conditions are agreed for a delivery in less than three (3) months ("Spot"). A long term sale ("Term") is one where the terms and conditions are agreed for a delivery in greater than twenty four (24) months and a mid-term ("Mid") sale is between the three months and two years.

18.9 Market Publications

In the Uranium market there are two (2) trade publications that are used as price reference points in U_3O_8 sales contracts: Ux Weekly published by the Ux Consulting Company LLC; and the Nuclear Market review published by TradeTech LLC. Both issue a weekly U_3O_8 Spot price and a Term month end price.

Figure 18-8: Spot versus Term price as published by Ux since 2004 - 2014.

18.10 Pricing Mechanisms

There are a variety of pricing mechanisms that can be used when negotiating and concluding a contract for the sale of U_3O_8, in general there are three (3) mechanisms that are favoured by the industry;

1) Fixed Price;

2) Market Related Price; and

3) Hybrid Price.

18.10.1 Fixed Price

Fixed pricing can also be divided into two categories:

1) Fixed price which is not subject to escalation; and

2) Based price escalated.

A fixed price contract is where the buyer and the seller agree to a specific price, which is not escalated by inflation indices. This type of pricing has been typically used for Spot or Mid sales, however more recently, buyers are now requesting fixed prices for longer term contracts.

Fixed pricing can be advantageous to the buyer and seller as both can easily forecast and measure cash flows, budgets and material inventory prices.

A base price escalated mechanism is traditionally used for longer term contracts. The base price is agreed and fixed in a contract between the buyer and seller and is escalated in line with an agreed escalation factor, the starting date of which is also agreed. A commonly used escalation factor is the US Gross Domestic Product Implicit Price Deflator (“GPDIPD”). The GPDIPD is one measure for the US annual inflation rate and is typically used in the nuclear industry.

Base price escalated approach has an element of unpredictability, but since the commonly used escalation factor or GPDIPD is used, it is unlikely that a huge variance will be seen year on year.

18.10.2 Market Related Price

Market related pricing is usually based on the Spot or Term price (as published by UxC and TradeTech or an average of both) near or at the time the delivery of U_3O_8 takes place. For Term contracts, it is not unusual for a seller to offer a buyer an agreed percentage discount against the Term price.

Market related pricing ensures that the purchase price will be more in line with the market conditions at the time of the purchase, but overall it gives a greater level of uncertainty for the buyer and the seller. This uncertainty can be mitigated somewhat by the use of price floors and ceilings, whereby the floor price protects the seller and the ceiling price protects the buyer. The level at which the floor and ceiling prices are fixed will depend on the market conditions at the time of contract negotiations.

18.10.3 Hybrid Price

In recent years buyers have looked at ways to optimise pricing mechanisms especially for Term contracts. As a result, hybrid prices have been agreed which is a combination of both the fixed price and market price mechanisms. The percentage split between the two varies

and is negotiated between buyer and seller. For example a Term sales contract could have sixty percent (60%) of the annual quantity delivered using a fixed price mechanism and forty percent (40%) delivered using a market related price mechanism.

19 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT (ITEM 20)

This section provides an update on the items raised during the feasibility studies, and the Environmental Social Impact Assessment (ESIA) submitted to the BEEEI on March 10, 2015, and approved July 31, 2015.

19.1 Environmental and Social Studies and Approvals

In Niger, the ESIA procedure starts when the project proponent submits a Notice of the project to the environmental authorities. The ESIA office ("BEEEI") is the link between the Minister in charge of environment (ME) and the project proponent. The BEEEI is responsible for reviewing the project Notice, deciding whether the project requires an ESIA and advising the ME on the decision. The ME then provides comments to the project proponent on the required ESIA process. The Notice of the Madaouela Project, prepared by GoviEx with the support of SRK and Legeni, was submitted to the BEEEI in January 2013 (Ref. V/L No. 004/GOV/ADM/AB/01/2013 of 11 January 2013). The Notice included a description of the Project, a preliminary impact assessment, a project schedule and other documents allowing the authorities to understand the Project and its potential impacts on the environment.

After revision of the Notice, an ESIA for the Madaouela Project was required by the ME. The ME invited GoviEx (letter No. 66/MH/E/SG/BEEEI/DEPI/M of 25 January 2013) to develop, in collaboration with the BEEEI, a terms of reference ("ToR") report outlining the ESIA needing to be undertaken for the Project. A preliminary ToR was prepared by GoviEx in concert with the BEEEI, and with the support of SRK and Legeni.

During the preparation of the preliminary ToR a support mission and a site visit was organised by the BEEEI and held in Arlit from 13 to 16 May 2013. Representatives of GoviEx, Legeni, SRK, BEEEI, Department of Mines, the Agadez and Arlit Departments of Mines and Environment and the Mayors of Arlit, Dannet and Gougaram participated in the mission. The observations and recommendations made during this mission (BEEEI, 2013) were addressed in a revised version of the ToR for the ESIA in June 2014.

Based on the ToR, GoviEx, with the support of SRK and Legeni and other stakeholders, prepared this ESIA report for submission to the ME. This report was developed in accordance with the legal requirements and the ToR commitments. As required by the ESIA Decree, public consultation with interested and affected parties has been initiated and will be carried out until the finalisation of the ESIA.

Public consultation mechanism includes the following steps:

- At the start of the ESIA process, to inform stakeholders of studies being undertaken for the potential develop of a project;
- At the completion of the ESIA report, public hearing organised by the BEEEI;
- Following completion of the ESIA report, the report will be available at the BEEEI to allow stakeholders to access it and the stakeholders will be informed and consulted on the content of the ESIA report.

According to the ESIA Decree, the BEEEI will review the provisional ESIA report within twenty-one days from the date of receipt. This phase includes the following:

- *Ad hoc* committee establishment by the ME to help the BEEEI analysing the provisional

report and providing feedback to the ME;

- On-site verification by the BEEEI together with the concerned authorities and feedback reporting to the ad hoc committee;
- Public hearing organised by the BEEEI together with the proponent and feedback reporting to the ad hoc committee based on the recommendations raised from the stakeholders;
- Provisional ESIA report validation workshop organised by the BEEEI at the Project location and ESIA validation reporting to the ME.

Based on the ME comments and recommendations, GoviEx will submit to the ME a revised ESIA report along with an authorisation application for the development of the Project. Then the BEEEI will prepare and sign the environmental and social terms of reference ("cahier des charges environnementales et sociales") requiring GoviEx to comply with the mitigation measures presented in the environmental and social management plan ("ESMP") along with the partnership convention between the proponent and the BEEEI to maintain the environmental monitoring and control of the Project activities. Following the BEEEI's comments, the ME will send out the final decision to GoviEx within seven days, which should result in granting the environmental compliance certificate of the Project. At the end of this period, the proponent may consider the Project as agreed from an environmental perspective.

Legeni was responsible for the implementation of the baseline studies (apart from the Water Quality study) with the technical support and expertise of SRK where required. SRK was responsible for the initial protocol development for the studies, review of the baseline studies and the development of an impact assessment and management plan.

Stakeholder input to the ESIA was also obtained with GoviEx, Legeni and SRK consulting with numerous stakeholders, including: the ESIA Office of Niger (Bureau d'Evaluation Environnementale et des Etudes d'Impact - "BEEEI"); traditional leaders; mayors; and members of the community allowing them to provide the study team with comments and suggestions on the ESIA process.

The ESIA report was submitted to the BEEEI on March 10, 2015, and approved by Minister in charge of environment July 31, 2015.

19.2 Existing Environmental and Social setting

19.2.1 Physical setting

The Project area is located in the Sahara desert climate zone, the majority of soils have a low organic nutrient content and are frequently eroded by wind and heavy rainfall events. There are currently no permanent abodes or industry on the Project area. Hence the levels of background noise and vibration are assumed to be low, this in conjunction with the distance to potential receptors meant these aspects did not form part of the specialist studies. The following sub-sections describe the baseline environmental findings of the specialist studies.

Climate

To record the existing climate conditions in the Project area, data for analysis has been obtained from two weather stations near the Project site named SOMAÏR and Arlit. Data from Arlit station are provided by the National weather service and SOMAÏR by the SOMAÏR mining operation.

Figure 19-1: A landform within the Project area

The Sahara desert climate zone, which is fully arid with precipitation less than evaporation so water available for storage, runoff or recharge is minimal. The minimum, average and maximum annual rainfall based on data collected from Arlit between 1972 and 2010 is 0.5 mm (1972), 56.5 mm and 190.8 mm (2010), respectively. The minimum, average and maximum annual rainfall based on the data recorded at SOMAÏR between 1981 and 2011 is 3.5 mm (1983), 45.7 mm and 114.3 mm (1997), respectively. There is little to no rainfall from November to May with peak rainfall generally in August.

Average annual temperature ranges between approximately 15°C and 35°C, with a mean of 28°C. There are three seasons: a relatively cold season, September to February, a hot season, March to June, and a humid season, June to September. In the hot season temperatures can exceed 40°C and in the cold season nights are generally cool with temperatures below 20°C.

The wind blows mainly from the north east and south east, with some easterly winds also observed.

Air Quality

The air was monitored for levels of fine dust in the air, levels of dust fallout onto land and directional dust. Two monitoring programmes took place at a number of locations across the Project area as can be seen in Figure 19-2. As Niger does not currently have air quality standards, the results were compared to various international guidelines. Levels of dust showed a strong seasonal trend, increasing as expected during the dry season. Naturally occuring levels of fine dust in the air are above international guidelines and levels of dust fallout from the air onto land are also high. Levels of nitrogen dioxide and sulphur dioxide (gases typically associated with industrial activity) were found to be low, which is likely to be due to the low concentration of industry in the Project area.

Figure 19-2: Air quality monitoring station

Soils

The Project is located in a desert and thus the soils tend to be poorly developed and undergo limited physical or chemical reactions. The evolution of soil is principally controlled by the actions of wind erosion with a systematic stripping of topsoil resulting in poor nutrient and organic matter content. This results in a hummocky terrain of sandy plateaus, plains, valleys and steep rolling hills. Land use within the Project area is restricted to episodic grazing which is prevalent across much of the region. No agricultural activities are undertaken within the Madaouela West tenement.

Water resources

The Project is situated in a region where surface water is scarce and the drainage network consists of ephemeral rivers also referred to as wadis. They are characterised by short duration flow events in response to heavy rainfall and remain dry for most of the year. Surface water usage is limited to supporting the sparse vegetation and opportunistic livestock watering in short lived pools.

The uranium ore bodies in the region are hosted within sandstone aquifers. The Madaouela and COMINAK ore deposits are hosted by the geological formations supporting the Guezouman aquifer and the SOMAÏR ore deposit is hosted by the Tarat aquifer formation. The Tarat aquifer supports the water supplies to the towns of Arlit and Akokan and has been impacted by dewatering in the past to facilitate mining of SOMAÏR. The communities to the east of the Project area have their own water supply boreholes, which are tapping older pre-carboniferous aquifers.

The groundwater monitoring data shows groundwaters are generally neutral to alkaline (pH 7.1 to 12.3) and can be classified as fresh. There was found to be minimal seasonal variation in groundwater quality across the six rounds of sampling (July 2012 to October 2014), this reflects the low levels of rainfall in the region resulting in virtually no infiltration to groundwater. In general the water quality was poorer in the vicinity of the ore bodies and could not be used for drinking water purposes without treatment. Trace metal concentrations were generally low, although elevated molybdenum concentrations were recorded in many of the samples collected, particularly those from wells in the vicinity of the army camp. Slightly elevated uranium concentrations were also recorded in boreholes in the vicinity of the army camp. These elevated uranium concentrations are thought to be occurring as these wells intercept the ore bodies of Marianne and Marilyn.

Figure 19-3: Photographs of types of community borehole

Biological setting

Approximately 40% of the Project area is covered by South Saharan Steppe and Woodlands ecoregion in the northwest and with 60% covered by the Sahelian Acacia Savanna ecoregion in the southeast.

The Southern Steppe and Woodlands ecoregion used to attract large herds of arid-adapted migratory herbivores after the rains but the area has been overgrazed by herds of domestic livestock and habitat degradation is widespread. This ecoregion serves as a transition from the Sahara to the Sahel.

There are no Legally Protected conservation areas, Internationally Recognised conservation areas or Priority Sites for biodiversity located within 50 km of the Project area.

The only endemic faunal species occurring in the two ecoregions on the Project area are small, arid adapted rodents. Only one species of vertebrate is strictly endemic to the South Saharan Steppe, the gerbil *Gerbillus dongolanus*. Other near endemic mammals found were two more species of gerbil *Gerbillus mauritaniae* and *G. principulus*. The Sahelian Acacia Savannah has four endemic species within the genus *Gerbillus (Gerbillus bottai, G. muriculus, G. nancillus and G. stigmonyx*).

Figure 19-4: Examples of the biological setting

Radiological Baseline

The Project area is located in a region of elevated background radiation due to the natural presence of high concentrations of uranium in the earth. The baseline study looked at the communities to the east of the Project area, the Project area and Arlit/Akokan to provide a holistic view of the current radiation doses received by community members.

A baseline estimation of the total radiation dose received by a person present in and around the Project area was calculated using a range of scenarios that produced a low, medium and high value for the dose received. The estimated radiation dose was calculated from several sources, including: external cosmic radiation; external radiation received from the ground; inhaled dust and radioactive gases; ingestion of radionuclides on foodstuffs; and drinking of water containing radionuclides. The low scenario predicted a total dose that was just below the world average, however the average and high calculated doses for the area were both above the world average total dose. The highest calculated does are to the east of the Project Area. It is worth noting that the calculated total doses were still lower than some other regions globally that have naturally elevated radionuclides in their soils.

19.2.2 Social setting

Population demographics

The Project is located southeast of the town of Arlit, the capital city of the Arlit Department, in the Agadez Region of the Republic of Niger. Arlit is about 200 km north of Agadez town, 800 km northeast of the capital Niamey and 170 km southeast of the Algerian border. The Arlit Department has less than one person per square kilometre, with more than 70% of the Department's population living in the Arlit Commune. The population is young with a median age of 15 years and 50% are between 0 and 14 years of age.

Arlit Commune has a multi-ethnic population of approximately 160,000 inhabitants, mainly associated with AREVA's mining operations whilst the villages in Gougaram and Dannet, in contrast, have a population of approximately 6,500 and 10,000 respectively. The rural communes are predominantly occupied by Touareg tribes and the occasional Fulani tribe. The Touareg are semi nomadic, travelling within the Department and Region whereas the Fulani can be considered more traditionally nomadic, travelling across the country and beyond.

Services and infrastructure

The proximity of the towns of Arlit and Akokan are an asset for the Project. The towns are well established, regionally important centres with some houses having electricity and potable water, with standpipes in other areas, and no electricity or access to piped water in the poorer areas. Every government administration is represented and the main banks have outlets. There are filling stations, bus transportation, repair shops; essentially all that is needed to sustain the community services that would provide for people associated with mine development; though it is noted some of these services are currently under pressure due to population growth. There are few paved roads in the region. The N25 connects Arlit to Agadez and then Niamey.

Figure 19-5: A classroom in Gougaram

There is no mains electricity in the rural areas and water is collected from wells and/or boreholes fitted with pumps.

The Arlit Commune has 43 primary schools (36 state controlled and 7 private) and 14 secondary schools (6 state and 8 private). In the rural communes of Gougaram and Dannet there are no secondary schools. There are 11 primary schools and 32 village classrooms in Gougaram (Figure 19-5) and 22 primary schools in Dannet

There are two hospitals located in Arlit and Akokan, owned by Areva. There is one public hospital currently under construction and 17 integrated health centres, of which just one has a doctor and one is a military facility. Of the 40 health clinics in the department, only 20 are fully functional. There are five doctors, 36 nurses and three midwives, all based in Arlit. Only one doctor, three nurses and one midwife are based in the public facilities; the others are private health workers. There are two ambulances and two medical motorbikes to cover the three communes. Most rural communities rely on traditional medicine practices.

Livelihoods

Most households in the urban communities in the study area are involved in horticulture, primarily for home use but extra crops are sold. Mining related employment is the second highest livelihood followed by traders and artisans. The rural villages are occupied by semi-nomadic tribes with livestock keeping as the dominant economic activity. The daily and annual activities in rural communities revolve around their livestock's feeding and breeding patterns. Women tend to be responsible for the processing of animal products, especially fermented milk foods such as cheese from camels milk.

Figure 19-6: Photographs of natural resource use in Project area

According to the social study, the current urban employee market is experiencing a large disparity between those seeking jobs and the number of positions available. This is compounded by the slowdown in the extraction of uranium at SOMAÏR and COMINAK.

Standard of living

Niger ranked 187th and last in the 2014 Human Development Index, with 76% of its people living on less than USD2 a day. Although the country has made gains in per capita GDP, infant mortality and education in recent years, poverty and social inequality have not decreased significantly. Poverty is deepest in rural areas and rural women and girls are among those worst affected.

The population of Arlit and Akokan is urban, multicultural, multi faith and has grown up around the mining industry over the last 40 years with the majority of livelihoods geared towards employment or work in mining and related sectors.

The majority of houses in Arlit, Akokan and Teslem are made from mud or clay brick, locally referred to as 'banco'. There are a few luxury homes found in the area built by SOMAÏR and COMINAK and around 2,000 mine employees live in estates, with a clubhouse and restaurants. In the poorer areas of the towns, dwellings are made from sticks and straw and scrap materials ranging from sheet metal to cardboard. The Touareg nomads, in rural areas, live in covered tents, while the Fulani live in small collapsible huts made of woven mats.

Cultural heritage

Through interviews and field surveys, sites of cultural importance were identified and mapped for the Project area. One hundred and forty seven heritage sites were visually identified within the Project area. The sites have been classified into three main groups, namely: funeral (tombs), settlement (remains of habitations such as ruins and various fragments of tools and potteries) and natural (fossils and ostrich eggs) sites. Of the sites identified only two funeral sites are within the proposed infrastructure footprint. There may be additional sites buried in the sand which would only be identified when earth works commence.

19.2.3 Stakeholder Engagement

Stakeholder engagement is required in accordance with Niger legislation and international good practice. As part of the scoping phase, meetings were held with local communities and authorities to identify issues that needed to be considered during the ESIA. LEGENI/SRK used this information to identify the studies needed to investigate the possible impacts of the Project and appoint appropriate experts to help with the impact assessment. Nigerien and international experts were assisted by community members with local knowledge.

A number of issues and concerns were raised during these meetings such as the existing negative legacy of mining activities in the area; the potential loss of local livelihoods due to competition for land and water resources, which could be exacerbated by population influx; possible pollution to the environment; potential infrastructure improvement and job creation; and the necessity for proper stakeholder consultation.

19.2.4 Key Environmental, Social Issues and Risks

Following a impact assessment and identification of appropriate management measures a number of key substantive issues associated with the project were identified. These included the following:

- Potential for unacceptable exposure to ionising radiation either via the inhalation of radioactive gas radon, the ingestion or inhalation of radioactive dust particles and/or direct exposure to radiation or directly through ingestion of radionuclides via surface or ground water or through the food web via bioaccumulation into edible plants that are either eaten or fed to animals;
- The possibility of water contamination and reduced water supply to Arlit residents and/or Nomadic herders are both issues that would need to be very carefully monitored by baseline hydrology and hydrogeology studies already underway. However, the geology and aquifer structures on the project concession in addition to the prevailing desert climatic conditions and engineering design of waste disposal facilities would all contribute towards reducing the likelihood of these issues presenting a major concern.
- Due to the sensitive nature of desert fauna there is the potential that some desert species may be disturbed by the mining activities.
- The disposal of high level radioactive waste at closure is an issue that needs to be addressed as part of the closure cost plan.

SRK undertook a risk assessment to identify key environmental and social risks to the project. GoviEx and Legeni participated in the risk assessment process. The following risks need to be taken cognisance of by the design engineers during the detailed design stage, construction and operations phases of the project:

- If GoviEx does not address the concern regarding unacceptable exposure to ionising radiation there is the risk that the issue will be publicised by an international NGO, which could have significant reputational risks for the company. This risk will continue post closure with the necessary disposal of project components that require decontamination, which may be costly and could have substantial impacts on onward sale of the project.
- Although the presence of sites of archaeological and cultural heritage are discussed in this section there may be sites on the project footprint that require permission to be moved on the project footprint and this could cause delays to the commencement of project construction.
- A number of the risks such as pressure on services caused by influx of job seekers, loss

of sense of place, perceived unequal benefits and reduced water supply and potential loss of threatened species should be possible to manage through appropriate data collection during the baseline studies, ongoing monitoring and applying international good practice principles for stakeholder engagement.

- The development of the appropriate environmental, social and health and safety policies and community development plans in addition to ensuring that equitable distribution of community benefits and proper grievance mechanisms are implemented will all assist in managing community expectations.

19.2.5 Way Forward

A number of activities were identified during the feasibility studies and have taken place during the Environmental and Social Impact Assessment stage of the project, these included:

- Ongoing baseline data collection for surface and ground water and air quality.
- Additional baseline studies including ionising radiation and possibly traffic. Air quality would also need to be undertaken.
- Baseline studies (such as fauna and flora, water and air quality) would need to demonstrate that they cover seasonal variation that will contribute towards ensuring compliance with IFC performance standards and access to Equator Bank Finance.
- Special attention needs to be paid to undertaking stakeholder engagement.
- Targeted searches need to be undertaken on the proposed project footprint for archaeological and cultural heritage sites and any species of conservation and/or local use value.
- GoviEx would need to maintain safety and security measures for any expats required to work on site.
- A number of permits would need to obtain Government approval prior to the commencement of any construction activities.
- GoviEx would need to develop Environmental, Social and Health and Safety policies which as well as completing the development of a Stakeholder Engagement Plan, Community Development Plan and ensuring that there are the appropriate staff to implement these plans.

19.3 Summary of project impacts

The impacts of the Project have been arranged in groups, as presented in Table 19-1, in order of importance for the decision-making process. The residual impact significance rating provided in the table assumes the management measures described within the ESIA report have been successfully implemented.

The first group of impacts in Table 19-1 represent the most important impacts for decision makers. Positive impacts are denoted with a “+ive” after them and represent the main benefits created by the Project from an environmental and social perspective and may be considered to balance some or all of the negative impacts. These include increasing government revenue during construction and operation, and the provision of employment and income for Nigerien people leading to reduced poverty. The only negative impacts with a high significance before management measures is related to the proposed wellfield, however by maximising re-use of the mine dewatering and implementing the other proposed mitigation measures the significance of this can be reduced to medium.

Table 19-1: Summary of impact significance ratings

Grouping	Identified impacts	Residual impact rating
Most important positive and negative impacts needing to be taken into account during decision making.	Water supply wellfield impacting on local aquifers and groundwater users	Medium
	Increased government revenue from fiscal and foreign exchange income	High +ive
	Direct and Indirect employment of Nigeriens residing in the Arlit Department	Medium +ive
Negative impacts having a high or medium significance without management and that require careful monitoring to ensure management measures are effective. If monitoring reveals additional management/remedial measures required, these must be implemented without delay.	Mine dewatering impacting on local aquifers and groundwater users	Medium
	Potential post-closure leaching of deleterious constituents from flooded underground mine workings to groundwater impacting on local aquifers and groundwater users	Medium
	Particulate matter from operations exacerbating health issues associated with the naturally high dust levels in the region	Low
	Land clearance for surface infrastructure causing loss, degradation and/or fragmentation of natural habitat and potential loss or disturbance of species of conservation value	Low
	Direct exposure to radiation through proximity to TSF and inhalation or ingestion of tailings material or process water	Low
	Increased demand on infrastructure and for services and goods	Low
	Change in community dynamics and social relationships	Medium
	Reduced access to the Project area for water and pasture	Low
	Increased infectious disease transmission between workers and the host community	Low
Impacts that can be managed readily through measures that are not difficult to implement and are known to be reliable.	Impacts of site infrastructure on storm water runoff patterns	Low
	Potential leaching of deleterious constituents from the waste facilities to groundwater impacting local aquifers and groundwater users	Low
	Seepage and discharge from waste water systems	Low
	Potential post-closure impacts of Miriam open pit on water users	Low
	Degradation and/or alteration of landscape resulting in loss of a soil resource and reduced land capability	Low
	Contamination of soils from solid or liquid waste or from aerial deposition	Low
	Gaseous emissions from operations exacerbating health issues in the region	Low
	Tailings dam forming pool during the rainy season and acting as an attractive nuisance to birds in the region	Low
	Direct inhalation of radon and radionuclides from the ventilation systems of the underground mines	Low
	Accidental spills of uranium concentrate during packaging and transport	Low
	Increased incidences of avoidable accidents and injuries	Low
	Accidental damage or loss of archaeological sites resulting from land clearance for construction of Project infrastructure and looting	Medium

Green shading indicates potential positive impacts, red and yellow shading represents potential medium and low negative impacts, respectively.

19.4 Implementation of Management Measures

The management measures committed to by GoviEx in its Project description and the additional mitigation measures identified during the impact assessment process have been used to develop an Environmental and Social Management Plan (ESMP). The Project will develop and implement an Environmental and Social Management System (ESMS) prior to the start of construction to ensure this plan is implemented. The objectives of the ESMS will be to:

- Undertake the ongoing identification and assessment of potential environmental and social impacts, both adverse and beneficial, building on this ESIA;
- Avoid, or where avoidance is not possible, minimise, mitigate or compensate for adverse impacts and enhance positive impacts on workers, affected communities, and the environment;
- Ensure affected communities are engaged on issues that could potentially affect them; and
- Promote improved social and environmental performance of the Project through the effective use of management systems.

Measures included in the ESMP will be put in place to achieve the following objectives:

- Recognise that social responsibility and environmental management are core corporate priorities;
- Establish and maintain relationships with internal and external stakeholders;
- Maintain information on legislative requirements and environmental and social aspects associated with the Project activities;
- Assign clear accountability and responsibility for environmental protection and social responsibility to management and employees;
- Provide a process for achieving targeted performance levels;
- Provide appropriate and sufficient resources, including training, to achieve targeted performance levels on an ongoing basis;
- Evaluate environmental performance and social responsibility against GoviEx environmental and other policies, objectives, targets and seek improvement where appropriate; and
- Establish a management framework to audit and review the GoviEx ESMS and identify opportunities to improve the system and resultant environmental and social performance.

19.4.1 Monitoring

The ESMP includes a monitoring programme which aims to:

- Provide measurements of environmental and social impacts of the Project;
- Ascertain and demonstrate compliance with conditions of approval and other legislation;
- Provide sufficient evidence to address any claims made against the Project in respect of environmental and social matters;
- Track performance of the ESMS and progress in the implementation of the ESMP;
- Track and measure key indicators and other performance measures over time to improve the Project's performance and reduce the likelihood of environmental incidents; and
- Inform decision processes for determining management actions.

19.4.2 Rehabilitation and Closure Strategy

A Rehabilitation and Closure Strategy has been produced that is a framework to enable GoviEx to develop a detailed closure plan that covers rehabilitation of the area and determine how much money it must put aside to do this. Closure related impacts, risk and benefits from the Project identified during the ESIA process will be re-assessed continually throughout the life of the Project, as part of the ESMS, with the aim of progressing the plan from conceptual in nature to a detailed implemental plan as the operation progresses. The objective of this review process is to use the outcome of ongoing operational monitoring and mine planning to refine the closure and rehabilitation measures so that at least two years prior to actual planned closure the actions required are thoroughly identified and relevant costs allocated.

The mine closure planning for the Project will follow good international industry practice and aim to:

- present a vision for closure, with clearly defined closure outcomes and completion criteria;
- should incorporate physical and socio-economic considerations and be an integral part of a project life cycle;
- include financial provisions to ensure that there are sufficient funds available to complete the prescribed closure activities;
- be regularly updated and refined to reflect changes in mine development and operational planning, as well as the environmental and social conditions and circumstances;
- include appropriate aftercare and continued monitoring of the site pollutant emissions and related potential impacts;
- include adjustments to closure funding arrangements to reflect any changes in mine closure requirements.
- all structures (e.g. tailings impoundments) should remain stable such that they do not impose a hazard to public health and safety as a result of physical failure or physical deterioration;
- design tailings structures should be decommissioned so that water accumulation on the surface is minimised and that any water from the surface of the structure can flow away via drains or spillways and these can accommodate the maximum probable flood event;
- ensure surface water and groundwater should be protected against adverse environmental impacts and leaching of chemical should be prevented to protect human health and ensure compliance with water quality objectives.

Where possible disturbed areas will be shaped so the land has a similar type of landform as before the Project. Some disturbed areas will not look the same as before mining commenced, such as the open pit, the waste rock dumps and the tailings storage facility. These disturbed areas will be contoured to ensure they are safe and the landforms are sustainable into the future.

19.4.3 Stakeholder Engagement Plan

A Stakeholder Engagement Plan (SEP) has been prepared to document what was done during the ESIA process. The first step in the engagement process was the identification of stakeholder groups, followed by identification of stakeholders within these groups; these stakeholders can be specific individuals, organisations, governmental departments, local businesses, media bodies etc. It is important to ensure the correct stakeholders are identified and engaged with in an appropriate manner during the ESIA and across the mine life. The list of interested parties will evolve as new stakeholders are identified and some existing

stakeholders are no longer involved during the life of the Project.

The SEP includes a grievance mechanism that enables GoviEx to plan a program of engagement activities, to utilise a range of engagement tools to communicate with key stakeholders and affected communities, and implement the grievance mechanism so the progress of addressing any complaints or issues is recorded on regularly.

19.4.4 Other plans

The other supporting documentation for the implementation of the ESMS that have been or will be produced includes:

- Developed as part of this ESIA
  - Environmental and Social Management Plan;
  - Stakeholder Engagement Plan;
  - Rehabilitation and closure strategy.
- Additional plans to be prepared during detailed design for construction include (note these may be individual or combined plans):
  - Construction Management Plan (addressing land clearance, water / waste management, air quality, noise, vibrations and other environmental impacts associated with construction);
  - Community Development Plan;
  - Economic Displacement and Livelihoods Restoration Plan;
  - Human Resources Management Plan;
  - Heritage Resources Management Plan;
  - Grievance Management Plan;
  - Health and Safety Plan; and
  - Security Management Plan.
- Additional plans to be prepared and / or reviewed during the construction phase ready for operation include (note these may be individual or combined plans):
  - Community Health and Safety Plan;
  - Water and Waste Management Plans;
  - Land and Wildlife Management Plan;
  - Air Quality Management Plan;
  - Emergency Preparedness and Response Plan;
  - Hazardous Materials Management Plan; and
  - Closure and Rehabilitation Plan based on the closure strategy.

19.4.5 ARDML

Introduction

Mining activities result in the disturbance and exposure of rock at the Earth's surface. This increases the likelihood of exposure of minerals within the rock to air and water, effectively accelerating natural weathering processes. If present in the rock, sulfide minerals such as pyrite (FeS_2) will oxidise, resulting in the generation of acidic conditions. Additionally accelerated weathering of minerals containing potentially problematic metals can result in generation of leachates that may be deleterious to human health, animals and the wider environment. Although these weathering processes would occur naturally over extensive geological timeframes they can be accelerated by mining.

SRK Consulting has undertaken a geochemical characterisation study to assess the Acid Rock Drainage and Metal Leaching ("ARDML") potential waste rock from the proposed development at Madaouela project, Niger. This report details the approach taken, presents the main findings of the work and draws conclusions on the potential for acid generation and metal release both during and after mining operations.

Methodology

Sample selection and geochemical testing

A series of short term geochemical tests were undertaken to allow the preliminary assessment of ARDML characteristics of waste rock that may potentially be excavated during mining. For the purposes of the ARDML assessment, 50 samples of waste rock and 4 samples of ore were collected which are representative of the spatial distribution of major waste rock material types that will be generated.

The following tests were undertaken:

- **Acid Base Accounting ("ABA") (BS EN 15875)** – analysis of potentially acid generating sulfur species and acid neutralising carbonate species within the sample to give an idea of its potential to generate acidity.
- **Multi element analysis of solids** – near-total geochemical assays using multi-acid digestion (4 acid mix with microwave assisted dissolution), to allow geochemical characterisation of the materials.
- **2 stage 10:1 contact leach** – using a standard European leach test (BS EN12457-3) the sample is washed with deionised water (at a 2:1 and 8:1 solid:liquid ratio) and the resulting leachates analysed for dissolved constituents. This gives an indication of the elements that are available for immediate release from the sample.
- **Static Net Acid Generation ("NAG") testing with NAG leachate analysis** – uses hydrogen peroxide to oxidise all exposed sulfide minerals in the samples. This gives a high end estimate of the acidity that may be produced through oxidative weathering of any exposed materials and allows the high end determination of elemental release through oxidative weathering.
- **Mineralogical examination** – by optical microscopy and X-Ray Diffraction (XRD). This allows the identification of potentially acid forming and potentially acid neutralising minerals in the waste.

Results

Acid Generating Potential

The results show that the ARDML potential of the Madaouela materials is generally low. All of the sampled core materials reported low sulfide sulfur concentrations (< 0.041 wt%). All of the acid-base and leach test procedures reported solutions of circum-neutral to slightly alkaline pH values, indicating a lack of net acid generating potential within the sample population. The surface samples reported moderately higher sulfide sulfur concentrations, but none show other acid forming characteristics.

The core materials are all either classifiable as non-acid generating, or as uncertain. The uncertain classification is conservative, and based upon there being limited neutralising potential in some samples. The samples report neutralisation potential values averaging 41 kg $CaCO_3$ eq/t, around 30 times greater than the average acid potential value, indicating a substantial excess of neutralising material. Overall, based upon the low sulfide sulfur concentrations it is unlikely that there is substantial potential for acid generation.

The test data was evaluated for differences between the same lithologies in different areas of the concession, but did not find any correlations between location and ARDML characteristics.

Metal Leaching Potential

The metal leaching potential of the waste rock samples was determined from the following:

- **Multi-element assay** – to determine which elements are present in the sample at concentrations that may prove to be potentially problematic (based on comparison with average crustal abundance)
- **Deionised water leach** – to identify parameters available for immediate release from the waste rock materials under short term weathering conditions
- **Hydrogen peroxide leach** – to indicate the maximum available release of potentially problematic elements through sulfide oxidation.

The rock digest followed by multi-element assay identified arsenic, antimony, manganese, molybdenum, uranium, zinc to be elevated in one or more material types. Whilst these metals occur at concentrations greater than the general crustal abundance, it is the mobility of these metals that determines whether there is potential for leaching of metals from exposed surfaces or waste rock.

The leach protocols released limited quantities of dissolved constituents into solution. Under comparable liquid:solid ratios contact waters would be expected to be of generally low salinity. The exception to this is the surface samples, which show higher major ion solute release than the core samples.

Due to the low acid release potential of the sample materials, the hydrogen peroxide leach did not result in low pH solutions, such that the hydrogen peroxide leach and the deionised water leach would be expected to report analogous results. Under both leach conditions the pH of the solutions were circum-neutral to slightly alkaline. Therefore, the hydrogen peroxide leach indicates that low pH conditions did not occur even under highly oxidising conditions, and that any metals release occurred under non-acidic conditions.

The highest metal release rates were observed for uranium in the ore samples. The ore samples also showed elevated concentrations of manganese and molybdenum. The core samples of the Guerzoman and Talak lithologies reported elevated concentrations of arsenic, cadmium, molybdenum and uranium. Molybdenum concentrations were particularly high; up

to 6,000 µg/L in leachate from one of the Talak core samples. At comparable liquid:solid ratios, contact waters from the Talak and Guezoman lithologies would be expected to exceed the WHO drinking water quality guideline value of 70 µg/L for molybdenum. Cadmium and arsenic could also be expected to exceed the WHO drinking water guideline values based upon the leach test data. Antimony, silver, mercury, tin, thallium and tungsten were generally below analytical detection limits in the leachates, indicating that these parameters are unlikely to be mobilised under natural weathering conditions.

The ARDML characteristics of Madaouela waste materials are summarised in Table 19-2.

Table 19-2: Summary of Madaouela ARDML characteristics

Sample type	Material type	ARD Class	Metal Leaching Potential	Constituents Commonly Elevated in Leachates
Core	Guezoman	Non Acid Forming	Moderate	Arsenic, cadmium, molybdenum, uranium
	Guezoman (altered)	Non Acid Forming	Low	
	Izegouandane	Non Acid Forming	Low	
	Talak	Non Acid Forming	Moderate	Aluminium, arsenic, cadmium, molybdenum
	Tarat	Non Acid Forming	Low	
	Tchinozogue (Blanchard)	Non Acid Forming	Low	
	Tchinozogue (inc TM)	Non Acid Forming	Low	
	Unit Terminal	Non Acid Forming	Low	Uranium
Surface	Guezoman	Non Acid Forming	Moderate	Iron, uranium
	Talak	Non Acid Forming	Moderate	Calcium, sulfate, iron
	Unite d'Akokan	Non Acid Forming	Low	
Ore	High Grade Ore	Non Acid Forming	High	Manganese, molybdenum, uranium
	Low Grade Ore	Non Acid Forming	High	Manganese, molybdenum, uranium

Preliminary Waste Rock Handling and Design Options

Waste rock storage and handling must be considered within the local climate conditions, where rainfall is limited and recharge to groundwater is understood to be negligible. Therefore it is currently likely that waste rock above ground will pose low risk to groundwater as there is limited potential for solutes released to reach groundwater and there are no surface water bodies in the region.

The majority of waste rock from the Madaouela deposit is likely to comprise relatively inert materials. The waste rock materials are likely to have a low acid generating potential and generally low metals release, except for molybdenum. Molybdenum is currently elevated within background groundwater, such that contact waters reaching groundwater may not significantly substantially alter groundwater composition. Uranium leaching is also observed, particularly from ore grade materials, but also from non-ore grade materials of the Guezoman and Talak lithologies.

20 CAPITAL AND OPERATING COSTS (ITEM 21)

The tables below summarises the capital and operating costs for the Madaouela Uranium Project. The detailed development of these individual costs are provided in the relevant sections.

20.1 Capital Expenditure

Total capital expenditure for the life of the operation are presented in Table 20-1. A three-year construction period ahead of plant production is envisaged. Beneficiation plant capital includes a 10 % contingency.

Table 20-1: Capital expenditure

Parameter	Units	Total amount
Project Capital		
Capitalised operating costs (incl year 1)	(USDm)	66.1
Mining	(USDm)	29.7
Processing	(USDm)	234.1
Tailings	(USDm)	3.1
Infrastructure	(USDm)	20.4
Water Management	(USDm)	5.2
Total Project Capital	**(USDm)**	**358.5**
Sustaining capital		
Mining	(USDm)	303.7
Tailings	(USDm)	4.8
Infrastructure	(USDm)	5.4
Water Management	(USDm)	3.4
Total Sustaining Capital	**(USDm)**	**317.3**
Total Capital Expenditure	**(USDm)**	**675.8**

20.2 Operating Costs

Life of mine operating costs have been presented in Table 20-2. By-product credits for the molybdenum are not included in Table 20-2.

Table 20-2: LoM operating costs

Operating Cost Item	Total amount (USDm)	Unit Cost (USD/t ore)
Mining	813	27.61
Processing[1]	694	23.57
Tailings	22	0.74
Site Infrastructure	16	0.54
Water management	15	0.50
G&A	105	3.56
Distribution cost	6	0.22
Closure Cost	15	0.51
Subtotal operating costs	**1,686**	**57.25**

21 ECONOMIC ANALYSIS (ITEM 22)

GoviEx and SRK completed a technical economic analysis on the Project. The various assumptions and inputs are described in the section below:

- Miriam open pit is mined first, with the preference on processing the higher grade ore ;
- Marianne-Marilyn and MSNE-Maryvonne underground mines follow Miriam;
- Remaining lower grade ore from Miriam is used to maximise mill throughput during the underground ramp up stages; and
- Ore processing is based on radiometric ore sorting followed by ablation and two-stage leach with recovery of uranium and molybdenum oxides through the use of Cyanex 600 solvent exchange.

Molybdenum is considered as a by-product from the processing plant, based on predominantly metallurgical testwork results, and is included as specific operating and capital costs have been included to recover it. SRK notes that the molybdenum is not included in the Mineral Resource or Ore Reserve statements.

21.1 Inputs

The following assumptions have been applied to the life-of-mine plans and the economic analysis:

- Mineral Resources as defined in line with the CIM guidelines;
- Maximum plant feed of 1.47 Million tonnes per annum;
- Overall uranium (eU, determined largely by radiometric analysis) recovery of 93.7%, and molybdenum recovery of 67.0% based an average 472 ppm grade;
- A ROM molybdenum grade of 472 ppm, results in approximately 592 t of MoO_2 per annum. The revenue generated by the molybdenum is presented in the below as a by-product credit to offset the operating costs. Operating and capital costs have been included in the processing plant estimates. SRK notes that Mo has not been modelled in the Mineral Resource estimate;
- A total project life of 24 years (3 construction, 21 revenue generating);
- An assumed U_3O_8 price of USD 70.00 /lb, and MoO_2 price of USD 11.00 /lb in line with market forecasts;
- Operating costs and capital expenditures as defined in Section 26;
- A 30 % income tax rate after a three year tax holiday;
- Royalty rate based on the Niger Mining Code which states a sliding scale royalty rate from 5.5 % up to 12 % for higher profit to revenue ratios. The royalty is assumed payable on both uranium and molybdenum revenue. SRK notes that the revenue associated with the molybdenum is presented as a by-product credit in the below, however for the royalty calculations is treated appropriately;
- A base case 8 % discount rate; and
- No provision for salvage value at closure is assumed in the analysis.

Total ore tonnages and grades for the three different deposits are presented in Table 21-1. Figure 21-1 and Figure 21-2 present the mining production schedule and the mill feed schedule, respectively, per source of material.

Table 21-1: Technical mining inputs

Parameter	Units	Base Case
LoM (ore production)	(years)	21
Marianne-Marilyn	(years)	11
MSNE-Maryvonne	(years)	7
Miriam	(years)	5
RoM Ore	(kt)	29,454
Marianne-Marilyn	(kt)	14,132
MSNE-Maryvonne	(kt)	7,813
Miriam	(kt)	7,510
RoM U Grade	(kg/t)	0.79
Marianne-Marilyn	(kg/t)	0.79
MSNE-Maryvonne	(kg/t)	0.76
Miriam	(kg/t)	0.82
RoM Mo Grade[1]	(kg/t)	0.47
U Content	(kt)	23.2
Marianne-Marilyn	(kt)	11.1
MSNE-Maryvonne	(kt)	5.9
Miriam	(kt)	6.2

[1]*Mo grade is not JORC compliant and is based on metallurgical testwork and not a resource estimation*

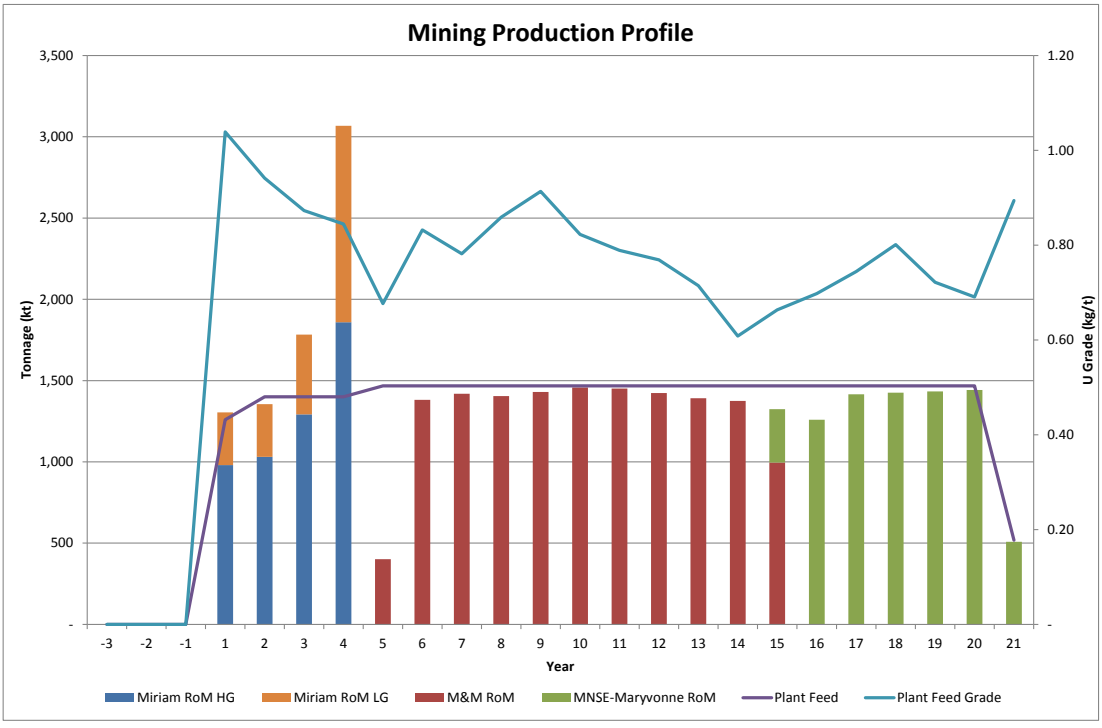


Figure 21-1: RoM Ore Production Profile

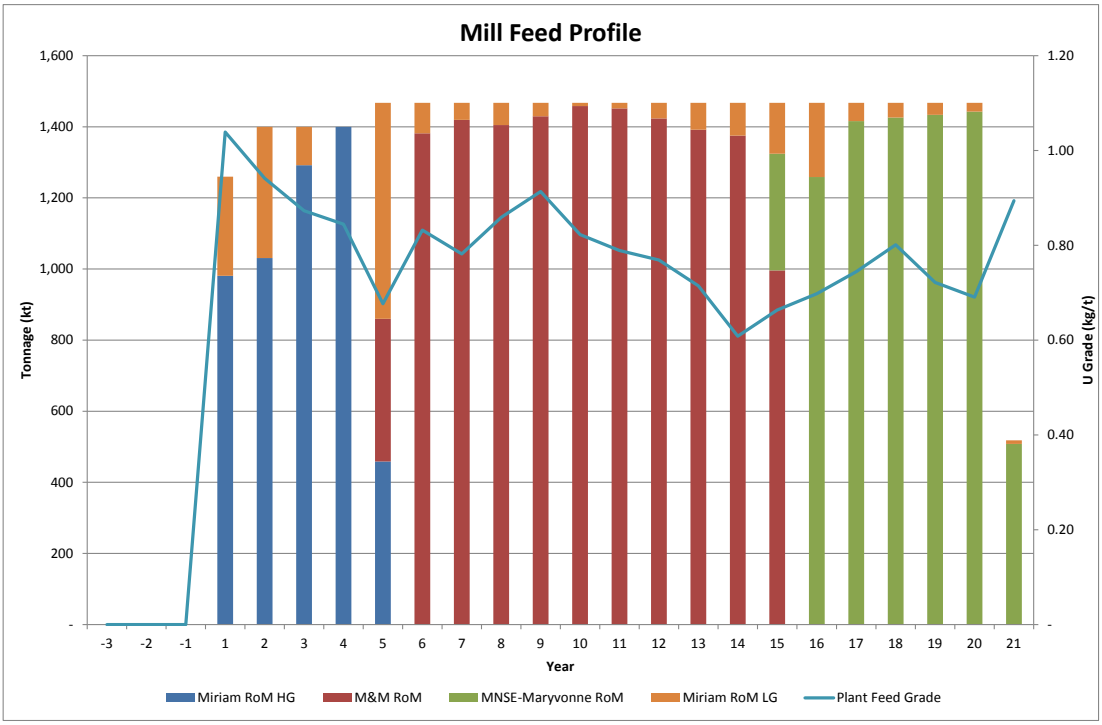


Figure 21-2: Mill Feed Production Profile

21.2 Results

The base case economic analysis results in a post-tax pre-finance NPV of USD 340 M (at 8 % discount rate), with an IRR of 23.5 %. Undiscounted payback will be in production year 3. Total LoM statistics are presented in Table 21-2. A year by year summary is presented in the technical economic model summary (Table 21-5).

Table 21-2: Technical economic model summary and results

Parameter	Units	Base Case
Mining		
RoM Ore	(kt)	29,454
U Grade	(kg/t)	0.79
U Content	(kt)	23.2
Processing		
U Recovery	(%)	93.7%
U Recovered	(lb)	47,955,179
Revenue		
U_3O_8 sales	(lb)	56,550,835
U_3O_8 price	(USD/lb)	70.00
U_3O_8 revenue	(USDm)	3,959
Operating Expenditure		
Direct Operating Costs	(USDm)	1,686
By-product credits (Mo)	(USDm)	(301)
Subtotal operating costs	(USDm)	**1,385**
Royalty	(USDm)	396
Total Operating Costs	**(USDm)**	**1,781**
Unit Operating Costs		
Subtotal operating costs	(USD/t ore)	47.01
	(USD/lb U)	28.88
	(USD/lb U_3O_8)	24.49
Royalty	(USD/t ore)	13.44
Total Operating Costs	(USD/t ore)	60.46
	(USD/lb U)	37.13
	(USD/lb U_3O_8)	31.49
Operating Profit – EBITDA	**(USDm)**	**2,178**
Corporate Profit Tax	**(USDm)**	**376**
Capital Expenditure		
Project	(USDm)	359
Sustaining	(USDm)	317
Total Capital Expenditure	**(USDm)**	**676**
Net Free Cash	**(USDm)**	**1,126**
NPV @ 8.00%	**(USDm)**	**339.8**
NPV @ 10.00%	**(USDm)**	**251.5**
IRR	(%)	23.5%
Break even price (NPV=0 @ 8%)	(USD/lb U_3O_8)	47.79

21.3 Sensitivity

Table 21-3 presents NPV and IRR sensitivity results for changes in price and discount rate, and Figure 21-3 graphically presents the NPV sensitivity to commodity price, operating costs and capital expenditures.

Table 21-3 presents NPV and IRR sensitivity results for changes in price and discount rate.

Table 21-3: NPV and IRR sensitivity to price and discount rate

Price (USD/lb U_3O_8)	Discount rate (%)	NPV (USDm)	IRR (%)
70	10%	251	23.5%
	8%	340	
75	8%	411	26.2%
65	8%	263	20.2%

The molybdenum grade has been taken from testwork as the current resource model does not account for by-product elements. The range of molybdenum concentrations in the deposit is high from 300 ppm to more than 4,200 ppm based on several studies (CEA 1965, GoviEx data unpublished, SGS 2009 2013, and Mintek 2012). The molybdenum occurs in close proximity to uranium but rarely overlapping as it is formed in more reducing environments than where the uranium is precipitated hence molybdenum grade cannot be reliably deduced from knowledge of uranium grade.

For purposes of determining reagent consumption related to molybdenum and possible product generation the grade of 1,200 ppm was taken from multiple analysis of the 8 t bulk metallurgical sample that was mined from M&M and analysed at Mintek. In addition testwork at Ablation technologies on Miriam ore gave an average of 850 ppm. In order to provide a conservative approach the 850 ppm number was used. When diluted in the ore feed this grade decreases to 472.

A sensitivity to Mo grade (base case 472 ppm) is presented in Table 21-4 at the base U price of USD70 /lb and 8% discount rate.

Table 21-4: NPV and IRR sensitivity to price and discount rate accounting for Mo

Mo grade	NPV (USDM)	IRR (%)
300 ppm	310	22.3%
472 ppm (Base case)	340	23.5%
600 ppm	362	24.3%

The sensitivity of project value (NPV) to uranium price is +USD 99 M for a +10 % change and –USD 109 M for a -10 % change on the base price of USD 70 /lb U_3O_8. Sensitivity of project value (NPV) to operating costs and capital cost are broadly similar at around ±USD 41 M for a ± 10 % change. Sensitivity to every ±10% change in molybdenum grade is ±USD 8 M.

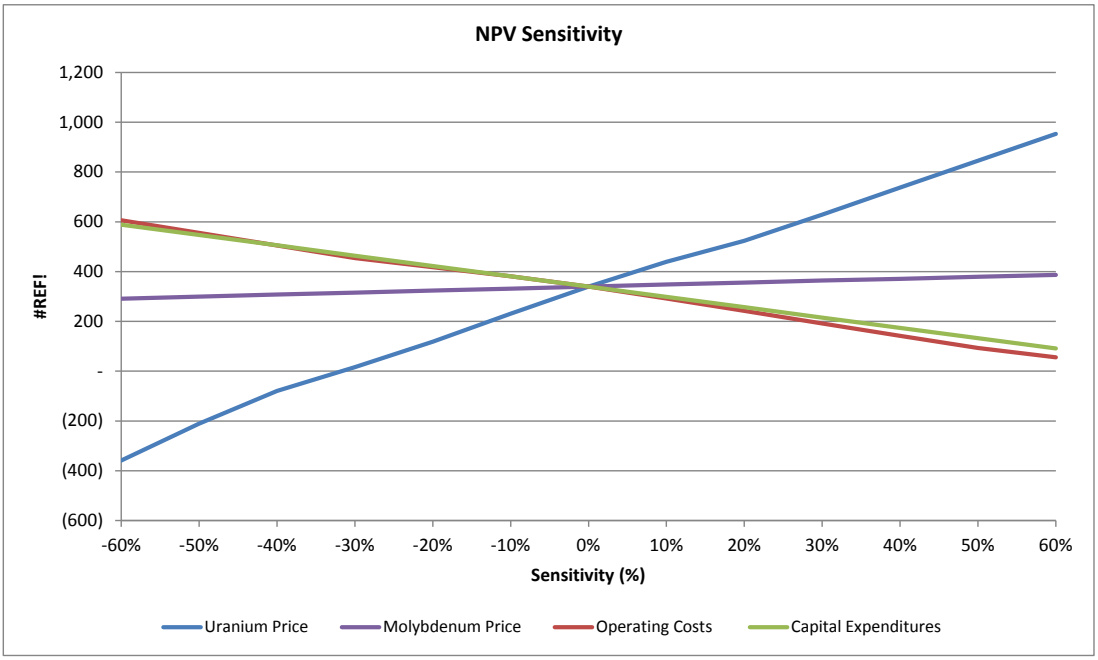


Figure 21-3: NPV (at 8%) sensitivity to uranium price, molybdenum price, operating costs and capital expenditure

Table 21-5: Technical economic model

Year	Units	Totals	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21
PRODUCTION																										
Total Mining																										
RoM Ore	(kt)	29,454	-	-	0	1,304	1,355	1,782	3,068	401	1,382	1,419	1,405	1,430	1,458	1,452	1,423	1,392	1,375	1,324	1,259	1,416	1,426	1,434	1,443	508
Grade	(kg U/t)	0.79	-	-	0.56	1.02	0.96	0.79	0.70	0.77	0.85	0.79	0.88	0.92	0.82	0.79	0.78	0.73	0.62	0.68	0.73	0.75	0.81	0.73	0.69	0.90
Content	(kt)	23.2	-	-	0.0	1.3	1.3	1.4	2.2	0.3	1.2	1.1	1.2	1.3	1.2	1.2	1.1	1.0	0.8	0.9	0.9	1.1	1.2	1.0	1.0	0.5
Total Sales																										
Uranium metal contained	(kt U)	21.8	-	-	-	1.2	1.2	1.1	1.1	0.9	1.1	1.1	1.2	1.3	1.1	1.1	1.1	1.0	0.8	0.9	1.0	1.0	1.1	1.0	0.9	0.4
U_3O_8	(kt U_3O_8)	25.7	-	-	-	1.4	1.5	1.3	1.3	1.1	1.3	1.3	1.4	1.5	1.3	1.3	1.2	1.2	1.0	1.1	1.1	1.2	1.3	1.2	1.1	0.5
	(lb U_3O_8)	56,550,835	-	-	-	3,185,562	3,208,484	2,975,780	2,878,907	2,417,963	2,972,059	2,792,767	3,068,449	3,262,556	2,938,641	2,818,934	2,747,185	2,552,777	2,172,654	2,370,902	2,492,879	2,658,662	2,861,947	2,578,907	2,466,857	1,127,964
Sales Summary																										
U_3O_8	(lb U_3O_8)	56,550,835	-	-	-	3,185,562	3,208,484	2,975,780	2,878,907	2,417,963	2,972,059	2,792,767	3,068,449	3,262,556	2,938,641	2,818,934	2,747,185	2,552,777	2,172,654	2,370,902	2,492,879	2,658,662	2,861,947	2,578,907	2,466,857	1,127,964
Product Prices																										
U_3O_8	(USD/lb)	70.00	-	-	-	70.0	70.0	70.0	70.0	70.0	70.0	70.0	70.0	70.0	70.0	70.0	70.0	70.0	70.0	70.0	70.0	70.0	70.0	70.0	70.0	70.0
Revenue																										
U_3O_8	(USDm)	3,959	-	-	-	223.0	224.6	208.3	201.5	169.3	208.0	195.5	214.8	228.4	205.7	197.3	192.3	178.7	152.1	166.0	174.5	186.1	200.3	180.5	172.7	79.0
Operating Expenditure																										
Mining	(USDm)	(813)	-	-	-	(45.7)	(47.3)	(44.9)	(39.2)	(20.5)	(42.8)	(40.0)	(40.7)	(40.3)	(40.0)	(40.0)	(41.2)	(41.0)	(41.7)	(39.6)	(35.6)	(39.6)	(39.4)	(39.6)	(39.6)	(14.2)
Processing	(USDm)	(694)	-	-	-	(30.8)	(33.0)	(33.0)	(33.0)	(34.1)	(34.1)	(34.1)	(34.1)	(34.1)	(34.1)	(34.1)	(34.1)	(34.1)	(34.1)	(34.1)	(34.1)	(34.1)	(34.1)	(34.1)	(34.1)	(19.1)
Tailings	(USDm)	(22)	-	-	-	(0.9)	(1.0)	(1.0)	(1.0)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(1.1)	(0.4)
Site Infrastructure	(USDm)	(16)	-	-	-	(0.6)	(0.6)	(0.6)	(1.0)	(0.8)	(0.8)	(0.8)	(0.8)	(0.8)	(0.8)	(0.8)	(0.8)	(1.1)	(1.1)	(1.1)	(0.6)	(0.6)	(0.6)	(0.6)	(0.6)	(0.6)
Water Management	(USDm)	(15)	-	-	-	(0.5)	(0.8)	(0.8)	(0.8)	(0.7)	(0.7)	(0.7)	(0.7)	(0.7)	(0.7)	(0.8)	(0.8)	(0.8)	(0.8)	(0.8)	(0.7)	(0.7)	(0.7)	(0.7)	(0.7)	(0.7)
G&A	(USDm)	(105)	-	-	-	(5.0)	(5.0)	(5.0)	(5.0)	(5.0)	(5.0)	(5.0)	(5.0)	(5.0)	(5.0)	(5.0)	(5.0)	(5.0)	(5.0)	(5.0)	(5.0)	(5.0)	(5.0)	(5.0)	(5.0)	(5.0)
Distribution cost	(USDm)	(6)	-	-	-	(0.4)	(0.4)	(0.3)	(0.3)	(0.3)	(0.3)	(0.3)	(0.3)	(0.4)	(0.3)	(0.3)	(0.3)	(0.3)	(0.2)	(0.3)	(0.3)	(0.3)	(0.3)	(0.3)	(0.3)	(0.1)
Closure	(USDm)	(15)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	(15.0)
Molybdenum (by product credits)	(USDm)	301	-	-	-	14.4	14.4	14.4	14.4	14.4	14.4	14.4	14.4	14.4	14.4	14.4	14.4	14.4	14.4	14.4	14.4	14.4	14.4	14.4	14.4	14.4
Subtotal operating costs	**(USDm)**	**(1,385)**	**-**	**-**	**-**	**(69.5)**	**(73.7)**	**(71.2)**	**(66.0)**	**(48.1)**	**(70.4)**	**(67.6)**	**(68.3)**	**(68.0)**	**(67.6)**	**(67.7)**	**(68.9)**	**(69.0)**	**(69.6)**	**(67.6)**	**(63.0)**	**(67.0)**	**(66.8)**	**(67.0)**	**(67.0)**	**(40.8)**
Royalty	(USDm)	(396)	-	-	-	(21.4)	(21.5)	(20.0)	(19.4)	(16.5)	(20.0)	(18.9)	(20.6)	(21.8)	(19.8)	(25.4)	(24.8)	(17.4)	(15.0)	(16.2)	(17.0)	(18.0)	(19.3)	(17.5)	(16.8)	(8.4)
Total operating costs	**(USDm)**	**(1,781)**	**-**	**-**	**-**	**(90.9)**	**(95.2)**	**(91.2)**	**(85.4)**	**(64.6)**	**(90.5)**	**(86.4)**	**(88.9)**	**(89.8)**	**(87.4)**	**(93.1)**	**(93.7)**	**(86.3)**	**(84.6)**	**(83.8)**	**(80.0)**	**(85.1)**	**(86.1)**	**(84.5)**	**(83.9)**	**(49.2)**
Operating Profit - EBITDA	**(USDm)**	**2,178**	**-**	**-**	**-**	**132.1**	**129.4**	**117.1**	**116.1**	**104.7**	**117.6**	**109.0**	**125.8**	**138.6**	**118.3**	**104.2**	**98.6**	**92.4**	**67.5**	**82.2**	**94.5**	**101.0**	**114.2**	**96.0**	**88.8**	**29.8**
Corporate Income Tax																										
Profit tax	(USDm)	(376)	-	-	-	-	-	-	(21.7)	(17.3)	(24.2)	(20.2)	(25.1)	(28.3)	(22.1)	(26.2)	(25.1)	(21.7)	(13.9)	(17.4)	(19.3)	(21.1)	(25.6)	(20.9)	(20.3)	(5.4)
Net Profit	(USDm)	1,802	-	-	-	132.1	129.4	117.1	94.4	87.4	93.4	88.9	100.8	110.3	96.2	78.0	73.5	70.6	53.7	64.8	75.2	79.9	88.6	75.1	68.5	24.4
Capital Expenditure																										
Capitalised pre-production operating costs	(USDm)	(66)	(0.2)	(24.7)	(41.2)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mining	(USDm)	(333)	-	(14.6)	(9.7)	(5.4)	(1.1)	(0.2)	(16.6)	(46.5)	(29.5)	(3.1)	(10.7)	(18.6)	(4.7)	(17.1)	(28.0)	(18.5)	(33.4)	(33.1)	(20.7)	(14.8)	(3.5)	(3.2)	(0.4)	-
Processing	(USDm)	(234)	-	(93.6)	(140.4)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Tailings	(USDm)	(8)	-	-	(3.1)	-	-	-	-	(1.2)	-	(1.2)	-	-	(1.2)	-	-	-	-	(1.2)	-	-	-	-	-	-
Infrastructure	(USDm)	(26)	(20.4)	-	-	-	-	(2.7)	-	-	-	-	-	-	-	(2.7)	-	-	-	-	-	-	-	-	-	-
Water Management	(USDm)	(9)	(0.3)	(2.0)	(1.5)	(1.5)	(1.6)	-	-	-	-	-	-	-	-	(1.7)	-	-	-	-	-	-	-	-	-	-
Total Capital Expenditure	**(USDm)**	**(676)**	**(20.9)**	**(134.9)**	**(195.8)**	**(6.9)**	**(2.7)**	**(2.9)**	**(16.6)**	**(47.7)**	**(29.5)**	**(4.3)**	**(10.7)**	**(18.6)**	**(5.9)**	**(21.5)**	**(28.0)**	**(18.5)**	**(33.4)**	**(34.3)**	**(20.7)**	**(14.8)**	**(3.5)**	**(3.2)**	**(0.4)**	**-**
Net Free Cash	**(USDm)**	**1,126**	**(20.9)**	**(134.9)**	**(195.8)**	**125.2**	**126.6**	**114.2**	**77.8**	**39.7**	**63.9**	**84.6**	**90.1**	**91.6**	**90.3**	**56.5**	**45.5**	**52.1**	**20.3**	**30.5**	**54.5**	**65.1**	**85.1**	**71.9**	**68.1**	**24.4**
Cumulative NFC	**(USDm)**	**-**	**(20.9)**	**(155.8)**	**(351.6)**	**(226.4)**	**(99.8)**	**14.4**	**92.2**	**131.9**	**195.8**	**280.5**	**370.5**	**462.2**	**552.4**	**609.0**	**654.5**	**706.6**	**726.9**	**757.3**	**811.8**	**876.9**	**962.0**	**1,033.9**	**1,102.0**	**1,126.4**
Reporting Statistics - Real	(USDm)	-																								
Cash Costs (excl royalty, incl by-product credits)	(USD/lb U_3O_8)	24.49	-	-	-	21.8	23.0	23.9	22.9	19.9	23.7	24.2	22.3	20.8	23.0	24.0	25.1	27.0	32.0	28.5	25.3	25.2	23.3	26.0	27.2	36.2
Total Cash Costs	(USD/lb U_3O_8)	31.49	-	-	-	28.5	29.7	30.7	29.7	26.7	30.4	31.0	29.0	27.5	29.8	33.0	34.1	33.8	38.9	35.3	32.1	32.0	30.1	32.8	34.0	43.6

22 ADJACENT PROPERTIES (ITEM 23)

There are currently two producing mines in the Arlit region that are on property positions immediately adjacent to the Madaouela Uranium Project:

22.1 Somaïr

Société des mines de l'Aïr (Somaïr, the mining company of the Aïr) was established in 1968. The company is operated by AREVA, which owns 63.4% of the share capital; the remaining 36.6% is held by Société du patrimoine des mines du Niger (Sopamin, the Nigerien national mining company). Somaïr has operated several uranium deposits near the town of Arlit since 1971. The ore is extracted from open pit mines and heap leached or processed mechanically at the front end of the Arlit mill. In both cases, the uranium solutions are processed at the back end part of the mill, whose capacity was raised to 3,000 metric tons of uranium per year in 2011 (7.8 million pounds of U_3O_8).

22.2 Cominak

Cominak (Compagnie Minière d'Akouta) is 34% owned by AREVA, which operates it. The other shareholders are Sopamin of Niger (31%), Ourd (25%), and Enusa Industrias Avanzadas SA of Spain (Enusa, 10%). The ore is extracted underground and is then processed in the site's mill, producing approximately 1,500 metric tons of uranium per year (3.9 million pounds of U_3O_8).

22.3 Imouraren project

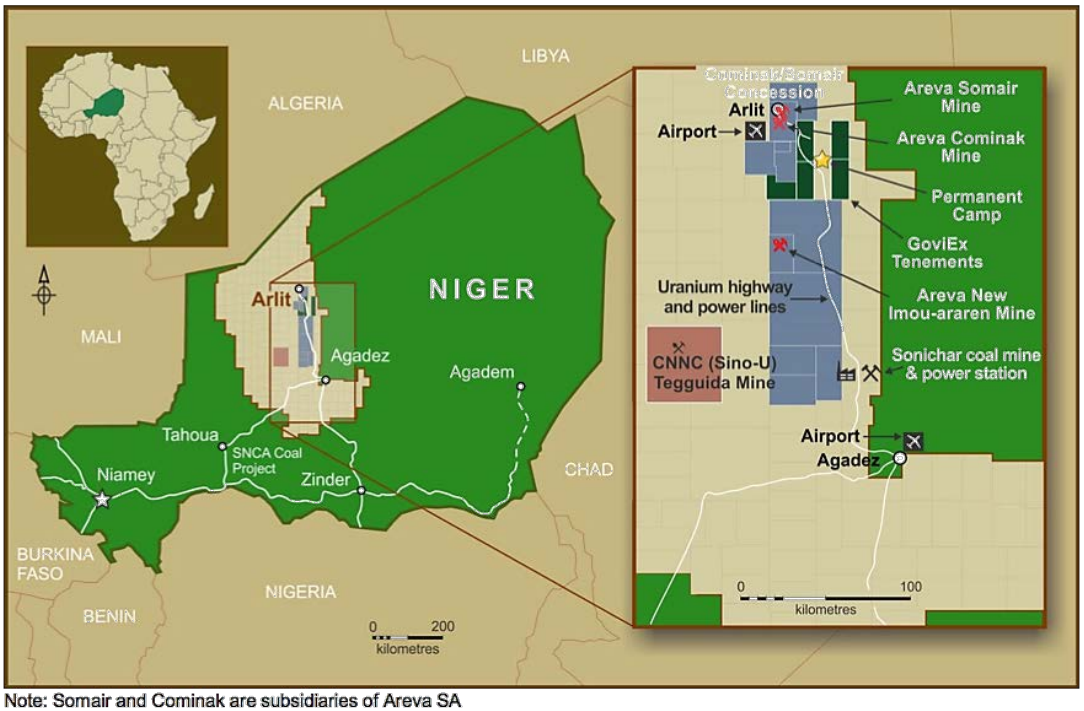
Located 80 kilometers south of Arlit, this deposit was discovered in 1966 and constitutes one of the largest deposits in the world today (174,196 metric tons of reserves). The feasibility study was completed in December 2007 and submitted in April 2008. AREVA received the mining permit for the deposit in early January 2009. The Imouraren SA mining company was established, with AREVA NC Expansion (86.5% AREVA and 13.5% Kepco/KHNP) holding a 66.65% interest and Sopamin of Niger holding the remaining 33.35%. In view of market conditions construction work was suspended. The site, equipment and facilities are currently put under care and maintenance, and all demobilization operations should be completed in the first quarter of 2015.

Table 22-1: Reported Production, Mineral Reserves and Resources for the Areva mines in Niger

	Production	Total Reserves		Measured and Indicated Resources		Inferred Resources	
Mine	TU	TU	Kg/t U	TU	Kg/t U	TU	Kg/t U
Somair	2331	7771	1.46	28919	1.41	21925	1.68
Cominak	1501	12100	3.62				
Imouraren		213772	0.70	62584	0.58	2879	0.66

Source: Areva – 2014 Reference Document

The above stated resources have not been reviewed by a Qualified Person, are unknown as to compliance with or correlation to CIM definitions for mineral resources, and therefore should not be relied upon.



Note: Somair and Cominak are subsidiaries of Areva SA

Figure 22-1: Location of Uranium Projects in Niger

22.4 Statement (SRK)

The above stated public information demonstrates that the Madaouela Uranium Project is located in an active uranium mining district that has been established and continually in production since 1970. There are two other uranium deposits in the Arlit region of north central Niger that are currently in development; however they are not immediately adjacent to the Madaouela Uranium Project.

23 OTHER RELEVANT DATA AND INFORMATION (ITEM 24)

23.1 Madaouela Ventilation Study

23.1.1 Introduction

The underground uranium mines Marilyn-Marianne and MSNE-Maryvonne, which are part of the Madaouela Uranium Mine are designed to be room and pillar operations.

An investigation into the ventilation requirements for Marilyn-Marianne and MSNE-Maryvonne at each stage of mine development has been undertaken. Modelling and simulation work uses CAE Mining's Ventsim Visual 3 software ("Ventsim") to determine the optimum ventilation network with respect to the standard ventilation requirements of uranium mines. The investigation covers the primary and secondary ventilation; intake and exhaust raise development; fan size and power requirements.

23.1.2 Underground Uranium Mines

The use of negative pressure ventilation systems is preferable when efficiently and effectively ventilating underground uranium mines. Single pass ventilation is essential in order to minimise levels of radioactivity in working areas (McPherson, 1993).

Because the rate of decay of radon into its radioactive daughter elements ("radon daughters") increases with time, very small quantities of radon gas can become problematic. Because of this, single-pass ("primary") ventilation should be used where possible and local secondary ventilation of working areas is considered essential.

Radon progeny is considered to have the highest risk potential for occupational exposure in high-grade Canadian uranium mines (Souza, 2009). Radon is highly soluble in water and there is usually a high occurrence of radon in the groundwater located near the mineralised zones. Although ventilation is essential for eliminating the potential for occupational exposures to radon and radon daughters, it must be used in combination with management and operating practices that prevent the release of radon-bearing water into the mine environment. For the purposes of this study, SRK does not considered the inflow of groundwater to be a significant source of radon to the mine atmosphere.

Removal of contamination at the source is regarded as the most effective strategy for dealing with radon progeny, rather than to deal with is within the air stream. Therefore, proper control and containment of radon sources is required, and will help to prevent unreasonable demands (large airflow volumes, high energy costs) on the ventilation system. This can be achieved by prevention of groundwater inflow, well ventilated sumps; low air residency times; effective ventilation management; robust and disciplined operating practices and extensive real-time instrumentation monitoring.

Airflow sensors, alpha nuclear prisms, and fan run indicators are essential instruments for predicting and preventing the occurrence of harmful conditions. These instruments are typically managed by a central control system and operate in real-time. Continuous working level monitors (alpha-numeric prisms) measure radon progeny and the results are represented by a set of prisms resembling a traffic light (green, amber and red light output).

If radon daughters are released into the mine atmosphere during operations, the methods currently available are to remove the contaminated air or dilute it to acceptable contamination levels. Effective management of the measurement and control system is critical, especially in working areas located near the mineralised zones.

In ideal situations, all development should be designed in waste rock above and below the mineralised horizon. In the case of the Madaouela Uranium Mine , the main infrastructure and access development will ideally be located above the mineralised horizon or in low-grade zones to minimise the introduction of radon gas into these areas. Panel development will be developed within the orebody, applying sufficient drainage and secondary ventilation until the primary ventilation circuit can be completed.

Operating practices for panel development and production requires measures to control radiation exposure levels in all working areas, which include:

- Primary ventilation;
- Exhaust raises located within the panels;
- Sufficient fresh air intake capacity provided by designated fresh air raises; and
- Utilisation of ventilation raises as both intake and exhaust within the ore zones during development and production.

Typically the most significant proportion of the ventilation requirements will be dictated by the number, capacity and power rating of diesel-powered equipment. This approach is used when determining the instantaneous airflow volume requirements and developing the ventilation designs for the Madaouela Project. The ventilation design criteria applied are based on the Australian uranium mine ventilation standards (WA Mines Safety and Inspection Act 1994) and industry standard practical ventilation rules (McPherson, 1993), details of which are given in Section 23.1.4.

23.1.3 Ventilation Analysis

The ventilation system is one of the most critical aspects of the long-term mine planning for the Madaouela Uranium Mine, and needs to be considered in all planning activities.

Both Marilyn-Marianne and MSNE-Maryvonne underground mines are assumed to have similar fleet capacities, production rates, water ingress and uranium grades. Therefore, the same ventilation design criteria, flow rate requirements and ventilation concepts have been used to model both mines.

The primary ventilation system is required to provide the mine with sufficient volumes of fresh air within acceptable velocity tolerances and allow a predetermined, logical and realistic sequence of extraction. Fresh air intake development is a critical activity for providing sufficient fresh air to the mine, and therefore has a high priority during development stages.

An auxiliary ventilation system will be used to divert fresh air away from the main ventilation circuit and direct it into panel development, production headings and other excavations. This system can either be forced air, an exhaust type system or a combination of both. A forced system would typically consist of fans, ducting, dust collectors and scrubbers to capture potentially contaminated air (radioactive dust, radon progeny, and diesel particulate matter) from working areas where mineralised material is being handled and deliver it to the exhaust system which will release it at a safe location on the surface. Exhaust raises equipped with exhaust fans can be constructed within the panel development drifts (at specified stages) to provide an exhausting system within the production areas, drawing fresh air from the main ventilation circuit.

The primary philosophies behind the ventilation strategy at the proposed Madaouela Uranium Mine are to;

- Apply primary ventilation to all production areas;

- Minimise air residency times within the underground operations; and
- Minimise the recycling of air underground.

The main ventilation system will consist of a combination of larger diameter fresh air intake raises within the barren waste rock, and smaller diameter exhaust raises within the production panels. This will allow primary ventilation to throughout the mine and if necessary, the exhaust raises can be used temporarily as fresh air intake raises to provide local fresh air during development stages.

Ventilation Requirements

Estimation of Regional Ventilation Air Requirements

An estimation of the primary ventilation air requirements has been made by considering the total instantaneous diesel powered equipment requirements planned for the underground mining activities, number of active panels and operating sumps.

A flow of 0.06 m^3/s per kW is assumed for diesel powered equipment. Due to downtime, maintenance and general operational conditions, it is assumed that the machinery will be operating at 75% overall utilisation.

SRK has assumed that a permanent exhaust raise will be developed during the early stages of the mine life and will draw continuous fresh air to the fuel storage and magazine area. The nearby workshop area can be ventilated by a secondary ventilation, connected to the main fresh air ventilation circuit.

Three sumps per working panel are assumed (15 active sumps at any time), each requiring 3.8 m^3/s, which will result in an additional 57 m^3/s of fresh air supply. It is assumed that shotcrete barricade will be constructed upon depletion of a panel to isolate mined out areas no longer requiring access.

A contingency of 25% is assumed to allow flexibility in the design.

The total maximum diesel fleet, power ratings and ventilation requirements are shown in Table 23-1. A summary of the overall ventilation requirements is given in Table 23-3.

Table 23-1: Summary of maximum diesel fleet requirements

Machine Description	Power Source	Power (kW)	Quantity	Total Power (kW)	Airflow Required (m^3/s)
Drilling					
Face Drill Jumbo 1	Electric	110	1	110	0.0
Face Drill Jumbo 2	Electric	125	4	500	0.0
Load and Haul					
Loader 1	Fuel - Diesel	186	4	744	44.6
Loader 2	Fuel - Diesel	144	5	720	43.2
Haul Truck 1	Fuel - Diesel	298	6	1788	107.3
Ancillary Equipment					
Shotcreter	Fuel - Diesel	96	1	96	5.8
Cement Carrier	Fuel - Diesel	170	1	170	10.2
Integrated Tool Carrier	Fuel - Diesel	48	2	96	5.8
Primary Bulk Explosive Loader	Fuel - Diesel	110	1	110	6.6
Secondary Bulk Explosive Loader	Fuel - Diesel	48	3	144	8.6
Grader	Fuel - Diesel	101	1	101	6.1
Light Vehicle	Fuel - Diesel	151	15	2265	135.9
Rockbolting Machine 1	Electric	58	1	58	0.0
Rockbolting Machine 2	Electric	55	4	220	0.0
Service/Fuel Truck	Fuel - Diesel	151	1	151	9.1
Flat Bed Truck	Fuel - Diesel	62	1	62	3.7
Machinery Sub-Total			51	7,335	386.8
Machinery Subtotal at 75% Utilisation				5,501	290.1

Based on this assessment, SRK estimates the overall airflow requirements for the Madaouela underground operations to be a minimum of 290 m^3/s to account for all diesel equipment proposed for underground mining.

Estimation of Local Ventilation Requirements

A typical operating machinery scenario and thus ventilation requirement was assumed for each working panel, whereby the maximum number of machines possible were assumed to be working in the panel at the same time. Based on this evaluation, each panel will require a minimum of 100 m^3/s (Table 23-2). Typical ventilations requirements would be expected to be lower than the figures quoted.

Table 23-2: Summary of typical diesel fleet requirements for a working panel

Machine Description	Power Source	Power (kW)	Quantity	Total Power (kW)	Airflow Required (m^3/s)
Loader 1	Fuel - Diesel	186	1	186	11.2
Loader 2	Fuel - Diesel	144	2	288	17.3
Haulage Truck	Fuel - Diesel	298	2	596	35.8
Face Drill Jumbo 1	Electric	110	1	110	0.0
Face Drill Jumbo 2	Electric	125	1	125	0.0
Rockbolting Machine 1	Electric	63	1	63	0.0
Light Vehicle	Fuel - Diesel	151	2	302	18.1
Explosives Loader	Fuel - Diesel	110	1	110	6.6
Machinery Sub-Total		1187	11	1780	88.9
Operating Sumps			3		11.4
TOTAL					100.3

Summary of Global Ventilation Requirements

For the purposes of this study, the overall ventilation requirements for the mine are modelled based on the number of active panels and additional panels assumed to be in development for future production. Each of which will be provided with the estimated maximum instantaneous ventilation requirement of 100 m^3/s.

A maximum of five working panels have been assumed to be in production at any given time. In addition, up to two additional panels are assumed to be being developed for future production simultaneously. Exhaust fans are to be installed in all developed panels to maintain a negative pressure in the mine, resulting in an additional fresh air intake of up to 200 m^3/s.

Three sumps have been assumed per active panel, with each sump requiring 3.8 m^3/s of fresh air ventilation.

Table 23-3 shows the summary of the estimated overall underground ventilation requirements.

Table 23-3: Summary of global ventilation requirements

	Quantity	Airflow Required (m^3/s)
Active Panels	5	501.6
Additional Developed Panels	2	200.6
Workshop Exhaust Raise	1	100.3
Operating Sumps	15	57.0
TOTAL		859.6
TOTAL + 25% Contingency		1074.5

When considering active and developed panels, operating sumps and the maximum overall fresh airflow requirement is approximately 1,075 m^3/s.

It is expected that ventilation management at an operational level could reduce the total

airflow by balancing air intake with requirements for specific underground activities to minimise unnecessary airflow into low activity areas.

Applying the diesel machinery capacity and utilisation instead of the maximum instantaneous airflow requirement for each panel will reduce the overall ventilation requirements are approximately 810 m^3/s, inclusive of the 25% contingency.

Primary Ventilation Infrastructure

The primary ventilation infrastructure required is based on the underground mine designs. The designs are modelled using the Ventsim software package which allows:

- Simulation providing a record of flows for an underground mine design;
- Selection of circuit fans for primary and secondary ventilation;
- Simulation of paths and concentrations of smoke dust, or gas;
- Full thermodynamic analysis of heat, humidity and refrigeration;
- Dynamic time based analysis of contaminants and concentrations of blast fumes;
- Recirculation checks; and
- Simulation of Diesel Particulate Matter (“DPM”) concentrations through the mine.

For this study the design, simulation, fan modelling, dynamic contaminant and recirculation functions of the Ventsim software were used. Further ventilation planning is recommended for future development of this project using Ventsim or an equivalent software to gain additional confidence in the ventilation strategy and better model the impact of the impact of radiation on the ventilation design.

Secondary Ventilation Infrastructure

The secondary ventilation infrastructure will consist of flexible ventilation ducting and auxiliary fans located in both the main access drives and panel development. The primary objective of this secondary ventilation is to provide fresh air to the development headings, and direct contaminated air to the return air system.

A ventilation duct with a diameter 1.2 m is considered to be effective at providing sufficient airflow to the panel development drives with 2 x 55 kW development fans.

SRK has considered the physical constraints of installing the secondary ventilation systems in the main access development and panel development drives. Figure 23-1 and Figure 23-2 shows the sectional area of the haul truck, heaped pile and ventilation duct within each profile.

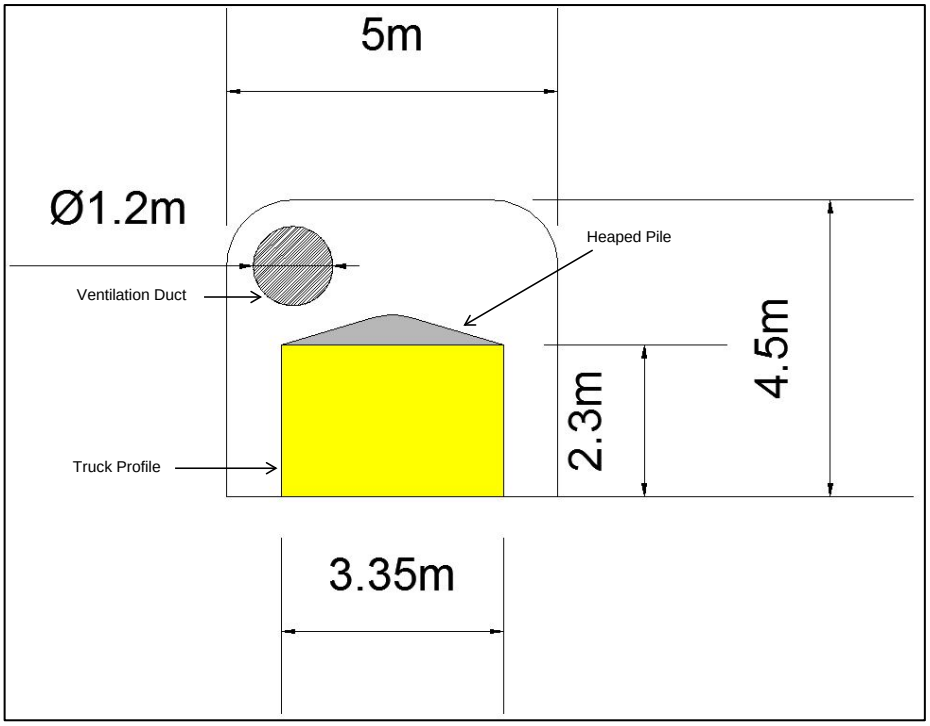


Figure 23-1: Main access development cross-section

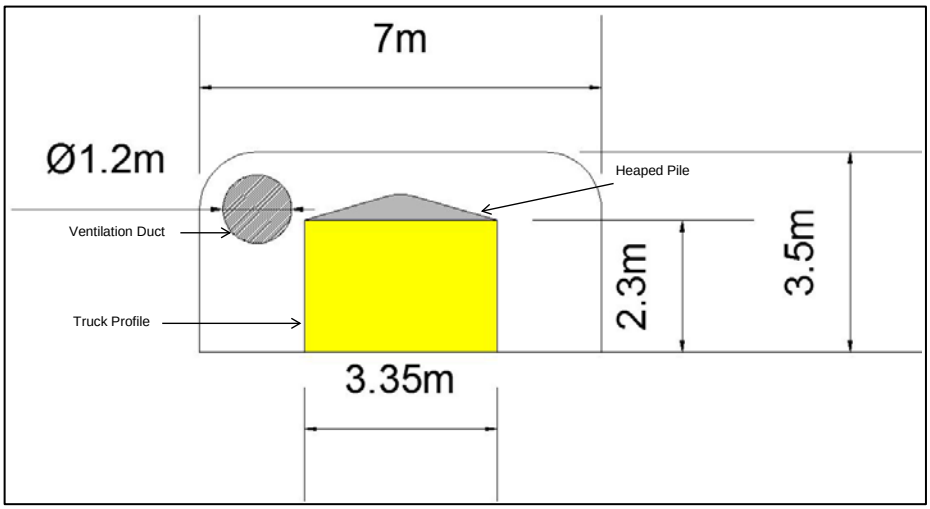


Figure 23-2: Panel development cross-section

23.1.4 Ventilation Design Criteria

Introduction

For the purposes of this study, natural ventilation and potential heat generation was not considered, however the air flow was assumed to be compressible.

Legislation

Niger has a long history of underground mining of Uranium so has an established regulatory environment for ventilation of radiation in underground mines. SRK has reviewed these regulations from 1975, which under article 6 defined the required air per worker at 7 m^3 ,

which must be maintained in a state of purity sufficient for a worker to complete his tasks. SRK decided to use the Australian standards and regulations are as a basis for evaluation, as they are more up to date and provide more detailed and stringent requirements.

Airflow Rates & Velocities

In developing the ventilation network, the following airflow design criteria are considered material:

- Airflow volumes must meet the requirements for the diesel machine fleet operating in the mine and each panel;
- Airflow volumes in specific areas should meet the requirements as described in Table 23-4 (Souza, 2009);
- Air velocities in specific areas should not exceed the recommended values stated in Table 23-5 (McPherson, 1993).

Table 23-4: Airflow requirements by area

Area	Airflow m^3/s
Active Extraction Areas	7.1 - 27.2
Development Headings	15
Miscellaneous Mucking	10.4
Ore Bin Area	5
Active Travelways	15.0 - 27.2
Storage Areas	5
Sumps	3.8 - 5
All other Areas	2.35

Table 23-5: Maximum recommended air velocities by area

Area	Air Velocity m/s
Working faces	4
Conveyor drifts	5
Main haulage routes	6
Smooth lined main airways	8
Ventilation shafts	20

Airflow velocities in wet upcast/exhaust raises should not lie between 7 to 12 m/s. Water blanketing may occur in this velocity range, and may results in large intermittent cascades of water falling to the shaft bottom due to the oscillating loads on main fans caused by variations in the shaft resistance. In the case of the panel exhaust raises, an air velocity of at least 13 m/s is to be targeted to help avoid water blanketing (McPherson, 1993).

Radiation & Radon Control

Understanding and controlling the radiation levels within the mine is key to providing a safe working environment in an underground uranium mine. The following key aspects relating to radiation in the mining environment are considered during the ventilation design process for the Madaouela project:

- Residency time of air within the mine;
- Primary ventilation system in production areas with no recirculation of potentially contaminated air;
- Ingress of radon bearing water into the mine and ventilation of sump areas;
- Removal of potentially contaminated air at point of source with either primary exhaust ventilation in production areas or forced secondary ventilation during development activities; and
- Volumetric flow rate into the mine, benchmarked against similar uranium projects.

An important method of controlling radon ingress into the mine environment will be the local application of shotcrete to walls and backs of the main access and panel development drifts. A shotcrete barrier should be applied in areas which are identified to be sources of elevated levels of radon gas emission. This will contain the radon gas behind the barrier and reduce its emission into the mine atmosphere.

Residency Time

The rate of radon progeny emission increases with time, and therefore the residency time of contaminated air in the designed ventilation network should be understood. It is essential to maintain low radon concentrations in the mine, which is achieved by dilution with fresh air and providing for low residency times of 10 to 15 minutes (C. Gherghel, 2008). In this study, the residency time is assumed to be the time taken from when the air enters to the mine, to when it is exhausted to surface, as during this time the air comes into contact with and accumulates radon progeny.

An analysis is carried out whereby a simulated source of radon-bearing gas is introduced to the mine atmosphere in the fresh air intake raises. Dynamic simulation is undertaken applying typical ventilation conditions to determine the approximate residency time of air entering every fresh air intake. The vast majority residency times for the two proposed underground operations do not exceed 15 minutes, varying according to the location of the nearest exhaust raise relative to the gas source. Residency times of the air in the proposed mining network are therefore considered to be sufficiently low.

Primary Ventilation System and Recirculation

Designated fresh air raises are located in barren waste rock or low-grade areas to reduce potential radon contamination of the incoming air. Exhaust raises located within the panels provide primary ventilation, allowing the contaminated air to be exhausted immediately from the panels. Multiple exhaust raises are designed for each panel to minimise the time to develop a panel before production can commence, which can double as intake raises as production progresses away from the access.

Fresh air enters a panel from the main fresh air circuit and is exhausted within the panel, minimising the potential for re-circulation and air/radiation leakage.

Water Control

The high solubility of radon in water makes groundwater ingress an important potential source of contamination. The most effective way of lowering the risk of contamination by ground water is to prevent contaminated water from entering the mine. This can be achieved by probe and grout covers ahead of development, and shotcreting the walls and backs of the drives. In the case of Madaouela, shotcrete will be available in selected areas, however as the infrastructure development is envisaged to be constructed above the mineralised horizon,

radiation issues due to water ingress within the infrastructure areas are perceived to be minimal.

Water which does enter the mine should be collected in sumps located within the panels and main development. Airflow of between 3.8 and 5 m^3/s should be allowed for in these sump areas to remove any contaminants generated.

Ventilation Network Design

Table 23-6 and Table 23-7 provide the development profiles, friction factors and maximum air velocities applied to the ventilation analysis of the mine design.

Table 23-6: Summary of horizontal development profiles

Development Type	Type	Height m	Width m	Area m^2	Friction F x 10^4 kg/m^3	Max Velocity m/s
Decline & Main Development	Arch	4.5	5	22.5	120	6
Conveyor Drive	Arch	3.2	4	12.8	120	5
Remuck	Arch	4.5	5	22.5	120	8
Panel Development	Arch	3.5	7	24.5	120	6

Table 23-7: Summary of vertical circular development profiles

Development Type	Type	Diameter m	Area m^2	Friction F x 10^4 kg/m^3	Max Velocity m/s
Intake Raise	Circular Raisebore	5	19.6	50	20
Exhaust Raise	Circular Raisebore	3.5	9.6	50	20

Fan Selection and Power Requirements

Fan selection is based on airflow capabilities at practical pressure ratings and the previously discussed airflow requirements. Table 23-8 shows model type, possible airflow capabilities and power ratings for the selected fans.

Table 23-8: Fan type and power requirements

Fan Type	Capable Airflows m^3/s	Typical Operating Electrical Power kW
Clemcorp L52C 100% solidity 50 Deg Blade Pitch	58 - 126	90
2 x 55 kW Staged Development Fan	13 - 40	80

The estimation of fan requirements assumes a single exhaust fan is required for each active production panel and two panels being developed. In addition, an allowance has been made for surplus fans for installation on a second raise within a panel to maintain constant exhaust for a production area during the transition from one exhaust raise to the next.

Auxiliary ventilation will be required during the development of main access and panel drives.

This will be provided by development fans and flexible ducting where needed. It is assumed that two main access headings, one conveyor drive and five panel drives could be developed simultaneously. With several permanent forced ventilation areas, and additional fans for mechanical contingency, a total of 16 auxiliary fans could be operating simultaneously. In this case, each auxiliary fan has been modelled by 2 x 55 kW Staged Development Fans.

Table 23-9 shows the total power requirements for the selected fans during operation.

Table 23-9: Total operating fan power requirements

Fan Type / **Typical Operating Scenario**	**Quantity No.**	**Operating Electrical Power kW**
Clemcorp L52C 100% solidity 50 Deg Blade Pitch	10	900
2 x 55 kW Staged Development Fan	16	1280
Total	**26**	**2180**

The maximum power requirement for all exhausting and auxiliary fans working simultaneously is approximately 2,180 kW. However it is likely that the average typical power requirements will be less than this value, as all fans will not be operating at every stage of the mine.

The ventilation power requirements during each phase of the mine will vary depending on the diesel machinery capacity, number of active panels and development headings.

Panel Scheduling

The fresh air raises are located to provide fresh air to multiple panels simultaneously. However, areas around fresh air raises are prone to experiencing high air velocities, and may exceed the maximum practical air velocity of 6 m/s if not properly managed. This is especially important where panels and access drives are connected to the main fresh air system via a single main access development drive, as large quantities of air may be forced through a relatively small cross-sectional area. In order to prevent exceeding a maximum air velocity of 6 m/s in main haulage routes, each intake raise can only provide fresh air to:

- Three active ore panels simultaneously if connected using three drifts (two access, one conveyor); and
- Only one panel can be ventilated if a single access or panel development drift connects to the main fresh air circuit or raise.

Robust scheduling of the panel development and ore extraction is required to maintain control of the air velocities underground.

Monitoring

To assist with the management of the ventilation underground, an effective monitoring system is key. Monitoring can be continuous with fixed installations providing “live” feedback of ventilation parameters or static requiring regular ventilation surveys and manual monitoring.

Ventilation monitoring should have the capacity to cover the following areas:

- Radiation;
- Airflow velocity;
- Noxious gases (CO, CO_2, etc.);
- Heat (wet and dry bulb);

- Humidity; and
- Air pressure.

Full time ventilation "officers" have been incorporated into the labour schedule to undertake regular surveys throughout the mine and regulate airflows.

Ventilation Controls

The principle means of controlling ventilation in the Madaouela underground deposits is to maximise the use of primary ventilation circuits and minimise the residency time of air underground. To manage these flows, a standard array of ventilation controls have been assumed (i.e. brattices, parachutes, ventilation walls, regulators, etc.).

Local conditions suggest that heat will require special consideration. Due to the shallow nature of the mine and the presence of groundwater, it is anticipated that working conditions underground should be more comfortable than on the surface. However, the possibility for extreme heat on the surface suggests that policies should be implemented to regulate working hours to minimise the risk of heat stroke, in line with Nigerien legislation.

To monitor worker exposure to radiation over time, thermoluminescent dosimetry ("TLD") badges should be provided for all workers. These can be connected to cap lamps or self-rescuers to provide a continuous measurement of radiation exposure for each workers. Should the annual exposure limits be breached, individual workers can then be removed from that environment for the remainder of the monitoring year. A similar system is in place at COMINAK's Akouta Mine. All attempts should be made to contain radiation exposure to levels where removal from the environment is not required.

In addition, a shotcreter is allowed for in the equipment schedule providing an effective means of containing radon emission at the source. The shotcrete acts as an airtight seal that can be sprayed on areas emitting high levels of radiation, creating a physical barrier and containing preventing the emission of radon gas.

Alpha radiation is simple to control as it has a low penetration distance. However, significant exposure levels can be experienced in the event that dust is ingested into the body. For this reason, smoking and eating underground should be discouraged unless in controlled areas, which are regularly cleaned and have facilities for washing hands. Dust control and respirators (in particular airstream helmets) will also play an important role in reducing the workers' exposure to radiation.

23.1.5 Ventilation Design

Method

Ventilation design, raise diameter optimisation and fan selection is undertaken using Ventsim software. The ventilation design criteria discussed previously and general initial assumptions on mining sequence are applied to the Ventsim model.

During the simulation of the staged airflow requirement, the following assumptions are made:

- Each mining panel requires approximately 125 m^3/s of fresh air during production, inclusive of 25% contingency, provided by a single exhaust fan in each panel;
- Airflow quantity is driven by the maximum instantaneous airflow requirement projected for the panel to prevent need for variable fan speeds;
- If an exhaust fan is installed in an active raise then it will be operating;
- The maximum length of development which can be ventilated by secondary ventilation is

1,200 m and is used as a basis for distance between fresh air intake raises; and

- Activities with the highest initial priority when designing the ventilation design stages are the installation of the first fresh air intake raise and the construction of the underground workshop; magazine; exhaust raise and fuel storage area.

Marilyn-Marianne

A visual representation of the proposed development network for the Marilyn-Marianne Mine is provided inFigure 23-3. Initial access to the mine is to be provided by a decline from the surface, which is to act as the initial source of fresh air into the mine.

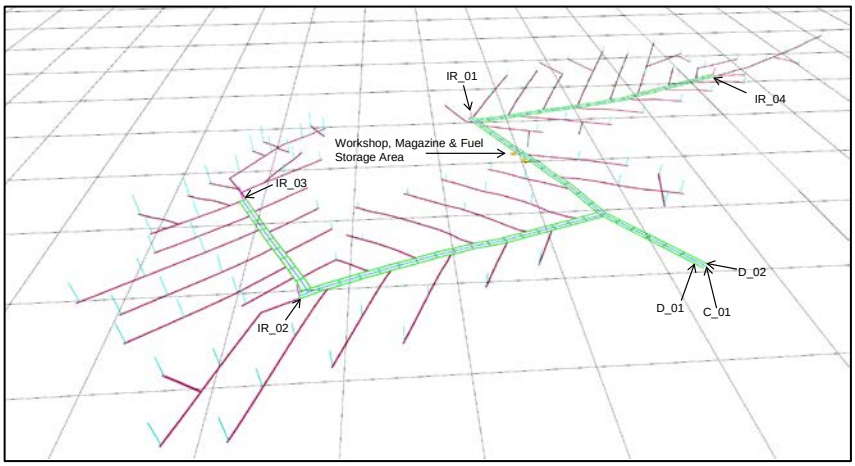


Figure 23-3: Isometric view of Marilyn-Marianne development & ventilation

The ventilation of the mine is split into a number of phases to represent the development of the mine over time:

- Phase 1 – The primary objectives in the first phase will be to develop the initial fresh air intake raise (IR_01), workshop area and the first five ore panels. Approximately 11,300 m of development length will be made in Phase 1. The two access declines, conveyor drive and panel access will initially require forced ventilation from the portals. Once several exhaust raises and surface fans are installed, the ventilation system can then draw fresh air from the portals and IR_01. The development fans can then be relocated and provide forced fresh air for Phase 2 development.
- Phase 2 – This phase models the ventilation requirements during production from the first four ore panels, as well as panel development towards the West.
- Phase 3 – The second intake raise (IR_02) will be developed during this phase, along with panel development in the central region of the mine. Five ore panels are available for simultaneous production, with several others undergoing development. Forced ventilation will be used to provide fresh air to the working face as access drifts are developed to the next fresh air intake location. Raises developed in the panels can be used as intake raises if required. This methodology will be used during each subsequent phase.
- Phase 4 – This phase consists of an extension of approximately 890 m in the access and conveyor development to the North West, and development of multiple production panels in this area. A third designated intake raise (IR_03) will be installed, providing fresh air to the new and existing panels.
- Phase 5 – The final phase of the mine consists of a 2,000 m extension of the access and conveyor drifts in the North East direction. It is envisaged that panel raises will be

developed in every second panel (500 m intervals) to act as intake raises to supply fresh air to the ventilation circuit. This fresh air will be utilised by the forced ventilation system to supply fresh air to the working face. A fifth intake raise (IR_05) will be developed at the end of the main access extension to provide fresh air to the production panels in the North East extension. The total length of all underground development including raises is approximately 60,000 m.

Phase 1

Up to 300 m^3/s of fresh air will be supplied by raise IR_01 at a velocity of 15.3 m/s. Approximately 135 m^3/s of fresh air intake is provided by each of the two access declines, and 64 m^3/s airflow is provided the access conveyor declines.

Figure 23-4 shows a plan view of the Phase 1 ventilation network including intake and exhaust raises. Figure 23-21 gives a summary of the intake and exhaust flow rates for Phase 1.

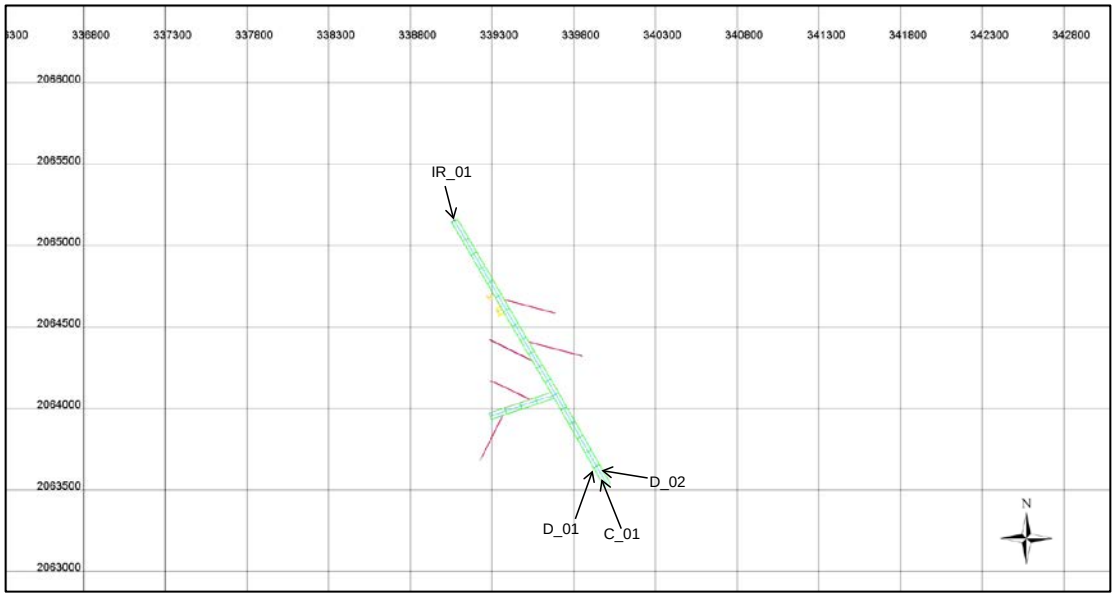


Figure 23-4: Plan view of Marilyn-Marianne phase 1 development & ventilation

Table 23-10: Summary of Marilyn-Marianne phase 1 primary ventilation

Phase 1					
Intake Code	**Description**	**Area m^2**	**Volume m^3/s**	**Velocity m/s**	
D_01	Access Decline	22.5	135	6	
D_02	Access Decline	22.5	131	5.8	
C_01	Conveyor Decline	12.8	64	5	
IR_01	Intake Raise	19.6	300	15.3	
Total			**629**		
Exhaust					
		Area	**Volume**	**Velocity**	**Quantity**
Code	**Description**	**m^2**	**m^3/s**	**m/s**	
ER	Exhaust Raise	9.6	127	13.2	5
Total			**635**		

Analysis of the Phase 1 ventilation network suggests that in order to maintain air velocities in the main development below the threshold of 6 m/s, only four active development panels can be ventilated simultaneously using primary ventilation. Additional panel development drifts can be ventilated using forced ventilation.

It is envisaged that during the early stages of the mine, panel development will be taking place without the extraction of ore, thus reducing the total diesel machine capacity at any one time. Therefore, forced ventilation can be used during panel development in combination with the exhaust raises, whilst still maintaining sufficient airflow into the mine.

Phase 2

Approximately 270 m^3/s of fresh air intake will be shared equally between the two access declines, with an additional 65 m^3/s being provided by the conveyor decline. Intake raise IR_01 will provide around 308 m^3/s of fresh air to the mine. Exhausts are to be provided by the production panels

In order to maintain air velocities below threshold of 6 m/s in main development drives, only four active extraction panels are allowed during Phase 2.

Figure 23-5 shows a plan view of the phase 2 ventilation network development. Table 23-11 gives a summary of the intake and exhaust flow rates for Phase 2.

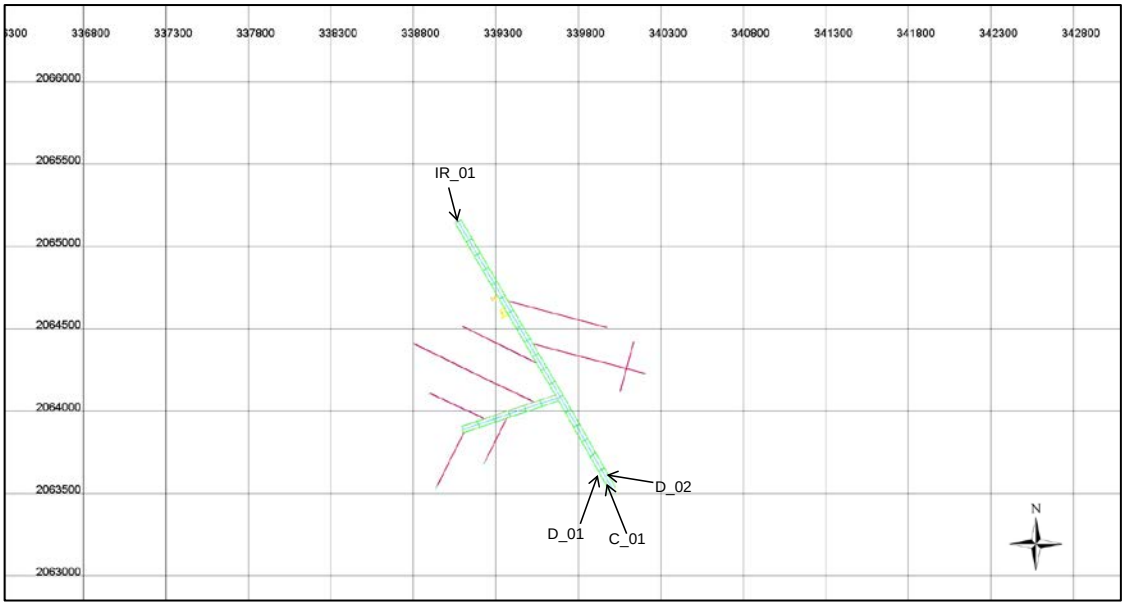


Figure 23-5: Plan View of Marilyn-Marianne phase 2 development & ventilation

Table 23-11: Summary of Marilyn-Marianne phase 2 primary ventilation

Phase 2					
Intake Code	**Description**	**Area m^2**	**Volume m^3/s**	**Velocity m/s**	
D_01	Access Decline	22.5	135	6	
D_02	Access Decline	22.5	133	5.9	
C_01	Conveyor Decline	12.8	65	5.1	
IR_01	Intake Raise	19.6	308	15.7	
Total			**641**		
Exhaust					
		Area	**Volume**	**Velocity**	**Quantity**
Code	**Description**	**m^2**	**m^3/s**	**m/s**	
ER	Exhaust Raise	9.6	125	13	5
Total			**625**		

Phase 3

Fresh air enters the mine through fresh air raises IR_01 and IR_02, at volumetric flow rates of 235 and 265 m^3/s, respectively. The access declines provide approximately 200 m^3/s between them, and 60 m^3/s from the conveyor decline.

The second intake raise increases the flexibility of the ventilation of mining areas, and allows the multiple panels to be ventilated effectively without exceeding the allowable air velocities in the access drives. During Phase 3, five active panels can be operated, with a number of access headings being developed simultaneously.

Figure 23-6 shows a plan view of the Phase 3 ventilation network development. Table 23-12 gives a summary of the intake and exhaust flow rates for Phase 3.

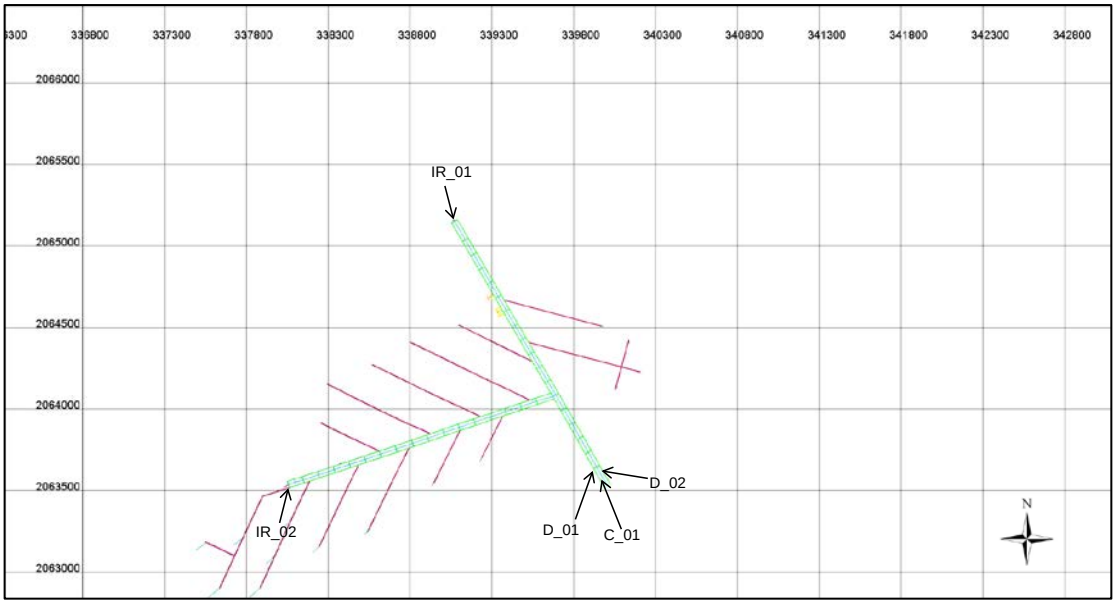


Figure 23-6: Plan view of Marilyn-Marianne phase 3 development & ventilation

Table 23-12: Summary of Marilyn-Marianne phase 3 primary ventilation

Phase 3					
Intake Code	**Description**	**Area m^2**	**Volume m^3/s**	**Velocity m/s**	
D_01	Access Decline	22.5	110	4.9	
D_02	Access Decline	22.5	92	4.1	
C_01	Conveyor Decline	12.8	60	4.7	
IR_01	Intake Raise	19.6	235	12	
IR_02	Intake Raise	19.6	263	13.4	
Total			**761**		
Exhaust					
		Area	**Volume**	**Velocity**	**Quantity**
Code	**Description**	**m^2**	**m^3/s**	**m/s**	
ER	Exhaust Raise	9.6	126	13.1	6
Total			**756**		

Phase 4

During Phase 4, a third fresh air intake is developed to the northwest of the mine. For this phase, 110 m^3/s is provided by IR_01, approximately 300 m^3/s by IR_02, and 350 m^3/s via IR_03.

The intake requirements from each main decline are reduced to around 50 m^3/s, and the conveyor decline to 26 m^3/s. shows a plan view of the Phase 1 ventilation network including intake and exhaust raises. Figure 23-7 shows a plan view of the underground development including intake and exhaust raises, and Table 23-13 gives a summary of the intake and exhaust flow rates for Phase 4.

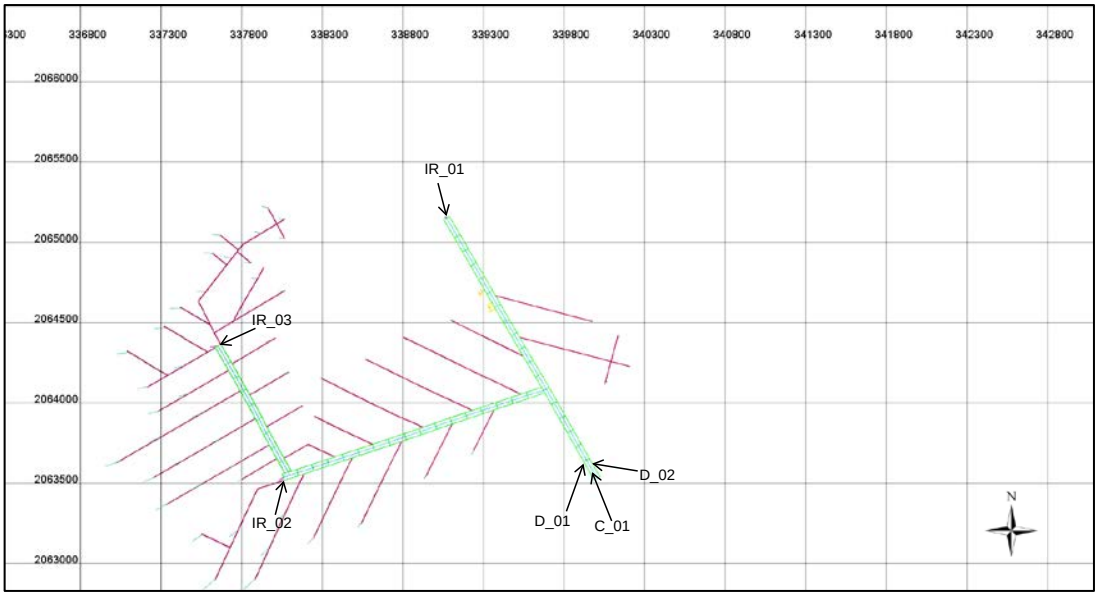


Figure 23-7: Plan view of Marilyn-Marianne phase 4 development & ventilation

Table 23-13: Summary of Marilyn-Marianne phase 4 primary ventilation

Phase 4					
Intake Code	**Description**	**Area m^2**	**Volume m^3/s**	**Velocity m/s**	
D_01	Access Decline	22.5	54	2.4	
D_02	Access Decline	22.5	50	2.2	
C_01	Conveyor Decline	12.8	26	2	
IR_01	Intake Raise	19.6	110	5.6	
IR_02	Intake Raise	19.6	292	14.9	
IR_03	Intake Raise	19.6	353	18	
Total			**884**		
Exhaust					
		Area	**Volume**	**Velocity**	**Quantity**
Code	**Description**	**m^2**	**m^3/s**	**m/s**	
ER	Exhaust Raise	9.6	126	13.1	7
Total			**882**		

Phase 5

During Phase 5, the maximum assumed five panels can be active along with several additional panel development headings.

IR_04 is developed to the East of the mine, and provides 221 m^3/s of fresh air. IR_01, IR_02 and IR_03 provide volumetric rates of 241, 108 and 171 m^3/s, respectively. The declines provide very similar fresh air flow rates as in Phase 4.

Figure 23-8 shows a plan view of the Phase 5 ventilation network development. Table 23-14 gives a summary of the intake and exhaust flow rates for Phase 5.

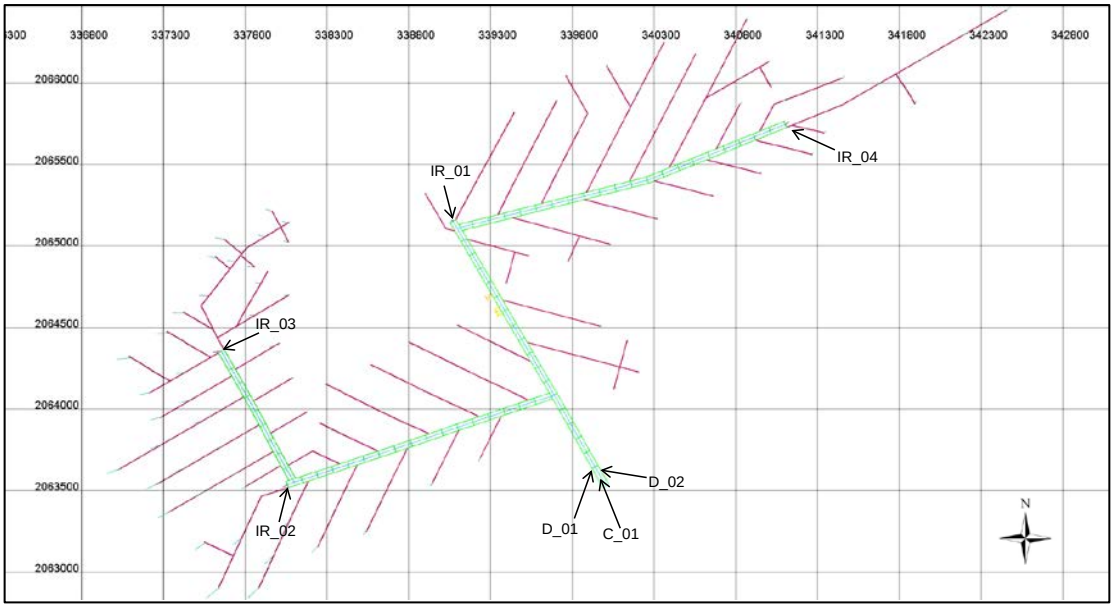


Figure 23-8: Plan view of Marilyn-Marianne phase 5 development & ventilation

Table 23-14: Summary of phase Marilyn-Marianne 5 primary ventilation

Phase 5					
Intake Code	**Description**	**Area m^2**	**Volume m^3/s**	**Velocity m/s**	
D_01	Access Decline	22.5	54	2.4	
D_02	Access Decline	22.5	50	2.2	
C_01	Conveyor Decline	12.8	26	2	
IR_01	Intake Raise	19.6	241	12.3	
IR_02	Intake Raise	19.6	108	5.5	
IR_03	Intake Raise	19.6	171	8.7	
IR_04	Intake Raise	19.6	221	11.3	
Total			**870**		
Exhaust					
		Area	**Volume**	**Velocity**	**Quantity**
Code	**Description**	**m^2**	**m^3/s**	**m/s**	
ER	Exhaust Raise	9.6	126	13.1	7
Total			**882**		

MSNE-Maryvonne

The ventilation design for MSNE-Maryvonne uses the same principles, machinery operating assumptions and flow airflow limitations as the Marilyn-Marianne ventilation design.

Figure 23-9 shows the isometric view of the MSNE-Maryvonne development and ventilation design and portal location.

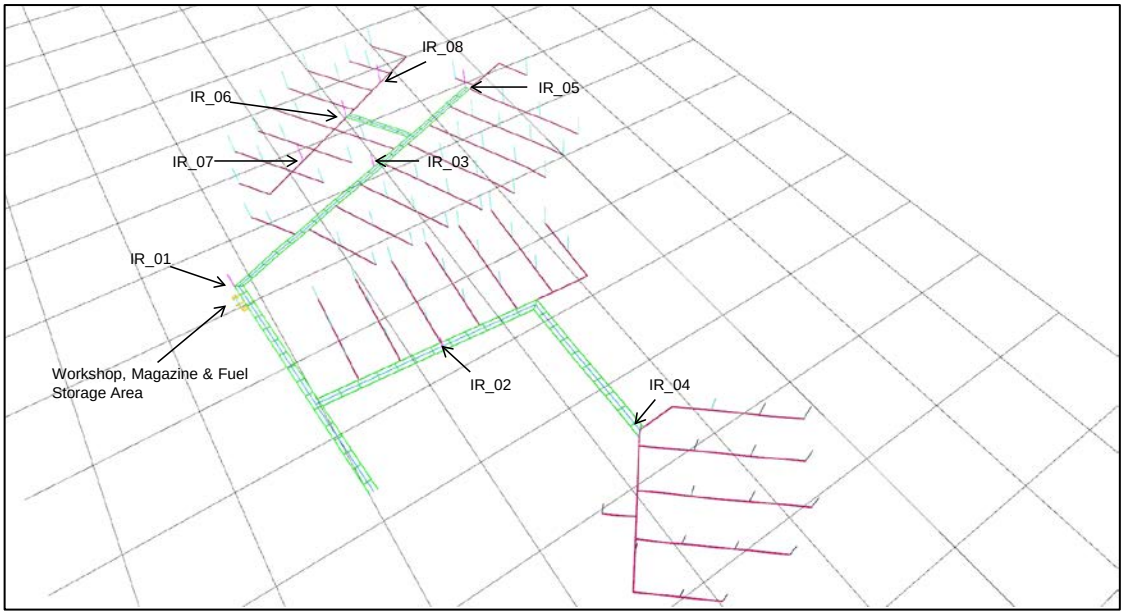


Figure 23-9: Isometric view of MSNE-Maryvonne development & ventilation

The ventilation of the mine is split into a number of phases to represent the development of the mine over time:

- Phase 1 – The initial phase consists of 10,800 m of underground development. The priorities during Phase 1 will be to install the first fresh air intake raise (IR_01) and workshop, and designated exhaust raise in the magazine area. During the initial development, fresh air will be drawn through the surface portals and provided to the working faces using a forced air system.
- Phase 2 – During this phase the first five panel access drives and the second fresh air intake raise (IR_02) are developed. It is envisaged that surface fans will be installed in the panel exhaust raises and used to draw fresh into the mine via the declines and intake raises.
- Phase 3 – The main access drives are extended towards the West and multiple production panels are developed. Forced air ventilation will be utilised during the development stages, drawing fresh air from the fresh air circuit fed by the declines, IR_01 IR_02. The third designated intake raise, IR_03, will be developed. The ventilation system modelled is capable of supporting five operating production panels, with several additional panels being developed simultaneously.
- Phase 4 – The underground access and conveyor drifts are extended approximately 920 m towards the North. Forced ventilation will provide fresh air during development, drawing fresh air from IR_02. The fourth intake raise IR_04 will be installed at the end of the development, which will then supply fresh air to the northern production panels. Five operational production panels will be ventilated during this phase, with several additional panels being developed.
- Phase 5 – Access and conveyor drifts are extended 960 m towards the South-West. Forced ventilation will draw fresh air from IR_03 and supply it to the working face. The fifth designated intake raise, IR_05, will be installed at the end of the development and will provide fresh air to the production panels in this area. Five operating and multiple developing production panels are modelled in this phase. At this point the mine is provided by five well distributed intake raises, which allows flexibility when managing and scheduling the production panels.

- Phase 6 – During the final phase the access and conveyor drifts are extended 480 m towards the South. This will require forced ventilation, drawing fresh air from the central main fresh air circuit. The sixth intake raise IR_06 will be installed at the end of this extension. A single access drift, of 1,500 m approximate length, will then developed to give access the production panels in the South. Two additional intake raises, IR_07 and IR_08, will be installed along the panel access drive. This is necessary to provide sufficient fresh air to the production panels whilst maintaining practical air velocities in the access drifts.

Phase 1

During the initial decline development, fresh air will be provided to the working face by a forced ventilation system. Each development heading will have an auxiliary fan installed; in this case 2 x 55 kW staged development fans. This can supply sufficient fresh air to the underground workings to develop access to the location of the first fresh air intake raise, IR_01.

The installation of the initial fresh air raise has a high priority, as this will establish the primary ventilation network. Figure 23-10 shows the development and ventilation arrangement for the initial stages of the mine, and assumes IR_01 has been developed. The exhausting raise is located in the magazine and fuel storage area.

Table 23-15 gives a summary of the intake and exhaust flow rates for Phase 1.

For Phase 1, the main intake raise IR_01 will provide approximately 80 m^3/s, with each access decline providing 20 m^3/s and an additional 10 m^3/s from the conveyor decline.

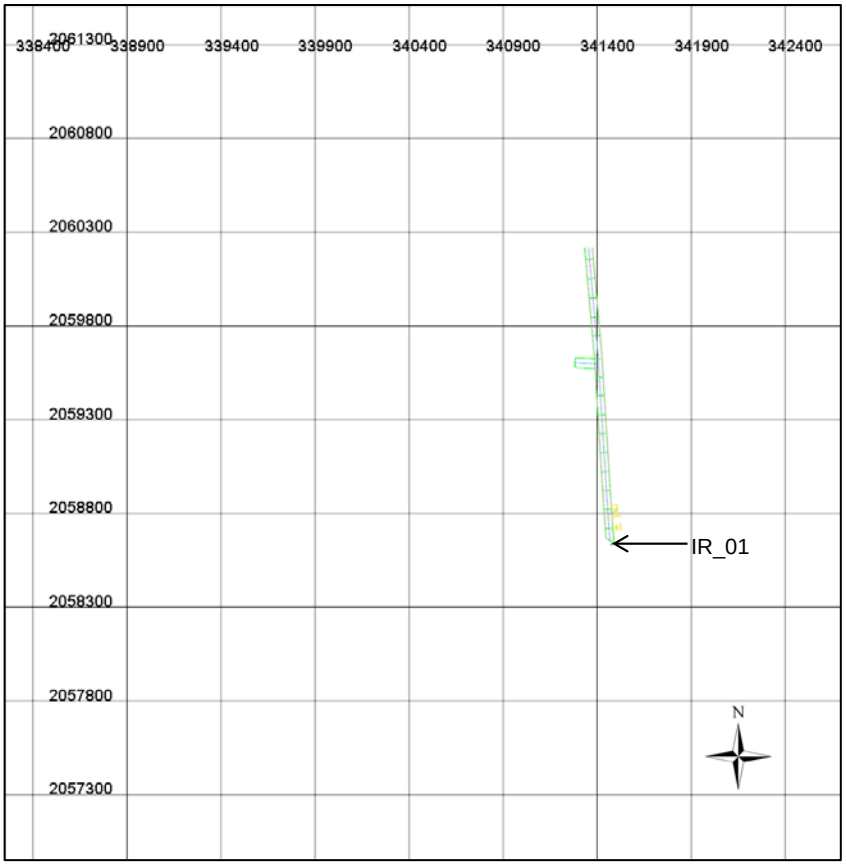


Figure 23-10: Plan view of MSNE-Maryvonne phase 1 development & ventilation

Table 23-15: Summary of MSNE-Maryvonne phase 1 primary ventilation

Phase 1					
Intake Code	**Description**	**Area m^2**	**Volume m^3/s**	**Velocity m/s**	
D_01	Access Decline	22.5	20	0.9	
D_02	Access Decline	22.5	20	0.9	
C_01	Conveyor Decline	12.8	10	0.8	
IR_01	Intake Raise	19.6	80	4.1	
Total			**131**		
Exhaust					
		Area	**Volume**	**Velocity**	**Quantity**
Code	**Description**	**m^2**	**m^3/s**	**m/s**	
ER	Exhaust Raise	9.6	127	13.2	1
Total			**127**		

Phase 2

Phase 2 shows the first panel development activities for MSNE-Maryvonne and utilises IR_01 at 302 m^3/s, and a second fresh air intake raise, IR_02 at 249 m^3/s fresh air intake.

The main declines provide 88 m^3/s each and the conveyor decline provides 44 m^3/s to fresh air intake to the mine.

Figure 23-11 shows the plan view of the Phase 2 development and ventilation network and Table 23-16 gives a summary of the intake and exhaust flow rates for Phase 2.

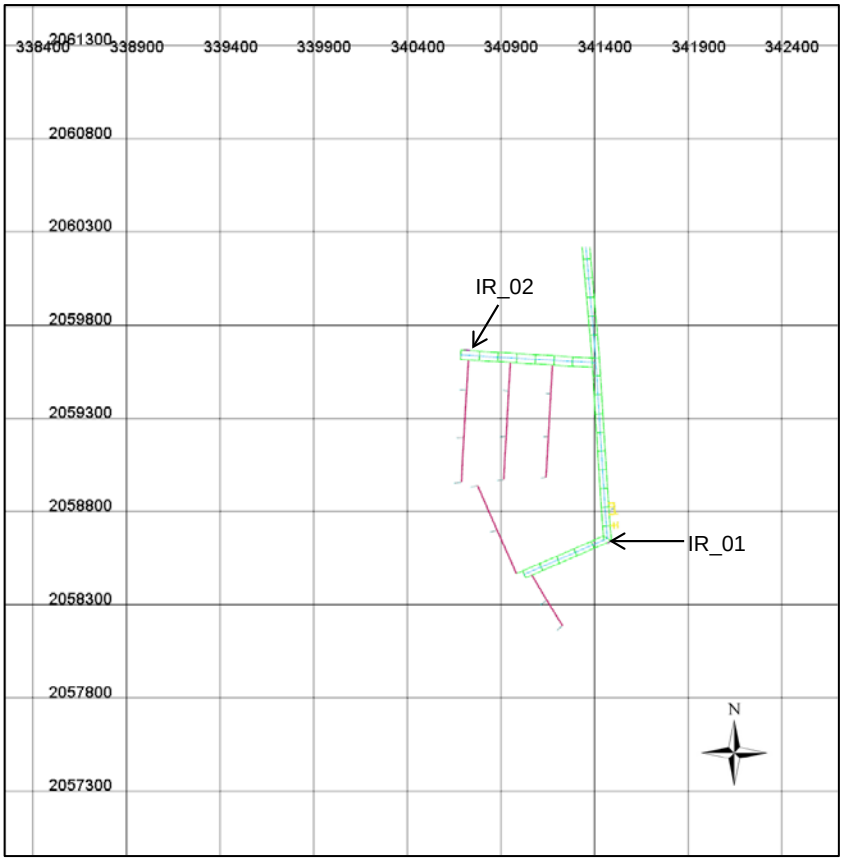


Figure 23-11: Plan view of MSNE-Maryvonne phase 2 development & ventilation

Table 23-16: Summary of MSNE-Maryvonne phase 2 primary ventilation

Phase 2					
Intake Code	**Description**	**Area m^2**	**Volume m^3/s**	**Velocity m/s**	
D_01	Access Decline	22.5	88	3.9	
D_02	Access Decline	22.5	88	3.9	
C_01	Conveyor Decline	12.8	44	3.4	
IR_01	Intake Raise	19.6	302	15.4	
IR_02	Intake Raise	19.6	249	12.7	
Total			**770**		
Exhaust					
		Area	**Volume**	**Velocity**	**Quantity**
Code	**Description**	**m^2**	**m^3/s**	**m/s**	
ER	Exhaust Raise	9.6	127	13.2	6
Total			**762**		

Phase 3

For Phase 3, 77 m^3/s of fresh air can be provided by each main access decline, and 37 m^3/s from the conveyor decline. 261, 267 and 300 m^3/s of fresh are provided by IR_01, IR_02 and IR_03, respectively.

A total of eight exhaust raises have been modelled, which allows for the workshop exhaust,

five active panels, and two additional developed panels to have working exhaust raises. This gives rise to a relatively high intake of fresh air; above that based of the diesel felt capacity alone. These additional panels and increased ventilation capability gives flexibility to the mining areas and increases the ventilation network's ability to deal with potential radon contamination.

Figure 23-12 shows a plan view of the Phase 3 ventilation network development and Table 23-17 gives a summary of the intake and exhaust flow rates for Phase 3.

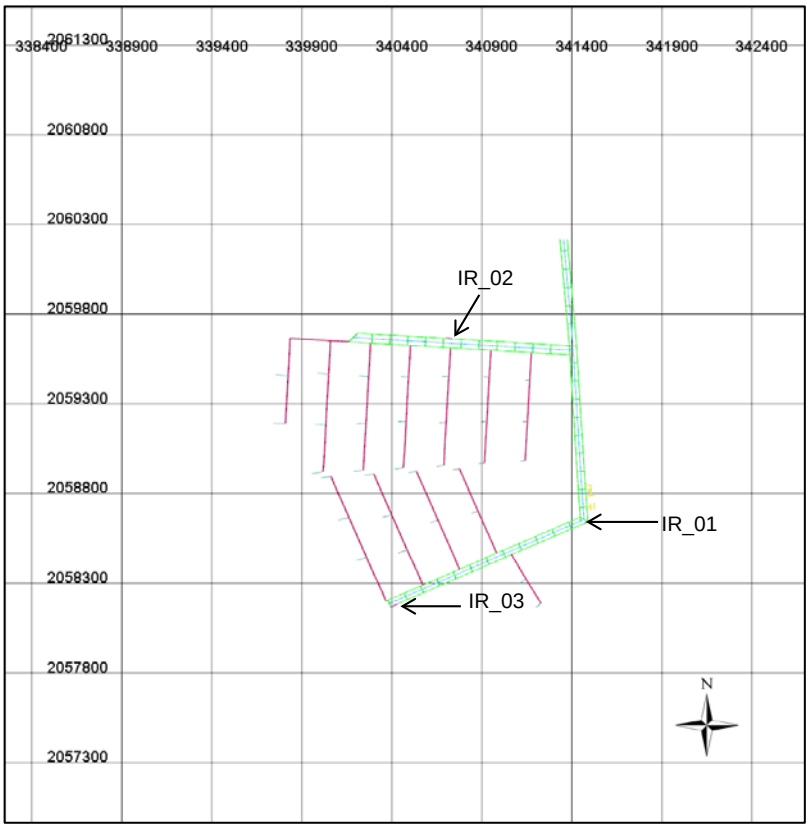


Figure 23-12: Plan view of MSNE-Maryvonne phase 3 development & ventilation

Table 23-17: Summary of MSNE-Maryvonne phase 3 primary ventilation

Phase 3					
Intake Code	**Description**	**Area m^2**	**Volume m^3/s**	**Velocity m/s**	
D_01	Access Decline	22.5	77	3.4	
D_02	Access Decline	22.5	77	3.4	
C_01	Conveyor Decline	12.8	37	2.9	
IR_01	Intake Raise	19.6	261	13.3	
IR_02	Intake Raise	19.6	267	13.6	
IR_03	Intake Raise	19.6	300	15.3	
Total			**1017**		
Exhaust					
Code	**Description**	**Area m^2**	**Volume m^3/s**	**Velocity m/s**	**Quantity**
ER	Exhaust Raise	9.6	127	13.2	8
Total			**1,016**		

Phase 4

For Phase 4, the majority of the fresh air is provided via four fresh air intake raises. IR_01 supplies 214 m^3/s of fresh air to the main fresh air circuit. IR_02 and IR_03 provide 188 m^3/s and 218 m^3/s respectively. The northern panels are supplied mainly by IR_04 which provides 255 m^3/s of fresh air to the mine.

The main access declines both supply 59 m^3/s of fresh air intake, and the conveyor decline supplies 28 m^3/s airflow.

Figure 23-13 shows a plan view of the Phase 4 ventilation network development including intake and exhaust raises. Table 23-18 gives a summary of the intake and exhaust flow rates for Phase 4.

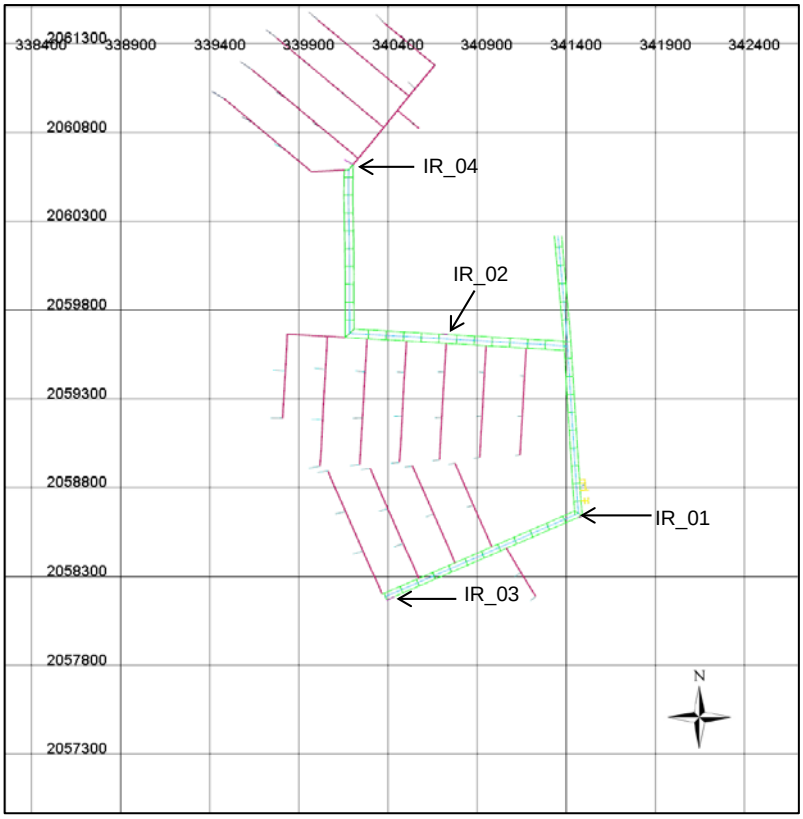


Figure 23-13: Plan view of MSNE-Maryvonne phase 4 development & ventilation

Table 23-18: Summary of MSNE-Maryvonne phase 4 primary ventilation

Phase 4					
Intake Code	**Description**	**Area m^2**	**Volume m^3/s**	**Velocity m/s**	
D_01	Access Decline	22.5	59	2.6	
D_02	Access Decline	22.5	59	2.6	
C_01	Conveyor Decline	12.8	28	2.2	
IR_01	Intake Raise	19.6	214	10.9	
IR_02	Intake Raise	19.6	188	9.6	
IR_03	Intake Raise	19.6	218	11.1	
IR_04	Intake Raise	19.6	255	13	
Total			**1019**		
Exhaust					
Code	**Description**	**Area m^2**	**Volume m^3/s**	**Velocity m/s**	**Quantity**
ER	Exhaust Raise	9.6	127	13.2	8
Total			**1,016**		

Phase 5

During Phase 5, IR_01 and IR_02 provide 149 and 174 m^3/s of fresh air, respectively; IR_03 and IR_05 supply 1,159 and 163 m^3/s of airflow primarily to the southern mining panels, and IR_04 provides 249 m^3/s of fresh air intake to the northern panels.

The main access declines provide 88 m^3/s of fresh air between them, and the conveyor drive supplies an airflow of 27 m^3/s.

Figure 23-14 shows a plan view of the Phase 5 ventilation network development including intake and exhaust raises. Table 23-19 gives a summary of the intake and exhaust flow rates for Phase 5.

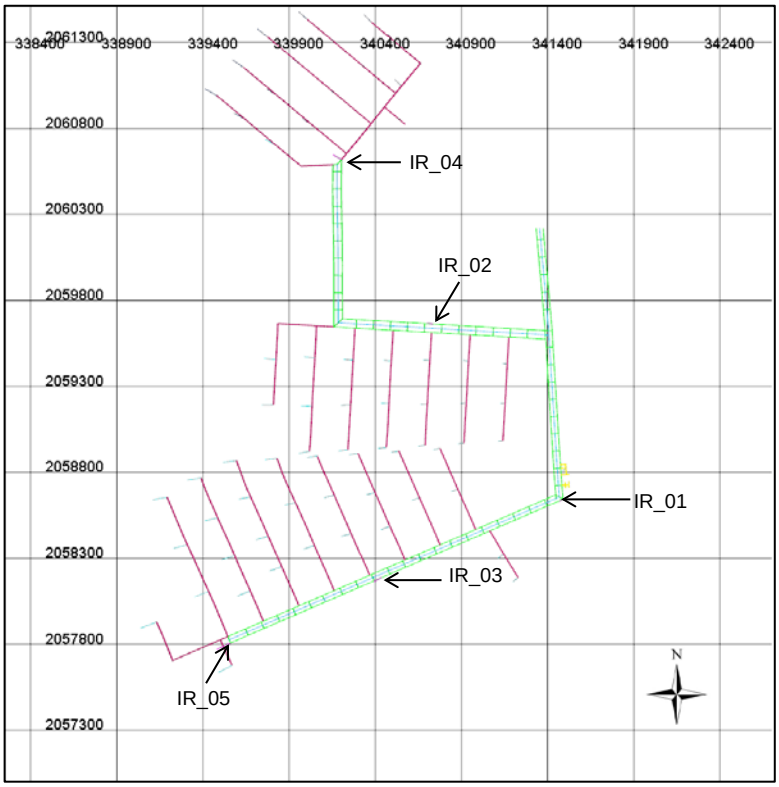


Figure 23-14: Plan view of MSNE-Maryvonne phase 5 development & ventilation

Table 23-19: Summary of MSNE-Maryvonne phase 5 primary ventilation

Phase 5					
Intake Code	**Description**	**Area m^2**	**Volume m^3/s**	**Velocity m/s**	
D_01	Access Decline	22.5	47	2.1	
D_02	Access Decline	22.5	41	1.8	
C_01	Conveyor Decline	12.8	27	2.1	
IR_01	Intake Raise	19.6	149	7.6	
IR_02	Intake Raise	19.6	174	8.9	
IR_03	Intake Raise	19.6	159	8.1	
IR_04	Intake Raise	19.6	249	12.7	
IR_05	Intake Raise	19.6	163	8.3	
Total			**1,008**		
Exhaust					
Code	**Description**	**Area m^2**	**Volume m^3/s**	**Velocity m/s**	**Quantity**
ER	Exhaust Raise	9.6	127	13.2	8
Total			**1,016**		

Phase 6

For Phase 6, a total of eight intake raises provide MSNE-Maryvonne with fresh air as described in Table 23-20.

Figure 23-15 shows a plan view of the Phase 6 ventilation network including intake and exhaust raises. Table 23-20 gives a summary of the intake and exhaust flow rates for Phase 6.

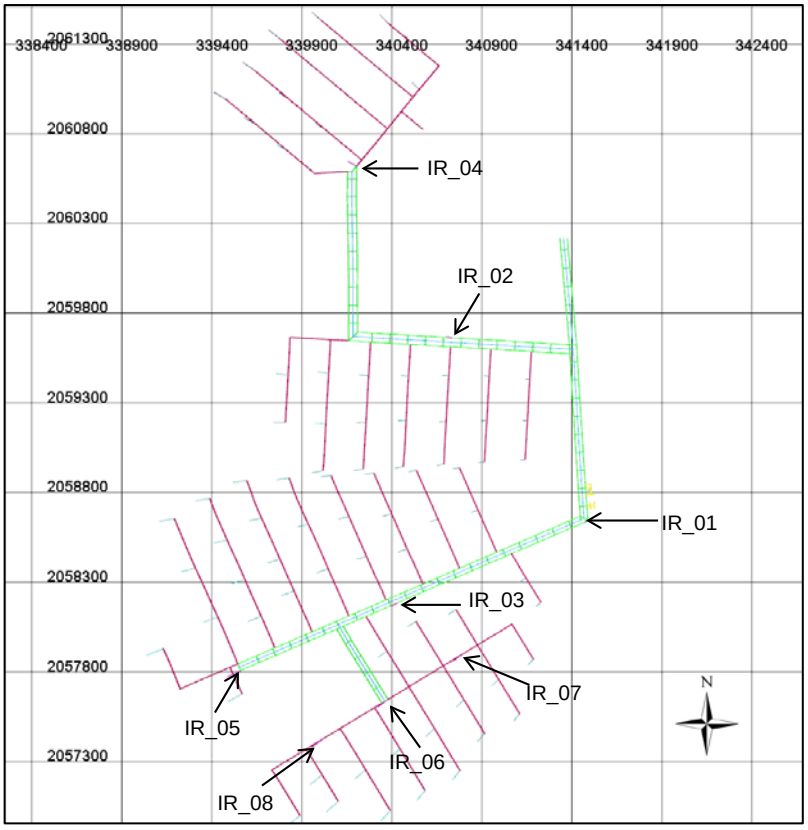


Figure 23-15: Plan view of MSNE-Maryvonne phase 6 development & ventilation

Table 23-20: Summary of MSNE-Maryvonne phase 6 primary ventilation

Phase 6					
Intake Code	**Description**	**Area m^2**	**Volume m^3/s**	**Velocity m/s**	
D_01	Access Decline	22.5	27	1.2	
D_02	Access Decline	22.5	27	1.2	
C_01	Conveyor Decline	12.8	14	1.1	
IR_01	Intake Raise	19.6	102	5.2	
IR_02	Intake Raise	19.6	84	4.3	
IR_03	Intake Raise	19.6	106	5.4	
IR_04	Intake Raise	19.6	153	7.8	
IR_05	Intake Raise	19.6	112	5.7	
IR_06	Intake Raise	19.6	153	7.8	
IR_07	Intake Raise	19.6	206	10.5	
IR_08	Intake Raise	19.6	159	8.1	
Total			**1,142**		
Exhaust					
Code	**Description**	**Area m^2**	**Volume m^3/s**	**Velocity m/s**	**Quantity**
ER	Exhaust Raise	9.6	127	13.2	9
Total			**1,143**		

23.1.6 Conclusions and Recommendations

Conclusions

The results from the iterative analysis of the ventilation design show that the proposed ventilation network can support five actively extracting panels, at least two additional developed panels and multiple access development drives.

The exact volumetric flow into the mine will depend on the number and location of active panels and diesel machines. The design phases suggested in this study should be considered examples of possible stages and therefore act only as conceptual guides to show that ventilation of the mine is achievable for the given ventilation design.

Given the low expected residency times, single-pass primary ventilation system and airflow design contingency the proposed ventilation design for the Madaouela Project is considered technically feasible.

A primary ventilation system is proposed, where potential contaminated air is exhausted promptly through exhaust raises located within the panel developments minimising the risk of radiation and other contaminants leaking into the central access development.

The approach to ventilation contamination control at the Madaouela underground uranium project should prioritise the removal of radon gas and radon progeny at its source. This is achieved by numerous exhaust raises located within the ore panels, and preventing the recirculation of contaminated air. The secondary ventilation should be used as a means of diluting and directing potentially contaminated air to the exhaust system, and providing fresh air supply to development headings.

Reducing the ingress or groundwater will minimise the potential for water-borne progeny, as well as effective ventilation management and operating practices, and properly utilised and extensive real-time instrumentation monitoring.

The required airflow rates for each phase of the mine are dependent on the emissions of radon gases and the planned diesel-powered equipment operating at that stage. This study has taken into account the maximum instantaneous airflow requirements for diesel powered equipment, operating sumps and accounted for a ventilation contingency of 25%. Analysis determined a maximum total airflow of 1,075 m^3/s is required. The phased ventilation modelling shows that 882 m^3/s of airflow for Marilyn-Marianne and 1,150 m^3/s for MSNE-Maryvonne is required.

Effective and robust scheduling of panels and development headings is critical in maintaining acceptable air velocities in haulage drives, conveyor drifts and working areas. Analysis found that each intake raise can only provide fresh air to three active ore panels simultaneously if connected using three drifts (two access, one conveyor), and only one panel can be ventilated if a single access or panel development drift connects to the raise.

23.2 Water resources study

23.2.1 Introduction

Background

This report summarises the assessment of available baseline hydrological and hydrogeological information and field data for the Madaouela Uranium Project. The information herein is based upon: data collection and analysis completed in support of the

studies during 2012 and early 2013 (SRK, 2014); further hydrogeological work conducted in late 2013 and early 2014; and from work completed in 2014 in support of the ESIA process.

Work Completed

To meet the objectives of the study an extended monitoring programme has been completed, which includes a detailed hydro census which incorporates: climate data review, catchment delineation and drainage mapping, spinner testing program, an open hole test survey and groundwater level monitoring and six rounds of groundwater sampling.

Information Sources

Reviewed information sources for this assessment comprise:

- Geological information for the Project area and adjacent areas
- Existing hydrogeological assessments for the Madaouela site:
  - SRK Consulting (UK) Limited (2013) - U4887 Madaouela Uranium Project studies
  - Atomic Energy Commission (1967); and
  - GoviEx/Legeni site investigation report from 2010.
- Two Areva EIA documents used to obtain historical groundwater abstraction data for the SOMAÏR and COMINAK holdings, along with hydraulic properties data and historical groundwater level data

23.2.2 Hydrology

An analysis of Shuttle Radar Topographical Mission (SRTM) data was carried out using the Archydro tool, within ArcGIS to determine the regional water catchments and drainage routes. To validate the drainage routes derived using this method, a site walkover was performed. Due to the resolution of the available topographical data, the results should be treated as preliminary. Further analysis to re-delineate more reliable results of catchments and drainage routes will take place when higher resolution topography information is available.

The drainage routes and catchment derived from the analysis are shown in Figure 23-16. In broad terms, the terrain in the region around the Project included in this analysis comprises relatively flat land with levels ranging between 350 and 560 m above sea level (masl). The higher elevations are associated with the foothills of the Aïr Mountains to the east.

The main drainage is associated with the runoff from the Aïr Mountains to the east of the Project and the drainage in the immediate Project is local. Surface water is scarce and the drainage network consists of ephemeral rivers also referred to as wadis. They are characterised by short duration flow events in response to heavy rainfall and remain dry for most of the year.

Since the drainage routes remain inactive, the site walkover produced mixed results with some of the mapped surface water drainage routes being clearly evident and other areas showing no evidence of surface drainage. The watercourses draining westward are well defined in the areas close to the Aïr Mountains and become indistinct on the plains.

The location of the ground-truthing points from the site walkover are shown in Figure 23-17. They have been classified into three categories: 1-obvious drainage: where signs of water routes such as sparse vegetation or obvious landscape features were found, 2-tentative (inconclusive) and 3-not obvious. In some locations where drainage routes seemed to be obvious, its path or direction were difficult to identify suggesting the path may vary from time

to time and the watercourse is not regular.

To have a better understanding of the topographical nature of the catchments, pictures of the ground-truthing points are shown in Figure 23-18, Figure 23-19 and Figure 23-20.

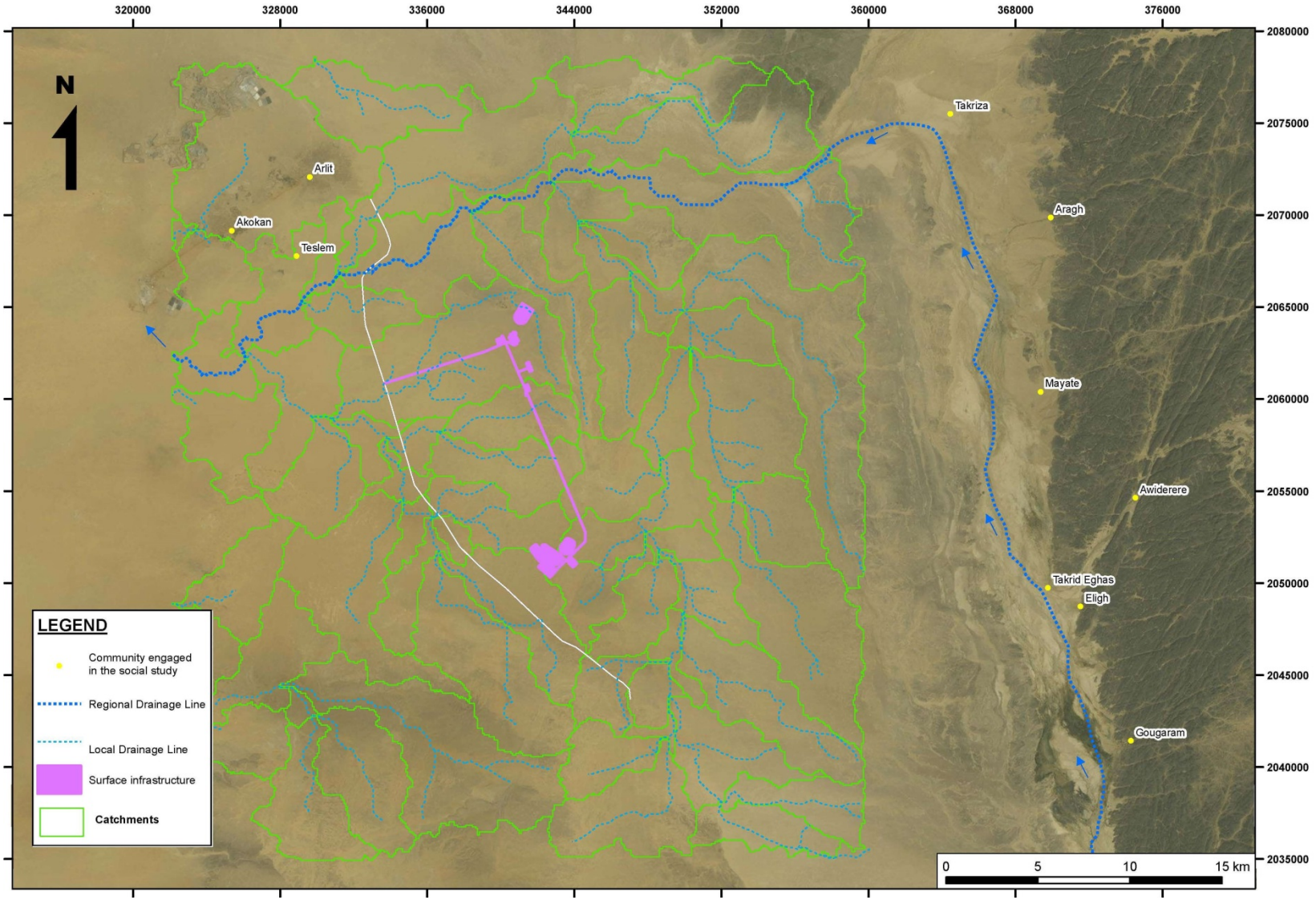


Figure 23-16: Surface water drainage lines and catchments map

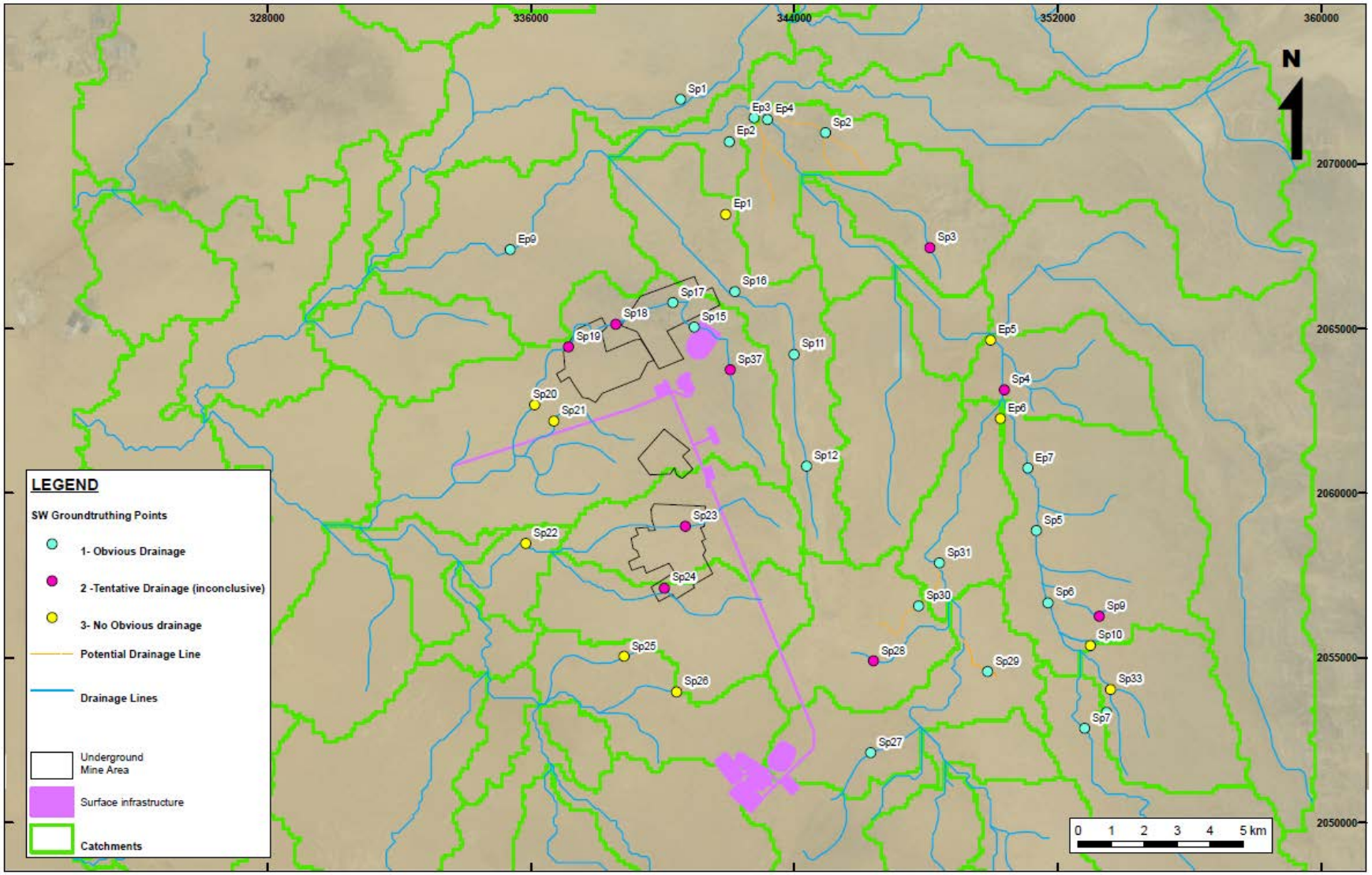


Figure 23-17: Ground truthing points

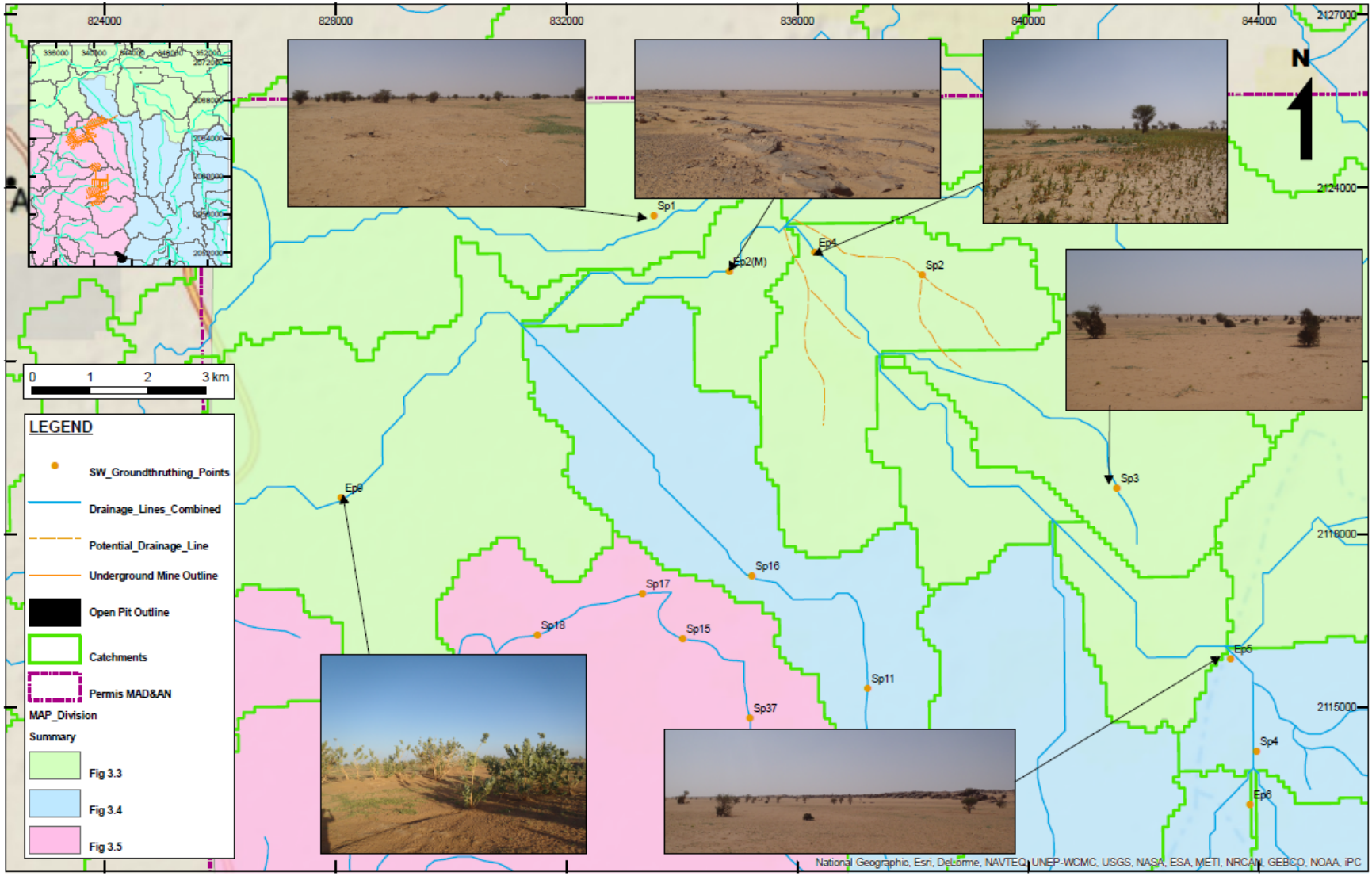


Figure 23-18: Surface water hydrology catchment characteristic 1

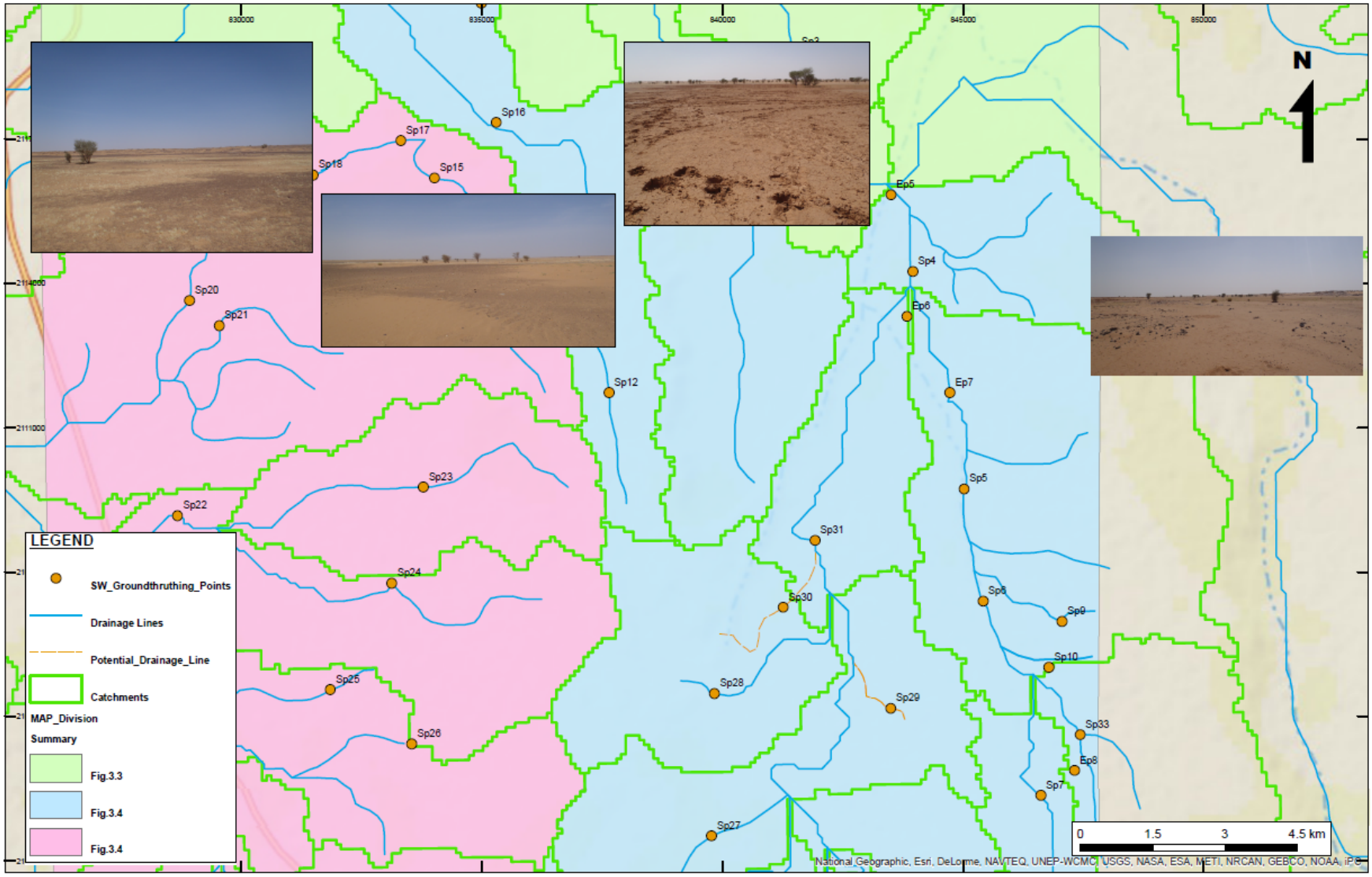


Figure 23-19: Surface water hydrology catchment characteristics 2

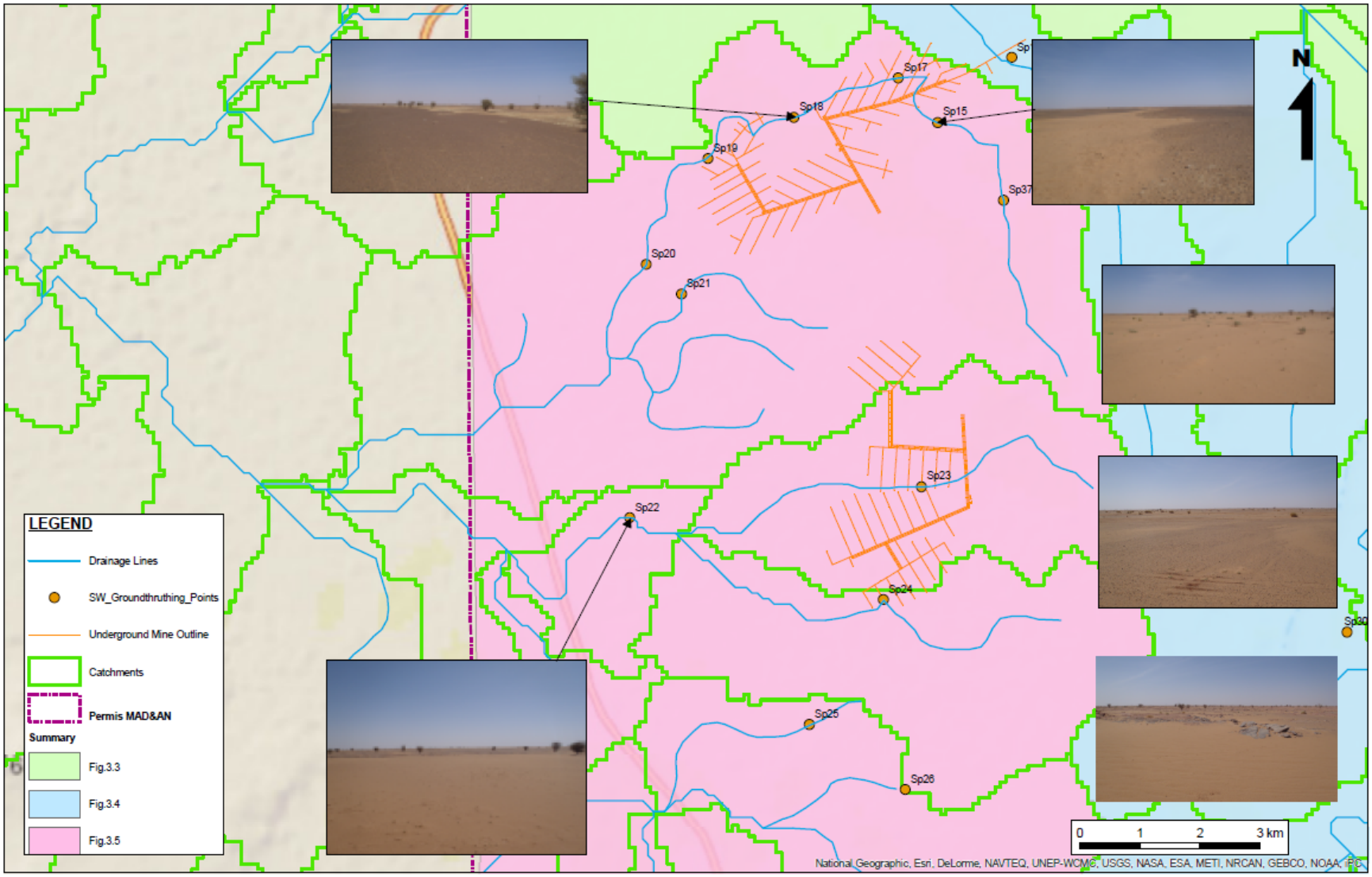


Figure 23-20: Surface water hydrology catchment characteristics 3

23.2.3 Hydrogeology

The following section provides a summary of the available hydrogeological data and the interpretations thereof. The conceptual hydrological model, which represents the overall interpretation of the groundwater system, is presented in Section 23.2.4.

Previous Groundwater Studies at Marilyn & Marianne

The 1967 Atomic Energy Commission "Completion Report of the Madaouela mine site" examines the hydrogeological and hydrochemical characteristics of the Guezouman aquifer. The following is a list of the most pertinent information taken from the report;

- The Guezouman formation consists of lenticular high permeability zones that may be identifiable by recording water strike information during drilling.
- ENE-WSW groundwater flow directions occur in the Guezouman formation, in the zone where mining occurred, following "channelized depressions". The regional north westerly movement of groundwater is described as still holding true.
- Artesian conditions (producing 30 m^3/h) were encountered in three boreholes at the base of the Guezouman close to the unconformable Talak contact, at a depth of 70.0 to 70.8 mbgl (meters below ground level), in a unit referred to as "L'unite Inferieure". It is believed the flows are associated with confined aquifers at the base of the Guezouman rather than true artesian conditions (which would infer boreholes flowing at surface). The confusion with the nomenclature is likely due to the fact the boreholes were drilled from within the workings. It is believed that what the report describes as the "L'unite Inferieure" may be the Unite Akokan (UA) channel.
- In "artesian" units, water pressure and temperature data suggests that vertical upward groundwater movement from deeper aquifer units occurs via fractures and high permeability layers. However this is likely to be localised; examples are also given where piezometer data suggests limited vertical hydraulic connectivity.
- Pumping rates of up to 50 m^3/h (subsequently reduced to 40 m^3/h) were attained in the FE1 borehole completed in the Guezouman formation (the exact location of this borehole is currently unknown).
- A localised ENE-WSW groundwater flow direction following "channelized depressions" is described that does not invalidate regionally established groundwater flow directions (to the northwest).
- Hydraulic conductivity (K) for the Guezouman formation range from 10^{-6} to 10^{-5} m/s; the highest of these values are generally an order of magnitude greater than those attained during pumping tests.

GoviEx/ Legeni Site Investigation Report (2010)

A series of packer tests were carried out and reported upon in March 2010 focusing on mineralised zones at the base of the Guezouman formation. The main objective was to evaluate permeability of the tested mineralised zones, and to provide an estimation of the potential for in-situ leaching as a method of uranium extraction.

A total of ten boreholes were selected for testing, and data was successfully collected from nine of these. After the completion of testing, each borehole was airlifted and converted to a standpipe piezometer.

Hydraulic conductivity (K) values for the base of Guezouman formation obtained from the packer tests range from 10^{-9} to 10^{-7} m/s. These values are low in comparison with K values obtained from pumping tests.

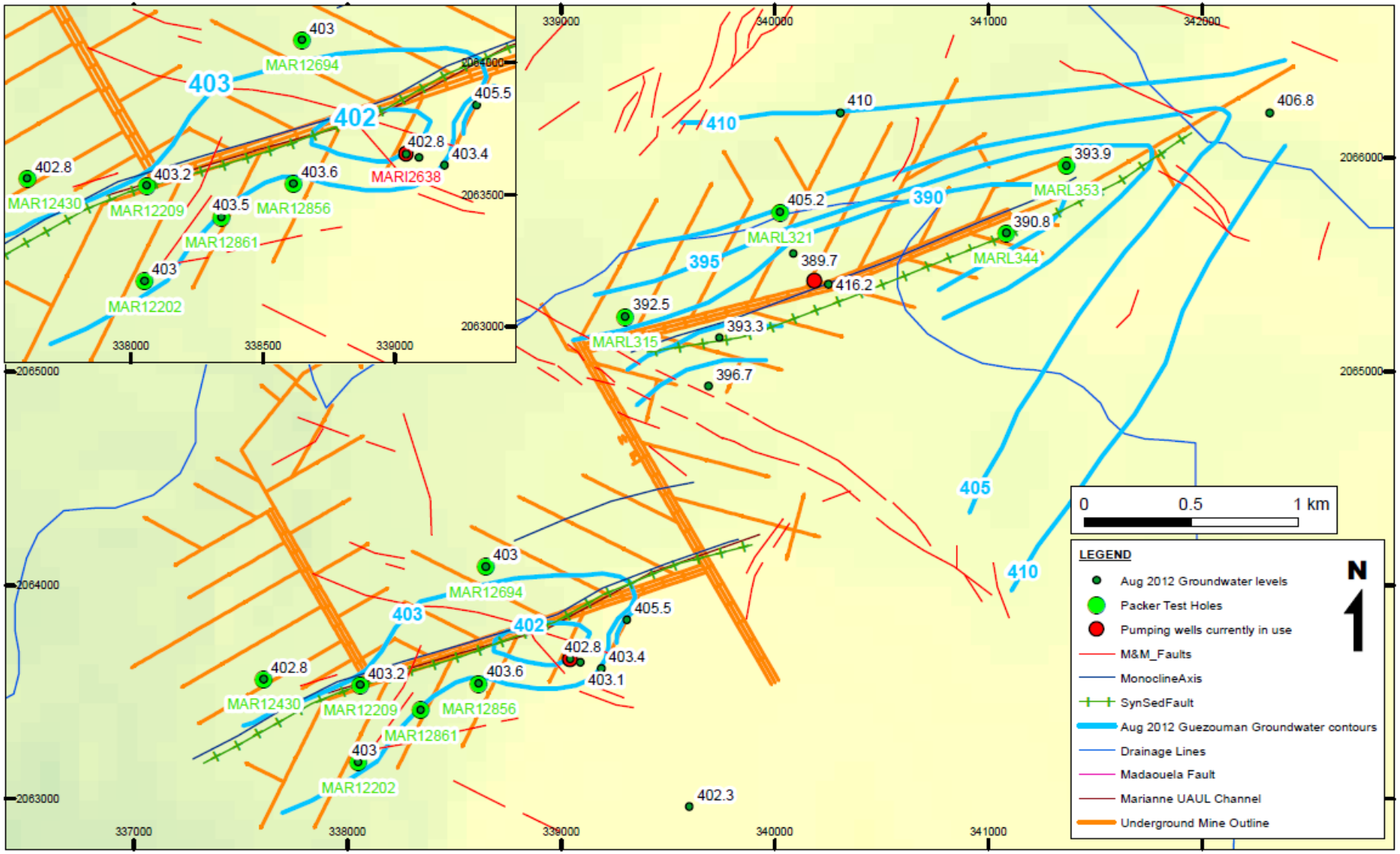


Figure 23-21: Map of Marianne and Marilyn zones with August 2012 groundwater contours for the Guezouman formation

Information from Neighbouring Areva Operations

Overview

Information on the neighbouring operations has been obtained from the Akouta and SOMAÏR EIA reports (Areva, 2004 and 2005). General information regarding the operations has also been obtained through discussions with GoviEx staff and is summarised as follows:

- COMINAK is an underground operation and the Guezouman is the ore bearing unit
- The principal (and possibly the only) dewatering method is abstractions from the workings themselves. SRK are not aware of any boreholes used specifically for dewatering at COMINAK.
- There have been periods when groundwater inflow rates have been problematic and impacted on the rate of mine advance. It is understood this situation has occurred during decline development.
- A major source of groundwater in the COMINAK mine is believed to be the overlying Tarat units. Old exploration boreholes form a conduit through the Tchinezogue and promote vertical downward flow from the saturated Tarat aquifer into the underlying dewatered Guezouman formation.
- SOMAÏR is an open pit operation. The Tarat aquifer is the ore bearing unit and the source of major groundwater inflows. Nuisance inflows were also encountered within the Izegouande.
- There is little information available on the method of dewatering employed at SOMAÏR. It is understood in-pit sumps are used to pump groundwater inflows. Water supply wells, which also target the Tarat, impact on water levels around the open pit.

Historical abstractions

Historical abstraction rates up to 2004 are provided in the two EIA documents. The Arlit and Akokan boreholes are tapping the Tarat aquifer and are by far the most significant groundwater abstractions in the region. Historical abstraction data presented in the AREVA EIA documents indicate SOMAÏR water supply and dewatering boreholes pumped at an average rate of 482 m^3/hr between 1969 and 2003. Boreholes owned by the COMINAK operation pumped at an average rate of 573 m^3/hr between 1975 and 2003.

Abstractions from the Tarat aquifer are far greater than those from the Guezouman formation. The total volume of water abstracted from the Tarat aquifer up to 2004 by Cominak, SOMAÏR and Cogema was 256,833,890 m^3, while abstractions from the Guezouman amounted to 7,759,454 m^3.

Hydraulic properties

The EIA documents also provide hydraulic properties data for the various aquifer units.

Groundwater Levels and Regional Flow Directions

A general regional groundwater pattern is described in the SOMAÏR EIA report. The regional groundwater flow direction is to the northwest (from the southeast) in the Izegouande, Tarat and Guezouman aquifers. Recharge occurs from unspecified zones to the South and discharge/evaporation theoretically occurs at formation outcrops at the Northern limit of these formations.

Regional groundwater contours developed by S. Bottero are provided for the Izegouande (1969) and the Tarat (1969 and 2004). Pre-mining groundwater level contours for the Guezouman formation are also referred to in the report, although no figure is provided. Table

23-21 summarises the main characteristics of each set of initial pre-mining contours.

Table 23-21: Summary of pre-mining groundwater contours

Aquifer	Year	Maximum piezometric elevation (masl)	Minimum piezometric elevation (masl)	Principal direction of Groundwater flow	General gradient of groundwater flow (m:km)
Izegouande	1969	401	394	SSE to NNW	0.5:1
Tarat	1969	401	393	SSE to NNW	0.35:1
Guezouman	Pre-mining	418	393	SE to NW	1:1

The 1969 (S. Bottero, 2004) groundwater contours for the Tarat formation indicate (Figure 23-22) a pattern of regional groundwater flow to the northwest. In 2004, updated groundwater contours (S. Bottero, 2004) in the same formation (Figure 23-23), show the drawdown effect of potable water pumping and open pit and underground mine dewatering. The general north-westerly flow pattern in the Tarat is over-printed by two cones of depression, focusing around COMINAK mining operations and Akokan. Drawdown at the open-pit SOMAÏR operation is less distinct; this is attributed to the thinning and shallowing of the Tarat formation in this area.

A general decline in groundwater elevations around the Artois sector (the northeast end of the SOMAÏR operation, the planned zone for open-pit operations in 2004) is observed between the readings taken in 2003 and 2004 in the range of 0.4 to 3.15 m. The data also indicates that groundwater elevations in the Izegouande formation are higher those in the Tarat formation; in the order of 8 m, indicating a downward hydraulic gradient.

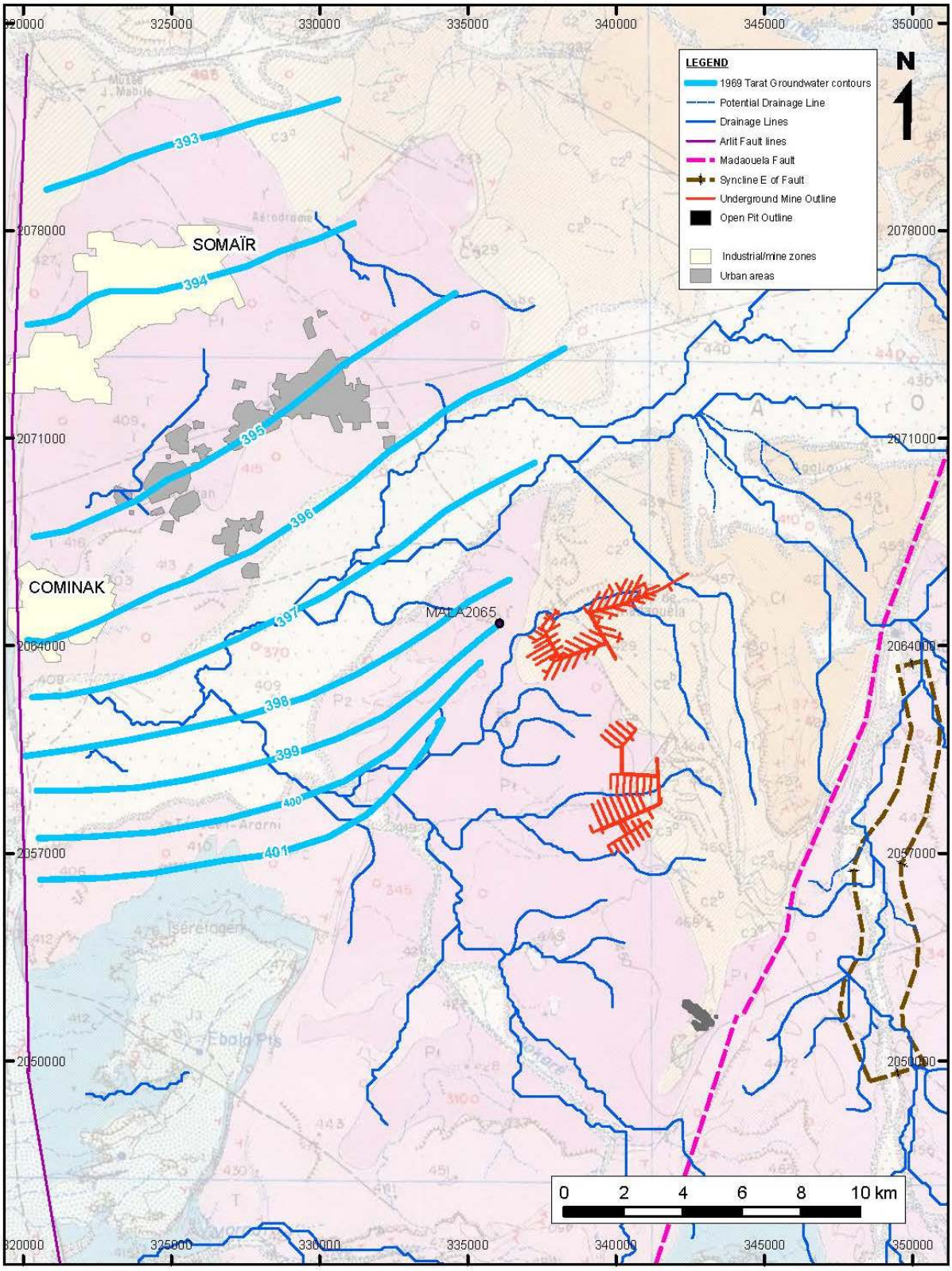


Figure 23-22: Regional map with 1969 groundwater contours for the Tarat formation (Source: Bottero, 2004)

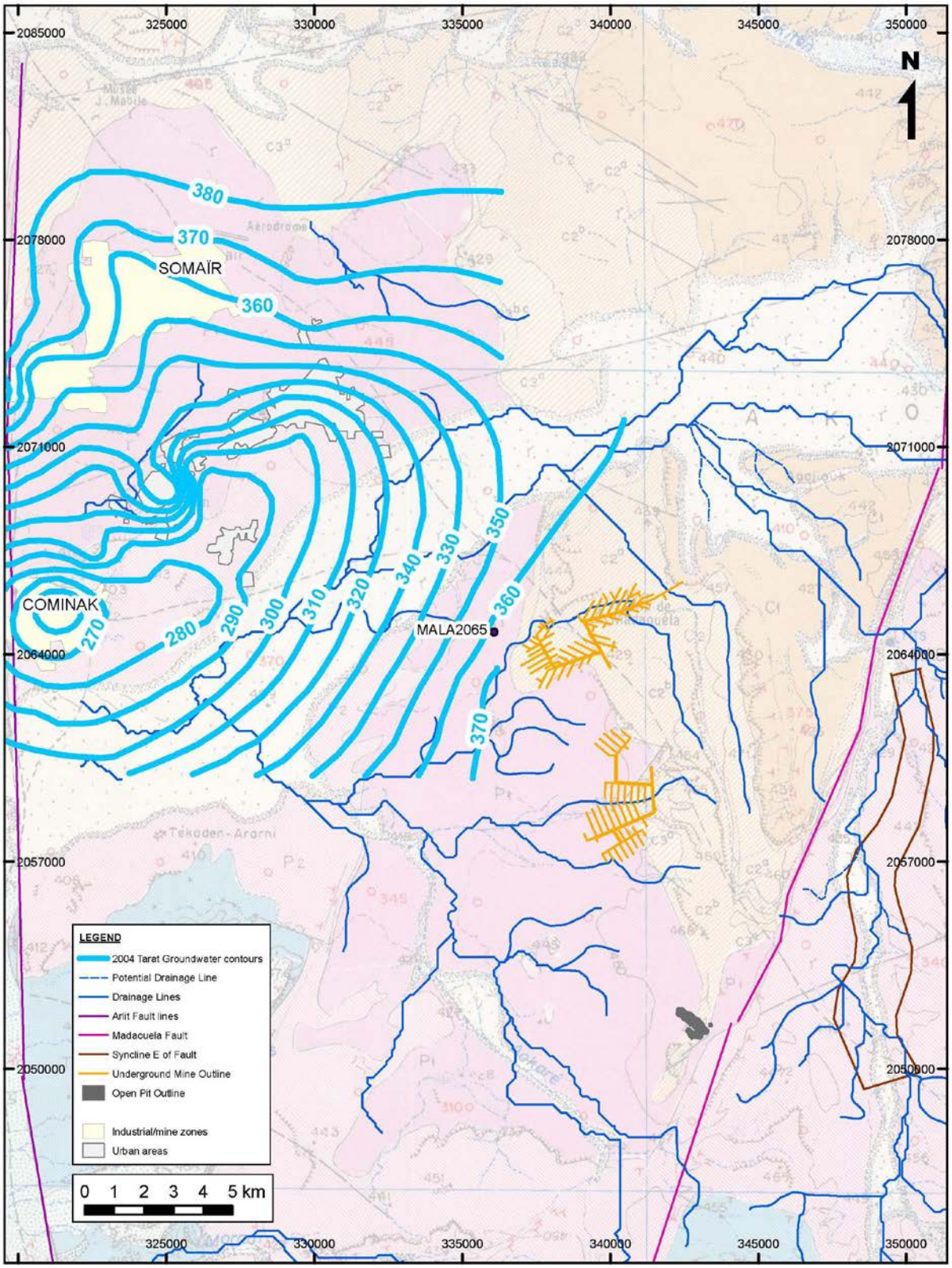


Figure 23-23: Regional map with 2004 groundwater contours for the Tarat formation

GoviEx Pumping Test Programme

A pumping test programme was completed by GoviEx. Table 23-22 presents a summary of completed pumping tests. Figure 23-24 shows the locations of the pumping and observation wells.

Table 23-22: Summary table of pumping tests

Test	Pumping Well	Pumping Well Coordinates (UTM, Z32)			Observation Wells (Distance from pumping well)
		Easting	Northing	Elevation (mRL)	
PT1	MSNE4400	341668.8	2052642	454.48	MSNE4892 (100m), MSNE4113 (200m)
PT2	MARI2638	339043.4	2063653	435.17	MARI2422 (50m), MARI2933 (150m)
PT3	MARI3048	335101.1	2066102	422.12	MARI3070 (100m), MARI3081 (200m)
PT4	MAD10227	352836.6	2051597	455.02	MAD10017 (100m)
PT5	MSNE4268	339881.2	2059841	438.84	MSNE5159 (100m), MSNE4264 (200m)
PT6	MSNE4326	340145.9	2055055	446.02	MSNE4789 (100m)
PT7	MSNE4931	342802.9	2051903	463.18	MSNE4996 (68m), MSNE4999 (71m), MSNE5151 (110m), MSNE5194 (50m), MSNE4947 (417m), MSNE4956 (708m), MSNE5424 (1193m)
PT8	MAD10003	345936.4	2052628	447.92	MSNE5387 (1364m), MSNE4595 (1515m), MAD10005 (1581m), MAD10030 (2125m), MSNE5424 (2642m)

Well Performance Testing

A step-test was conducted at each pumping well with the exception of MSNE4400 (PT1) and MARI2638 (PT2). Pumping was increased through a set of a minimum of four pumping rates, as derived from a calibration test. Each step was continued for 120 minutes until an approximate stabilisation in the pumping well before the next rate was applied.

At the end of the final step, the pump was switched off (but not removed) and the recovery measured. The pumping well water level was then allowed to recover to within a minimum of 5% of the pre-pumping water level before any further testing was undertaken. The results of the step test were used to derive an appropriate pumping rate for the constant rate test.

The majority of tested wells were efficient at the tested rates. However MARI3048 and MSNE4268 produced lowered efficiencies, particularly MARI3048. These low efficiencies indicate these boreholes may be experiencing some permeability reduction at the wellbore, most likely caused by insufficient development.

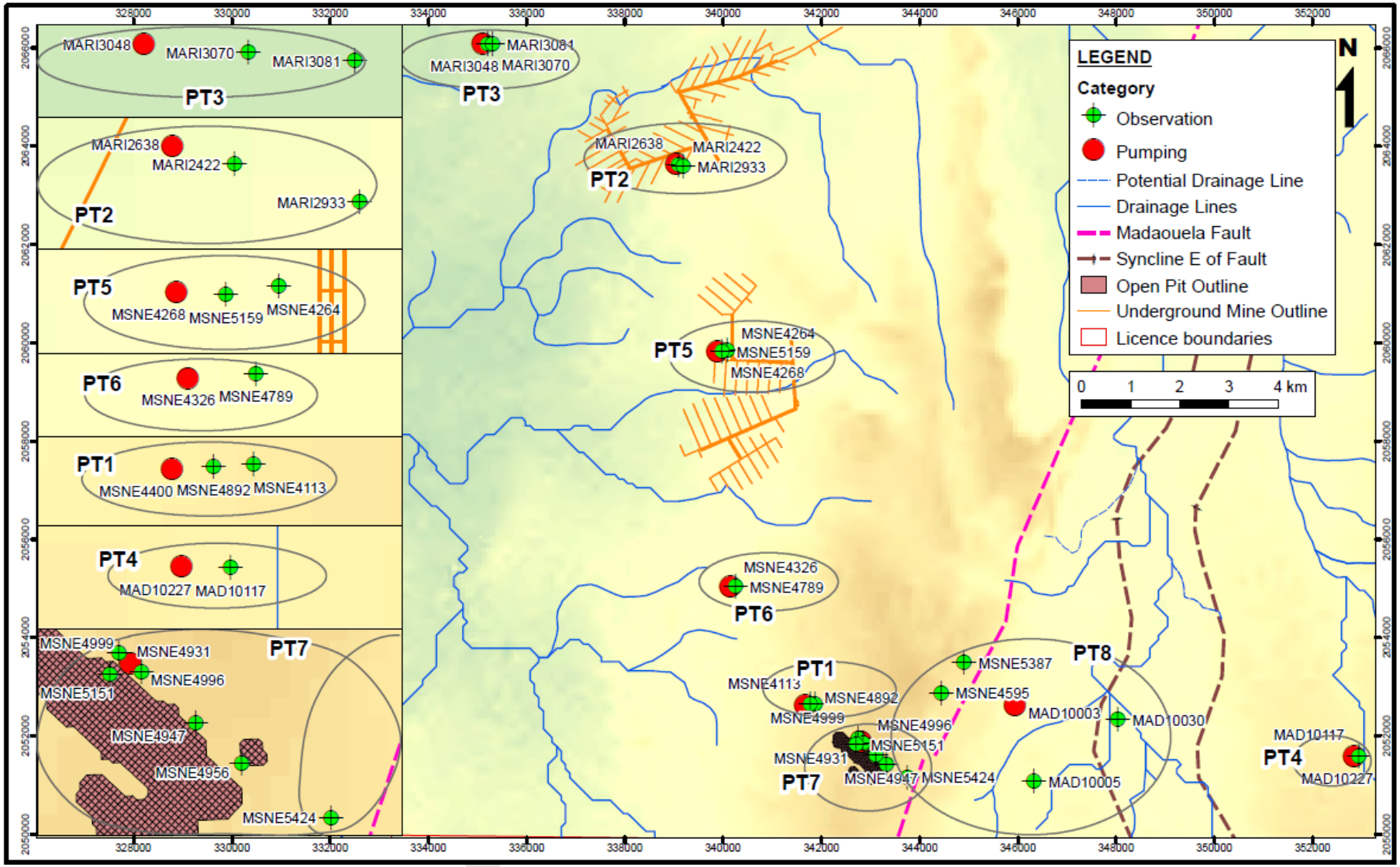


Figure 23-24: Locations of test production and observation wells

Constant Rate Testing

Numerous methods of analysis were used to verify the validity of the chosen method of drawdown analysis. The drawdown results for the eight pumping tests (PT) are summarised below and presented in Table 23-23:

- *Pumping Test 1 (MSNE4400)* – A constant rate pumping test of 1.77 L/s was performed in a 151 m deep well fully penetrating the 30 m thick Guezouman aquifer.
- *Pumping Test 2 (MARI2638)* - A constant discharge test of 2.38 L/s was conducted on a 96 m deep well fully penetrating the 29m thick Guezouman aquifer
- *Pumping Test 3 (MARI 3048)* - A constant-rate pumping test of 0.66 L/s was performed in a 268 m deep well tapping the full thickness of the 39 m Guezouman aquifer.
- *Pumping Test 4 (MAD1 0227)* - A pumping test was performed on a 181 m deep well at an average discharge rate of 2.03 L/s. The well penetrates the full thickness of the Madaouela formation to the east of the Madaouela fault
- *Pumping Test 5 (MSNE4268)* - A constant rate pumping test of 0.46 L/s was performed in a 140m deep well fully penetrating the Guezouman aquifer.
- *Pumping Test 6 (MSNE 4326)* - A constant-rate pumping test of 2.7 L/s was performed in a 172m deep well fully penetrating the Guezouman aquifer.
- *Pumping Test 7 (MSNE4931)* - A constant rate discharge test was conducted on a 120 m deep well at 1.98 L/s. The drawdown data in both pumping and recovery periods were recorded in the pumped well and seven piezometers. The configuration of the piezometers south-east (SE) of the pumping well aimed to measure the propagation of the cone of depression as it extended towards the Madaouela fault. The results of the test indicated that the Madaouela fault was likely to be acting as a barrier to groundwater flow.
- *Pumping Test 8 (MAD10003)* - A constant rate discharge test was conducted on a 372 m deep well at a rate of 3.6 L/s. This well taps the formations of the Madaouela, Tejia, Izegouande, Tarat and Guezouman; however spinner testing detected that the Tarat was the only productive aquifer. Therefore, this test is regarded as representative of the Tarat aquifer. The pumping test was carried out immediately east of the Madaouela fault and the impact of the fault is evident in the results.

Table 23-23 presents a summary of aquifer properties based on the recommended values for each test.

Table 23-23: Final summary of constant rate aquifer analysis

ID	T (m^2/d)	K (m/d)	S	Aquifer	Confidence		
					High	Medium	Low
PT1	7.21	0.26	$2.49x10^{-5}$	Guezouman	•		
PT2	22.89	0.73	$1.33x10^{-5}$	Guezouman		•	
PT3	1.33	0.038	$1.11x10^{-4}$	Guezouman	•		
PT4	4.52	0.30	$5.61x10^{-5}$	Madaouela	•		
PT5	1.09	0.045	$1.08x10^{-5}$	Guezouman		•	
PT6	7.44	0.25	$2.16x10^{-5}$	Guezouman	•		
PT7	4.57	0.16	$4.88x10^{-5}$	Guezouman	•		
PT8	5.26	0.17	$2.32x10^{-4}$	Tarat		•	

Spinner Testing Programme

Spinner testing is a borehole logging technique used to determine vertical variations in hydraulic properties in an aquifer. The objectives of the tests conducted as part of the hydrogeological investigations were to assess which lithological units are the most permeable and whether there is any heterogeneity of permeability within any of the units. On the west side of the Madaouela fault the Guezouman formation was of interest.

On the west side of the Madaouela fault the spinner test indicated only primary permeability within the sandstone matrix of the Guezouman formation with a marked absence of permeability in the Talak. The permeability was observed in the lower portion of the Guezouman (from 94 m to the base of the formation), which is to be anticipated given the fining upward sequence (courser sandstones at the base and finer grained deposits towards the top of the unit) that characterises this formation. The conductivity log shows evidence of deteriorating water quality at depth with the clear rise in electrical conductivity between the Guezouman and the underlying Talak.

On the east side of the Madaouela fault the spinner tests found that the Izegouande formation was the main contributor to permeability by the relatively high specific drawdown values for these inflow zones. The permeability was primary within the rock matrix. There are occasionally more discrete variations in permeability within rock units which are likely to reflect the presence of more permeable course grained beds. The permeability in the Izegouande occurred at different depths within the formation in each borehole, ranging across its full thickness.

There is an indication from some boreholes on the east side of the fault that there is ambient upward flow within the boreholes, with discharge into the unsaturated zone near the surface. This indicates an upward hydraulic gradient from the deeper units to the shallow units. The quality of data at holes where this is indicated is relatively poor and the observations should therefore be treated with some caution. It does however support the general pattern of upward hydraulic gradients observed through the area.

Open Hole Pumping Test Survey

A survey of open exploration boreholes east of the Madaouela fault was completed in November 2012. The objective of the survey was to obtain water level information and to identify targets for small scale single well pumping tests. A total of ten suitable exploration holes were identified for testing. Holes were pumped at a discharge rate of 0.75 L/s until water levels stabilised. The final drawdown was recorded and hydraulic conductivity estimated. The results indicate hydraulic conductivity ranges between 0.14 and 2.88 m/d, which is indicative of a productive aquifer.

The relatively small permeability range suggests that the area has potential to be high yielding, and hydraulic properties are likely to be broadly consistent across the zone.

Review of GoviEx Groundwater Level Monitoring

Regional Groundwater Contours

Groundwater contours within the Guezouman formation, to the west of the Madaouela fault, were generated using dipmeter water level data collected during November 2012 from the MadSud (MAD S) and Miriam zones. A general pattern of decreasing groundwater elevations from southeast to northwest is apparent from the data and indicates north-westerly regional groundwater flow consistent with the AEC report (1967). Water levels collected during August 2012, although not as extensive as the November dataset, show a similar regional pattern of groundwater flow to the northwest.

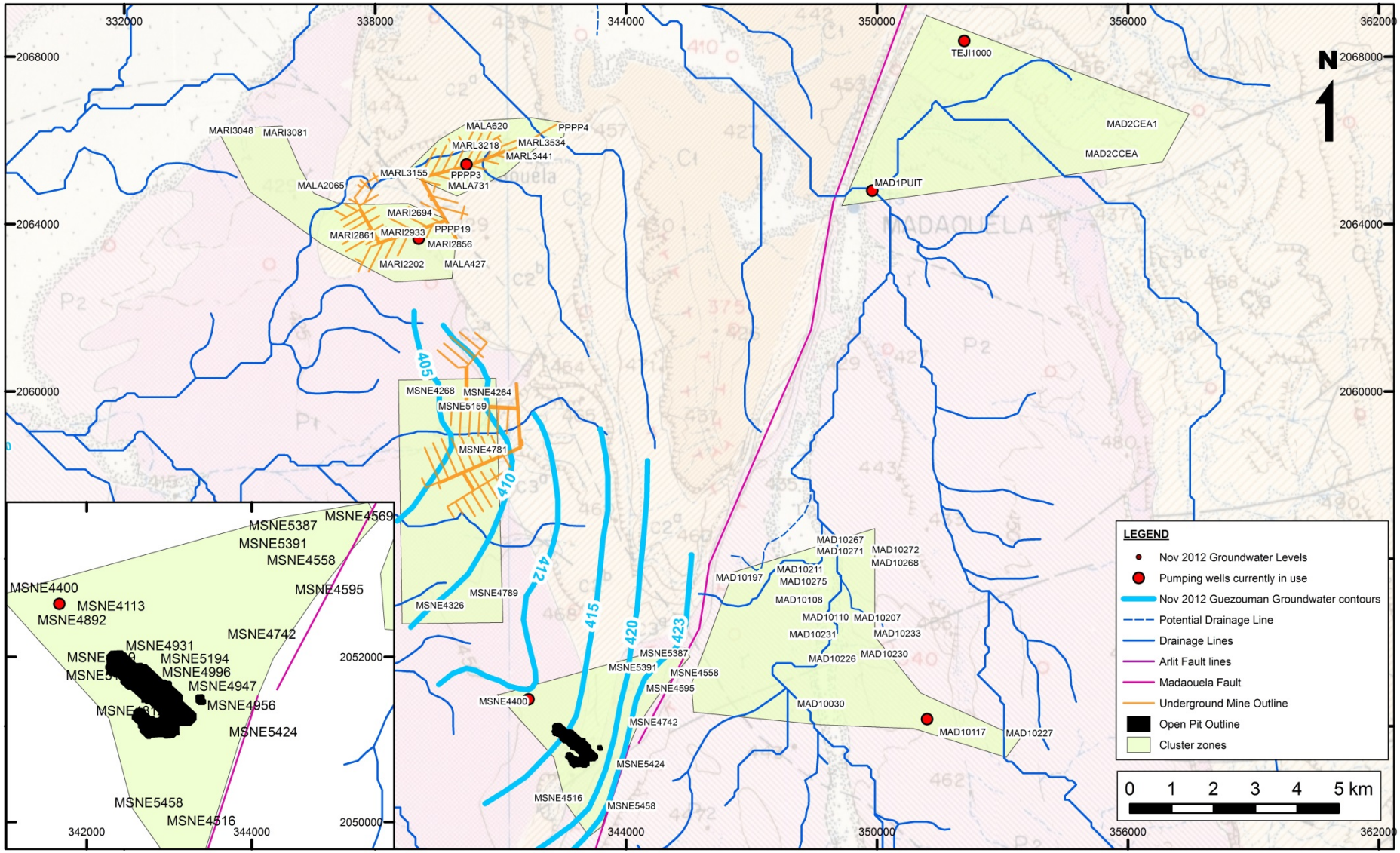


Figure 23-25: Regional map with November 2012 groundwater contours for the Guezouman Formation

To the east of the Madaouela fault, limited groundwater dips were collected from the Madaouela formation in November 2012. Groundwater elevations are reasonably flat within this zone, although the north-westerly groundwater flow direction is still discernable. The step change in water levels across the Madaouela fault presents strong evidence that this is a low permeability feature. Groundwater levels in shallow boreholes in the north of this zone (TEJI1000, MAD1PUIT, MAD2CCEA, MAD2CEA1) indicate flow is from south to north on the east side of the Madaouela fault. The water levels in the north do not show a consistent trend and there are other factors requiring consideration to enable a meaningful interpretation of these water levels. These include: upward hydraulic gradients from the deeper units into these near surface deposits, the influence of pumping, and uncertainties over the collar elevations.

Localised Groundwater Contours; Marianne and Marilyn Zones

Data from the Guezouman formation in the Marianne and Marilyn zones suggests a localised ENE to WSW groundwater flow direction (Figure 23-21) similar to that described in the AEC 1967 report. It is believed that slump and shear features occur in the UA channel, the uppermost unit of the Talak formation. Groundwater flow may occur preferentially along such features. The exact spatial extents of the features are difficult to define; these are thought to be soft sediment related with little or no displacement.

Groundwater elevations from the Guezouman in the Marilyn zone are generally deeper immediately adjacent to inferred structures; on the range of 390 – 394 mARL. Groundwater levels then become shallower to the north of the UA channel. The rise in groundwater levels correlates with the rising limb of the monoclinal structure in that area.

The groundwater elevations to the south of the structure are less well constrained as there is relatively little groundwater level data.

The groundwater contours in Marilyn are believed to be driven by the abstraction from the army camp water supply well, although connections between the Marilyn area and the Tarat aquifer to the west may also impact on the observed groundwater level distribution. The army camp water supply well is not monitored but readings taken on 20th November 2012 suggest the rest water level is approximately 384 mARL, and the pumping water level is 372 mARL. These elevations are approximate as the surveyed elevation of the collar is not currently available, so the elevation of the adjacent MALA9999 has been used to provide a preliminary estimate of the groundwater elevations in the pumping well.

The influence of the UA channel in the Marianne zone is less pronounced than at Marilyn, although localised WSW flow may be occurring along the UA channel. Pumping from borehole MARI2638 in this area is believed to cause a localised cone of depression.

Open Hole Surveys 2013

Following completion of the 2013 study, further water level surveys were carried out across the Project area in order to validate the assumptions and findings and to provide additional data to support dewatering and water supply studies.

A total of 503 exploration holes were surveyed for water level and total depth between February and April 2013. 31 additional holes were put forward by SRK for surveying in October 2013, focussing on the western portion of the Project (an area not covered by the previous open hole surveys). A further 23 holes were also surveyed by GoviEx during the survey. Of the 54 holes surveyed; 28 were blocked at surface, and 18 were collapsed above the water table, leaving only 8 holes where a water level could be recorded.

Figure 23-26 shows the water level distribution across the Project area. The coloured circles represent water level data from piezometers and open hole water level surveys. The open hole water level surveys have greatly increased the available water level data across the area, especially in the MSNE, Maryvonne and Marianne deposit areas.

Dry hole elevations are also included on the figure as coloured squares. Dry hole data can be useful where water level data is sparse, or where dry holes are found to be deeper than recorded water levels in the vicinity. The cluster of holes to the west of MSNE is a good example, where the westernmost hole is dry at an elevation that is lower than measured water levels immediately to the east.

The results of the open hole water level surveys support the findings of previous hydrogeological work. The overall groundwater gradient is towards the west and north-west, and driven by the abstractions at SOMAÏR and COMINAK, and the strong upward hydraulic gradients are evident at a number of locations. The impact of the dome structure at Marianne on groundwater water levels is more clearly evident. There is also some evidence of localised groundwater mounding at MSNE.

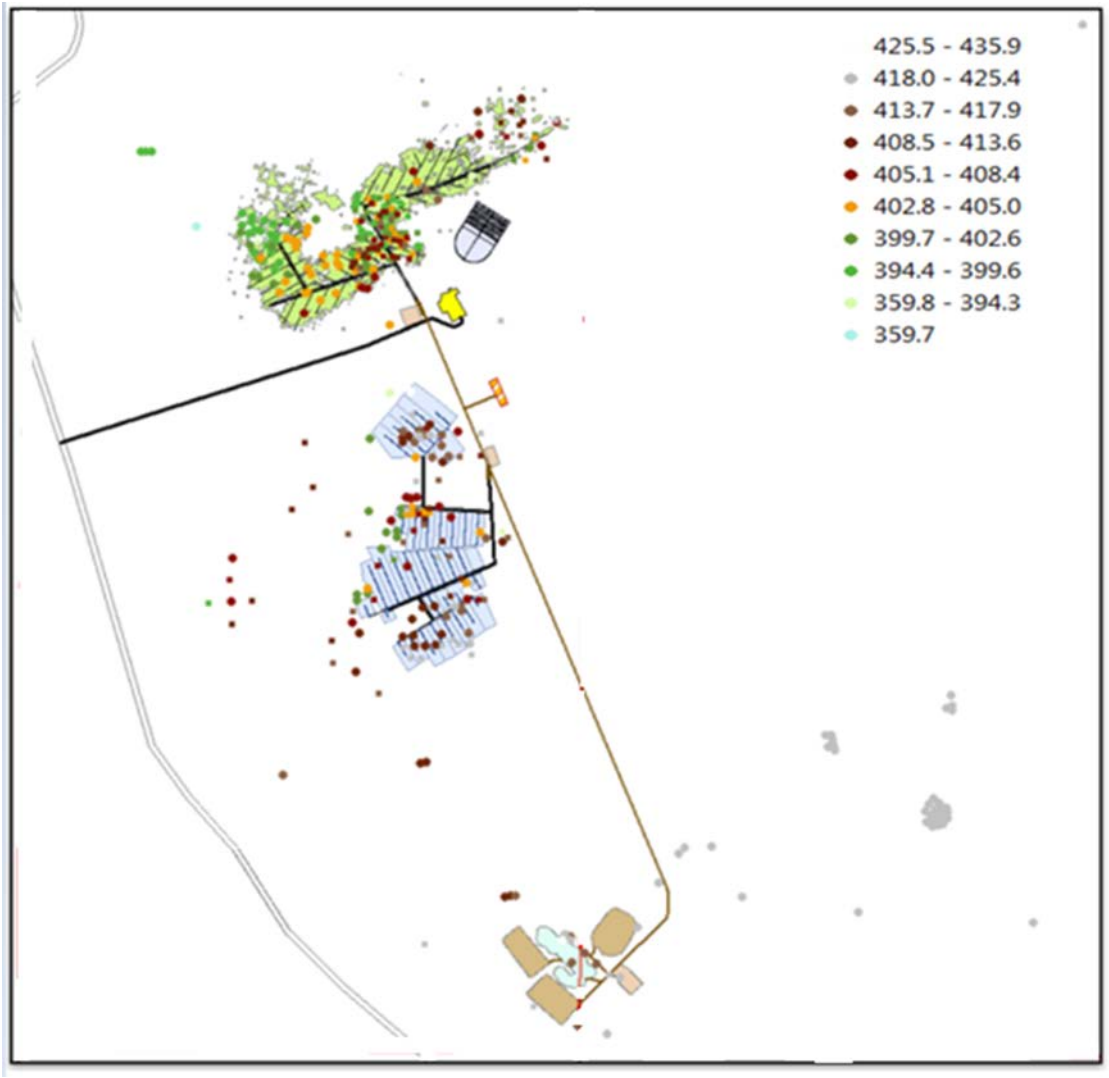


Figure 23-26: Water level data across the Madaouela deposits

Hydrograph Analyses

The details of the monitoring holes used in the 2012 baseline groundwater level monitoring are included below. The baseline monitoring data from all monitoring points has been plotted by area in order to analyse the responses. Groundwater level fluctuations were compared to rainfall data from the SOMAÏR meteorological stations and no discernable recharge responses were noted. A series of pumping tests have been completed and a number of pumping test responses are observed in the routine water level monitoring. There are also number of apparent groundwater level fluctuations which are associated with a change in water level monitoring equipment during October 2012. Groundwater level fluctuations within that period have therefore been largely.

Marianne Cluster

Groundwater elevations for the Marianne cluster are plotted in Figure 23-27. The key observations from the Marianne cluster are discussed below.

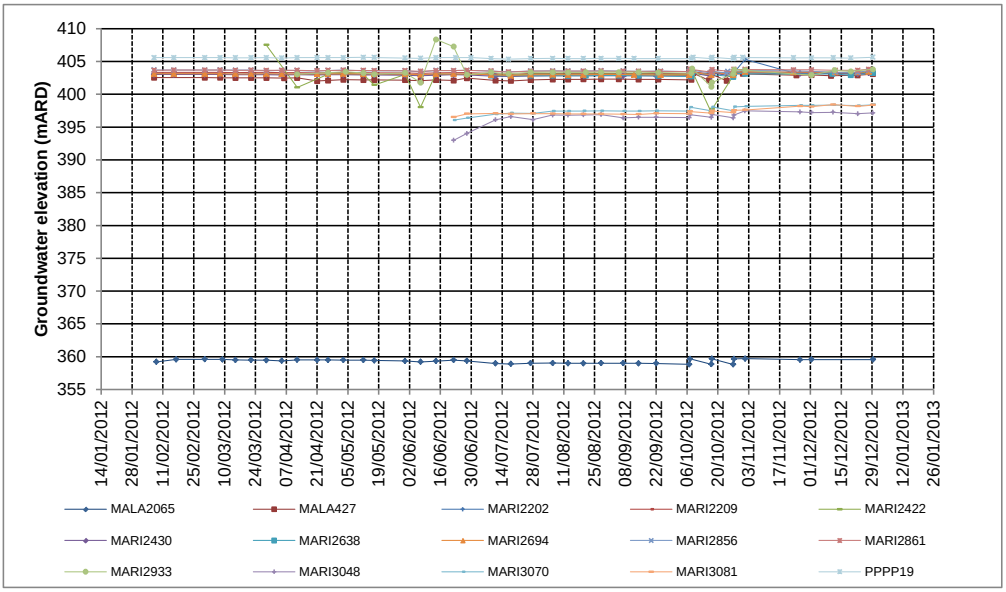


Figure 23-27: Groundwater hydrographs in the Marianne cluster

MALA2065 records groundwater elevations of around 359 mARL, which is around 40m lower than other water levels in the area. It is likely this hole intercepts the Tarat formation. At 150 m it is of significantly shallower depth than the holes to the northeast tapping the Guezouman (these holes range from 262 m to 268 m deep). The depth of the Tarat formation intercepted in these deeper holes is on the range of 145 – 150 mbgl.

The lower groundwater elevations at MALA2065 appear to be associated with the propagation of drawdown within the Tarat formation from the mining operations to the west. MALA2065 has a current groundwater elevation of 359 mARL and sits directly on the 360 mARL contour. This would suggest the cone of depression from the COMINAK and SOMAÏR operations had approached a steady state by 2004, although it is not clear how well constrained Botterro's contours were in the MALA2065 area.

Boreholes MARI3048, MARI 3070 and 3081 are completed in the Guezouman to the far northwest of the monitoring network and record lower groundwater elevations than the Guezouman water levels around Marianne. This is in line with the general pattern of decreasing groundwater elevations towards the north-west. The early data from these holes

clearly shows the water levels are recovering from the pumping test on MARI3048.

MALA427 consistently records deeper groundwater elevations than a group of 10 holes located 900m to the northwest (Figure 23-27). The reasons for this anomaly are currently being evaluated further. Confirmation of the units intersected and screened in this hole is necessary. It is thought to be completed within the Guezouman and possibly the Talak formation. It is possible MALA427 has a stronger connection with the Marilyn boreholes and is therefore impacted by the abstraction in that area.

Marilyn Cluster

Groundwater elevations for the Marilyn cluster are plotted in Figure 23-28. Key observations from Marilyn are discussed below.

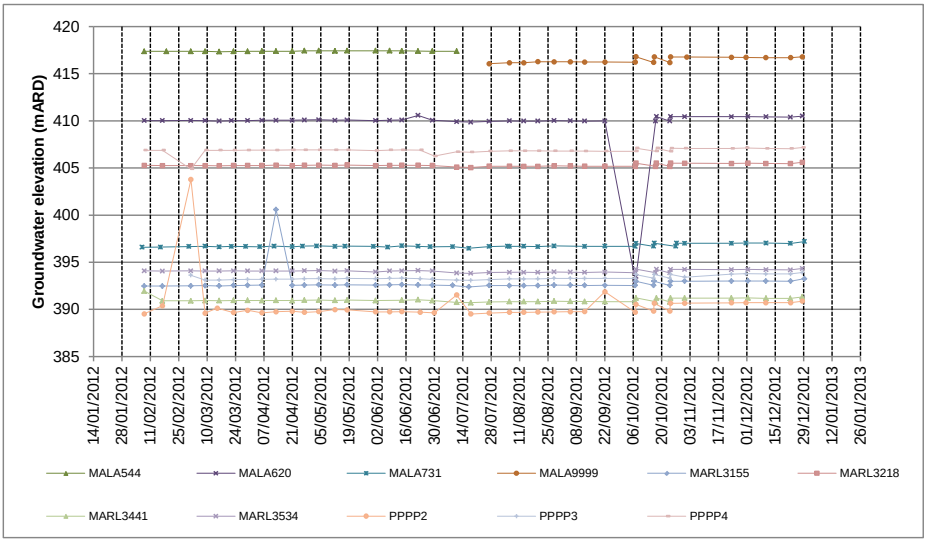


Figure 23-28: Groundwater hydrographs in the Marilyn cluster

MALA544 has high groundwater elevation values (on the range of 417 mARL); this hole is much deeper than surrounding holes (at 450 m it is some 300 m deeper than the next deepest hole) and so the readings here are likely to represent piezometric heads in deeper formations (Figure 23-28).

High groundwater elevation readings recorded at MALA9999 (located near slump and shear features associated with the UA channel) are on the range of 416 mARL (Figure 23-28). This hole has a total depth of 150 m, while most other holes in the immediate vicinity are 44 – 70 m deep; deeper formations of high piezometric head may be intercepted here.

The higher groundwater elevations recorded in these deep boreholes suggest there may be upward hydraulic gradients from the deeper units occurring in this area.

MAD-S Cluster

Limited monitoring data is recorded from this area. Monitoring holes in the North of this cluster MSNE 4268, 4264 and 5159 record similar groundwater fluctuations on the range of 404 mARL to 407 mARL (Figure 23-29) and fit in with the general conceptualisation of north-westerly regional groundwater movement within the Guezouman formation.

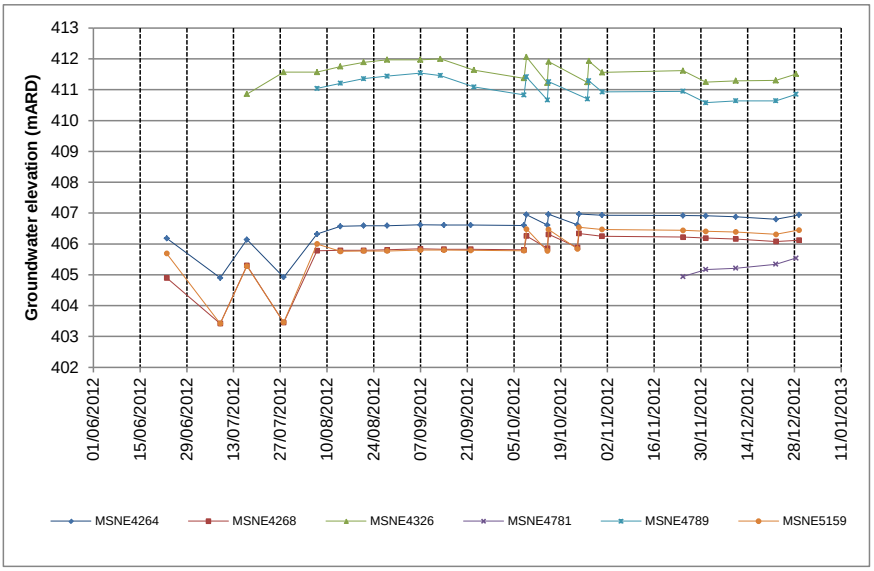


Figure 23-29: Groundwater hydrographs in the MAD-S cluster

Miriam Cluster

Groundwater elevations recorded here, in the Guezouman formation, consistently decrease to the northwest. Elevations rise steeply towards the Madaouela fault to the southeast. Figure 23-30 presents the hydrograph for this cluster. Groundwater elevations at MSNE 4892, MSNE 4113 and MSNE 4400 appear lower than expected and cause a kink to the southeast in the regional groundwater contours. This is likely to be a result of the drawdown effect from pumping at MSNE 4400. Due to the low frequency of data collection, no drawdown effect from pumping can be clearly established in the hydrographs.

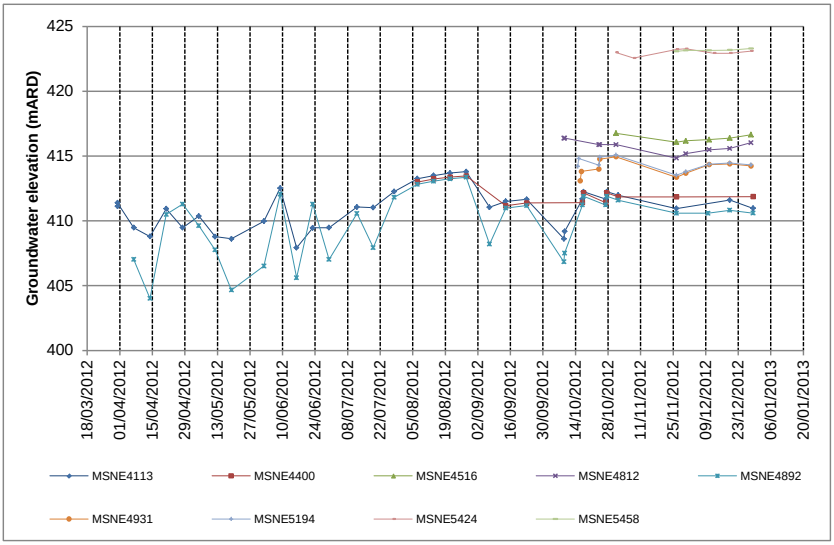


Figure 23-30: Groundwater hydrographs in the Miriam cluster

MAD1 Cluster

Regular readings in this area are only available from piezometers MAD10227 and MAD10117, both of which tap the Madaouela formation. These holes show consistent hydrograph readings on the range of 424 mARL to 425 mARL (Figure 23-31). No data is available from the nearby pumping well MAD10001 to examine drawdown effects.

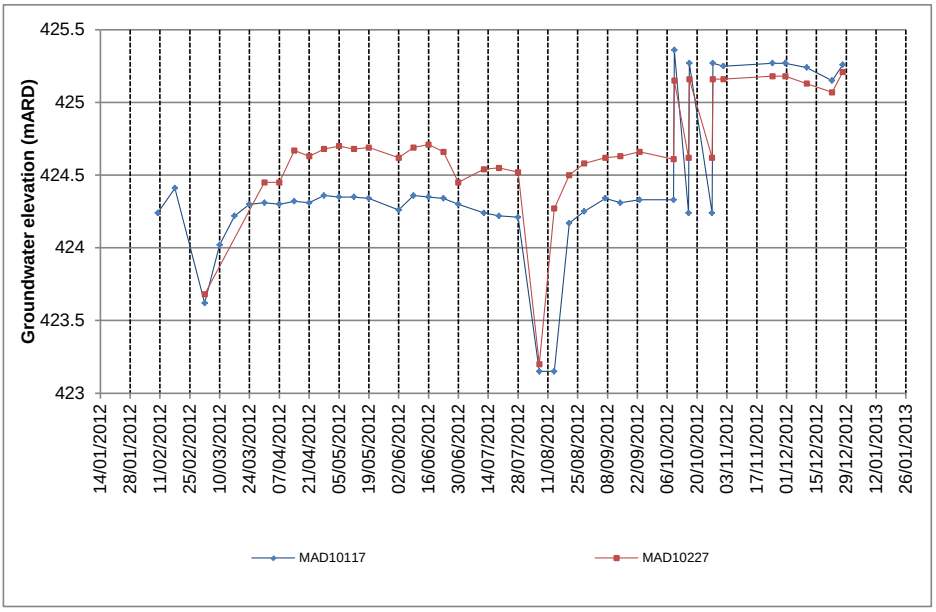


Figure 23-31: Groundwater hydrographs in the MAD-1 cluster

North-East Cluster

Limited data is available from this zone; confirmation is needed on what formations are tapped by each of the four monitoring points. There are conflicting reports of the total depths of monitoring points in this cluster; for the purposes of this analysis it is assumed that the correct depths are as follows;

- MAD1PUIT (Madaouela Well) 20 mbgl
- TEJI1000 (Tejiat Well) 26 mbgl
- MAD2CEA1 (piezometer) 400 mbgl
- MAD2CCEA (piezometer) 400 mbgl

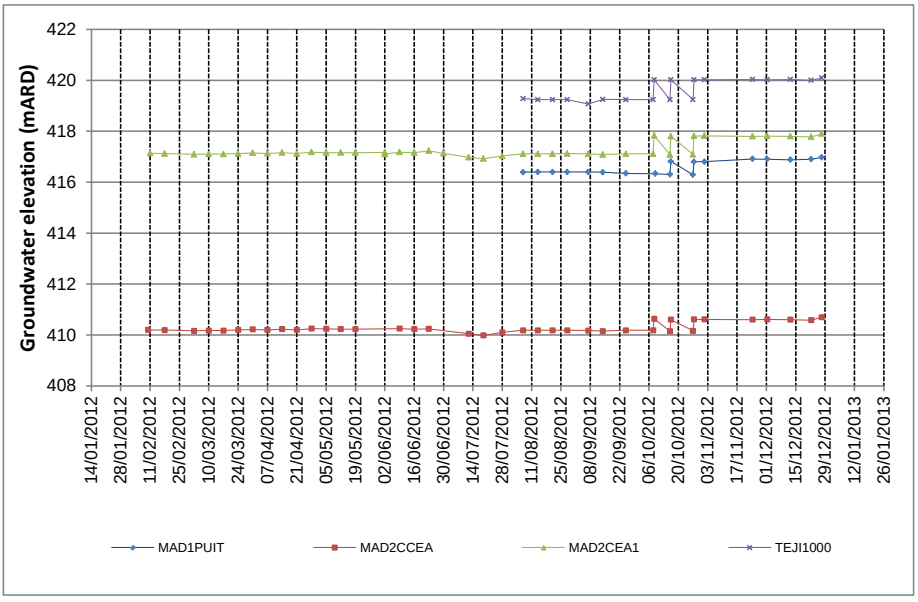


Figure 23-32: Groundwater hydrographs in the north-east cluster

The difference in groundwater elevations between MAD2CEA1 and MAD2CCEA may indicate that groundwater drawdown is being induced in deep formations close to or at MAD2CCEA

(Figure 23-32). It is likely that groundwater elevation readings from MAD1PUIT and TEJI1000 represent much shallower formations.

Groundwater Abstractions

A map showing the locations of the abstraction wells inventoried as part of the baseline studies is presented in Figure 23-33. There are a total of six current groundwater abstraction wells within the Project area:

- Army camp borehole: This is the water supply for the army camp; it will cease to be operational upon relocation of the camp. It suffers from poor water quality. The well taps the Guezouman aquifer. The borehole has been operational since the 1960's and has contributed to the depletion of groundwater levels in the Guezouman.
- MARI2638: This is the GoviEx exploration water supply borehole used to supply drill rigs and the core shed. This hole has been used to supply nomads and their animals in the past but it is not currently operational. The well taps the Guezouman aquifer.
- MSNE4400: This borehole was equipped with a pump to supply the GoviEx drilling rigs. It is not used by communities and is not considered a receptor. The well taps the Guezouman aquifer.
- MAD10001 (also known as the Ebargaz well): This well is owned and operated by GoviEx and is used to supply the Ebargaz camp. The well is used by the livestock herders to water their livestock, and taps the same aquifers as the proposed Project wellfield.
- MAD1-PUIT (also known as the Madaouela Well): This is a shallow large diameter well. The well is used by livestock herders but frequently runs dry. It is tapping shallow - and probably perched - groundwater and is not believed to have any connection with the main aquifers of the area.
- TEJI1000 (also known as the Tejiat well): This well is owned and operated by neighbouring mining operations. It was installed as part of a communities program and is used to supply livestock herders and their animals. Construction details are uncertain but it is believed to be tapping the same aquifers as the proposed wellfield.

The Arlit and Akokan boreholes are tapping the Tarat aquifer and are by far the most significant groundwater abstractions in the region.

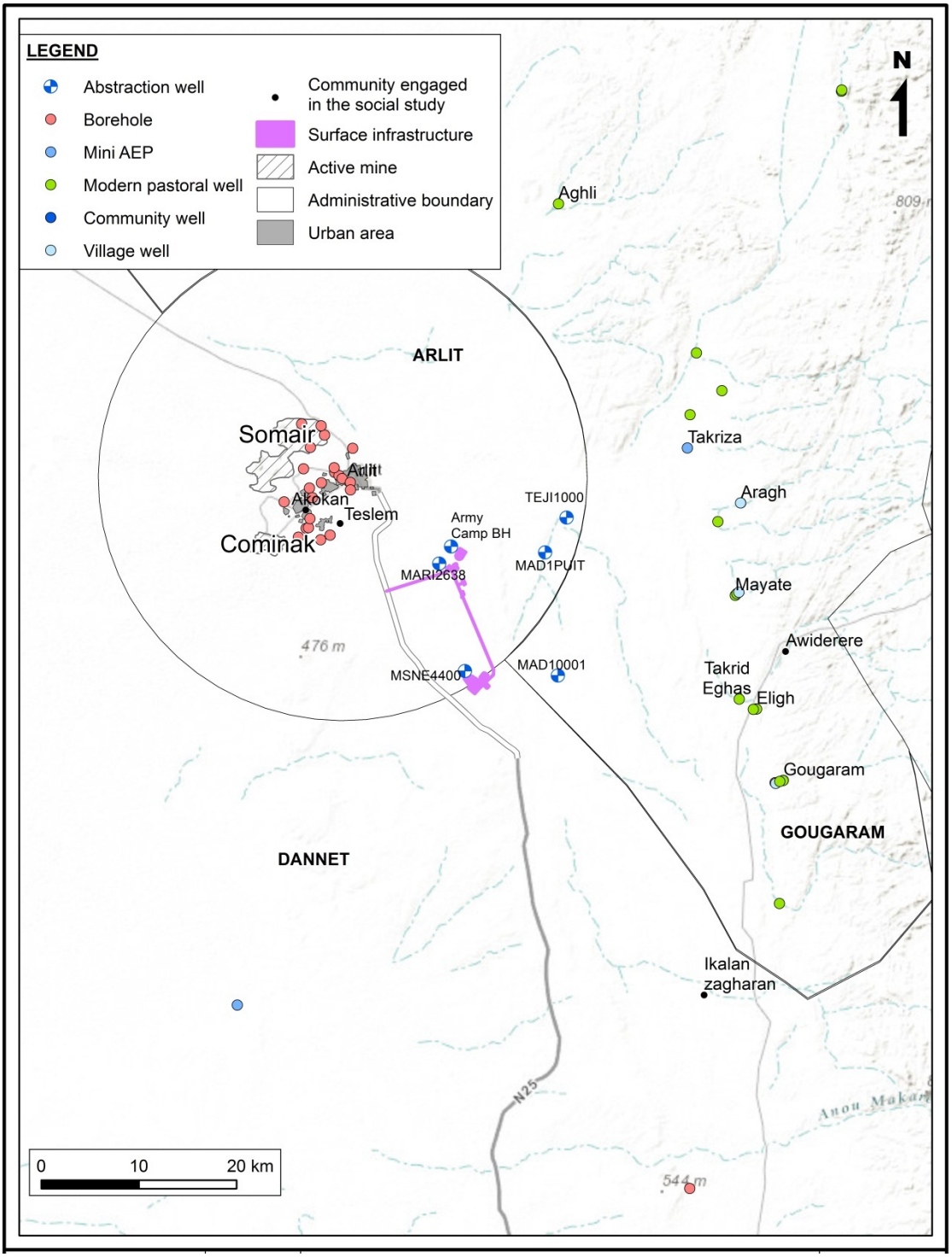


Figure 23-33: Groundwater abstraction well locations

Hydraulic properties laboratory testing

A HQ core hole (TSRK0038) was completed to the base of the Guezouman in the wellfield area east of the Madaouela fault during January 2013. The purpose of the hole was to:

- Confirm the properties of individual aquifer units from water table to base of Guezouman; and
- Provide core for laboratory testing of hydraulic properties testing (vertical hydraulic

conductivity; horizontal hydraulic conductivity, porosity, specific yield).

The hole is located approximately 7.5km east north-east of the MSNE deposit at 349689 E; 2060788 N, and is also known by hole ID MAD10279. A total of 22 samples were collected for testing, the details of which are summarised in Table 23-24 (the colours in the table represent the different formations intercepted).

Table 23-24: Samples collected from hole TSRK0038 for hydraulic properties testing.

Sample No.	From	To	Formation
Hydro-01	231.47	231.70	TARAT
Hydro-02	239.23	239.51	TARAT
Hydro-03	250.80	251.04	TARAT
Hydro-04	264.33	264.67	TARAT
Hydro-05	281.50	281.80	TARAT
Hydro-06	288.07	288.27	TCHI-UTT
Hydro-07	298.10	298.46	TCHI-BV
Hydro-08	299.24	299.54	TCHI-GB
Hydro-09	306.98	307.23	TCHI-GB
Hydro-10	312.00	312.35	TCHI-BL
Hydro-11	326.04	326.41	TCHI-BL
Hydro-12	338.65	339.00	ALT-GUEZ
Hydro-13	345.00	345.79	GUEZ-SL
Hydro-14	349.86	350.19	GUEZ-SL
Hydro-15	363.00	363.38	GUEZ-SL
Hydro-16	368.57	368.81	UA
Hydro-17	368.47	368.65	UA
Hydro-18	126.33	126.64	ARLIT
Hydro-19	147.52	147.80	MADAOUELA
Hydro-20	179.12	179.40	MADAOUELA
Hydro-21	57.50	57.80	IZEG
Hydro-22	101.50	101.80	IZEG

The first round of testing was carried out in May 2013 and focussed on the main aquifers of interest, the Tarat and the Guezouman. The results of the hydraulic testing are shown in Table 23-25. The core samples tested are shown in Figure 23-34.

Table 23-25: Summary of hydraulic properties data for a selection of samples from TSRK0038

Sample number	Summary lithology	Lithology	Approx depth	Gas permeability	Hydraulic conductivity	Hydraulic conductivity	Minipermeability plug end	Grain density	Porosity	Sy
				(mD)	(m/d)	(m/s)	mD	(g/cm3)	(%)	
H12	Laminated silt	Guezouman	339	0.029	1.87E-05	2.17E-10	<0.05	2.64	9.52	0.01
H13	Sst (speckled)	Guezouman	345.3	0.098	6.29E-05	7.28E-10	<0.05	2.66	14.38	0.01
H14	Sst	Guezouman	350.2	76.1	4.89E-02	5.66E-07	40	2.65	12.17	0.07
H15	Sst	Guezouman	363.4	182	1.17E-01	1.36E-06	55	2.66	14.73	0.07
H03	Sst-grit	Tarat	250.8	128	8.23E-02	9.53E-07	100	2.65	13.91	0.08
H03	Sst-grit	Tarat	251	136	8.75E-02	1.01E-06	102	2.65	12.02	0.07
H05	Fine sst	Tarat	281.8	0.042	2.68E-05	3.10E-10	N/T	2.59	7.6	N/T

1. Hydraulic conductivities were calculated from gas permeability
2. Specific Yield was calculated using the centrifuge method. All samples were run for 2 hours. The low Sy values (H12 & H13) are potentially overestimated due to surface draining

Figure 23-34: Photograph of core samples tested in the BGS laboratory

A series of mini-permeameter readings were also conducted on the four Guezouman samples and the H03 sample from the Tarat. A mini-permeameter is a gas-based device for measuring permeability in porous rocks. The results of these tests are shown in Figure 23-35. The mini-permeameter is a less accurate instrument than the gas permeameter, but it does enable a rapid evaluation of the variability in permeability along a length of core. The tests indicated permeability was within the same order of magnitude across 3 of the 4 Guezouman samples, with H14 showing a significant reduction in permeability towards the lower end of the core sample. The Tarat sample (H03) was more variable with permeability ranging from 6.23 to 117.8 mD across the sample.

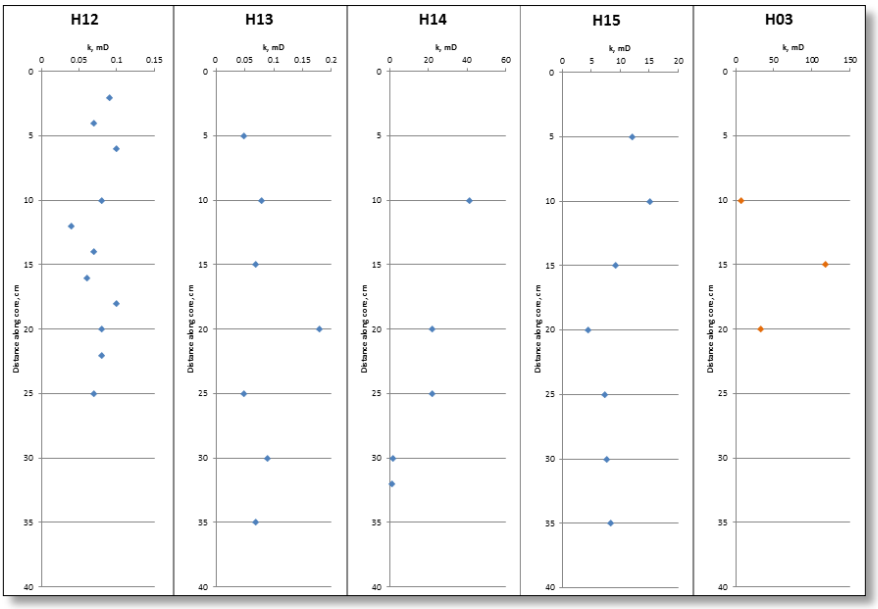


Figure 23-35: Mini-permeameter readings four Guezouman samples and the Tarat H03 sample

Additional pumping test program

Wellfield pumping test programme

A total of three production wells east of the Madaouela fault were planned to support the water supply study. The first borehole (MAD1_0003) was located close to the Madaouela fault and achieved a yield of only 13 m^3/hr with a drawdown of 80.44 m, substantially less than the 40 m^3/hr planned. This low yield was attributed to it being too close to the Madaouela fault where permeability is known to be lower.

The second borehole (MAD1_0025) situated at the northern limit of the wellfield recorded a yield of 65 m^3/hr with approximately 38 m drawdown during the short duration step test. The third borehole (MAD1_0103) was also successful. Measured yield was approximately 65 m^3/hr with approximately 50 m drawdown during the short duration step test.

The currently available information suggests that boreholes yields of more than 40 m^3/hr are achievable in the proposed wellfield area.

Expanded open hole pumping test survey

Additional to the open hole water surveys a total of 41 open holes (38 from the original survey and 3 from the November 2013 survey) were identified as targets for short-term pumping tests with the GoviEx MP1 sampling pump. Successful tests were completed on 19 of these holes. 18 holes were found to be blocked, and a pump failure occurred during one test. The remaining three holes were low priority targets and were not tested.

As is typically the case with tests conducted on uncased and undeveloped holes, a series of difficulties were encountered due to hole collapse and residual drilling muds remaining in the holes. Despite these set-backs, the tests have successfully identified high yielding zones that warrant further testing.

A summary of results from the successful tests is presented in Table 23-26.

Table 23-26: Summary of results from MP1 pumping tests

HoleID	Pumping Rate (l/s)	Drawdown (m)	Specific Drawdown (l/s/m)	*Indicative discharge for 30m drawdown (l/s)**
MARL3627	0.20	2.98	0.07	2.04
MARL3078	0.17	9.6	0.02	0.54
MSNE4849	0.26	10.76	0.02	0.74
MSNE4858	0.27	7.15	0.04	1.11
MSNE4853	0.25	7.27	0.03	1.04
MSNE4069	0.21	5.54	0.04	1.12
MALA856	0.24	7.25	0.03	1.00
MALA851	0.21	5.52	0.04	1.16
MSNE4073	0.27	0.87	0.32	9.48
MSNE4534	0.23	2.85	0.08	2.39
MSNE4557	0.20	7.29	0.03	0.81
MSNE4138	0.20	8.74	0.02	0.68
MSNE5445	0.12	3.16	0.04	1.16
MARI2668	0.16	0.21	0.78	23.41
MARI3076	0.18	2.8	0.07	1.97
MARI2762	0.07	4.77	0.01	0.45

HoleID	Pumping Rate (l/s)	Drawdown (m)	Specific Drawdown (l/s/m)	*Indicative discharge for 30m drawdown (l/s)**
MARI2144	0.12	9.45	0.01	0.38
MARI3054	0.17	3.4	0.05	1.53
MARI3053	0.15	4.94	0.03	0.93

* These figures are purely to illustrate the potential discharge rate if the holes are completed as pumping wells. The figure assumes the specific drawdown is linear as pumping rate increases. This is likely to be optimistic so the figures must be treated with caution, but the values does give a reasonable order of magnitude estimate of potential well yield.

The data presented in the final column of Table 23-26 suggests the majority of holes tested are comparatively low yielding which is consistent with previous results of the pumping tests in the Guezouman. Yields within the Guezouman range from 0.46 to 3.6 l/s. This is substantially lower than the historical dewatering well at Marilyn, which yielded 14 l/s. By comparison the two high yielding holes drilled in the wellfield east of the Madaouela fault (MAD1_103 & MAD1_025) have yields in the order of 20 l/s.

The two exceptions to this are holes MSNE4073 and MARI2668. These holes appear to be tapping high permeability zones within the Guezouman.

23.2.4 Conceptual groundwater model

A conceptual groundwater flow model is an idealised picture of a groundwater system, which is developed from interpretation of available groundwater data (including water quality), and is typically based on a simplification of the true groundwater system. At a given scale, the hydrogeological system can be broken down into four fundamental components that governs the behaviour of the groundwater system:

- The hydraulic properties of the geological units, which governs their ability to transmit and store groundwater
- Interactions with the adjacent groundwater systems (i.e. boundary conditions)
- The recharge to the groundwater system governing the amount of water entering the groundwater system
- The discharge mechanisms, which includes both natural discharge and groundwater abstraction

Conceptual models are typically developed as a precursor to numerical flow modelling. The conceptual model is used to provide a summary of the current interpretation of the groundwater flow systems in the area. The numerical groundwater model is constructed based upon this conceptual model and then used as a tool to prove or disprove the concepts outlined. The process is somewhat iterative in the fact the conceptual model should be continuously revisited and revised to reflect the findings of the numerical model.

Hydraulic Properties

Groundwater flow is primarily controlled by the both lithology and structure, as is typical for sedimentary groundwater systems. The majority of groundwater flow occurs within the coarse sedimentary sandstones. The major faults, including the Arlit and Madaouela faults, result in discontinuity within the high permeability sandstone beds and form low permeability barriers within the regional groundwater system.

The interbedded sandstone and shale sequence leads to a highly anisotropic groundwater system with hydraulic conductivity that is several orders of magnitude higher parallel to

bedding than it is perpendicular. The bedding planes are typically shallow dipping so this results in high horizontal hydraulic conductivity and low vertical hydraulic conductivity. This permeability contrast gives rise to limited vertical groundwater flow and strong vertical hydraulic gradients.

The absence of vertical connectivity also results in strong vertical water quality gradients with depth. There is limited throughput in the deeper groundwater units which subsequently show increased dissolved solids content due the longer resident times.

The low permeability nature of the major regional structures (the Arlit and Madaouela faults) is confirmed by available field data. It is supposed that vertical hydraulic conductivity is enhanced, and horizontal conductivity is reduced around the smaller post-sedimentary structures although this is yet to be confirmed by field data. Syn-sedimentary structures can lead to zones of higher hydraulic conductivity where they result in a comparatively high energy depositional environment and hence zones of courser grained sediments. It is possible this situation arises towards the base of the Guezouman in close proximity to the UA channels at a Marianne, Marilyn and MSNE although this is yet to be confirmed.

The impact of exploration and mining activities on hydraulic properties can also be significant. Old exploration boreholes will result in dramatic increases in vertical hydraulic conductivity, and the mine workings themselves will also alter the hydraulic properties.

The available field data has been used to characterise the hydraulic properties of the major lithological units in the Project area, and this information is summarised in Table 23-27.

Table 23-27: Summary of hydraulic properties by stratigraphic unit

Unit	Description	Hydraulic Conductivity Range (m/s)	Storage Properties
Moradi, Tamamalt and Tejia	These units have not been considered in any detail as they are typically absent across the Project area or within the unsaturated zone. They do not constitute viable aquifer, and where they are present they are assumed to have the same properties as the underlying Izegouande.	N/A	N/A
Izegouande	Generally considered to be a poor aquifer comprising interbedded sandstones and shales. It is described as series of perched aquifers and “water pockets” in the Akouta EIA. Sandstone units becoming more abundant towards the base (SOMAÏR EIA). The GoviEx spinner testing programme has indicated that there is significant permeability with this unit, although the tests are not of sufficient duration to confirm whether the permeability is extensive or localized.	$1.22x10^{-08}$ to $1.65x10^{-06}$ (Akouta EIA, pumping tests) $7.5x10^{-07}$ to $1.2Ex10^{-06}$ (SOMAÏR EIA, pumping tests)	Specific Storage of 8.6×10^{-6} m^{-1} calculated from pumping test result in SOMAÏR EIA

Unit	Description	Hydraulic Conductivity Range (m/s)	Storage Properties
Madaouela	Considered a poor aquifer of interbedded sandstones and shale, similar to the Izegouande in terms of hydraulic properties, although spinner testing completed to date by GoviEx suggests it may be slightly less permeable.	3.5×10^{-6} (GoviEx pumping test)	Specific Storage of 3.7×10^{-6} m^{-1} calculated from GoviEx pumping test.
Tarat	Principal sandstone aquifer in the area. The SOMAÏR and Akouta EIA's describe it as the only exploitable aquifer to meet the water supply requirement for those operations. A decreasing permeability from the north to the southeast is noted in the Akouta EIA. Pumping test and spinner test results from the GoviEx water supply investigations also suggest it is likely to be the most productive aquifer to the east of the Madaouela fault.	$1.00x10^{-06}$ to $8.00x10^{-04}$ (Akouta EIA, pumping test) $1.95x10^{-07}$ to $4.5x10^{-04}$ with most values ranging from $5.0x10^{-06}$ to $5.0x10^{-05}$ (SOMAÏR EIA, pumping tests) $2x10^{-6}$ (GoviEx pumping test, single test close to Madaouela fault; higher permeability encountered in subsequent holes - analysis pending)	Specific Storage of $7.5x\ 10^{-6}$ m^{-1} calculated from GoviEx pumping test. Specific Yield of 15% inferred from Akouta EIA
Tchinezogue	Comprises silts, fine sandstones and clays, and is an aquitard. It is considered that there may be some permeability parallel to bedding but vertical permeability is negligible.	No data	
Guezouman	Comprises a fining upward sequence of conglomerate, sandstones and some shales. Zones of elevated permeability at the base of the unit were encountered during exploration mining at Marilyn. Considered a relatively poor aquifer on a regional basis but localized pockets of high permeability do exist.	$2x10^{-5}$ – $3.6x10^{-5}$ (AEC pumping tests; Marilyn area) $1.7x10^{-9}$ – $3.2x10^{-7}$ (Legeni packer testing report; Marilyn area) $4.9x10^{-06}$ to $1.7x10^{-05}$ (Akouta EIA, pumping tests) $1.3x10^{-07}$ to $3.0x10^{-05}$ (SOMAÏR EIA, pumping tests) $4.4x10^{-7}$ to $8.4x10^{-6}$ (GoviEx pumping test)	Specific Storage ranges from $4.2x\ 10^{-7}$ to $3.2x10^{-6}$ m^{-1} calculated from GoviEx Y pumping tests. Specific Yield of 7% inferred from Akouta EIA
Unite Akokan	The Unite Akokan (UA) is conformably deposited on the Talak argillites. It is composed of medium grain feldspathic sandstone, developed to the north of the basin, and grading toward sandstone and clay to the south. The UA channel in the Marianne-Marilyn area is described as thinly bedded mudstones and siltstones, suggesting comparatively low permeability.	No data	

Unit	Description	Hydraulic Conductivity Range (m/s)	Storage Properties
Talak	Comprises a argillic rich sequence and forms the basal aquitard to the local aquifer system.	No data	

Boundary Conditions

For the purposes of this study the Talak is assumed to form an impermeable basal unit to the regional groundwater system. This assumption is reasonable as the deeper Farazekat aquifer, which sits below the Talak, is not exploited in the area and is largely disconnected from the overlying aquifer systems due to the low hydraulic conductivity of the Talak. The Talak outcrops to the north and east of the project area, and is also outcropping between M&M and the Madaouela fault. The Arlit fault is taken as the western limit of the model area as it is known to form a barrier to groundwater flow based on the experiences of the COMINAK operation. The southern extent of the groundwater system in the Madaouela is not well constrained as the principal aquifers continue to plunge to the south.

Groundwater Recharge & Discharge

Historical Recharge & Natural Discharge Mechanisms

Groundwater recharge and discharge mechanisms in the Sahara are complex due to the fact present day recharge rates are negligible, but discharge of fossil groundwater continues to occur. This results in the absence of a natural equilibrium between groundwater recharge and discharge. It is therefore necessary to consider how the groundwater systems may have evolved since the onset of desertification during the mid-Holocene (around 5,500 years ago).

Previous studies have indicated that groundwater within the Tarat aquifer around Arlit can be attributed to humid periods around 10,000, 25,000 and 30,000 years ago (Dodo & Maria Zuppi, 1999). The most recent recharge event corresponds with the African Humid Period, which continued from around 14,500 to 5,500 years before present. During these humid periods groundwater recharge rates would have been high. Groundwater elevations would also have been higher and would likely have resulted in the discharge of groundwater to perennial river systems during that period.

With the onset of desertification the groundwater recharge rates have reduced to negligibly low levels across the majority of the Project area. The natural groundwater gradients that were observed before the major abstractions were likely to be partially driven by historical groundwater recharge. Prior to the initiation of dewatering and water supply abstraction in the 1960's evaporation of groundwater was the primary groundwater discharge mechanism. This evaporation from groundwater gives rise to upwards hydraulic gradients from the deeper groundwater units to the shallow units, and lateral groundwater flow towards areas of outcrop.

Present Day Recharge

The only potential source of groundwater recharge at present is considered to be from infiltration through dry stream beds following intense storm events. There is greatest potential for this to occur to the east of the Project area where the runoff from the Air Mountains collects in the dry river channels. There is no available data on the magnitude or frequency of these recharge events, and for the purposes of groundwater resource evaluations recharge is conservatively assumed to be zero.

Direct recharge from infiltration of precipitation is unlikely to occur as soil moisture deficits are so high that infiltrating water is retained as soil moisture and then evaporated back to the atmosphere. The proliferation of vegetation that is observed along stream channels during the wetter months would also suggest that much of the infiltration through dry stream channels is also lost to evaporation. There is no evidence of hydraulic gradients away from stream channels in any of the available data which supports the assumption of negligible or zero recharge.

Groundwater Abstraction

Groundwater discharge in the Project area is now dominated by the activities of the SOMAÏR and COMINAK operations and the associated water supplies to the towns of Arlit and Akokan. The majority of the abstraction is from the Tarat aquifer. This has led to a reversal in vertical groundwater gradients with downward vertical gradients now being observed from the Izegouandane to the Tarat around the SOMAÏR mine.

Drawdown within the Tarat has also extended to the south and east of Arlit, extending into the Project area. Drawdown in the Tarat is estimated at 40 m immediately to the west of Miriam (compared to over 100 m in the vicinity of the COMINAK mine). The extent of drawdown further south is yet to be confirmed.

Drawdown within the Tarat will have led to increases in the rate of discharge from Guezouman into the overlying Tarat, and may be responsible for depletion of groundwater within the Guezouman. Current estimates suggest the Guezouman has experienced about 10 m of drawdown on average, and possibly around 20 m in the Marilyn area. Local abstractions (historical dewatering at Marilyn and the army camp water supply well) are also responsible for the observed drawdown but increased rates of leakage from the Guezouman into the overlying Tarat is also likely to have an influence.

The principal objective of the conceptual and numerical modelling is to determine the impact GoviEx will have on the regional groundwater system, and what would the implications be for the GoviEx operations and other groundwater users in the area.

- Dewatering of the Guezouman has the potential to reverse groundwater flow directions between the Guezouman and the Tarat. The extent of the hydraulic connection between the Guezouman and the Tarat is thought to be limited but this is the subject of on-going evaluation.
- Water supply abstractions to the east of the Madaouela fault will result in drawdown in the aquifer systems in that area. The low permeability nature of the Madaouela fault means it is extremely unlikely any impacts will be felt on the western side of the fault.

Conceptual Model Map & Sections

A series of plans and sections have been drawn to describe the conceptual groundwater models under natural conditions, current conditions, and during future GoviEx operations.

Figure 23-36 shows the regional groundwater flow directions under natural conditions and is based on groundwater level data collected during the 1960's. Groundwater flow directions are towards to the northwest and are driven by evaporation from the low lying outcrop areas around Arlit. Evaporative flux is also assumed to occur from the Moradi, Tamamalt and Tejia units where depth to groundwater is low (i.e., in the lower lying areas). It is assumed that there is relatively little evaporative flux in the Madaouela area as surface elevations are higher in that area and depth to groundwater is therefore greater. SRK has no data from east of the

Madaouela fault for this period but it is assumed that a similar pattern of northerly flow occurs, with negligible flow across the Madaouela fault.

Figure 23-37 presents a regional 3D overview of Marianne/ Marilyn, whilst Figure 23-38 shows a cross section (labelled A-A') from east of the Madaouela fault, extending through Marianne/Marilyn to COMINAK and the Arlit fault in the west. This cross section also closely characterises the MSNE & Maryvonne deposits located south of Marianne and Marilyn. Under natural conditions there is relatively little lateral groundwater movement along this section with the principal groundwater fluxes being perpendicular to the section, towards the outcrops in the north. There are strong vertical gradients from the deeper layers to the near surface layers but the flux is very low due to the low vertical hydraulic conductivity.

Figure 23-36b shows the present day regional groundwater flow directions, with the impact of the SOMAÏR and COMINAK abstractions clearly evident. Figure 23-39 shows the present conceptualisation of groundwater flow for section B-B' (shown in Figure 23-36b). The drawdown within the Tarat is shown to result in downward seepage from the overlying Madaouela and Izegouande deposits and increased rates of upward flow from the Guezouman to the Tarat. The influence of the abstractions in the Marilyn area are also evident within the Guezouman. The impact on water levels east of the Madaouela fault is minor although some leakage across the fault is occurring in the Miriam area due to drawdown within the Guezouman.

Figure 23-36c shows a plan of projected groundwater flow during GoviEx operations. The drawdown associated with the GoviEx wellfield results in a change in groundwater flow directions east of the fault. The impact of dewatering from GoviEx mining areas is also evident, although the dominant control on groundwater movement to the west is still the abstractions around COMINAK and SOMAÏR.

Figure 23-40 shows the conceptualisation of groundwater flow on the section line C-C' (presented in Figure 23-36c). There is some flow reversal between the Guezouman and the Tarat, and significant drawdown within the Guezouman itself. The water levels within the Tarat continue to be controlled by the abstractions from COMINAK and SOMAÏR due to the limited connection between the Guezouman and Tarat aquifers. The impact of wellfield abstractions within the Tarat to the east of the Madaouela fault are also shown on this figure, with vertical gradients now resulting in flow towards the Tarat aquifer. There is no communication between the mining areas and the wellfield area on this section however, due to the presence of the Talak formation and the Madaouela fault itself.

Figure 23-41 presents a 3D regional overview of the proposed Miriam deposit, whilst Figure 23-42 shows cross section D-D' (presented in Figure 23-36c) through the Madaouela fault in the Miriam area. Drawdown in the Guezouman is significant which results in a major increase in the hydraulic gradient across the fault. Flow rates across the fault are however minor due to the low hydraulic conductivity of the fault zone. The impact of wellfield abstractions within the Tarat to the east of the Madaouela fault also serve to reduce hydraulic gradient (and hence the flux) across the fault. As with Figure 23-41, vertical gradients within the wellfield area now result in flow towards the Tarat aquifer.

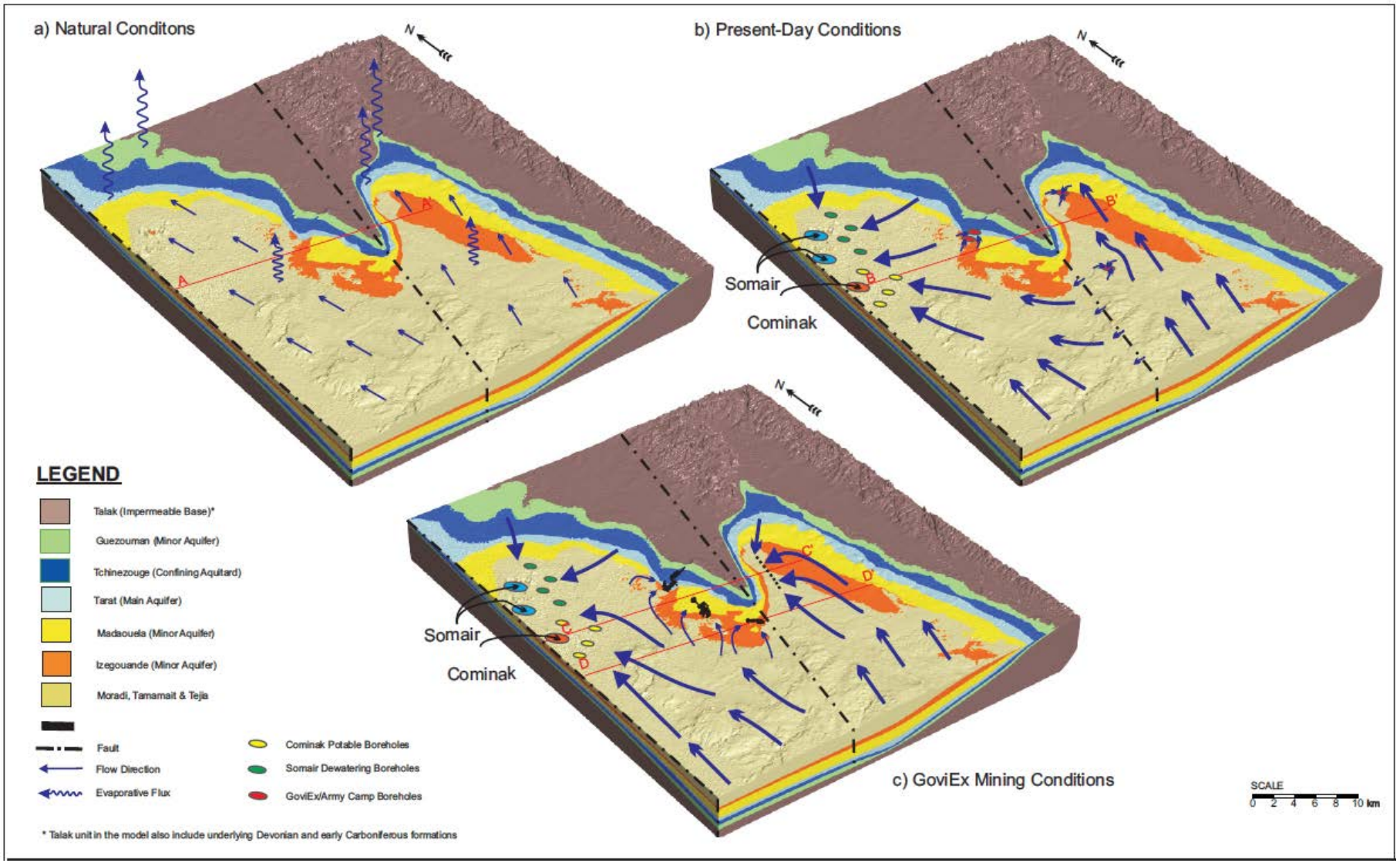


Figure 23-36: Regional groundwater directions

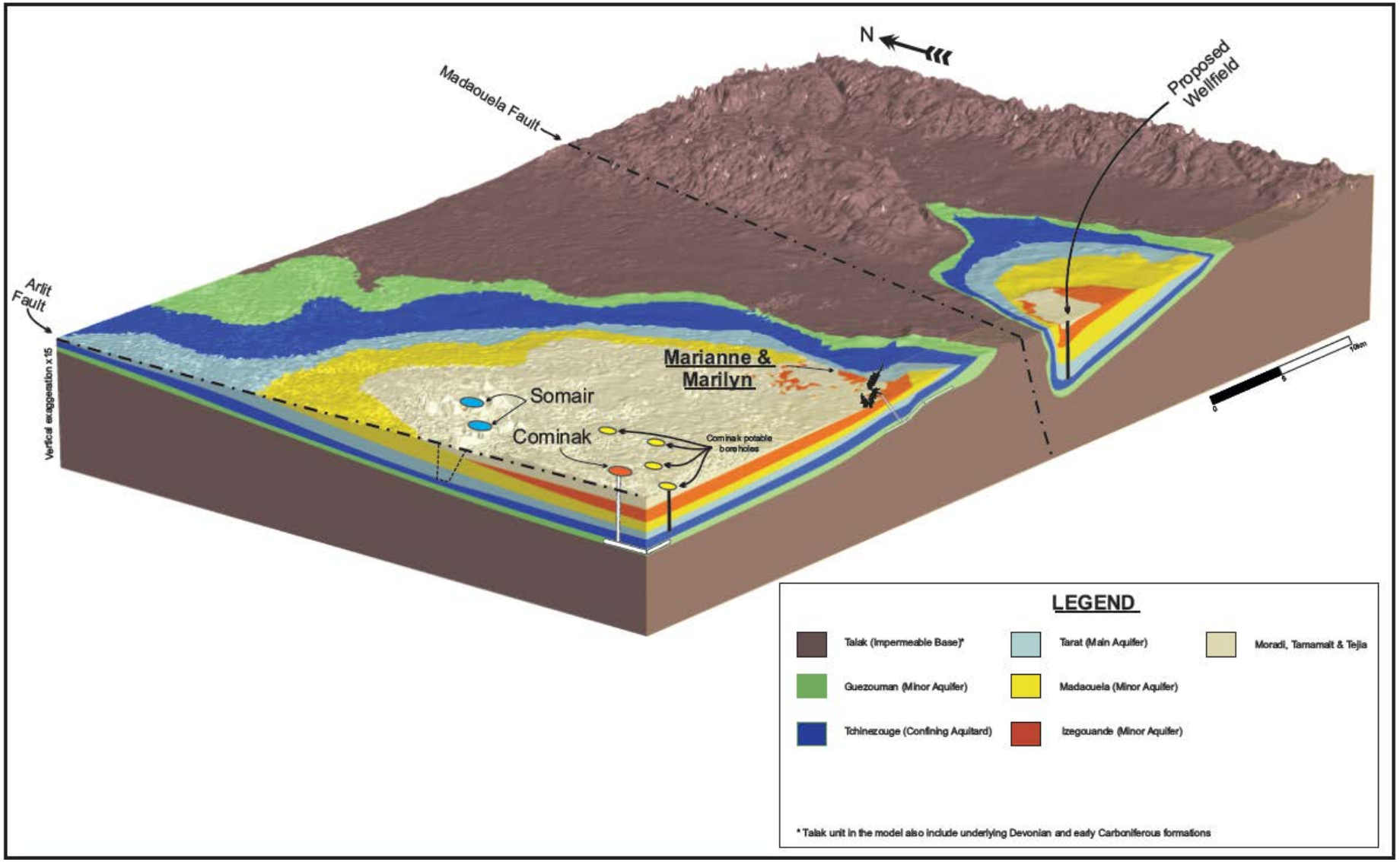


Figure 23-37: Conceptual groundwater model: 3D regional overview of Marianna and Marilyn

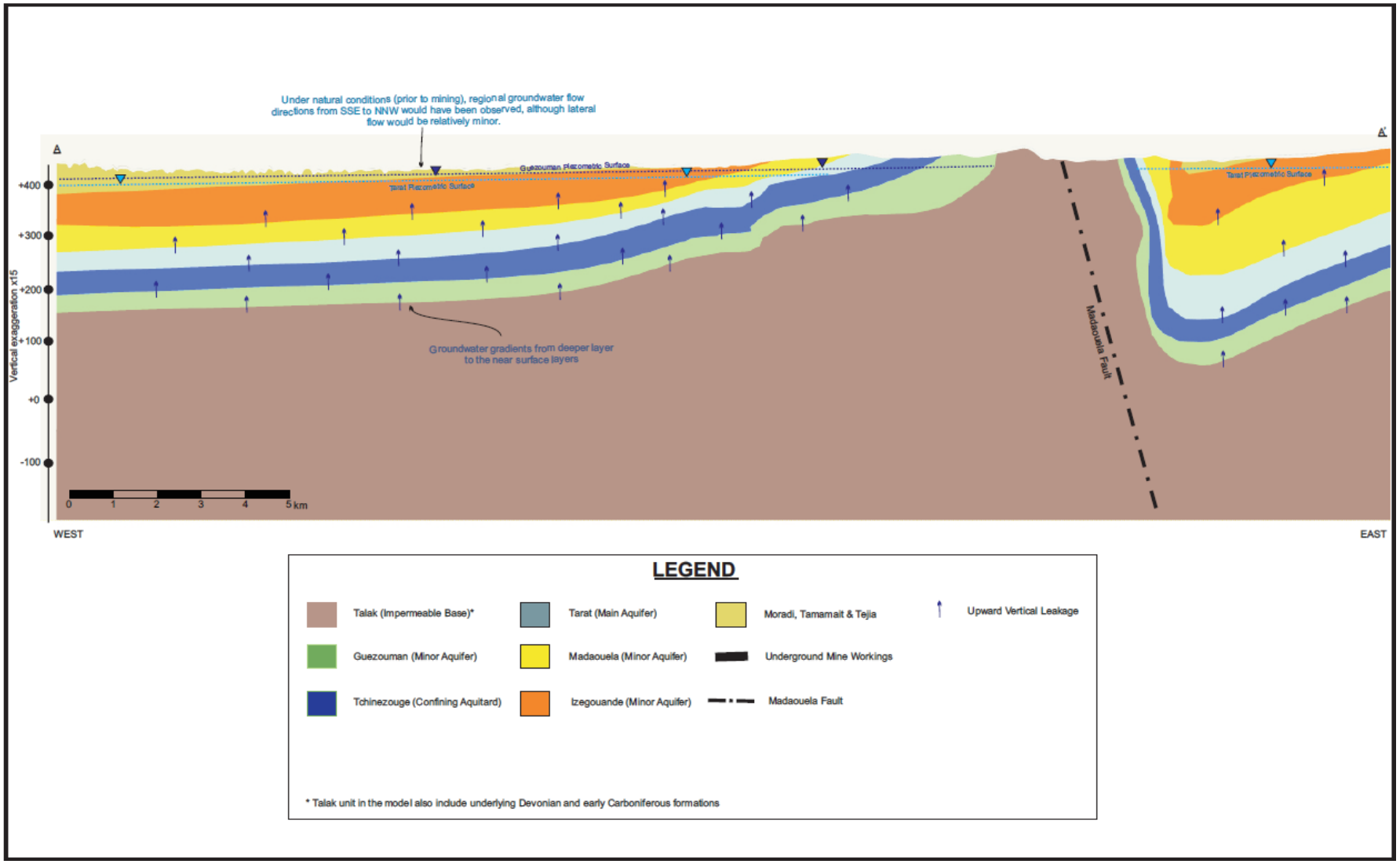


Figure 23-38: Conceptual groundwater model section A-A': Marianne, Marilyn, MSNE & Maryvonne natural conditions prior to mining (pre-1960s)

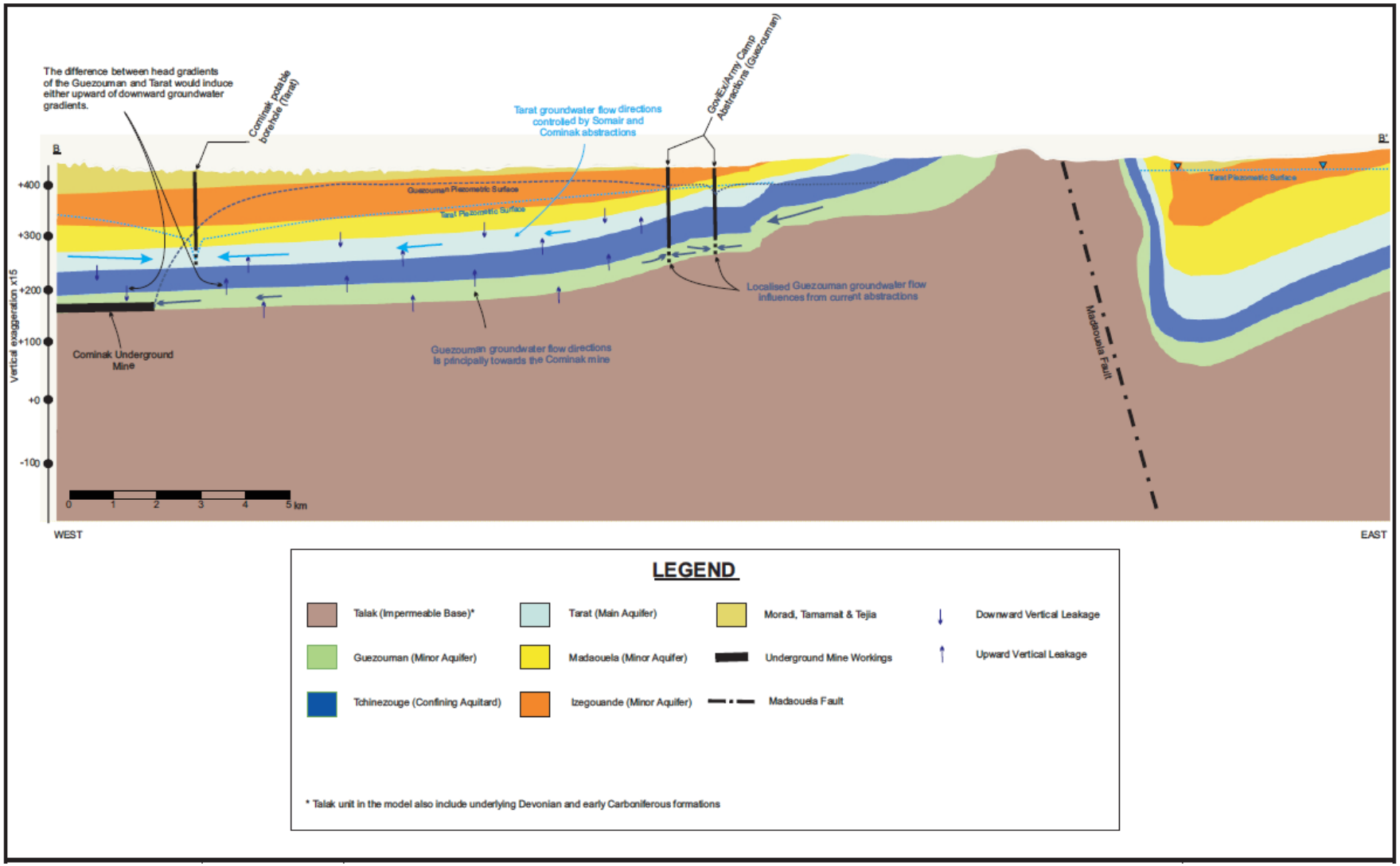


Figure 23-39: Conceptual groundwater model section B-B': Marianne, Marilyn, MSNE & Maryvonne present day conditions (2013)

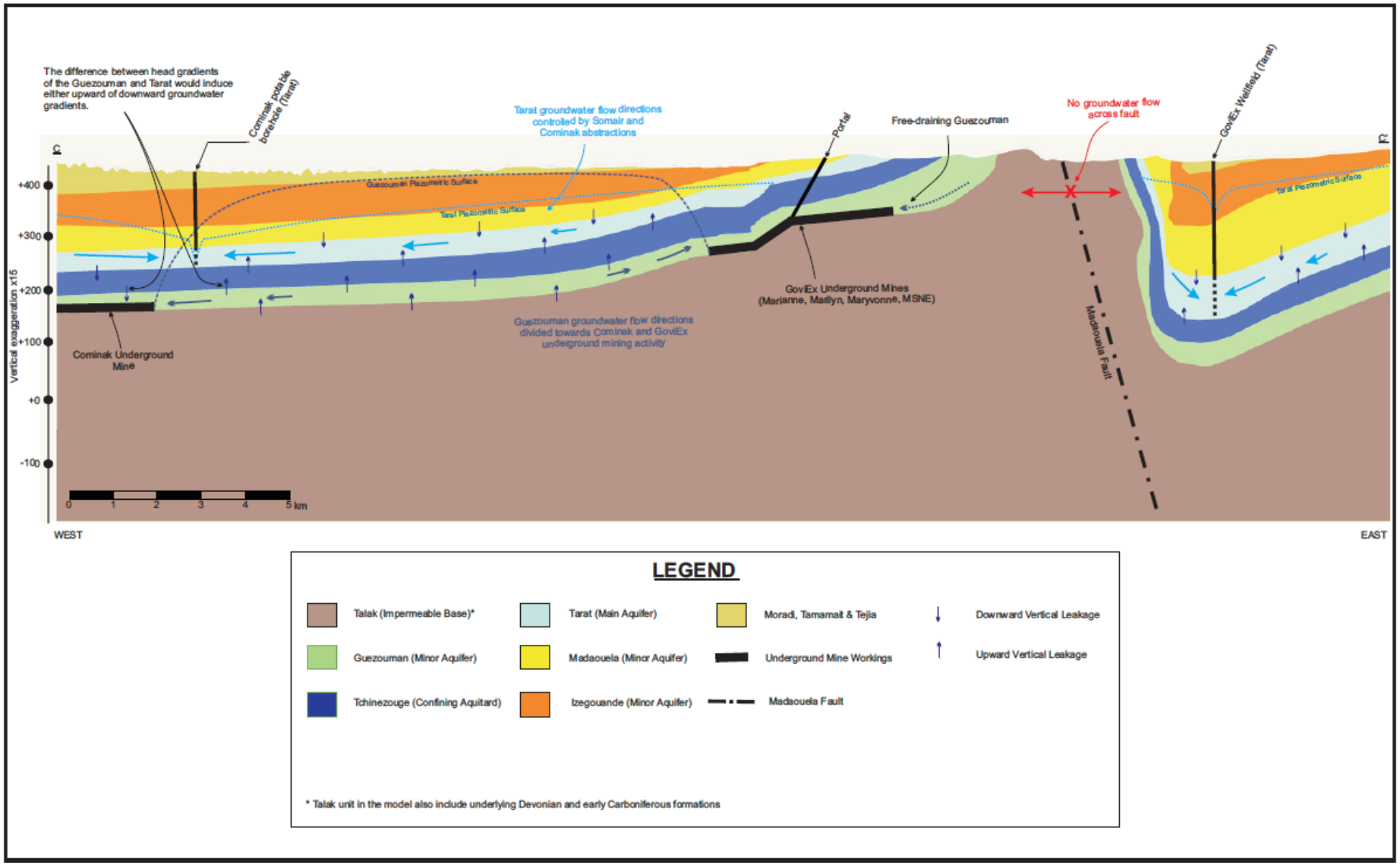


Figure 23-40: Conceptual groundwater model section C-C': Marianne, Marilyn, MSNE & Maryvonne projected post mining

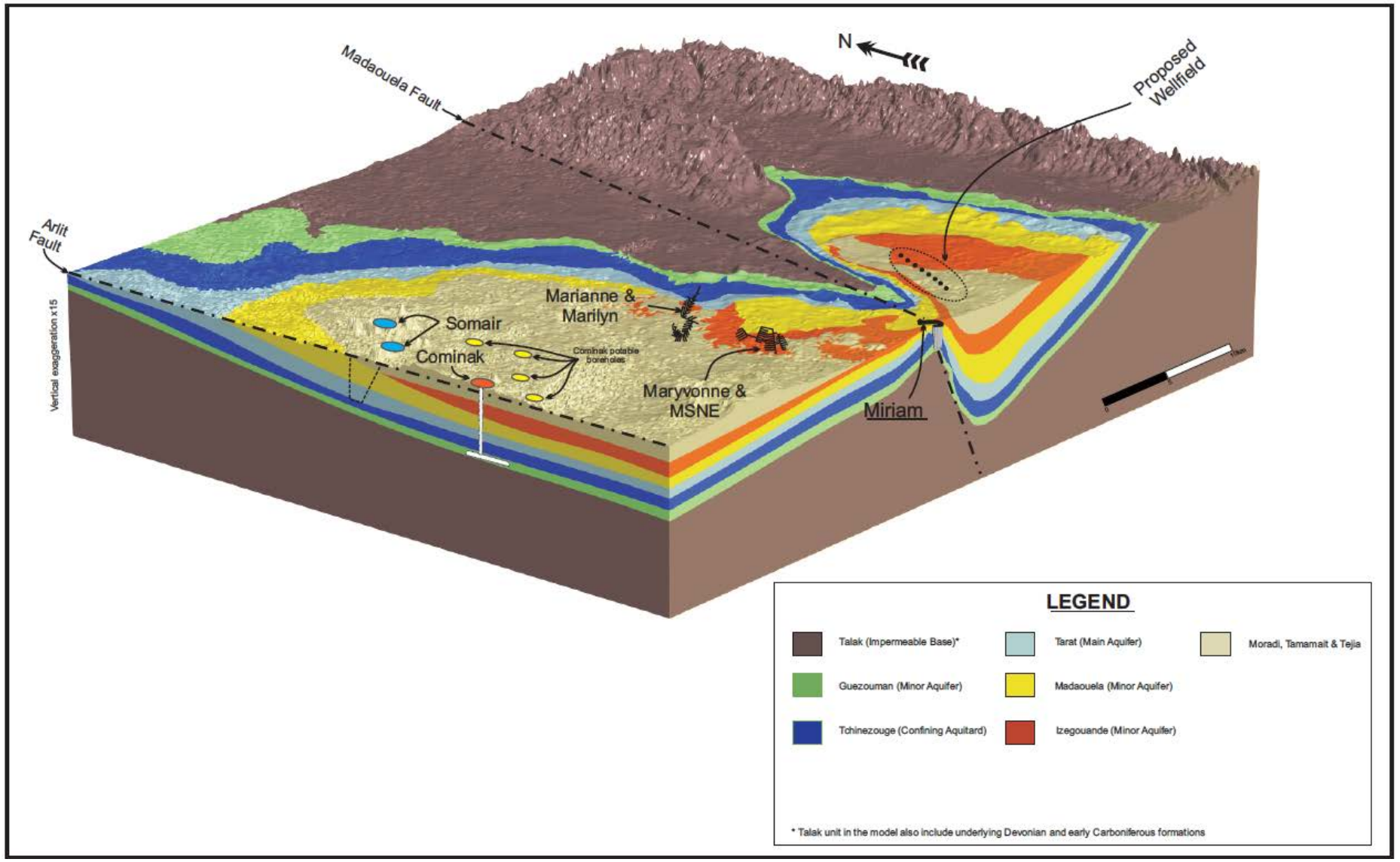


Figure 23-41: Conceptual groundwater model: MSNE, Maryvonne & Miriam 3D overview

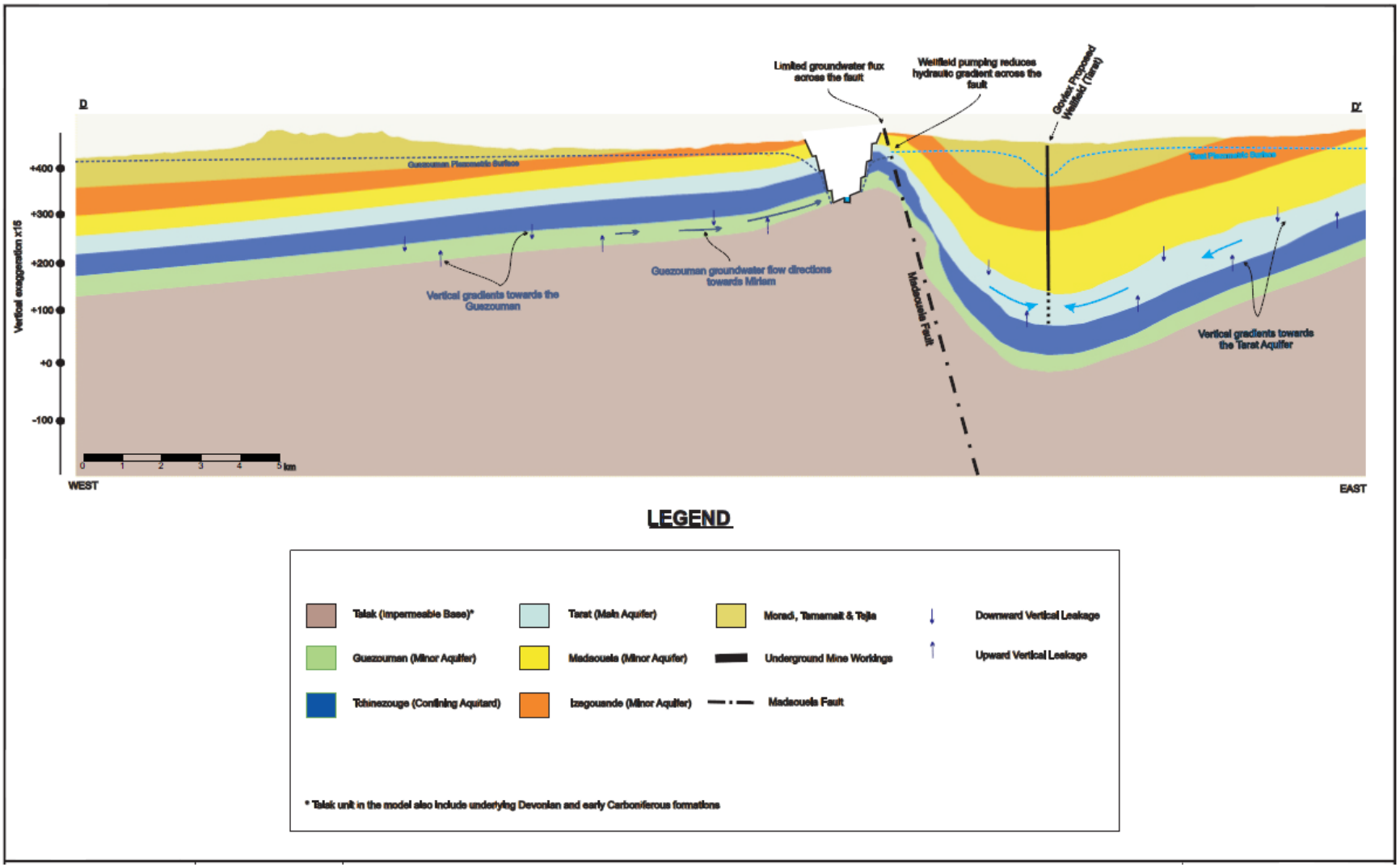


Figure 23-42: Conceptual groundwater model section D-D': Miriam post-mining cross section

23.2.5 SUMMARY

The ore horizons at Madaouela are located at the base of the Guezouman, which is an aquifer and must be dewatered to facilitate extraction of the ore. The project will operate within the Mining Code and is therefore committed not to waste water Careful planning of the dewatering programme will avoid wastage of groundwater, especially during mine development when process water demand is zero.

There is no legislative requirement to pay for abstracted groundwater. It is also understood that the exploration and mining licences come with a right to abstract groundwater within that licence area. The right to abstract groundwater is however dependent on that abstraction having no adverse impact on existing groundwater users within or adjacent to that licence area.

Groundwater flow is primarily controlled by both lithology and structure, as is typical for sedimentary groundwater systems. The majority of groundwater flow occurs within the sedimentary sandstones. The major faults, including the Arlit and Madaouela Faults, result in discontinuity within the high permeability sandstone beds and form low permeability barriers within the regional groundwater system.

Groundwater recharge and discharge mechanisms in the Sahel are complex due to the fact that present day recharge rates are negligible, but discharge of fossil groundwater continues to occur. Groundwater discharge in the project area is now dominated by the activities of the Somair and Cominak operations and the associated water supplies to the towns of Arlit and Akokan. The majority of the abstraction is from the Tarat aquifer. Drawdown within the Tarat has also extended to the south and east of Arlit, extending into the Madaouela I concession area. Drawdown in the Tarat is estimated at 40 m immediately to the west of Miriam (compared to over 100 m in the vicinity of the Cominak mine). The impact on water levels east of the Madaouela Fault is minor although some leakage across the fault is occurring in the Miriam area due to drawdown within the Guezouman.

Drawdown within the Tarat will have led to increases in the rate of discharge from Guezouman into the overlying Tarat, and may be responsible for depletion of groundwater within the Guezouman. SRK is yet to obtain accurate baseline (i.e. 1960's) groundwater elevations for the Guezouman but current estimates suggest the Guezouman has experienced about 10 m of drawdown on average, and possibly around 20 m in the Marilyn area. Local abstractions (historical dewatering at Marilyn and the army camp water supply well) are also responsible for the observed drawdown but increased rates of leakage from the Guezouman into the overlying Tarat is also likely to have an influence.

Groundwater levels in the Marilyn area are heavily influenced by groundwater abstractions. The Marilyn deposit was dewatered for a period of approximately 12 months during the 1960's and the Guezouman is still used to provide a water supply for the army camp. This had resulted in a cone of depression that is concentrated along the UA channel in the Marilyn deposit. The phreatic surface sits at approximately 390 mRL along the centre of the UA channel, and rises to between 405 mRL and 410 mRL to the north and south. The eastern extreme of the Marilyn deposit is in close proximity to the dry limit of the Guezouman; the deepest section of the Marianne deposit is approximately 80 m below water level.

The phreatic surface ranges from approximately 405 to 410 mRL across the MSNE deposit,

approximately 80 to 90 m above the elevation of the workings. Groundwater flow direction is from the southeast to the northwest in the MSNE and is influenced by the abstractions at Marilyn and the regional impact of dewatering at the Somair and Cominak mines to the northwest.

The Miriam open pit will be excavated through the Tchinezogue and Guezouman formations. Dewatering of the open pit will be required. The phreatic surface sits at approximately 415 mRL across the pit area, approximately 45 m below surface and 65 m above the final pit floor. The Tchinezogue formation comprises silts, fine sandstones and clays. The unit will yield very little groundwater although some nuisance inflows from the sandstone beds may be expected. The lower 25 m of the pit walls will comprise the Guezouman formation.

SRK has developed a 3D numerical groundwater flow model to support the dewatering and water supply evaluations. A finite-difference software package, Modflow2005 has been used for the model. The model has been used to predict LoM dewatering rates and the drawdown around each mine area through LoM, and the recovery period post-closure.

For the purposes of design it is assumed that an installed dewatering capacity of 200 m^3/hr is required to handle groundwater inflows at both Marianne-Marilyn and MSNE-Maryvonne. This means pumps capable of delivering 200 m^3/hr are installed in each area. For the purposes of OPEX calculations, average inflows of 100 m^3/hr are assumed for both M&M and MSNE-Maryvonne. An installed pumping capacity of 60 m^3/hr is assumed for the Miriam open pit operation. An average long-term inflow rate of 30 m^3/hr is assumed for OPEX calculations.

The dewatering strategy comprises a combination of dewatering boreholes (eighteen in total) and conventional centrifugal dewatering pumps for underground and open pit operations. In the medium to long-term all dewatering discharge will be pumped to the process plant for use in the process. All dewatering wells are assumed to be completed to 175 m depth and completed with 6” internal diameter borehole casing. The above-ground reticulation system has not been designed as dewatering hole locations have not been defined. A nominal 1 km of pipeline per well has been assumed. The total costs for the dewatering wellfield are calculated at USD 2.95 M. The Miriam open pit dewatering system has been calculated at USD 250 K excluding power costs.

23.3 Geotechnical

23.3.1 Introduction and Scope of Work

Geotechnical design to support the study has been completed for four deposits at the Madaouela Uranium Project. These deposits are Marianne-Marilyn, MSNE, Maryvonne and Miriam. The Marianne-Marilyn, Maryvonne and MSNE deposits are to be mined by conventional room and pillar methods. Miriam will be mined by open pit. SRK undertook geotechnical site investigation work from Q3 2012 to Q1 2013. This report details the findings from the site investigation program and the subsequent design and analysis work undertaken for the four deposits.

Geotechnical recommendations and opportunities have been made for each deposit.

23.3.2 SRK Geotechnical Field Program

Program Summary

Geotechnical site investigation work included the following:

- Oriented core, triple tube drilling of 40 geotechnical/structural holes;
- Detailed geotechnical logging of drilled core by Legeni following training and supervision by SRK;
- Acoustic Televiewer (ATV) surveys of all holes with some Optical Televiewer (OTV) surveys in selected holes above the water table;
- Point Load Testing of drilled core. Testing done both axially and diametrally continuously downhole;
- QA/QC of all data by SRK staff;
- Structural interpretation of Quickbird (false coloured satellite photographs) imagery;
- Field mapping and ground-truthing of structures inferred from Quickbird imagery and drilling; and
- Review of 3D Geology wireframes provided by GoviEx from exploration drilling.

Geotechnical Drilling Program

Drilling was undertaken by GeoDrill over a 24 hour period per day under the supervision of SRK and GoviEx staff. Holes were drilled using two truck mounted drills with HQ3 coring. Drilling mud and polymer was used to cool the drill bit and improve core recovery.

Upon completion, each hole would be flushed with water for at least one hour or until water return was clean. Holes would then be surveyed with acoustic televiewer (ATV). Holes were not surveyed for deviation since hole depths were relatively shallow.

Hole locations were sited using a hand-held GPS accurate to +/-5m. Siting of the drills and alignment of the drill mast (azimuth and dip) was undertaken by GoviEx and Legeni staff and checked by SRK staff.

Marianne-Marilyn

The Marianne-Marilyn deposit is the primary economic focus for the study. 30 geotechnical holes were drilled to understand rock mass and structural conditions across the deposit. All holes were drilled a minimum of 10 m into the Talak to ensure sufficient distance into the footwall.

Hole locations were chosen based on:

- 3D geological wireframes provided by GoviEx;
- Preliminary structural interpretation from Quickbird imagery;
- Leapfrog 250 ppm U grade shell; and
- Preliminary mine layout.

Percussion (air rotary) drilling was done from surface down to a minimum of 30 m above the ore zone. Drilling method then changed to triple tube, oriented core drilling through the hanging wall and 15 m into the footwall. This improved drilling rates. Locations of the geotechnical holes are shown in Figure 23-43.

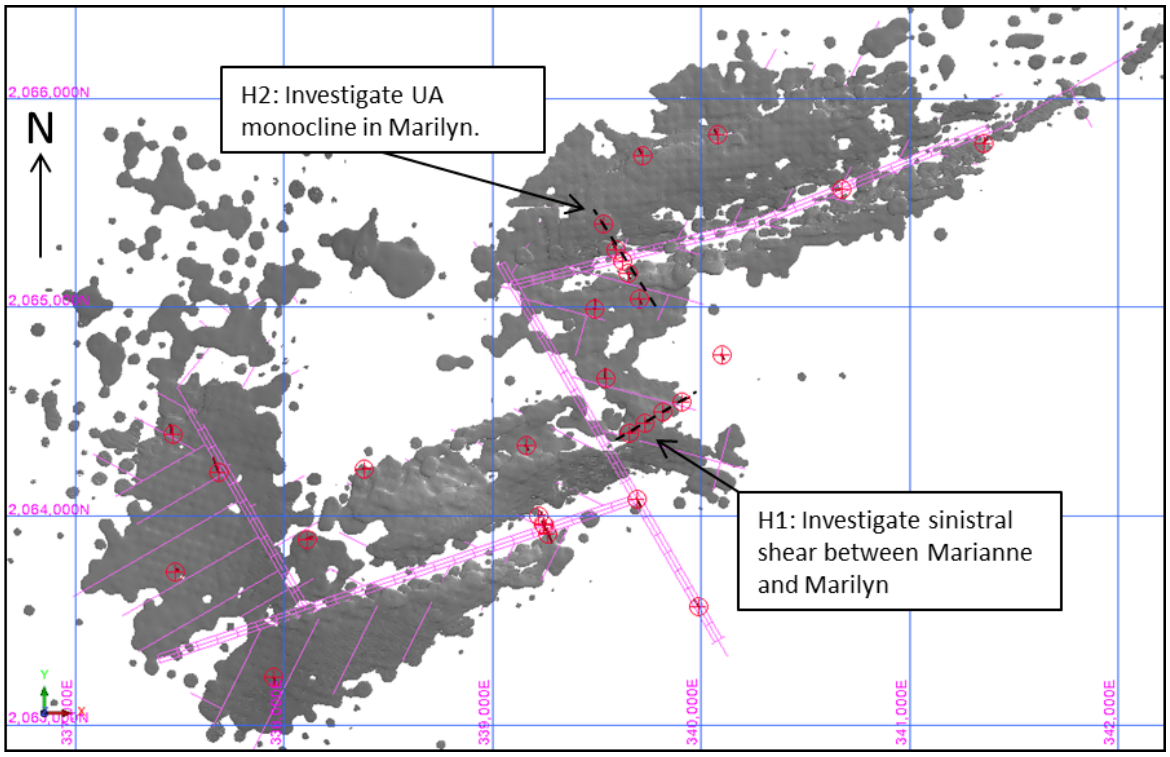


Figure 23-43: Plan view of the geotechnical holes (red) relative to the proposed mine layout at Marianne-Marilyn, and the 250 ppm U Leapfrog Grade Shell.

MSNE

Twelve holes were drilled in MSNE to understand the general geotechnical conditions. Holes were drilled a minimum of 10 m into Talak (footwall) to ensure sufficient depth past the ore zone. Hole locations were based on the structural interpretation done from Quickbird Imagery, the proposed mine layout and the Leapfrog 250 ppm U grade shell.

Hole locations relative to the Leapfrog 250 ppm U grade shell are shown in Figure 23-44.

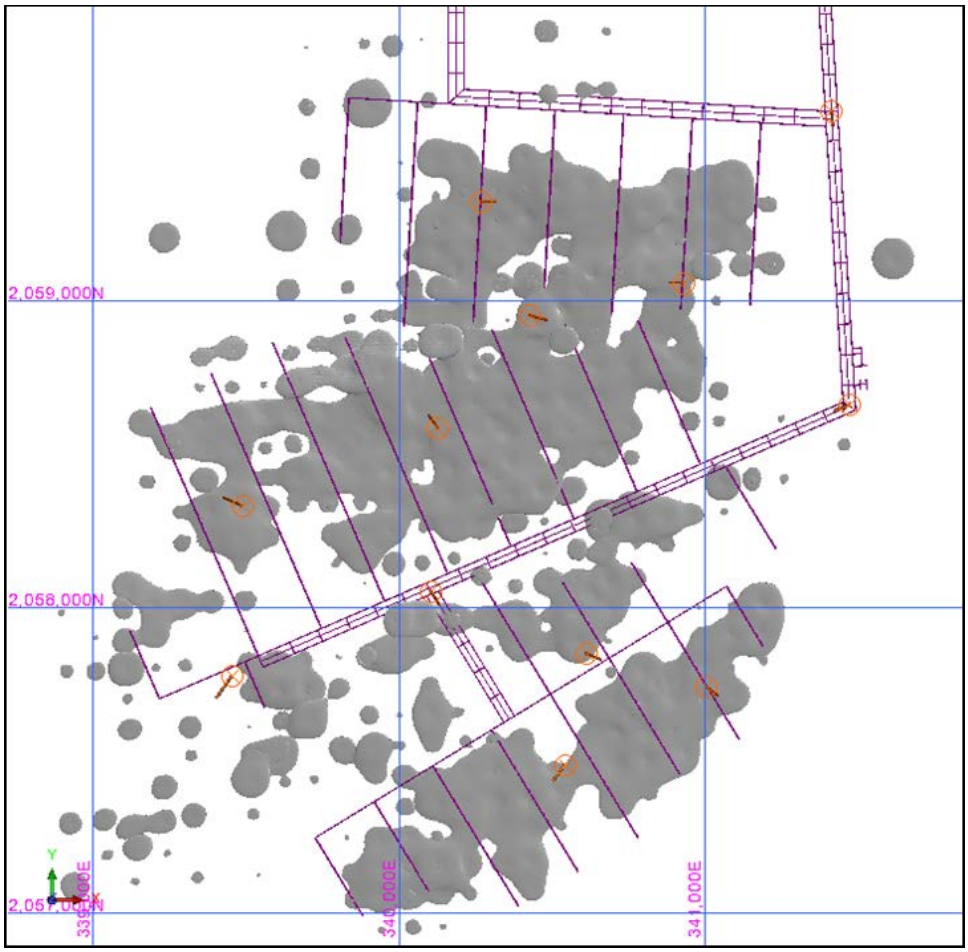


Figure 23-44: Plan view of geotechnical hole locations, proposed mine layout and the Leapfrog 250 ppm U grade shell at MSNE. Access to Maryvonne will be from the north central drive.

Miriam

A total of 12 geotechnical holes were completed in the Miriam deposit. Hole locations were based on the PEA Whittle pit shell, the 3D geological wireframes provided by GoviEx and preliminary structural interpretation from Quickbird imagery. Nine holes were focused on geotechnical variation around the pit, while three resource/structural holes were added by GoviEx to control possible shearing associated with rich vertical high grade zones. Triple tube oriented core drilling and logging was completed from surface on all holes. Since the deposit will be mined by open pit a continued geotechnical profile of the pit wall(s) is required in order to complete the design. .

Locations of the drillholes relative to the April 2013 pit shell are shown in Figure 23-45.

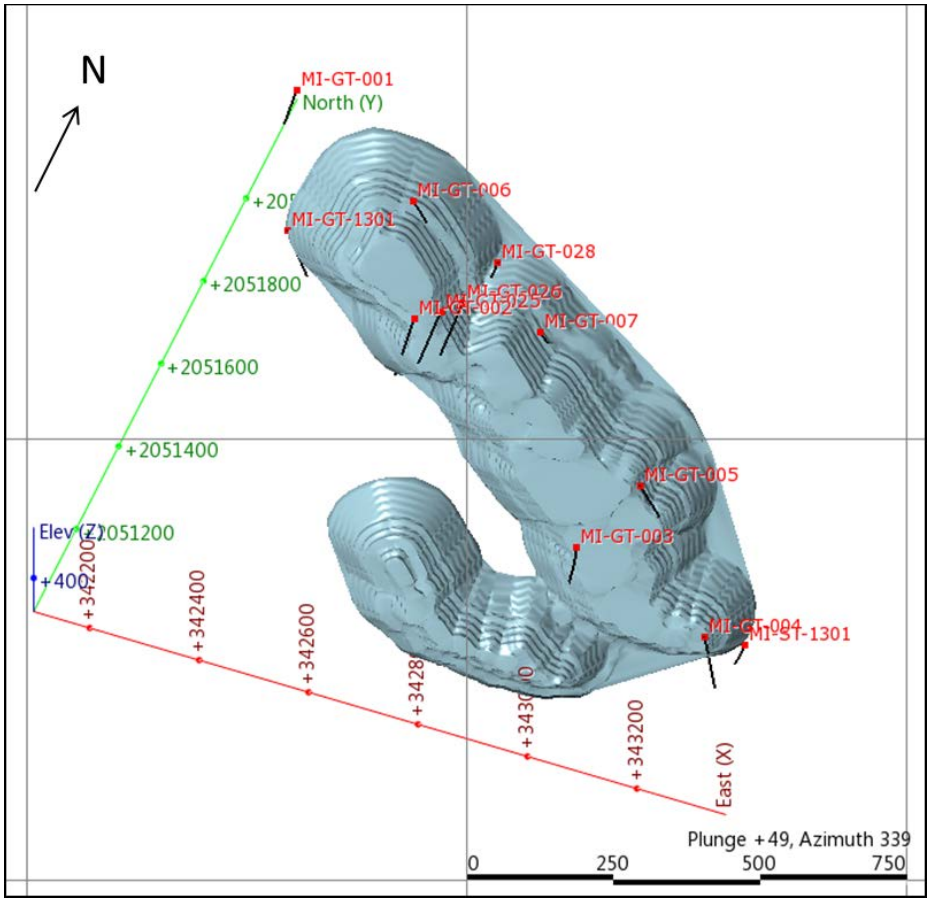


Figure 23-45: Locations of the geotechnical holes in the Miriam deposit relative to the April 2013 pit shell.

Core Orientation

Core orientation was done for all geotechnical holes using the ACT II Reflex Tool. The tool was operated by GeoDrill staff under supervision of Legeni staff. SRK Site senior trained Legeni and GeoDrill in how to use the ACT II Tool. The purpose of orienting the core is to characterize the orientation of discontinuities in the proposed underground mining areas and pit walls.

Geotechnical Core Logging

Core logging was undertaken by Legeni staff under the training and supervision of SRK. Legeni staff were logging core at the drills and instructed to complete the log of each run while the core was in the split tubes so as to most accurately replicate true ground conditions.

A site specific geotechnical core logging manual was produced by SRK and provided to Legeni which details the core logging procedure. Data was recorded at the drills on paper then typed up into an excel spreadsheet before being provided to SRK for Quality Assurance and Quality Control (QA/QC). The following information was recorded for each run where possible:

- Run Length; From, To (m);
- Total Core Recovery (TCR);
- Rock Quality Designation (RQD);

- Fracture Frequency (FF/m);
- Number of discontinuity sets;
- Added joints for domains of poor rock quality;
- Discontinuity characteristics including discontinuity type; infilling type, infilling properties, roughness, aperture, wall alteration and spacing;
- Alpha and beta angle for each discontinuity; and
- Presence of any argillite bands.

Based on the above parameters, a Rock Mass Rating (RMR) value has been generated using the classification scheme proposed by Laubscher (1990). RMR_{90} values for each run are presented in the geotechnical drillhole logs for Marianne-Marilyn and MSNE. For Miriam, the data was converted into RMR_{89} (Bieniawski 1989) values before the logs were generated. RMR_{89} is more applicable to the design of open pits.

Lithology data was recorded by GoviEx exploration geologists. This data was provided to SRK and added into the geotechnical logs.

Data Quality Assurance/Quality Control

Quality assurance / quality control (QA/QC) checking of drillhole logs against drilled core was done by the SRK Site Senior for each hole drilled. The purpose of this procedure was to ensure data consistency between SRK loggers and to minimize errors. The QA/QC process consists of general reviews of the core drilled and the geotechnical log, plus detailed checks on specific runs chosen at random intervals. Alterations to the geotechnical log would then be made as necessary by the Site Senior.

Televiewer Surveys

Acoustic Televiewer (ATV) surveys were carried out on all geotechnical holes excluding SRK-002, SRK-003 and SRK-004. Surveys were conducted by Legeni under the supervision.The acoustic televiewer identifies all discontinuities such as joints and faults and allows comparisons to be made with oriented core data. ATV surveys require water in the borehole therefore surveys typically start at 35m depth (water table depth).

For each feature identified at a specific depth, the orientation and aperture thickness is interpreted. ATV data was interpreted by JWRL Geomatics in Vancouver following specific requests from SRK regarding how discontinuities were classified. The features identified and how they are classified is detailed below:

- Joints - trace visible in both amplitude log and two-way travel time log;
- Cemented Joints - trace visible in amplitude log and not two-way travel time log;
- Bedding - trace visible in amplitude log;
- Tool marks - trace visible in both amplitude log and two-way travel time log; and
- Water level - trace visible in both amplitude log and two-way travel time log.

Point Load Testing

Point Load Testing (PLT) was undertaken on site at core shack to provide a continued estimate of intact rock mass strength. Approximately four to six tests were done on core samples every 15 m down each drillhole. In Marianne-Marilyn and MSNE, additional testing was done above and

below the ore zone to further quantify any strength variations for room and pillar design. Testing was done using a PIL-7 machine bought by GoviEx from RocTest. Testing was done by Legeni geologists following training by SRK staff.

Testing was carried out in accordance with ISRM guidelines (ISRM 1985). Both diametral and axial tests were undertaken to understand any anisotropy which may exist in the lithologies.

The following system was utilized to indicate failure mode of each point load test:

- T1 – good test (failed across diameter through intact rock);
- T2 – Failure along fabric (foliation/bedding; >50% along plane);
- T3 – Failure along cemented joint or vein (<50% along plane);
- T4 – Failed test; slipped, chipped, or rock mass indent (soft); and
- T5 – Refusal; exceeds 30 MPa gauge value.

The following parameters were collected as part of the point load testing:

- Depth;
- Lithology description;
- Failure description (planar, planar along vein, slipped, chipped, multiple mode, no break) and mode using the “T” system described above; and
- Gauge Pressure at failure (MPa).

The gauge reading was converted to an Is(50) value as per ISRM guidelines (ISRM, 1985). Adjacent to PLT tests, core samples were taken for unconfined compressive strength (UCS) testing in the laboratory (Section 2.7). This allows for correlation between Is(50) and UCS values.

Laboratory Testing Program

Laboratory testing on drilled rock core has been undertaken in Niamey by Legeni and in South Africa by RockLab.

Samples were selected by SRK for unconfined compressive strength (UCS) testing and direct shear box testing. Objectives of the UCS testing include:

- To determine how intact rock strength varies in the hanging wall, foot wall and ore body at Marianne- Marilyn and MSNE;
- Understand how intact rock strength varies as a function of lithology and depth at Miriam pit; and
- Correlate PLT results with UCS test results to determine suitable correlation factors.

The majority of UCS testing was done by Legeni with some tests done in South Africa to add confidence to the results.

Base friction angles were estimated by shear box testing on saw cut joints. Testing was done in South Africa by Rocklab.

UCS Correlation with PLT Data

UCS values have been plotted against Is(50) values from PLT data as a function of lithology. This enables a correlation factor (cf) to be generated to estimate UCS values from PLT tests such that:

$$UCS = cf \times Is(50)$$

The conversion factor was generated by plotting Is(50) values against UCS values for each lithology and fitting a linear regression line. Based on these plots and the work by Rusnak and Mark (2000), "cf" factors have been generated and are given in Table 23-28. For geotechnical analysis, all PLT data has been converted to UCS values based on the correlation factors given.

Table 23-28: Correlation factors relating PLT data to UCS values for each lithology.

Lithology	Correlation Factor (cf)
Tchinezogue	15
Guezouman	17
UA	19
Talak	21

Structural Geology Investigation

Objectives

The objective of the structural geology investigations were to understand, as far as possible, the geometry and nature of the structures that will likely impinge on the geotechnical design of the mine plans. Due to the nature of the geological setting, the principal structures of interest to this investigation are brittle fault zones. Additionally, however, decimetre-scale monoclinal structures affect the lowermost sedimentary formations (Talak and Guezouman) within the deposit areas, and later inversion-related dome structures affect the stratigraphic dips of the sedimentary pile over significant areas of the planned mine workings. The optimum way to evaluate these structures is by modelling and analysing them in 3D.

Data and Methodology

The focus of the structural investigation concentrated on the structure and geometry of the Guezouman, Talak and Unite d'Akokan (UA) Formations mainly, as these comprise the three main units affected by mining, particularly in the underground mines (Marianne-Marilyn and MSNE) where they constitute the main ore host, ore footwall and location of the development drives, respectively. Constraints on the stratigraphic dips in these units, salient to the underground developments, was gained from oriented drillcore and acoustic televiewer data from the geotechnical holes drilled during the field programme. Additionally, stratigraphic dip maps of the main ore footwall, the Talak Formation, were generated from the stratigraphic depths.

Table 23-29: Data used in the structural interpretations.

Data	Source	Use
Quickbird satellite imagery	GoviEx	Interpretation of linear surface traces of the faults and stratigraphic discontinuities (bedding and contacts)
3D modelled stratigraphic horizons	GoviEx/SRK	Determination of sub-surface offsets or breaks in the elevation of units
3D grade models (Leapfrog 3D)	SRK (2012)	Determination of sub-surface offsets or breaks in the elevation of units and rectilinear margins of the mineralisation were used as a guide to place some discontinuities
Geotechnical drill core photos	GT drilling	Provision of depth intervals of structural zones and the nature of the deformation
Orientation data of structures	GT drilling	Orientation of intersected significant structures

23.3.3 Marianne-Marilyn

Geology

Four main sub-horizontal geological formations exist in the Marianne-Marilyn project area. From surface they are described:

- Tchinezogue is a fluvio-deltaic sedimentary unit with thin sub-horizontal bedding. A degree of heterogeneity exists, and areas near the surface are weakened due to weathering effects;
- Guezouman is a massive, relatively competent fine to coarse grained sandstone unit with a relatively low density. The base of this unit, situated at depths ranging from 50-160 m from surface, hosts the uranium mineralization;
- Unite d'Akokan (UA) exists in channels between the Guezouman and Talak, and is characterized as thinly bedded mudstones and siltstones. The two NE-SW running central drives are planned directly above the UA channels; and
- Talak unit is fine grained claystone and mudstone lying at the bottom of the project area. It is weak and relatively isotropic.

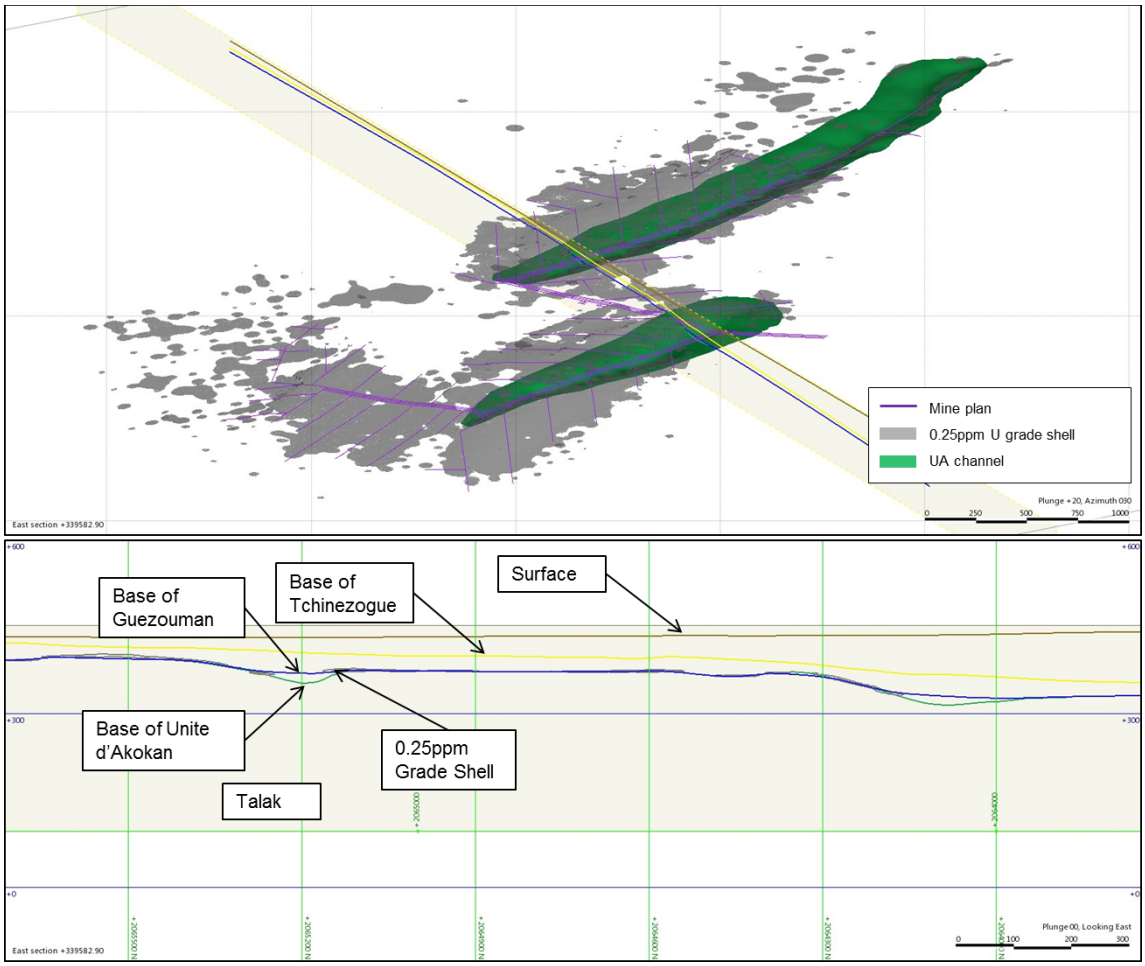


Figure 23-46: Isometric view and Cross section view of Marianne-Marilyn geology.

Structural Geology (Marianne-Marilyn)

Structural Setting

The Marianne-Marilyn deposits are the northernmost mineralised zones within the Madaouela Project. The mineralisation is elongate along a WSW-ENE trending structural axis which comprises a series of domes, faults and monoclines (Figure 6-5).

The major structures affecting the deposit and their potential implications for the geotechnical evaluation are outlined below.

Major Structures

Monoclines

The M&M deposits are affected by two well-defined monoclinal structures that control the location of elongate *barres* of UA sediments (Figure 6-5; Yahaya & Lang, 2000). Based on subtle gradient changes in the base of the Guezouman/Top Talak horizon a further, smaller, monoclinal-type structure has been interpreted to the north of the large southwestern UA-bearing structure (Figure 23-48).

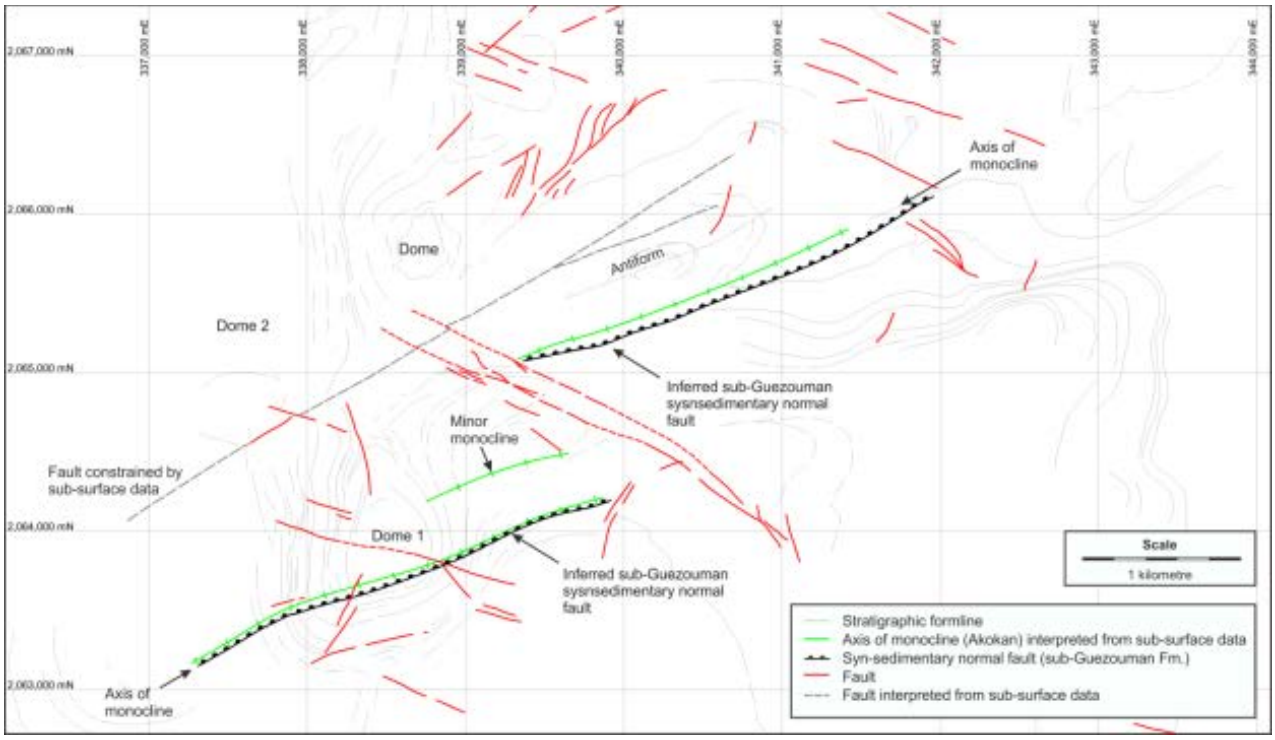


Figure 23-47: Plan view of stratigraphic formlines and structures interpreted at Marianne-Marilyn.

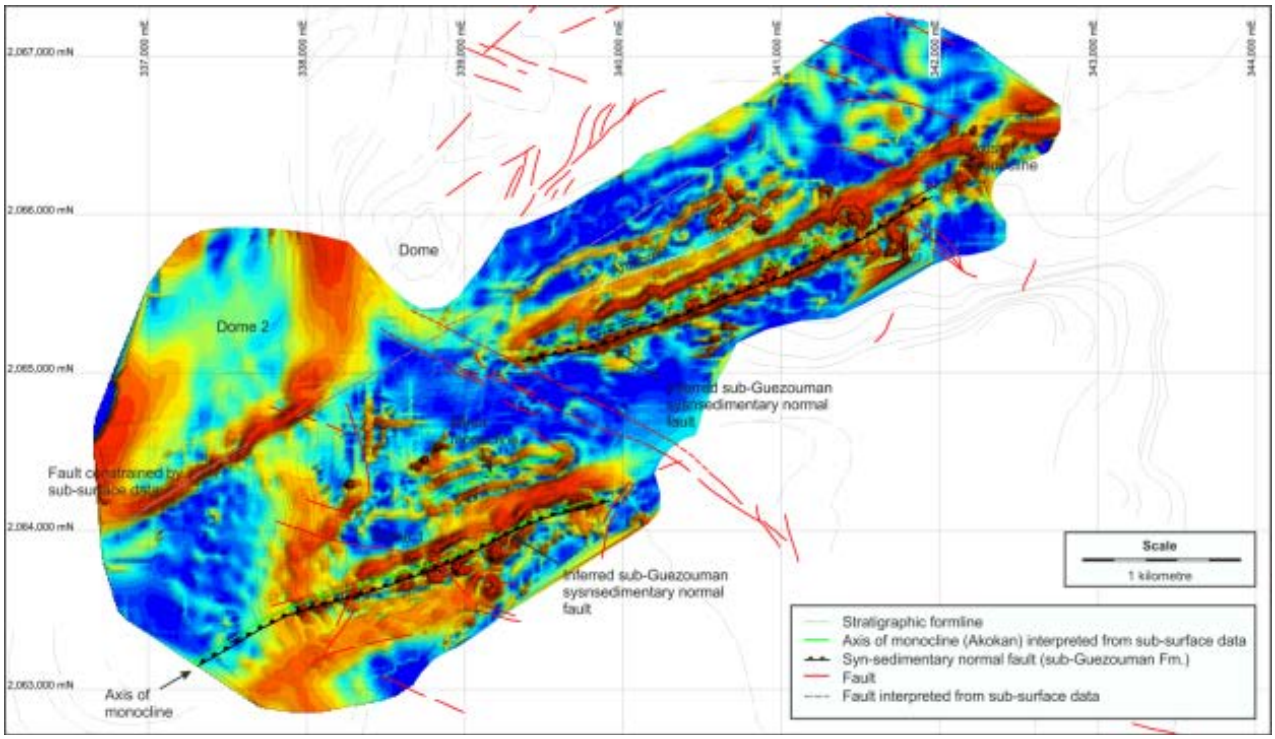


Figure 23-48: Plan view of stratigraphic formlines and structures interpreted at Marianne-Marilyn relative to a dip map of the top Talak/base Guezouman.

Domes

The west of the M&M deposit is dominated by a dome, where the stratigraphy defines a radially gently-dipping zone of uplift (Figure 6-5). Stratigraphic dips are accentuated on the flanks of this structure but near-horizontal at the apex (Figure 23-48). In general the stratigraphic dips on the flanks are approximately 5° or less, with the exception of the southern flank of the dome which reaches up to approximately 8°.

Faults

In total eight faults have been interpreted to be through-going structures at M&M. All of these structures have been interpreted primarily on the basis of linear traces on the Quickbird image. The majority of these traces are associated with small displacements of the stratigraphic traces on the surface.

Fault Rock Characteristics

Fault rock characteristics are known for a number of faults from diamond core drilling in the Marianne-Marilyn deposits. The principal characteristics of these are described below.

Akokan Faults

The Akokan Faults have a syn-sedimentary origin and are likely to have originally consisted of a localised zone of soft-sediment shear. Evidence for small-scale sedimentary faults and convolute slump structures are evident throughout the UA and Guezouman drill core intervals.

Steep Faults

The nature of the steep faults affecting Marianne-Marilyn is less constrained than the Akokan faults due to the uncertainity in dip of these structures. The nature of the faults vary markedly, with some uncorrelated faults comprising zones of clean slip surfaces with negligible fault rock to zones of variable recovery and broken rock.

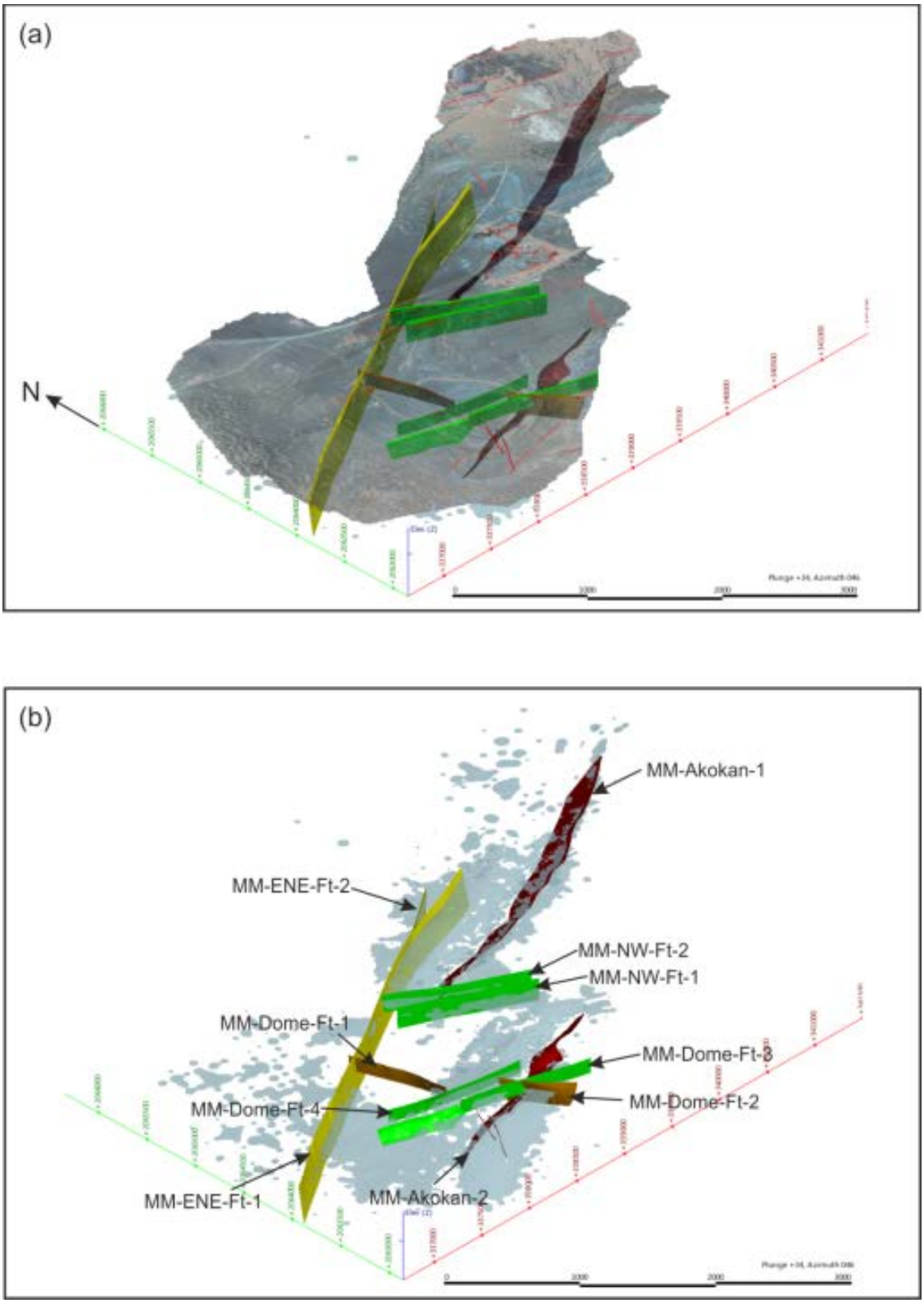


Figure 23-49: 3D view of interpreted faults at Marianne-Marilyn: (a) relative to topography with interpreted Quickbird image drape; (b) relative to Leapfrog 400 ppm U grade shell.

Rock Fabric

Oriented core drilling and ATV surveys give good information regarding the orientation and properties of discontinuity sets in the rock mass. The main discontinuity sets identified in both the hanging wall and footwall are:

- **Closed Bedding planes**; sub-horizontal; generally smooth and likely to persist for approximately 3-10 m (considered to be dominant discontinuity set). These are considered closed features (i.e. would not affect fracture frequency) and therefore their strength component has a "bond strength" (cohesion) as well as a friction angle.
- **Open Bedding Planes (bedding parallel joints)**; sub-horizontal; similar to bedding planes but are open, i.e. have aperture typically ranging from 0.1-1.0 mm. Some surface staining is present, but generally there is no infill and joint wall strength is equal to the intact rock strength.
- Sub-vertical joints; secondary joints, likely cut by bedding, aperture typically ranging from 0.1-1.0 mm

Rock Mass Model

A rock mass assessment has been undertaken to review the variability of rock mass characteristics of the geological units that form the hangingwall, orebody and the footwall of the Marianne-Marilyn deposit. Guezouman sandstone will form the pillars and the immediate roof, UA and Talak units will form the floor and lower portion of the pillars depending on the presence of UA channel. Data from the geotechnical site investigation program has been used to determine the rock mass design parameters.

Four geotechnical domains have been defined at Marianne-Marilyn. Analysis shows that these domains are primarily governed by lithology.

Means and ranges of rock mass properties per geotechnical domains, including intact UCS, rock mass unconfined compressive strength (RMS), Laubscher RMR_{90}, and estimated Geological Strength Index (Hoek et al. 2005) (GSI) from observations of core photographs are listed in Table 23-30.

Table 23-30: Rock mass properties per geotechnical domain.

Geotechnical Domain	Rock Type	Density (kg/m^3)	Design Intact UCS (MPa)	RMS (MPa)	RMR_{90}	GSI
Guezouman (Hangingwall)	Sandstone	2200	40	13.9	71	65 - 75
Guezouman (Orebody)	Sandstone	2200	40	13.9	63	60 - 70
UA (Footwall)	Silt/Mudstone	2250	50	9.4	60	55 - 65
Talak (Footwall)	Clay/Mudstone	2450	30	3.9	56	50 - 60

Geotechnical Design

A two-staged approach involving the use of both empirical design methods and numerical modelling has been used for underground geotechnical design at Marianne-Marilyn. Empirical analysis was conducted in the first stage, to determine acceptable range of span, bedded roof

bolting system design, and pillar design. These results were then complimented by numerical modelling, where 3D and 2D conceptual models were developed to confirm the expected stress state, and rock mass behaviour of the roof, floor and the pillars.

Empirical Analysis

Span Design

Empirical method for assessing the stability of excavation proposed by Ouchi (2004) was used to examine the allowable span for man-entry openings at Marianne-Marilyn.

Ouchi's updated span design curve (Figure 23-50) for weak rock masses (Ouchi et al. 2004) compares the RMR_{76} (Bieniawski 1976) and critical span to categorize the design into stable, potentially unstable and unstable. The RMR_{76} rating for the Guezouman domain was derived using 'GSI ≅ RMR_{76}' correlation. A correction factor of minus 10 was then applied to the RMR_{76} to account for shallow dipping bed planes in the immediate roof. This results in a stable design span of 10 m, which is equivalent to 7 m opening width in room and pillar layout.

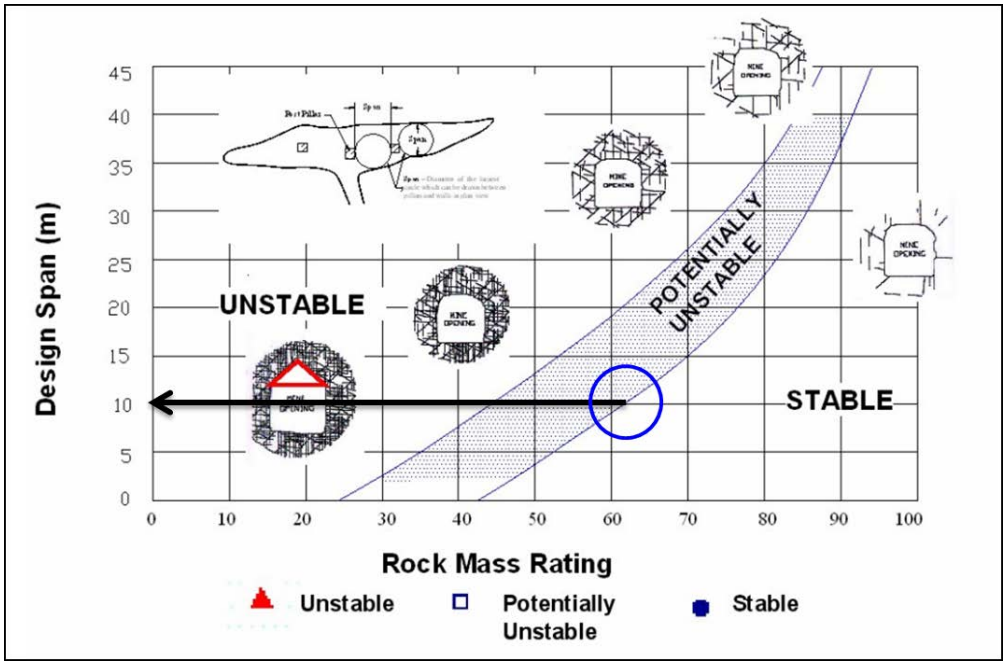


Figure 23-50: Underground span design guideline curve (Ouchi et al. 2004)

Roof Stability Analysis

The results of empirical assessment on design span indicate that 7 m opening width is a suitable design for Marianne-Marilyn project area. SRK have examined the stereonets obtained from oriented core logs and acoustic televiewer logs, and identified two main joint sets in majority of the deposit.

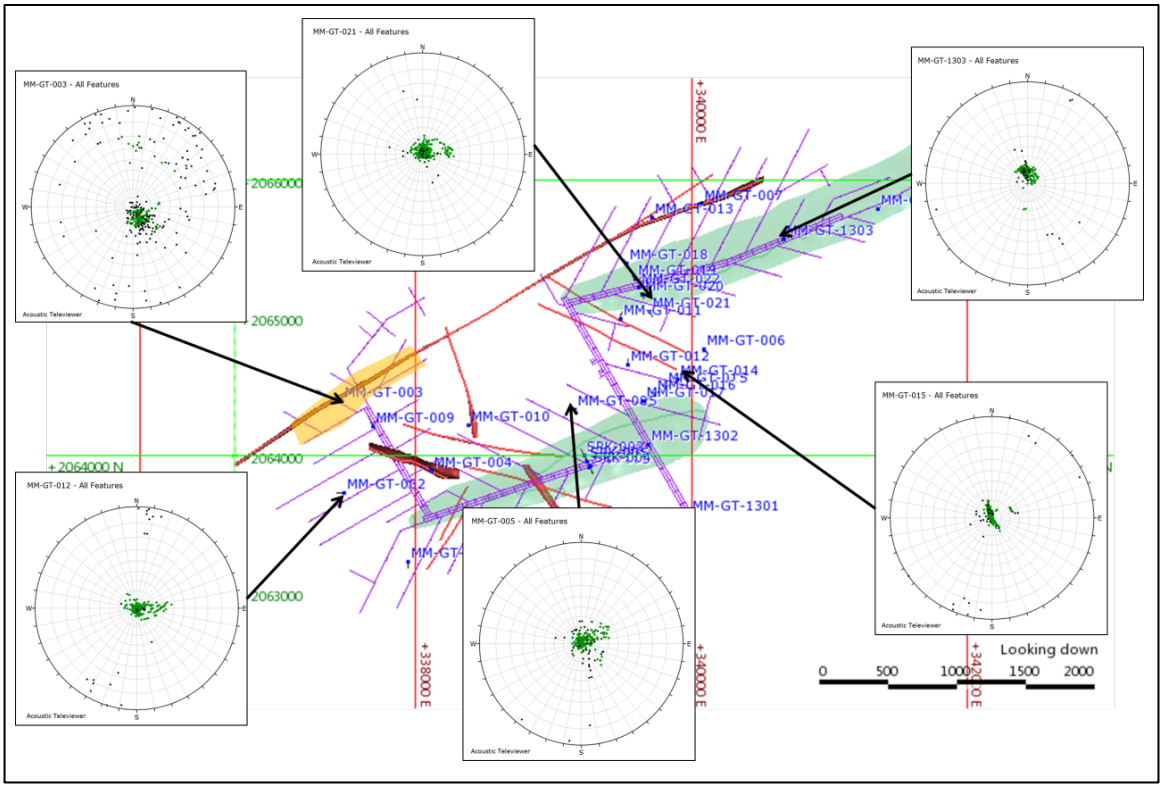


Figure 23-51: Stereonet plots representing discontinuities across Marianne-Marilyn deposit. An increased bedding dip and additional intermediate dipping joint set is seen in Hole MM-GT-003 due to intersection with a fault structure.

The combinations of the joint sets do not form wedges. The flat lying bedding planes and sub-parallel joints in the immediate roof could form layers of rock that act as “beams” and deflect into the excavation under loading. Pattern roof bolting is recommended to increase the stiffness of the immediate roof.

Ground Support Design

It is important to distinguish between the safety and stability of an excavation. The stability of an underground opening is concerned with the design of the excavation and ground support that is installed to prevent falls of ground. The potential for instability of the rock around development and stopes has been identified. These instabilities can, in general, be managed using levels of ground support typical for a Poor to Fair rock mass.

In the case of excavations that need to be accessed by personnel, it is essential that not only is it stable, but also that other factors, such as the development of ‘loose’ rock, which affects the safety of the workplace, be addressed. It is quite possible for an excavation to be stable such that no major ground falls will occur, and yet the development of small blocks and ‘loose’ rock presents a considerable hazard.

For liability reasons, most mines are moving towards the installation of mesh and rock bolts throughout all parts of the mine in which personnel access is required. Without the use of mesh, a formal documented scaling process is required and must be incorporated into the support

standards for the mine. Welded wire mesh, with a maximum mesh size of 100 mm (4") square, constructed from #8 gauge wire or thicker, is preferred to chain link or "chicken wire" type mesh.

If mesh is not used on a mine wide basis, then, as a minimum, it should be used in areas that are accessed on a proportionately higher basis, such as meeting stations, explosive magazines, and workshops.

The roof support design is based upon accepted bolting design criteria tempered by SRK's recent experience in design and review of bolting systems in similar ground conditions and rock types. SRK recommends full column resin grouted 1.5 m long #6 (20 mm nominal diameter) rebar bolted on a 1.2 m by 1.2 m square pattern for excavations with spans up to 7 m. Rockbolts should be forged head and be installed with a minimum 150 mm by 150 mm domed washer plate and appropriate spherical washer to provide adequate surface support of the excavation and transfer load into the back.

Pillar Design

Pillar design at the Marianne-Marilyn is based on the empirical design methodology described by Lunder and Pakalnis (1997).

The empirical formulae requires input parameters of unconfined compressive strength of the intact rock, average unit weight of the overburden rock, the depth from surface, room width, and the dimensions of the pillar. The values used in the analysis were:

- Unconfined Compressive Strength: 40 MPa
- Overburden Rock Unit Weight: 24 kN/m^3 (estimated from density)

Depth of overburden at Marianne-Marilyn varies from 50 m in the northeast area to 160 m in the southwest area. This depth was categorized into three ranges that best describes the boundaries of the proposed mining panels; under 75 m, 75-120 m, and 120-160 m. The 7 m width and 2 m height of the rooms in production panels were kept constant at all depths.

The design criteria for the pillars at Marianne-Marilyn were:

- Minimum factor of safety of 1.5; and
- Minimum width-to-height ratio of 2

The width-to-height ratio limits the potential for pillar foundation failure (pillars punching into the soft roof or floor), and shear failure along weak joint or bedding plane. The designs are considered to be relatively conservative, and further investigation of ground behaviour may allow for an increase in extraction ratio. Table 23-31 shows the current design configuration based on empirical analysis.

Table 23-31: Production panel pillar designs from empirical analysis.

Depth Range (m)	< 75		75 to 120		120 to 160	
Pillar Type	**Initial**	**Final**	**Initial**	**Final**	**Initial**	**Final**
Pillar Dimension (m)	15 x 15	4 x 4	17 x 17	5 x 5	19 x 19	6 x 6
Pillar Height (m)	2	2	2	2	2	2
Room Width (m)	7	7	7	7	7	7
Depth from surface (m)	50	50	100	100	140	140
Pillar Width-Height Ratio	7.5	2	8.5	2.5	9.5	3
In-Panel Extraction Ratio	54%	87%	50%	83%	47%	79%
Factor of Safety	13.6	2.2	7.4	1.6	5.7	1.5

Numerical Modelling

Numerical modelling assessment consisting of mine scale 3D stress analysis and detailed 2D pillar stability analysis was conducted to confirm and optimize the design obtained from empirical study. Numerical modelling complements the empirical design by simulating complex excavation shapes, material heterogeneity, and plastic deformation of rock mass.

3D Boundary Element Stress Analysis

A conceptual 3-dimensional mine scale elastic model of room and pillar layout, using the Map3D boundary element code, was built to examine the mining-induced stresses around the excavation. Of particular interest was the stress state in various types of pillars considered in the mine plan. The pillar types include the square in panel pillars, barrier pillars that separate the adjacent mining panels and central drift, and secondary barrier rectangular pillars.

The conceptual model consists of 8 mining panels, and is 900 m long by 650 m wide. The entire model dips by 7° allowing the depth to range by 50-160 m. The excavation phase was constructed using 7 m room width, 2 m mining height, and 5 m square pillars. 15 m barrier pillars divide the mining panels, and 10 by 15 m secondary barrier pillars were positioned every 100 m of development to ensure stability for ventilation infrastructure. Three parallel central drifts connect the mining panels with 25 m spacing in between. Figure 23-52 shows isometric view and plan view of the 3D conceptual model.

A simplified model of two geotechnical domains was considered. The geotechnical parameters (rock mass Young's modulus and Poisson's ratio) of the ore bearing Guezouman domain were assigned to all of the model space, while characteristics of Talak were assigned to the green blocks representing the floor of the excavation. The blue blocks are the excavated void space. The input parameters are:

- Guezouman: E = 15.6 GPa, v = 0.40
- Talak: E = 2.3 GPa, v = 0.21

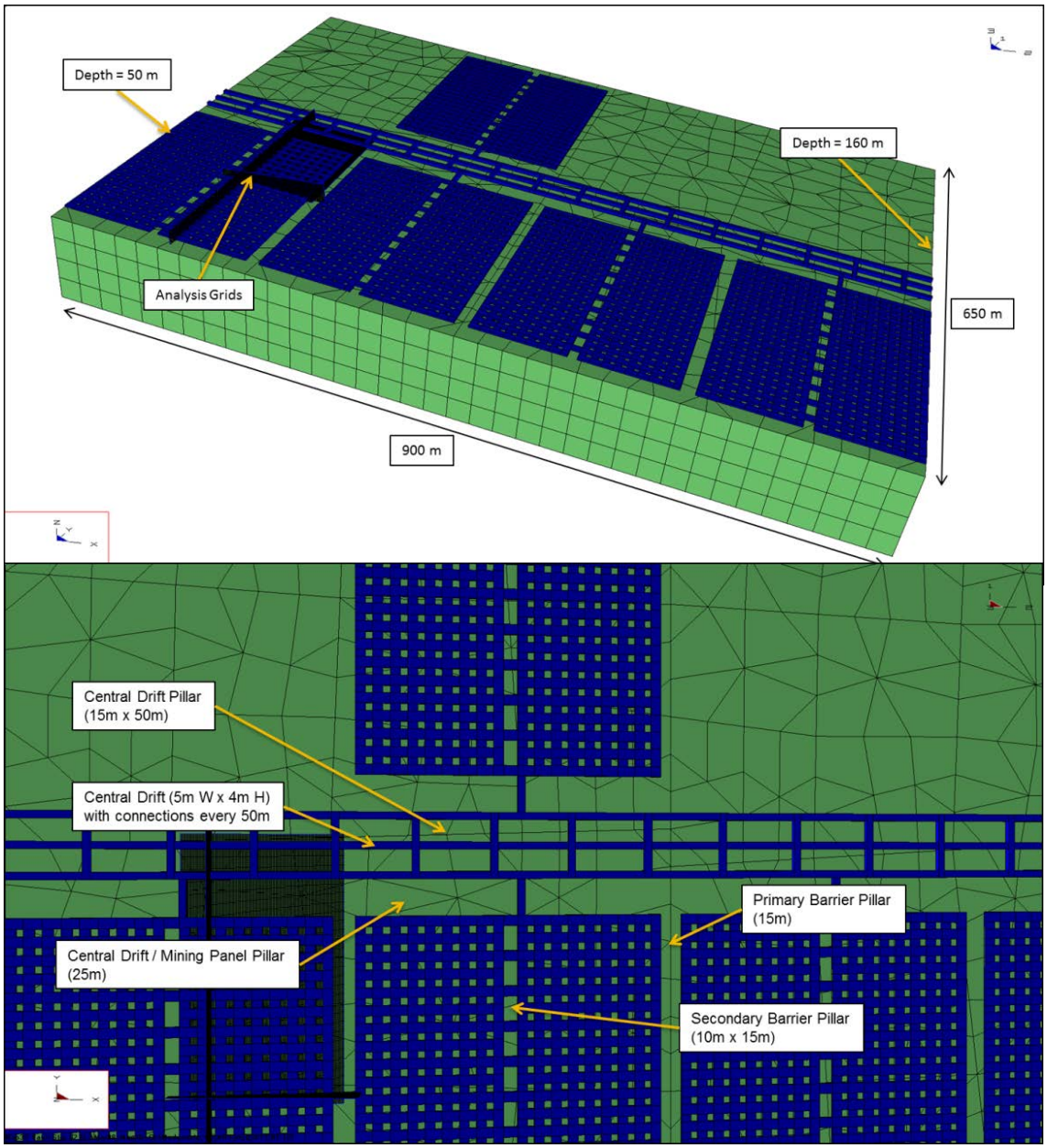


Figure 23-52: 3D Conceptual model and plan view of mining layout.

Results indicate higher stress conditions at the outer edges of the pillars, and lower stress in the core area. It is expected that some fracturing would occur at the outer wall of the various pillar types, but would not influence the load on pillar core significantly. This fracturing would be controlled by localised bolting.

Table 23-32 summarizes the 3D boundary element stress analysis results. The results are given in terms of the maximum stress at the pillar core.

Table 23-32: Summary of 3D stress modelling results.

Pillar Type	Depth (m)			
	75	100	125	160
	Maximum Stress (MPa)			
Final pillars (5 x 5m)	7	11	13.5	16
Primary barrier pillars	3	4	5	7
Secondary barrier pillars	3.5	7	8	10
Central drift / Mining panel pillar	2	3.5	4	4.5

2D Finite Element Pillar Modelling

Pillar stability numerical modelling was conducted using Phase2 two dimensional finite element code by Rocscience.

Three base case geometries of square pillars were modelled comprising of a single pillar with half a room width on both sides. The pillar sizes were 4 x 4 m, 5 x 5 m and 6 x 6 m with a constant room width of 7 m, as determined in the empirical analysis. Displacement of the lower boundary was fixed, and the side boundaries were fixed in horizontal direction. Displacement in the upper boundary was allowed and a distributed load was applied gradually in multiple stages. The distributed load was adjusted to account for change in overburden depth.

Figure 23-53 shows the modelling results of a 5 x 5 m pillar at an overburden depth of 125 m.

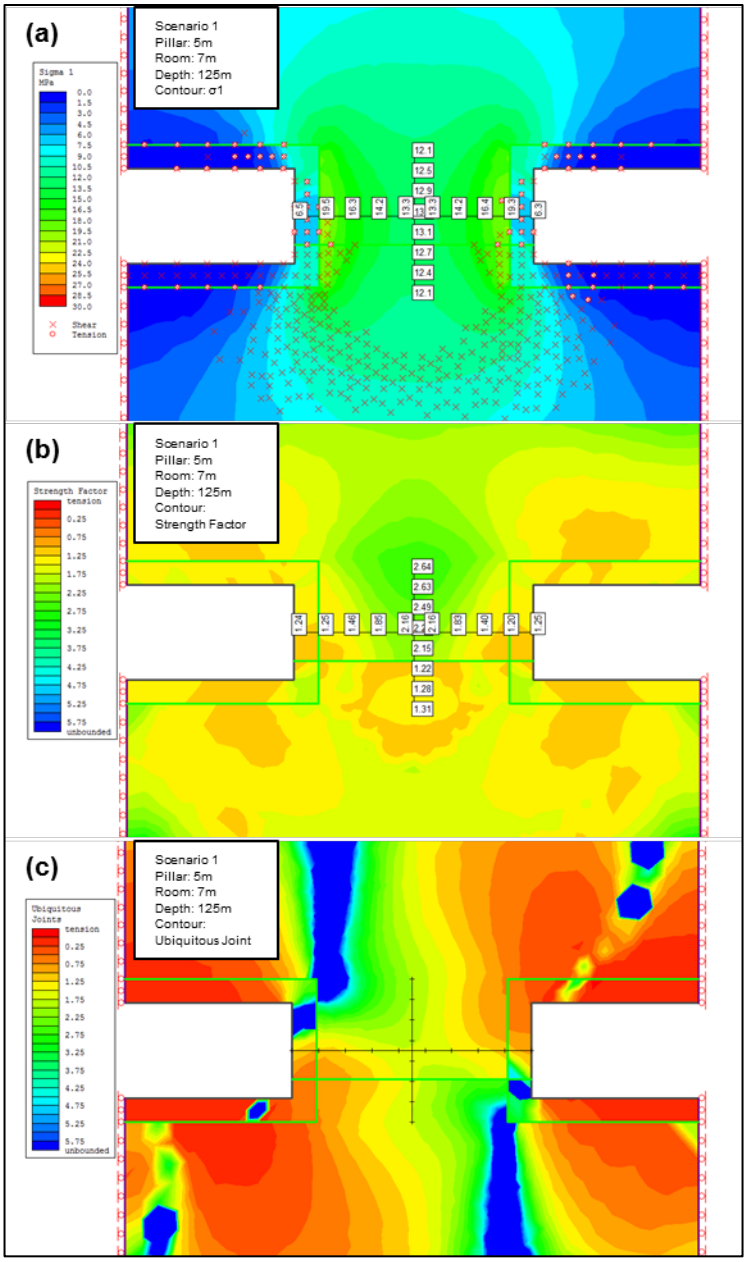


Figure 23-53: 2D Finite element pillar analysis results of 5 x 5 m pillar at 125 m depth. Contours of sigma1 (a), strength factor (b), and ubiquitous joint strength factor (c)

Table 23-33 summarizes the results in terms of average strength factor at the pillar core. It is noticeable that 4 x 4 m pillars provide acceptable strength factor at depth. However, weak foundation material, possibility of further blast damage and variability of Guezouman sandstone intact rock strength limits our confidence to design pillars solely based on the numerical modelling. The results reinforce the opportunity for design optimization.

Table 23-33: Summary of 2D pillar numerical modelling.

Pillar Type	Depth (m)			
	75	100	125	160
	Strength Factor at Pillar Core			
Final pillars (4 x 4m)	1.6	1.5	1.5	1.4
Final pillars (5 x 5m)	2.2	2.1	2.0	1.9
Final pillars (6 x 6m)	2.4	2.4	2.3	2.2
Primary barrier pillars	10.6	9.9	8.9	7.8
Secondary barrier pillars	5.3	4.8	4.7	4.3
Central drift / Mining panel pillar	6.9	6.5	6.1	5.2

Geotechnical Design Sectors

The Marianne-Marilyn geotechnical design sectors are presented in Figure 23-54. The production mining panels were grouped in terms of the depth of overburden and anticipated dip of the mineralization. The subsection of each sectors (labeled with an 'a') indicate unfavourable dip and/or depth. Each design sectors are assigned with suggested pillar dimensions considering the results from empirical and numerical modelling analyses, as well as practical issues associated with the mining process. Barrier pillars are also recommended to limit the propagation of potential unstable zone. The recommendations are summarized in Table 23-34.

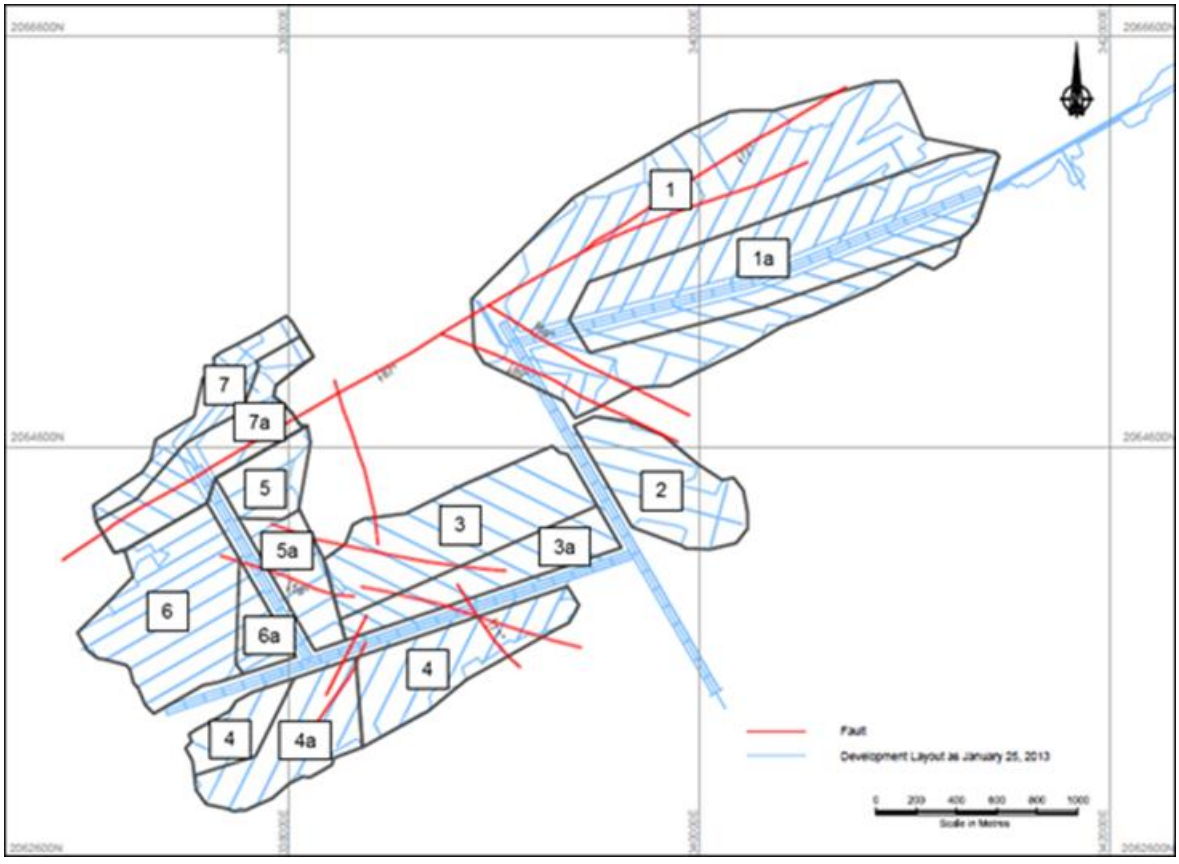


Figure 23-54: Marianne-Marilyn geotechnical design sectors.

Table 23-34: Geotechnical design summary.

Design Sector	Depth	Dip	Pillar Dimension			Panel Width	Extraction Ratio	Comments
	m	°	Width (m)	Length (m)	Height (m)	m	%	
1	50-75	0-7	4	4	2	7	87	Shallow and minor dipping ore; conventional R&P with 4 m square pillars
1a	50-75	7-10	4	4	2	7	87	Shallow and moderate dipping ore; apparent dip R&P with 4 m square pillars
2	50-75	0-7	4	4	2	7	87	Shallow and minor dipping ore; conventional R&P with 4 m square pillars
3	50-75	0-7	4	4	2	7	87	Shallow and minor dipping ore; conventional R&P with 4 m square pillars
3a	50-75	7-10	4	4	2	7	87	Shallow and moderate dipping ore; apparent dip R&P with 4 m square pillars
4	75-120	0-10	5	5	2	7	83	Moderate depth and dip; apparent dip R&P with 5 m square pillars
4a	120-160	10-16	6	6	2	7	79	Deep and moderate-to-steeply dipping; 6 m square pillars; some areas require stepped R&P
5	75-120	0-7	5	5	2	7	83	Moderate depth and minor dip; conventional R&P with 5 m square pillars
5a	75-120	7-10	5	5	2	7	83	Moderate depth and dip; conventional R&P with 5 m square pillars
6	100-120	0-7	5	5	2	7	83	Moderate depth and minor dip; conventional R&P with 5 m square pillars
6a	100-120	7-10	5	5	2	7	83	Moderate depth and dip; conventional R&P with 5 m square pillars
7	75-120	0-6	5	5	2	7	83	Moderate depth and dip; conventional R&P with 5 m square pillars
7a	75-120	7-16	6	6	2	7	79	Moderate depth and moderate-steeply dipping; 6 m square pillars; apparent dip R&P and some areas require stepped R&P

The following excavation criteria apply to the production panel designs:

- Production panels are developed with initial pillar dimension, which is equivalent to two times the length of final pillar plus an opening width e.g. Sector 1 would be (2 x 4m) + 7m = 15m;
- Final pillars are formed on retreat by splitting the initial pillars through the center of the pillar length;
- Mining height at production panels is kept at 2 m with the use of low profile mining equipment;
- Pattern roof bolting is to be applied in the production area with 1.2 m by 1.2 m spacing using

1.5 m long #6 (20 mm nominal diameter) full column resin grouted rebar;

- 15 m barrier pillars are to be left between adjacent mining panels, and secondary barrier pillars with a minimum 10 x 15 m dimension should be left every 100 m span within the panel;
- Minimum of 20 m spacing (25 m if depth > 100 m) should exist between the central access drift and production panels; and
- 20 m rib pillars should be designed around identified fault structures.
- Appropriate offset of the central drives from the Akokan faults may be required if these faults propagate above the UA channels.

Decline

Access to Marianne-Marilyn will be through a surface portal into a central drive in the south-centre of the deposit. The decline is approximately 700 m in length, and plunges at 8° to reach the mineralized depth. Two geotechnical holes (MM-GT-1301, MM-GT-1302) were drilled to characterize the likely encountered ground conditions. Figure 23-55 shows the location of mine access decline respective to the mine plan.

As the decline excavations are a life of mine critical infrastructure excavations it is considered prudent to increase the level of ground support above that recommended for the rest of the mine. Addressing ground control issues resulting from potential deterioration of these excavations throughout the mine life and the impact that this would likely have on production the following ground support is recommended; 2.4 m long #8 (25 mm) resin grouted rebar on 1.4 m centres through welded wire mesh in the back. Ribs to be supported with 1.5 m long #6 (20 mm) resin grouted rebar installed on 1.5 m centres through welded wire mesh that has been extended to within 1 m of the floor. The maximum mesh size of the welded wire mesh should be 100 mm (4") square, constructed from #8 gauge wire or thicker. Smooth wall blasting techniques are strongly recommended in permanent infrastructure excavations

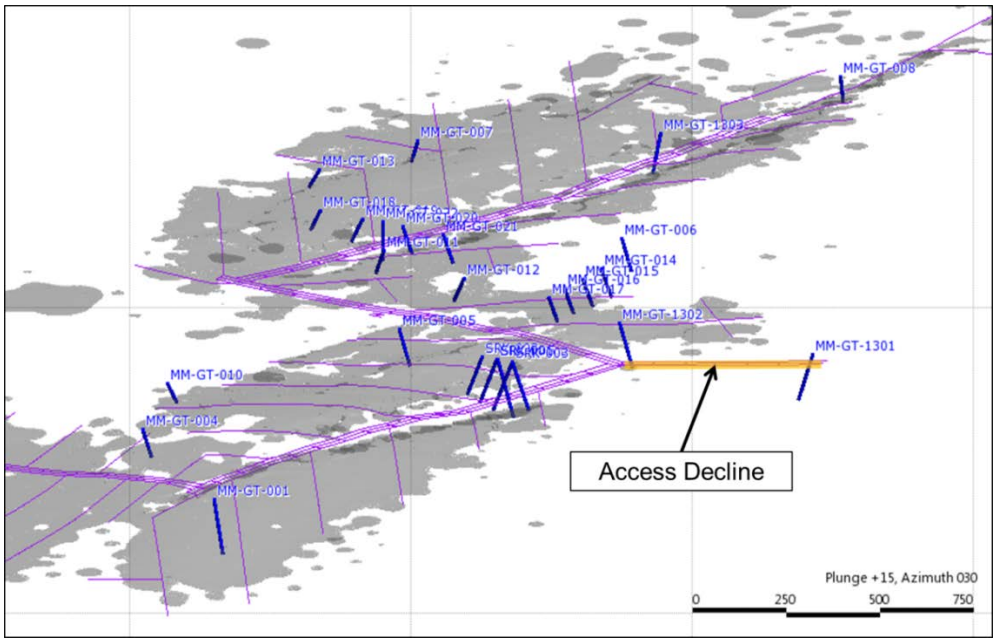


Figure 23-55: Marianne-Marilyn deposit access decline.

From surface the access decline passes through the Tchinezogue formation, which is a sub-horizontal bedded fluvio-deltaic sedimentary unit. The thickness of this unit is approximately 80 m, and the top 20 m is weathered and oxidized due to fluctuation of the groundwater table. The expected ground conditions and the geotechnical parameters along the length of the decline are given in Table 23-35.

Table 23-35: Expected lithologies and geotechnical parameters along the decline.

Decline length	Formation	RQD Design Value (%)	RMR_{90} Design value
0m - 0+150m	Weathered Tchinezogue	79	59
0+150m - 0+360m	Fresh Tchinezogue	96	65
0+360m – 0+700m	Guezouman SST	96	80

23.3.4 MSNE Geotechnical Design

Geology

Exploration and geotechnical drilling suggests that the geology at MSNE comprises Guezouman sandstone overlying UA channels overlying Talak mudstone formation. The Guezouman dips sub-horizontally across the whole deposit and forms the hanging wall and host rock. The UA formation exists in the southern area of the deposit and will form the footwall. To the north, where UA is absent, the Talak unit will form the footwall. Figure 6-13 shows isometric view of MSNE project area and an N-S cross section through MSNE project area with the geology wireframes provided by GoviEx.

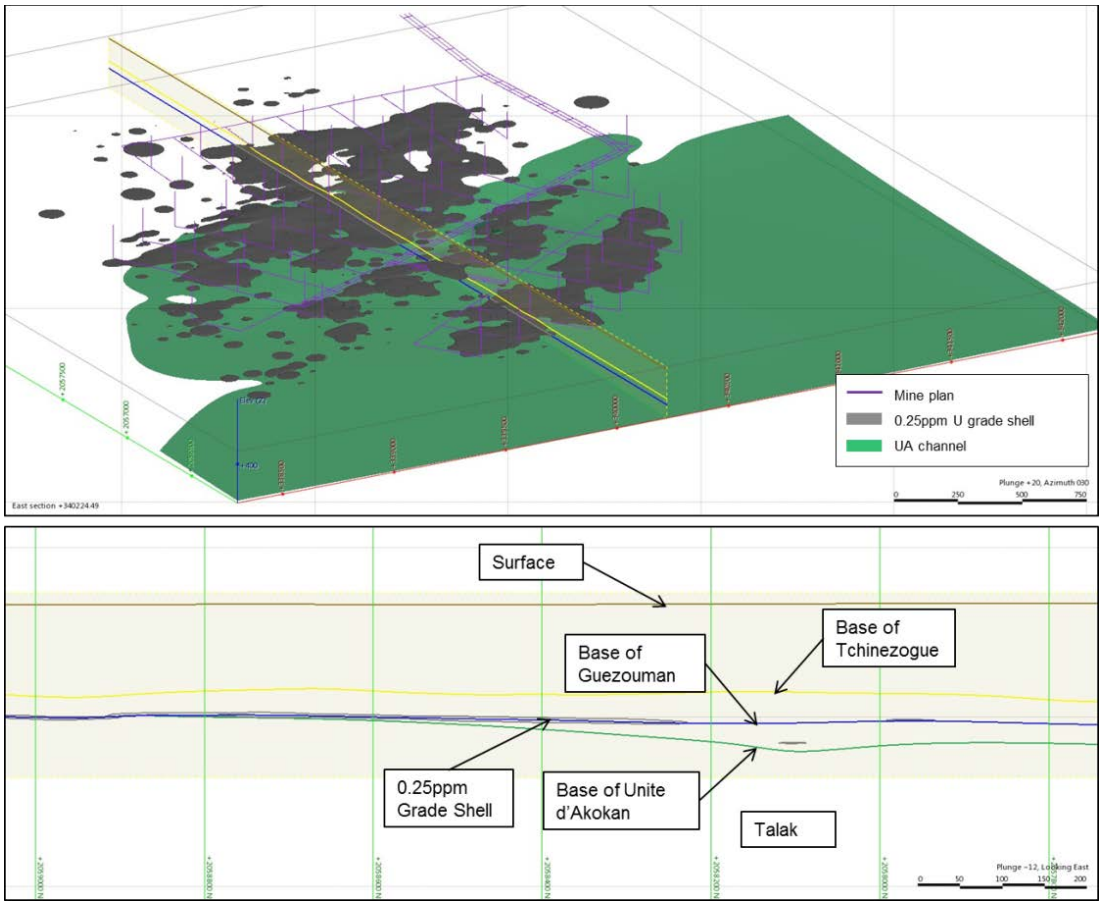


Figure 23-56: Isometric view and Cross section view of MSNE geology.

Structural Geology

Structural Setting

Faults

In total six faults have been modelled to be through-going structures at MSNE. MSNE-NE-Ft-1 appears to belong to a relatively significant NE-SW fault trend. This fault trace appears to bound the deposit area to the west and accommodates a significant west-side-down dip-slip displacement that juxtaposes Tchinozogue Formation on the west against Tarat Formation on the east. Fault displacement is interpreted to be transferred to an adjacent segment to its south over a broad zone of fault overlap, which may constitute a relay zone.

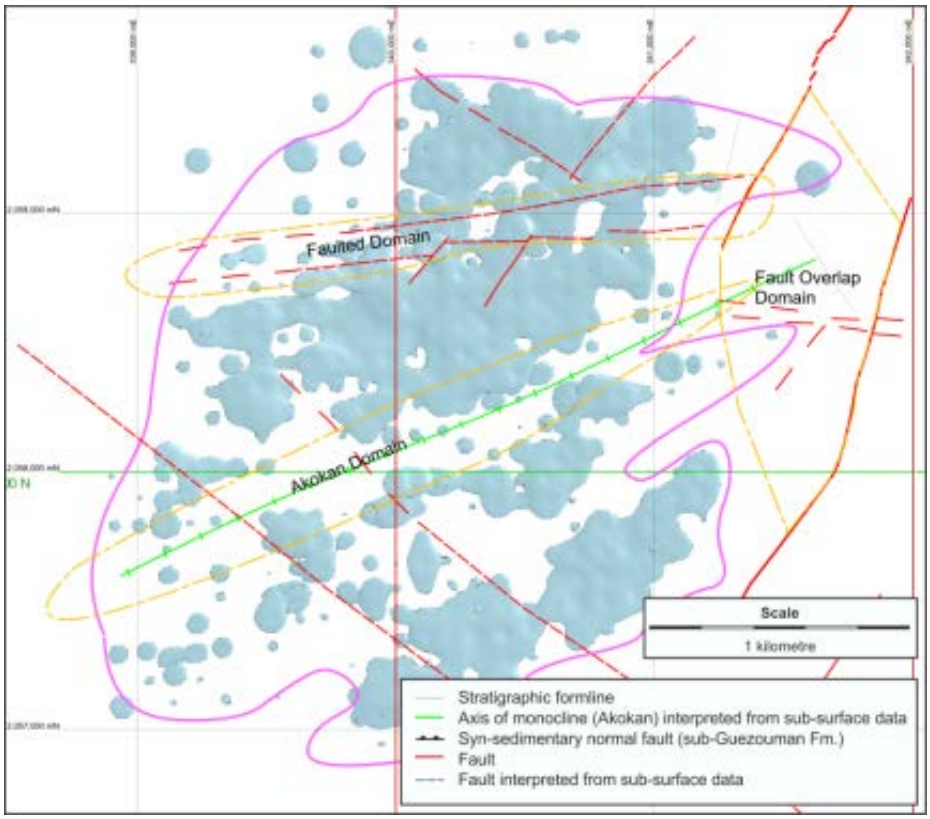


Figure 23-57: Plan view of structural domains relative to the Leapfrog 250 ppm U grade shell for MSNE.

Stratigraphic Dip

The stratigraphic dips in the MSNE deposit area are relatively subdued away from significant structures. Dips calculated on the basis of horizon intercepts are commonly 0-3° and rarely exceed 5°.

Rock Fabric

ATV and oriented core data identify discontinuities present within the rock fabric. Comparisons of all data across the deposit (Figure 23-58) have identified the following main discontinuity sets within the hanging wall, footwall and orebody:

- **Closed Bedding planes**; sub-horizontal; generally smooth and likely to persist for approximately 3-10 m (considered to be dominant discontinuity set). These are considered closed features (i.e. would not affect fracture frequency) and therefore their strength component has a “bond strength” (cohesion) as well as a friction angle.
- **Open Bedding Planes (bedding parallel joints)**; sub-horizontal; similar to bedding planes but are open, i.e. have aperture typically ranging from 0.1-1.0 mm. Occasionally some surface staining is present, but generally there is no infill and joint wall strength is equal to the intact rock strength.
- **NW-SE striking sub-vertical joints.** These are likely secondary joints which are cut by bedding in all lithologies. Aperture typically ranges from 0.1-1.0 mm.

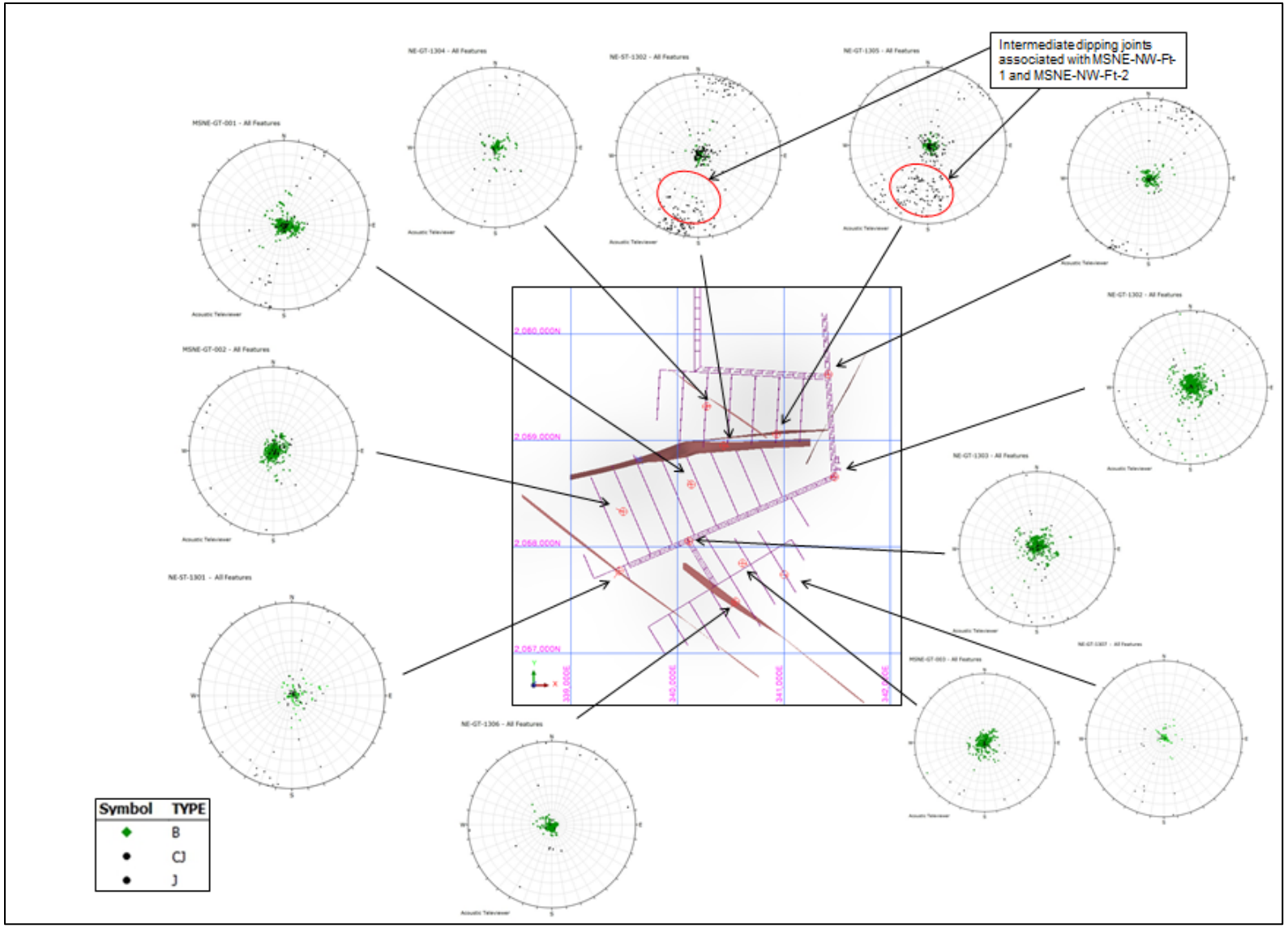


Figure 23-58: Stereonet plots of ATV data showing variation in rock fabric across the MSNE deposit.

Rock Mass Model

Rock mass assessment has been conducted to review variability of rock mass characteristics of the three lithology units that form the orebody and immediate hangingwall (Guezouman), and footwall (UA or Talak) of the MSNE deposit.

Guezouman Unit – Hangingwall and Orebody

The Guezouman sandstone unit is relatively massive with an average fracture frequency per meter of 0.16 and RQD of 99%. The intact UCS from Point Load Testing ranges from 15-70 MPa, and the laboratory UCS test results average at 35 MPa. A relatively low average unit weight of 21 kN/m^3 is likely the reason for the spread in intact rocks strength. 40 MPa was selected as design intact UCS. The average RMR value is 75 and GSI observed in the core box photographs range between 65-75, indicating a good quality rock mass. Very little variation of rock mass parameters is observed at the hangingwall and mineralized depths.

UA Unit - Footwall

A single UA channel exists between the Guezouman and Talak in the southern area of MSNE deposit with thickness reaching up to 40 m.

Although thinly bedded mudstones are frequently visible, very few discontinuities exist as indicated by average fracture frequency of 0.11 per meter and RQD of 100%. In terms of intact rock strength, six PLT results exist within the UA unit that ranges from 10-25 MPa. This result fits in the lower range of the intact UCS observed in UA channels in the MM project area. A lower design intact UCS of 30 MPa has been chosen. The average RMR is 63 and estimated GSI is in the range of 55-65.

Talak Unit - Footwall

The Talak unit, which forms the footwall for the northern section of the MSNE project area, is relatively isotropic rock mass with an average fracture frequency of 0.09 per meter. The intact rock strength is considered extremely weak. The PLT and laboratory UCS testing suggests a typical UCS value of 30 MPa. The average RMR obtained from geotechnical logging is 63. The estimated GSI varies between 50-60.

Geotechnical Domains

Geotechnical domains at MSNE are governed by lithology. Therefore geotechnical design parameters have been chosen for each formation and are given in Table 23-36.

Means and ranges of rock mass properties per geotechnical domains, including intact UCS, rock mass compressive strength (RMS), Laubscher RMR, and estimated GSI from core photograph observations are listed in Table 23-36.

Table 23-36: Geotechnical design parameters for MSNE

Geotechnical Domain	Rock Type	Density (kg/m^3)	Design Intact UCS (MPa)	RMS (MPa)	RMR_{90}	GSI
Guezouman (Hangingwall/Orebody)	Sandstone	2200	40	13.9	75	65 - 75
UA (Footwall)	Silt/Mudstone	2200	30	5.6	63	55 - 65
Talak (Footwall)	Clay/Mudstone	2450	30	3.9	63	50 - 60

Geotechnical Design

Empirical Analysis

The first stage of design involved the use of empirical methods to determine the roof span, roof support system, and the pillar dimensions. Below are the methodologies:

- Span design: The empirical span design curve by Ouchi (2004) was used to determine a stable roof span in the Guezouman domain;
- Roof stability analysis: Structural sets identified from oriented core logging and acoustic televiewer surveys were assessed for potential wedges forming in the roof. Sub-horizontal discontinuity features within the immediate 10 m of the roof were also observed for potential for slip or separation under load;
- Ground support design: The results of roof stability analysis were considered to determine a suitable ground support design. The design is based on SRK's recent experience in design and review of bolting systems in similar ground conditions; and
- Pillar design: The pillar load, estimated using Tributary area method, was compared to pillar strengths calculated using empirical formulae by Lunder and Pakalnis (1997). The design criteria considered are for factor of safety ≥ 1.5, and width to height ratio ≥ 2.

Results from the empirical analyses are as follows:

- Maximum span remains at 10 m (7 m room and pillar openings) as per Marianne-Marilyn;
- Two main joint sets were identified across the MSNE deposit; sub-horizontal open bedding planes and NW-SE striking sub-vertical joints. Additional north, northeast intermediate dipping joint sets are encountered in holes NE-ST-1302 and NE-GT-1305 that are associated with the NW fault system. Thin wedge will forms in north area (trending 020°) of the faulted zone;
- Full column resin grouted 1.5 m long #6 (20 mm nominal diameter) rebar bolted on a 1.2 m by 1.2 m square pattern is recommended to support the bedded hangingwall and potential wedges; and
- Empirical analysis indicates that 4 x 4 m square pillars are inadequate at mining depths of MSNE. Instead, 5 x 5 m or 6 x 6 m pillars are stable for cover depths of 120 m and 160 m respectively.

Numerical Modelling

The second stage of geotechnical design involved the use of numerical model analysis to confirm and optimize the design obtained from empirical study. Below are the methodologies:

- Map3D was used to build a mine scale 3D conceptual model for examining the mining induced stresses around excavations. Since geotechnical design parameters of Guezouman

and Talak are equal to that of Marianne-Marilyn, the same computer model was used.

- Phase2 (Rocscience) was utilized to determine the factor of safety of different types of pillars at varying depth and dip of mineralization. The behaviour or roof and floor are also observed for potential punching of pillar. The design input parameters are shown in Table 23-37.

Table 23-37: Rock mass properties used in Phase2 numerical modelling.

Geotechnical Domain	GSI	m_i	Intact Rock UCS (MPa)	Young's Modulus (GPa)	Poisson's Ratio	Generalized Hoek-Brown Constants		
						m_b	s	a
Guezouman	70	17	40	15.6	0.40	5.823	0.0357	0.501
UA	60	7	30	7.0	0.26	1.678	0.0117	0.503
Talak	55	4	30	2.3	0.21	0.802	0.0067	0.504

The results are as follows:

- Table 23-38 summarizes the mine scale 3D stress analysis in terms of maximum stress at the pillar core.

Table 23-38: Summary of 3D stress modelling results.

Pillar Type	Depth (m)		
	100	125	160
	Maximum Stress (MPa)		
Final pillars (5 x 5m)	11	13.5	16
Primary barrier pillars	4	5	7
Secondary barrier pillars	7	8	10
Central drift / Mining panel pillar	3.5	4	4.5

The results of 2D pillar modelling are summarized in terms of strength factor at pillar core in Table 23-39. Taking into consideration roof and floor behaviour, stable ground conditions can be achieved by using 5 x 5 m or 6 x 6 m pillars at all depths.

Table 23-39: Summary of 2D pillar modelling results.

Pillar Type	Depth (m)		
	100	125	160
	Strength Factor at Pillar Core		
Final pillars (4 x 4m)	1.5	1.5	1.4
Final pillars (5 x 5m)	2.1	2	1.9
Final pillars (6 x 6m)	2.4	2.3	2.2
Primary barrier pillars	9.9	8.9	7.8
Secondary barrier pillars	4.8	4.7	4.3
Central drift / Mining panel pillar	6.5	6.1	5.2

Room and Pillar Design Sectors

Room-and-pillar design sectors (Figure 23-59) have been developed based on depth of cover and dip of the orebody. Pillar dimensions, panel widths and extraction ratios for each design sector are given in Table 23-40.

Figure 23-59: MSNE geotechnical design sectors.

Table 23-40: Geotechnical design summary.

Design Sector	Depth	Dip	Pillar Dimension			Panel Width	Extraction Ratio	Comments
	m	°	Width (m)	Length (m)	Height (m)	m	%	
1	100-120	0-7	5	5	2	7	83	Moderate depth and minor dipping ore; conventional R&P with 5 m square pillars
2	120-140	0-7	5	5	2	7	83	
3	120-140	0-7	5	5	2	7	83	
3a	120-160	7-16	6	6	2	7	79	Deep and moderate-to-steeply dipping; apparent dip R&P or stepped R&P with 6 m square pillars
4	140-160	0-7	5	5	2	7	83	Deep and minor dipping ore; conventional R&P with 5 m square pillars
5	120-140	0-7	5	5	2	7	83	Moderate depth and minor dipping ore; conventional R&P with 5 m square pillars
6	140-160	0-7	5	5	2	7	83	Deep and minor dipping ore; conventional R&P with 5 m square pillars

The following excavation criteria apply to the production panel designs:

- Production panels are developed with initial pillar dimension, which is equivalent to two times the length of final pillar plus an opening width e.g. Sector 1 would be (2 x 5m) + 7m = 17m;
- Final pillars are formed on retreat by splitting the initial pillars through the center of the pillar length;
- Pattern roof bolting is to be applied in the production area with 1.2 m by 1.2 m spacing using 1.5 m long #6 (20 mm nominal diameter) full column resin grouted rebar;
- 15 m barrier pillars are to be left between adjacent mining panels, and secondary barrier pillars with a minimum of 10 x 15 m dimension should be left every 100 m span;
- Minimum of 25 m spacing should exist between the central access drift and production panels; and
- 20 m rib pillars should exist around identified fault structures.

Decline

Access to MSNE and Maryvonne deposits will be through a surface portal into a central drive in the northeast area of the MSNE deposit. The decline is approximately 700 m in length, and plunges at 8° to reach the mineralized depth. Geotechnical hole NE-GT-1301 was drilled to characterize the likely encountered ground conditions. Figure 23-60 shows the location of mine access decline respective to the mine plan.

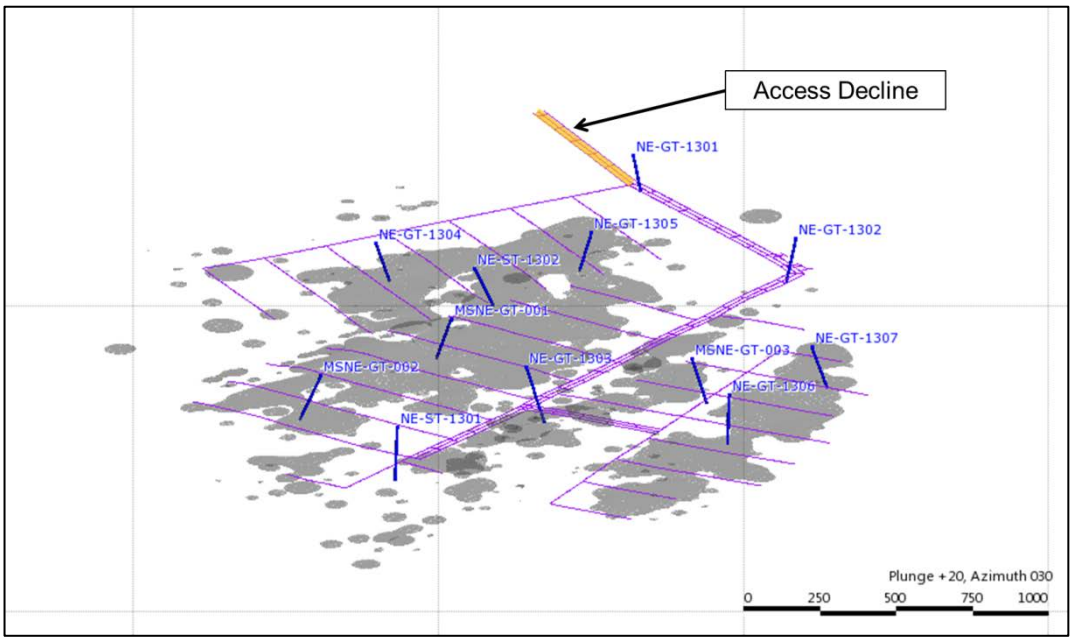


Figure 23-60: MSNE deposit access decline.

From surface the access decline passes through the Tchinezogue, formation which is approximately 70 m thick in this area. The upper 40 m is weathered and oxidized due to fluctuation of the groundwater table. Underlying the Tchinezogue is the Guezouman sandstone formation. The expected ground conditions and the geotechnical parameters for the length of the decline are given in Table 23-41.

Table 23-41: Litholgies and geotechnical parameters along the length of the MSNE decline.

Decline length	Formation	RQD Design Value (%)	RMR_{90} Design Value
0m - 0+290m	Weathered Tchinezogue	97	62
0+290m - 0+500m	Fresh Tchinezogue	99	67
0+500m – 0+700m	Guezouman SST	100	77

The following ground support is recommended for the decline; 2.4 m long #8 (25 mm) resin grouted rebar on 1.4 m centres through welded wire mesh in the back. Ribs to be supported with 1.5m long #6 (20 mm) resin grouted rebar installed on 1.5 m centres through welded wire mesh that has been extended to within 1 m of the floor. The maximum mesh size of the welded wire mesh should be 100 mm (4") square, constructed from #8 gauge wire or thicker.

23.3.5 Maryvonne Geotechnical Design

Maryvonne deposit will be mined by conventional room and pillar method with access from the south via MSNE (Figure 23-61). No geotechnical data exists for this deposit. Data currently available for analysis is:

- GoviEx exploration database;
- Wireframes of geology contacts generated by SRK from exploration drilling;
- Leapfrog mineralisation grade shells; and
- Quickbird Imagery.

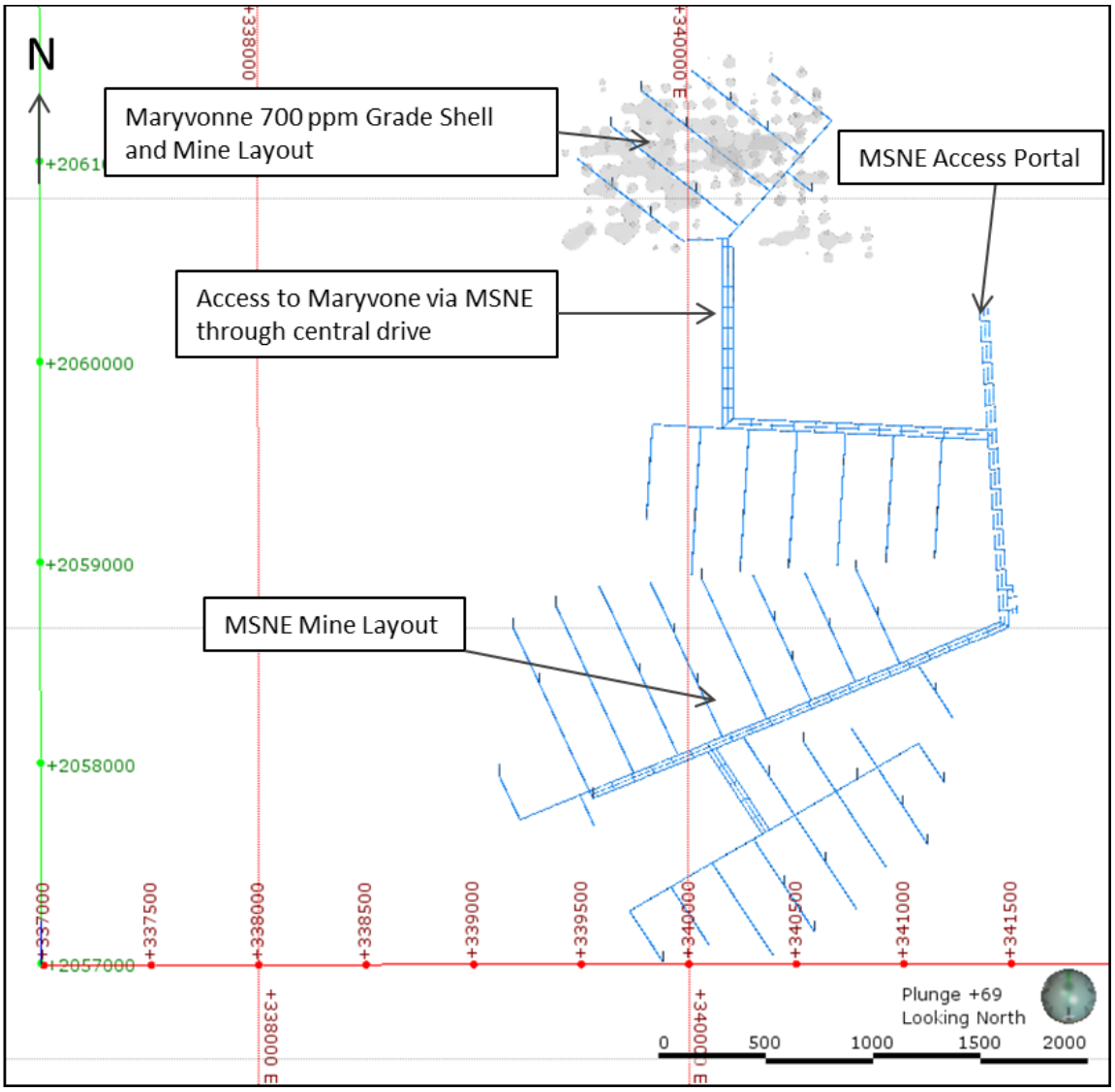


Figure 23-61: Location of Maryvonne in relation to MSNE.

Structural Geology

At surface, Maryvonne is located in a relatively sandy area with interrupted exposure of the Izegouande Formation, similar to MSNE.

Structurally the deposit is located 6 km west of the regional NNE-SSW striking Madaouela Fault. Unlike Marianne-Marilyn, to the north, domes do not affect the Maryvonne deposit area.

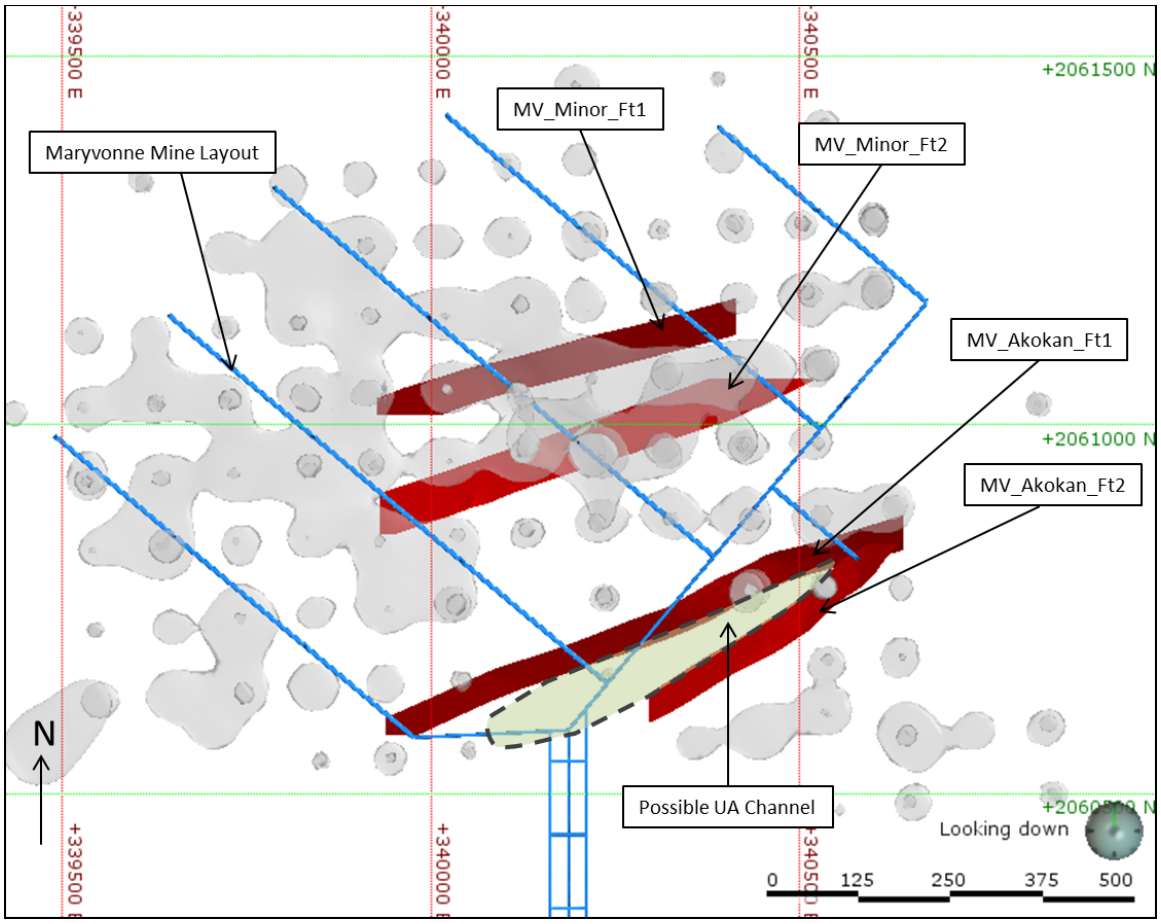


Figure 23-62: Positions of the interpreted faults and mine layout of Maryvonne. The inferred UA channel and Leapfrog 700 ppm grade shell is shown.

Rock Mass Conditions

In the absence of geotechnical data, rock mass conditions have been extrapolated from the MSNE deposit since the lithological conditions are similar. Rock mass properties for each lithology are assumed to be the same as those at MSNE given in Table 23-36.

Design Sectors

Geotechnical design of Maryvonne is based on analysis done at Marianne-Marilyn and MSNE, and applying the existing design sectors generated.

Maryvonne is expected to be similar to Design Sector 2 at MSNE. However, a reduced extraction ratio and stope size is incorporated into this design due to the absence of geotechnical data. Recommended pillar dimensions, panel widths and extraction ratios are given in Table 23-42.

Table 23-42: Recommended pillar dimensions, panel widths and extraction ratios

Depth	Dip	Pillar Dimension			Panel Width	Extraction Ratio	Comments
m	°	Width (m)	Length (m)	Height (m)	Opening Width (m)	%	
120-140	0-10	6	6	2	7	79	Limited geotechnical data within Maryvonne means a conservative design approach is taken.

The following excavation criteria apply to the production panel designs:

- Production panels are developed with initial pillar dimension, which is equivalent to two times the length of final pillar plus an opening width e.g. Sector 1 would be (2 x 5m) + 7m = 17m;
- Final pillars are formed on retreat by splitting the initial pillars through the center of the pillar length;
- Pattern roof bolting is to be applied in the production area with 1.2 m by 1.2 m spacing using 1.5 m long #6 (20 mm nominal diameter) full column resin grouted rebar;
- 15 m barrier pillars are to be left between adjacent mining panels, and secondary barrier pillars with a minimum of 10 x 15 m dimension should be left every 100 m span;
- Minimum of 25 m spacing should exist between the central access drift and production panels; and
- 20 m rib pillars should exist around identified fault structures.

23.3.6 Miriam Geotechnical Design

Mining Method

Mineralization at Miriam is shallower, lower grade and thicker than the Mineralization at Marianne-Marilyn. The mineralisation is generally near surface (upper 110 m) and will be mined by open pit. The proposed April 2013 pit is approximately 1.5 km long by 250 m wide and 110 m deep.

Geology

Geology within the Miriam deposit consists of three main sub-horizontal sedimentary units. 3D wireframes of geological contacts have been generated by GoviEx from exploration drilling and supplied to SRK. From shallowest to deepest, the main units are Tchinezogue, Guezouman and Talak and can be described as follows:

- Tchinezogue is a fluvio-deltaic sedimentary unit. Bedding is generally thin and sub-horizontal. In upper 30-40 m this unit is weathered and oxidised, as a result of exposure caused by retreat of the groundwater table.
- Guezouman formationis a coarse grained sandstone mainly composed of quartz with K-feldspar, plagioclase, and clay minerals. It is relatively massive with low fracture frequency, although some “disking” is occasionally observed due to bedding planes being opening by drilling. It is equigranular and has a low density due to the relatively large percentage of pores between grains. Some fine-grained sub-formations exist within the Guezouman Sandstone.
- Talak is a grey-black fine grained shale/argillite unit and forms the base of the stratigraphic

sequence at Miriam. The majority of mineralisation sits within the Guezouman Sandstone immediately above the Talak argillite. The Talak will form the floor of Miriam pit.

Structural Model (Miriam)

Structural Setting

Miriam is located within the southern extent of the Madaouela license area. Structurally the deposit is located directly to the west of the regional NNE-SSW striking Madaouela fault, which accommodates approximately 300m of vertical down-to-the-east displacement.

23.3.7 Major Structures

Faults

Unlike the other deposits considered herein, Miriam is not associated with any notable domes or monoclinal structures. Only two faults have been explicitly modelled in 3D in the area of the Miriam deposit.

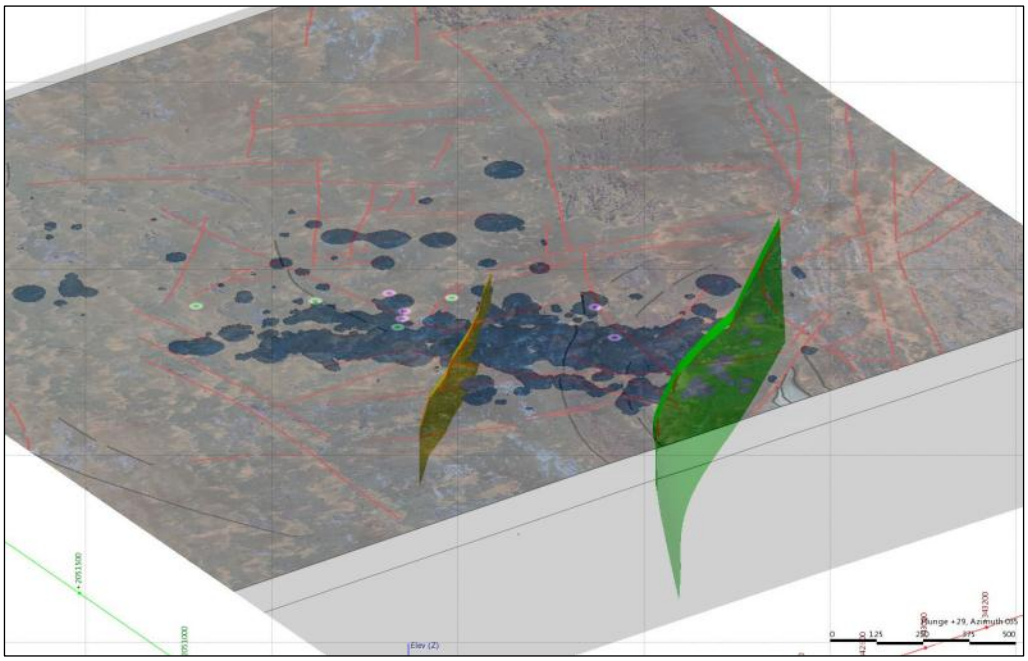


Figure 23-63: Miriam faults shown relative to topography with Quickbird drape and Leapfrog 400 ppm U grade shell. (Green fault - MI-NE-Ft-1; Yellow fault - MI-NE-Ft-2).

Stratigraphic Dip

Stratigraphic dip based on drillhole interval elevations are low, predominantly 0-4° over the main pit area. Towards the south of the planned open pit, there is a minor decrease in the average dip and dip variability.

Rock Fabric

Stereonet plots of oriented core drilling and ATV surveys have been reviewed to assess variation in rock fabric across the pit (Figure 23-64). There exists a good correlation between ATV and oriented core drilling.

The main discontinuity sets identified in all lithological units at Miriam include:

- **Closed Bedding planes**; sub-horizontal; generally smooth and likely to persist for approximately 3-10 m (considered to be dominant discontinuity set). These are considered closed features (i.e. would not affect fracture frequency) and therefore their strength component has a “bond strength” (cohesion) as well as a friction angle.
- **Open Bedding Planes (bedding parallel joints)**; sub-horizontal; similar to bedding planes but are open, i.e. have aperture typically ranging from 0.1-1.0 mm. Some surface staining is present, but generally there is no infill and joint wall strength is equal to the intact rock strength.
- **NW-SE striking sub-vertical joints**; mutually perpendicular to bedding. These are less common (in part due to orientation being sub-parallel to drillhole plunge), and likely to be cross cut by bedding.

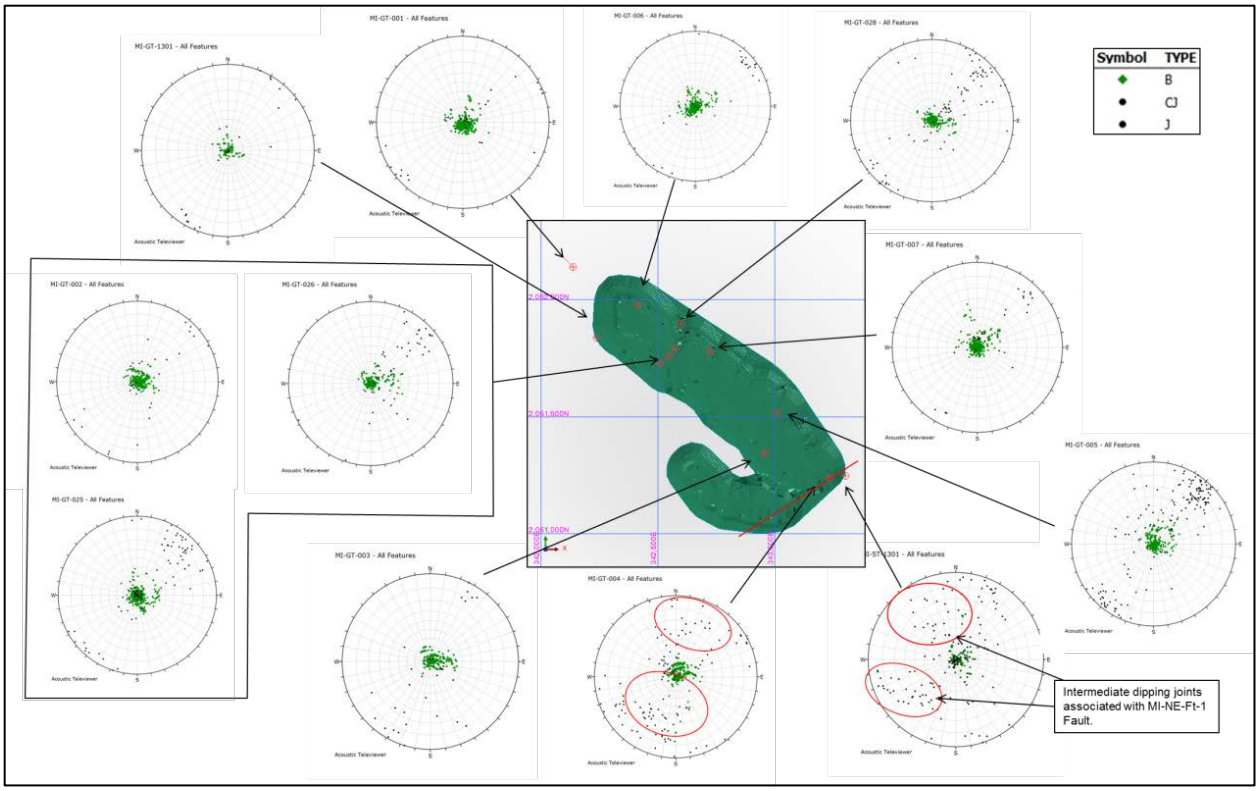


Figure 23-64: Stereonet plots of ATV data for each hole to indicate any variation in rock fabric across the pit. Holes in the southeast show an intermediate dipping set likely due to intersection with the MI-NE-Ft-1 Fault.

Structural Domains

Interpretation of fault structures and rock fabric indicates that two structural domains exist at Miriam pit.

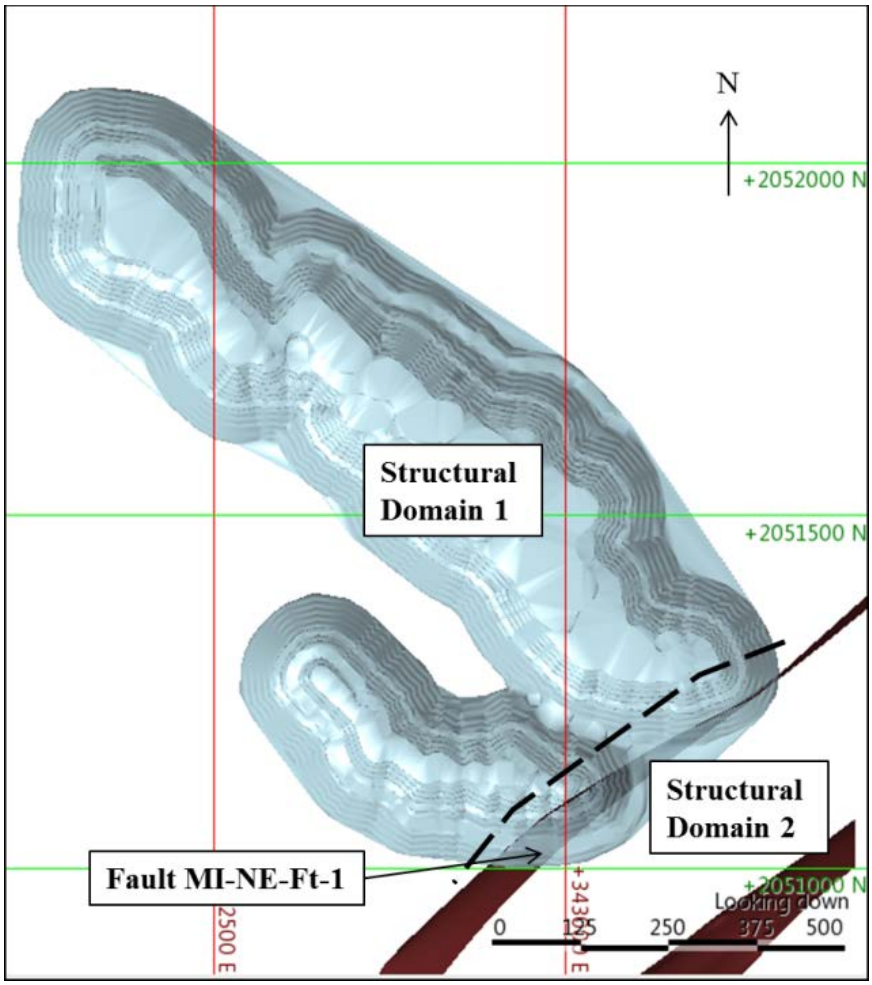


Figure 23-65: Structural domains of preliminary Miriam Pit; governed by presence of Fault MI-NE-Ft-1 in the south-east of the pit.

Rock Mass Model

The rock mass model is a representation of the engineering properties of the rock mass for use in stability analysis. This includes the properties intact rock pieces, the structures cutting through the rock mass, and the overall rock mass itself.

Rock Mass Domains

Rock mass domains have been generated based primarily on lithology and location of the water table causing weathering and oxidation of the Tchinezogue unit. Details of each domain and associated geotechnical parameters are given in Table 23-43. Domains cut to the April 2013 pit shell are shown in Figure 23-66.

Table 23-43: Rock Mass domains and corresponding design parameters for Miriam Pit.

Domain	UCS (MPa)	RMR_{89}	GSI
Tchinezogue (weathered)	50	65	60
Tchinezogue(unweathered)	70	70	65
Guezouman	40	75	70
Talak	25	55	50

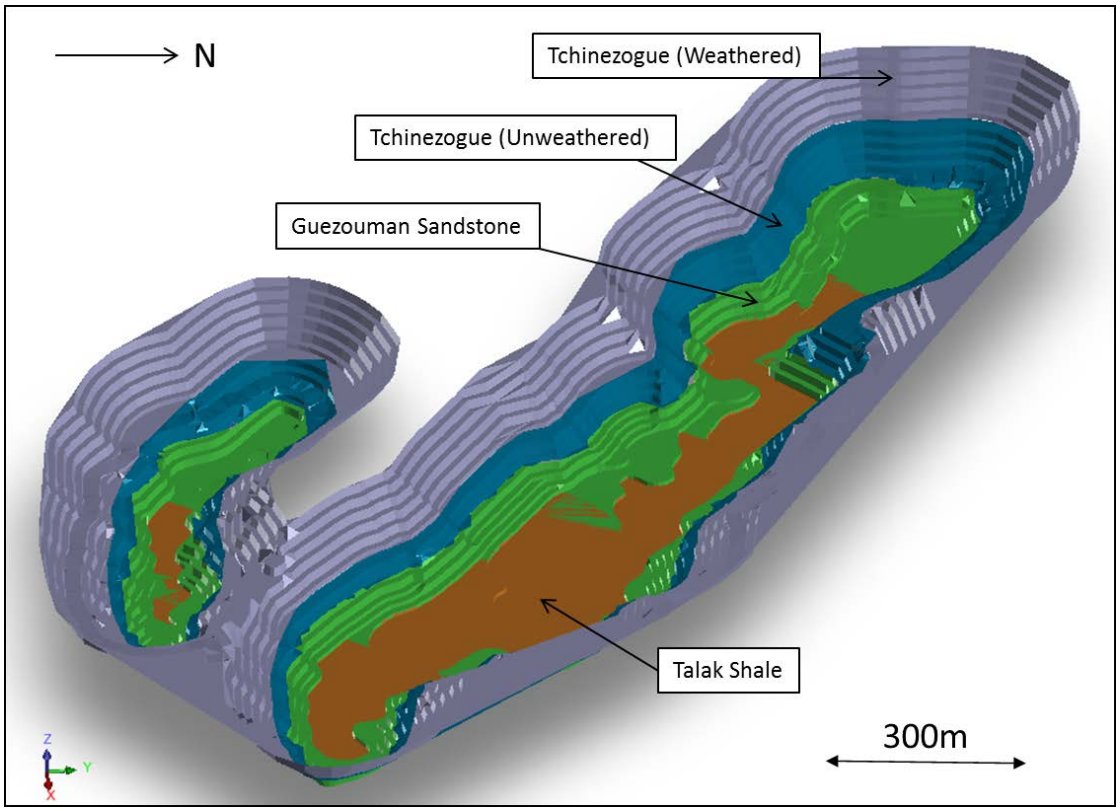
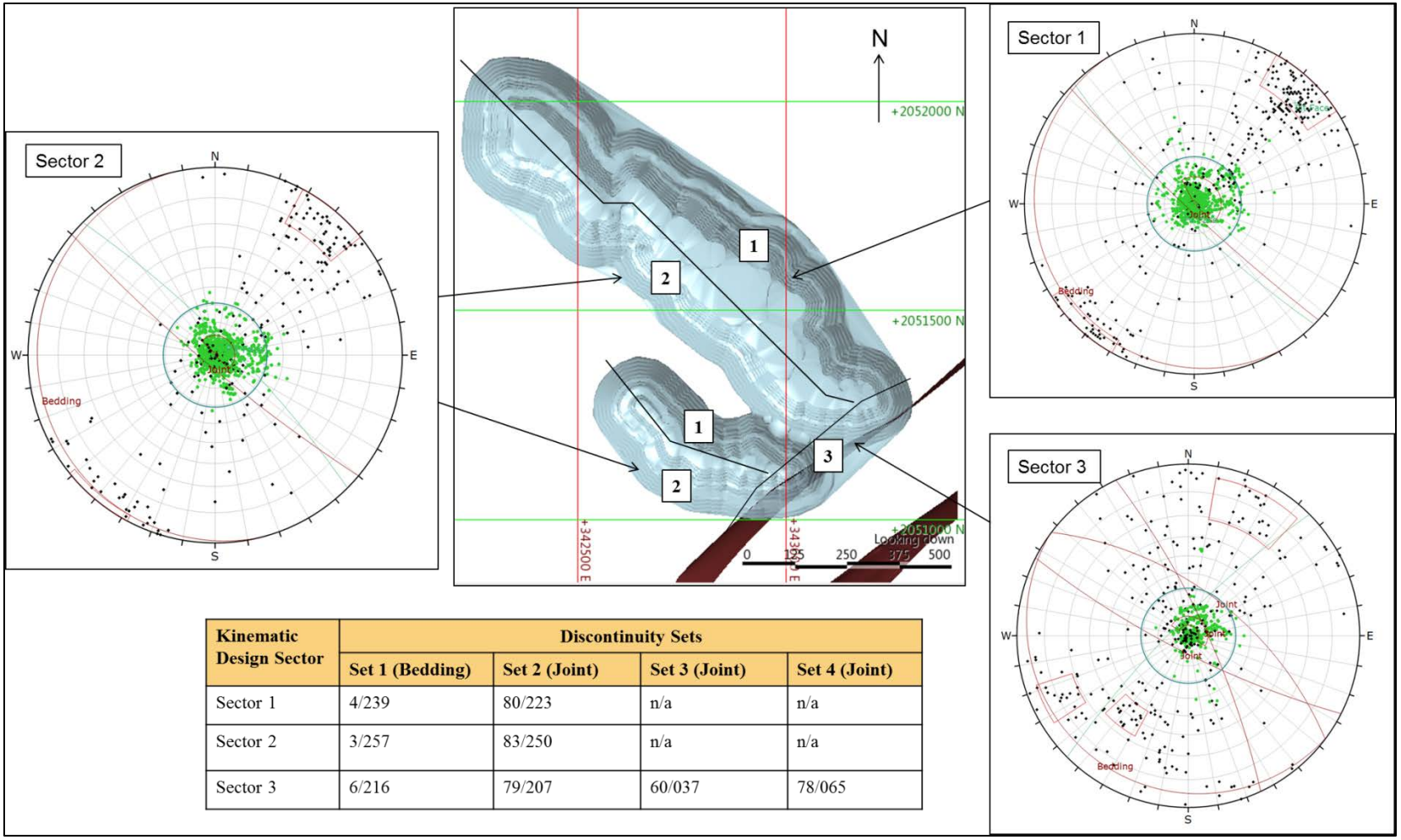


Figure 23-66: Rock mass domains cut to the Miriam pit shell.

Kinematic Evaluation

Kinematic slope modelling assesses the structurally controlled failures (planar, wedge toppling). This has been undertaken to assess bench and inter-ramp stability using the joint sets. For kinematic analysis the pit is divided into three design sectors based on structural domains and pit face orientations. These design sectors and the main joint sets in each sector are detailed in Figure 23-67. Details of each assessment are given in the following sub sections.



Kinematic Design Sector	Discontinuity Sets			
	Set 1 (Bedding)	Set 2 (Joint)	Set 3 (Joint)	Set 4 (Joint)
Sector 1	4/239	80/223	n/a	n/a
Sector 2	3/257	83/250	n/a	n/a
Sector 3	6/216	79/207	60/037	78/065

Figure 23-67: Kinematic design sectors for Miriam Pit. Stereonets of ATV data indicate the main joint sets present in each sector.

Sliding Failure

Sliding failure is kinematically possible where a discontinuity plane is inclined less than the slope face (daylights) and at an angle greater than the friction angle. Failure analysis has been undertaken using the software RocPlane (Rocscience).

Table 23-44: Discontinuity strength input parameters.

Base Friction Angle (°)	JRC	JCS
31	6	50

Analysis was done for bench and interramp scale planar failure using the strength parameters in Table 23-45. Because of blast damage and relatively low stresses involved, cohesion along discontinuities was ignored. Results are shown in Table 23-45.

Table 23-45: Factor of safety values of planar failure for each Sector at a bench and inter-ramp scale.

Kinematic Design Sector	Joint Set (dip/azimuth)	Bench Scale (10m height; 85° slope angle)	Inter-ramp Scale (60m height; 60° slope angle)
Sector 1	80/223	0.32	Joints do not daylight – no failure observed
Sector 2	83/250	0.25	Joints do not daylight – no failure observed
Sector 3	79/207	Joints do not daylight – no failure observed	Joints do not daylight – no failure observed

Results of the model indicate that planar failure is very likely on a bench scale in the NE and SW sidewalls of the pit. However, since the joint orientations are sub-parallel to bench faces, the size of the wedge is relatively small and will likely break back during blasting. Also, joint persistence is not expected to be high since bedding will likely cross-cut the joint sets.

In summary, localised sliding, and backbreak is expected along sub-vertical joints during blasting, but by using single benching plus suitable berm widths, steep (85°) bench face angles can be used for pit design in all walls.

Wedge Failure

Very few wedges are formed by the discontinuity sets in all Kinematic Design Sectors; therefore the likelihood of wedge failure is low throughout the pit.

Toppling Failure

There is a low likelihood of toppling failure across the pit. Although sub-vertical joint sets exist, they will likely be cut by bedding and therefore are not expected to persist for significant distances. Also, sub-vertical joint spacing is not expected to be close enough for toppling to occur, benches may break back along these sub-vertical joints during blasting.

Overall Slope Stability

Numerical modelling has been undertaken to assess the potential for inter-ramp and overall slope failures where the failure mechanisms are controlled by strength of the rock mass.

Based on the geotechnical domains and modelling results, two Design Sectors are required for Miriam Pit:

- The Weathered Zone in upper 40 m around the whole pit; and
- Unweathered Zone below 40m depth across the entire pit.

Slope Configurations have been calculated for each Design Sector and are given in (Table 23-46). For a pit depth of 110 m the overall slope angle will be 48° incorporating one 20 m wide geotechnical bench along each wall.

Table 23-46: Slope configurations for each design sector.

Design Sector	Pit Depth (m)	Bench Height (m)	Bench Face Angle (°)	Bench Width (m)	Interramp Angle (°)	Comments
Weathered Zone	0-40	10	75	6.5	47	Joint frequency higher in Weathered Zone, therefore bench face angle limited to 75° to reduce back-break along sub-vertical joints
Unweathered Zone	40-120	10	85	6.5	54	Bench stack height limited to 60 m i.e. 20 m wide geotech berm or ramp required every 60m to decouple slopes

In addition to the slope design configurations, SRK make the following comments:

- Target Factor of Safety for overall slope is 1.3;
- Bench stack height is limited to 60 m on all walls. A geotechnical berm (or ramp) of 20 m minimum width is required every 60 m to de-couple the slope;
- Fault MI-NE-Ft-1 is thought to dip sub-vertically and intersects the large pit endwall close to the crest. Within the smaller pit shell this fault may intersect in the lower third of the endwall. Although favourably oriented, this structure may result in weak material present behind the smaller pit endwall. De-coupling the slope at 60 m height (as per previous requirement) will reduce the likelihood of failure associated with this fault zone. Also, the endwalls are relatively small and likely to have good confining stresses which will increase stability;
- Bench width is large enough to account for some back-break and minor bench-scale sliding along the sub-vertical joint sets;
- Reduced quality rock mass exists in the Weathered Zone therefore a lower bench face angle required; and
- Good pre-split blasting to minimize wall damage.

Operational Considerations

Blasting

SRK recommend good, controlled blasting to maintain rock mass integrity during mining and achieve the required bench face angles. Controlled blasting is recommended on final walls and involves smaller diameter blast holes detonated as a post shear (cushion) line; typically applicable to weak rock such as the Guezouman and Talak. For harder rock, blasthole lengths should be staggered such that the holes do not intercept the crest of the bench below.

The requirements for interim pit slopes are less rigorous due to the shorter operating life of the walls. Trial blasts are recommended in the initial pit phases in order to evaluate and optimise the best performance.

Bench Maintenance

SRK recommend that the benches be kept clear of fallen material and bench faces be maintained regularly so they remain functional during mine operations. Routine scaling is recommended after blasting to achieve stable bench face angles and may allow bench widths to be minimised by reducing the volume of material to be controlled.

Pit Floor Heave

Some pit floor heave can be expected within the Talak unit due to relatively high pore pressures and low normal stresses. The amount of heave is expected to be less than 1m in total and can be managed on an ad-hoc basis with suitable machinery.

Groundwater Control

Natural drainage of the wall rocks will begin to occur once the pit is excavated below the water table. The passive (natural) drainage from the Tchinezogue and Guezouman units is expected to be sufficient for achieving the design slope angle. The rate of pore pressure dissipation in the Talak unit by passive dissipation alone may not keep up with mining advance rates. However, at present slope depressurisation is not considered necessary because:

- The overall slope height is relatively small;
- The Talak unit is only present in the toes of final walls and the final pit floor; therefore minor heave may occur but slope stability not likely to be significantly impacted;
- The economic benefit of an increased slope angle (and subsequent reduced stripping ratio) may not be significant; and
- Since the climate is dry and recharge is low, trafficability issues are not expected to outweigh the need for a depressurisation program.

Slope Monitoring Program

Pro-active geotechnical slope monitoring is recommended as the pit develops. Typical monitoring requirements include, but are not limited to:

- Installation and regular monitoring of prisms along NW and SE highwalls. Prism surveying should be undertaken at regular intervals to develop a comprehensive record of any wall deformation;
- Detailed geotechnical mapping along newly formed benches on the pit highwalls. This data should feed into a database which is used to update and modify the geotechnical model and stability analysis as necessary;
- Regular slope inspections and identification of potential instabilities (e.g tension cracks); and
- Installation of Vibrating Wire Piezometers in units where the rate of pore pressure dissipation may influence slope stability.

23.4 Project Alternatives

The purpose of this chapter is to provide a description of the possible alternatives to the project in accordance with the requirements of Art. 7 of ESIA Decree (No. 2000-397/PRN/ME/LCD of 20 October 2000). The project site layout was determined by the location of the targeted deposits (Marianne, Marilyn, MSNE, Maryvonne, Miriam). Infrastructures and facilities were designed as close as possible to each other to minimise construction disturbed footprint, obstruction to land access and transportation. The section

first covers location, technical and operational alternatives considered during the project development process to date; then, concludes with an assessment of the no-mine alternative. Technical and operational alternatives, including mining and processing, have been described in greater detail in the appropriate sections.

23.4.1 Project alternatives

The specialists worked closely with the project design team to identify and analyse project alternatives during the study (SRK, 2014). Decisions of preferred alternatives were influenced by environmental and social considerations, as well as cost and technical criteria. The main alternatives investigated concerned mining method, processing options, waste management options and location of the infrastructures and facilities. These are synthesized in Table 23-47.

For the majority of alternatives (mining method, processing options, waste management and, option and location of the infrastructures and facilities) the preferred environmental and social option (based on a qualitative assessment) has been selected. However, in the case of open pit mining method proposed for the Miriam deposit, the preferred option identified by the qualitative environmental and social assessment is not the option selected by the design and engineering team due to economic and technical reasons (as explained below). The open pit mining method is not considered a 'fatal flaw' from an environmental and social perspective because although it is a less preferred option, appropriate management measures will be applied to reduce the negative environmental and social impacts to an acceptable level.

Table 23-47: Main project alternatives considered

Topic	Alternatives considered	Factors influencing decisions on alternatives		Selection
		Technical / financial factors	Environmental / social factors	
Mining method				
Mining method	In-Situ Leaching	**Positive** • restricted mining waste • minimum footprint • restricted explosive use • smaller labour force required **Negative** • loss of pregnant leach solution to the host aquifer • very low permeability, very low uranium recovery, not commercially viable • high water consumption • high leaching solution used underground • requires extended surface roads and wellheads	**Positive** • minor land disturbance • safer ground disturbance practice **Negative** • high water requirement in desert environment • potential for soil and groundwater contamination from leaching solution • less employment opportunities • vulnerability of aquifer	Not preferred, not selected

Topic	Alternatives considered	Factors influencing decisions on alternatives		Selection
		Technical / financial factors	**Environmental / social factors**	
	Underground Room and pillar	**Positive** • confined footprint • less mine waste generated • less explosive required • appropriate mining method to access narrow deeper deposits **Negative** • higher operating cost • higher labour force • ventilation costs	**Positive** • confined land disturbance • limited waste generation • low dust generation • limited visual impact • minor GHG emissions related to ore and waste transportation • simpler rehabilitation **Negative** • Point source for Radon emissions	Preferred technical, economical environmental and social option selected for Marianne-Marilyn and MSNE-Maryvonne deposits
	Open pit	**Positive** • access to shallower and larger bulk deposits **Negative** • larger mine waste generated • lower operating costs • extended footprint • higher explosive use	**Negative** • high land disturbance • higher dust and visual impact • increased GHG emissions and dust generation related to transport of waste	Preferred technical and economical option selected for Miriam deposit, which is close to surface
Processing options				
Comminution /Leaching	Milling followed by two stage tank leach	**Positive** • easier to design and operate • high acid use • higher processing waste • higher uranium recovery through processing **Negative** • higher water consumption	**Negative** • high water demand in desert environment	Not preferred, not selected
	Heap leaching of crushed ore	**Positive** • less acid use • lower operating costs **Negative** • higher water consumption • extended footprint required	**Positive** • minimum acid use **Negative** • high water demand in desert environment • potential for seepage to vulnerable aquifer • higher land disturbance	Not preferred, not selected
	Dry milling and acid pugging	**Positive** • lower water consumption **Negative** • higher capital and operating costs related to higher acid consumption	**Positive** • lower water demand **Negative** • significant corrosion and noxious fumes potential related to acid used • dust generation from dry milling	Not preferred, not selected

Topic	Alternatives considered	Factors influencing decisions on alternatives		Selection
		Technical / financial factors	**Environmental / social factors**	
	ROS and Ablation followed by two stage tank leach	**Positive** • Much lower contaminated waste generation • Lower operating and capital costs • Lower leach plant throughput • Low water use • Reduced plant footprint	**Positive** • Low radiation ROS and ablation rejects can be used for concurrent rehabilitation • Low tailings generation • Low water use • Reduced GHG emissions	Preferred technical, economic and environmental option selected for the Project
Downstream uranium recovery (from the pregnant leach solution)	Resin in pulp and solvent extraction	**Positive** • robust uranium recovery option • lower operating and capital costs • conventional commercially available reagents **Negative** • solvents cross contamination problems • potential damage to the resin by coarser particles • high water consumption	**Negative** • higher water demand in desert environment • significant corrosion potential	Not preferred, not selected
	Solvent extraction (Cyanex 600)	**Positive** • extraction reagent capable of extracting Mo in addition to U, enabling the recovery of a Mo product (98%) and lower impurity U product (99%) **Negative** • use of phosphoric acid	**Positive** • lower water demand • Molybdenum extracted as a product, reduced concentration sent to TSF	Preferred technical and environmental option selected for the Project
Mine waste management				
Tailings storage facilities	Conventional impoundment (slurry)	**Positive** • lower operating costs • higher ore throughput **Negative** • higher collapsing risk • extended footprint • higher process water consumption • higher reclamation costs	**Positive** • low potential for dust generation **Negative** • higher spillage and seepage risk • higher risk of failure • higher water demand and loss • higher land disturbance	Not preferred, not selected

Topic	Alternatives considered	Factors influencing decisions on alternatives		Selection
		Technical / financial factors	**Environmental / social factors**	
	Paste/ thickened disposal	**Positive** • high water volume recovered • evaporation losses minimised • reduced dam construction capital costs • storage capacity maximised • minimised footprint • reduced water storage **Negative** • higher capital operating costs • difficult to design • high energy use	**Positive** • moderate water demand • reduced seepage and spillage risk • reduced potential for acid generation and metal leaching • minimised land disturbance **Negative** • higher GHG emissions due to energy demand	Not preferred, not selected
	Dry stacked	**Positive** • process water consumption minimised • water storage eliminated • higher storage capacity • reduced closure costs • better recovery of process chemicals **Negative** • higher capital and operating costs • higher energy use	**Positive** • lowest risk of failure • lowest water demand • seepage and spillage risk eliminated • progressive rehabilitation possible • least land disturbance **Negative** • higher GHG emissions due to energy demand and transportation • higher potential for dust generation	Preferred technical and environmental option selected for the Project
Waste Rock	Dumping	**Positive** • Need to access ore without possibility to backfill **Negative** • extended footprint	**Positive** • reduced GHG emissions associated with transport **Negative** • potential for higher dust generation • visual impacts of dump • greater land disturbance	Preferred technical and environmental option selected for Miriam
	Underground backfilling of voids	**Positive** • reduced footprint • reduced closure costs • maximise extraction of ore • reduced underground and surface subsidence **Negative** • higher operating costs due to transport of waste rock from pit to underground workings	**Positive** • lower land disturbance • potential for dust generation restricted to road entrainment • visual impacts of dump minimised **Negative** • higher GHG emissions associated with transport • higher risk of traffic incidents	Not preferred, not selected for Miriam open pit waste rock; Preferred technical and environmental option limited to a waste rock fraction from underground operations

Topic	Alternatives considered	Factors influencing decisions on alternatives		Selection
		Technical / financial factors	Environmental / social factors	
Site layout				
Infrastructure	Permanent	**Positive** • Life cost value • Better quality **Negative** • higher needs for maintenance • higher cost at closure	**Positive** • increased local procurements potential • potential to be handed over at closure (though limited need in middle of desert) **Negative** • increased materials to decommission and dispose of at closure	Preferred for operational stage technical and environmental option selected for the Project
	Prefabricated	**Positive** • predesigned, avoid permanent foundations **Negative** • Limited lifespan	**Positive** • simpler remediation at closure including option of dismantling and handing over to third party	Preferred for construction stage technical and environmental option selected for the Project
Power supply	Separate self-generation at the mine	**Positive** • higher security of supply **Negative** • high capital costs • increased fuel/gas consumption and transportation	**Positive** • avoids the need for a power line corridor (to connect to existing grid power) **Negative** • increased localised GHG emissions from oil/gas combustion and transportation • higher risk of traffic incidents, including potential fuel spills, associated with transportation • renewable power generation would require a very large footprint	Not preferred, not selected
	Grid power supply to mine site with backup power generation	**Positive** • lowest net present costs **Negative** • backup power generation still required to assure security of supply • higher security of supply risk due to transmission line outages	**Positive** • increase grid development potential with potential access for local communities and industry • avoid local GHG emissions • avoid risk of spills associated with road transport **Negative** • new transmission line corridor between source and the mine area with related land disturbance	Preferred environmental and social option and selected for project

Topic	Alternatives considered	Factors influencing decisions on alternatives		Selection
		Technical / financial factors	Environmental / social factors	
Ore transport within the mine area	Belt conveyor	**Positive** • low operating costs • smaller workforce (relative to haul road and trucks) **Negative** • higher capital costs • failure in one section will halt the entire conveyor • higher closure costs	**Positive** • lower GHG emissions • lower risk of traffic incidents • less fuel transported to site **Negative** • smaller workforce • potential barrier to people and animals passage • increased sabotage risk relative to haul road • higher volume of material to decommission at closure	Preferred environmental and social option and selected for underground operations to transport ore and waste to the surface and on to the processing facility
	Dedicated haul road	**Positive** • truck breakdowns do not hinder or halt overall transport of ore between mine and process plant • smaller stockpiles required to buffer interruptions to transport **Negative** • high operating costs • increases overall fuel supply requirements	**Positive** • increased employment opportunities for local people • limited sabotage risk or community damage **Negative** • higher GHG emissions • increased risk of traffic incidents	Preferred environmental and social option and selected for open pit operation

23.4.2 No-mine alternative

The “no-mine alternative” has been considered throughout the impacts assessments for each of the Project aspects. The main conclusion drawn for the majority of the aspects are:

- The wider region is already impacted by 50 years of mining;
- The soil in the impacted footprint of the Project is of low agricultural productivity.

Some of the existing operations in the Arlit area have already began to ramp down production and are potentially going to ramp down production further. The positive impact of the job creation, royalty and tax incomes in a poor country are significant and are more significant for the local area when cognisance is taken of the ramping down of the adjacent operations.

Direct benefits of this Project will include employment opportunities, tax and royalty income, training and educational benefits, and hard currency from export earnings. Indirect benefits that stem from direct capital investment and operating expenditures will include the production, employment and income growth occurring in other businesses supplying inputs to the Project. Wherever practicable, services and materials will be procured in the local area.

Unless there are significant environmental and social impacts that cannot be effectively prevented or managed, the ‘no project’ alternative can be discounted for the following reason:

- None of the direct and indirect benefits noted (such as increased state revenue, poverty alleviation and offering alternative livelihoods to reduce pressures on natural resources) would accrue in the absence of development; the creation of alternative non-mining industrial developments in the permit area are unlikely to offer the same level of revenue

generation, employment and other benefits associated with mining. In this context, existing levels of poverty, unemployment and associated negative social impacts would continue.

23.5 Tailings Evaluation

SRK has developed designs for the Tailings Storage Facility (TSF), being considered for the disposal and storage of tailings and has provided cost estimates for each for use in the project economic model. The designs have been completed based on the design criteria for the current project as developed by others, applicable national and international standards, and where appropriate, based on SRK's experience.

For the TSF, SRK has selected a dry stacked TSF as the most appropriate concept. The TSF with a production rate of 2,250 tpd, assuming Radiometric sorted waste material does not have to be treated as tailings, has been designed to store a total of 16.5 Mt, or 10.9 Mm^3 of tailings. Dry stacked tailings, similar to the disposal methodology used at Somaïr and Cominak was selected as the most appropriate TSF.

The uranium recovery process will also generate ableed of process solution to eliminate high total dissolved solids (TDS) (Al, Mg, Fe and Mo). The waste bleed water, in addition to yellowcake thickener overflow and minor solvent extraction wastes, are to be sent to dedicated evaporation ponds. SRK has developed a TSF design including facility sizing, construction methodology, water balance, evaporation pond design and a closure plan.

23.5.1 Design Criteria

The TSF design criteria are summarised in Table 23-48. The source of the design criteria is listed in the tables.

Table 23-48: TSF design criteria

	Design criteria	Value
1	Target production rate	2,250 tpd
2	Life of mine	21 years
3	Maximum tailings storage required	16.5 Mt / 10.9 Mm^3
4	Tailings solids concentration	80% w/w
5	Final moisture content	20% w/w
6	Tailings beach slope	10%
7	Tailings deposited dry density	1.6 t/m^3
8	Tailings specific gravity	2.65
9	Embankment freeboard	0.5 m in addition to storm allowances
10	Design storm (24 hour, 1 in 100 AEP)	61 mm
11	Run-off coefficients (disturbed catchments)	0.75
12	Minimum embankment crest width	5 m
13	Peak Ground Acceleration (PGA)	N/A
14	SG of Bleed Stream	1.15

23.5.2 Background Data

Climate

The Madaouela Project is located in northern Niger, with a climate characterised as Saharan desert. The area has an extremely dry climate with hot temperatures and low rainfall all year round. Average monthly precipitation and daily evaporation data utilised in the design have been sourced from the Somaïr operation located near the town of Arlit, approximately 15 km from the Madaouela site.

Precipitation and evaporation

The wet season generally occurs between May and October, with an annual average rainfall of approximately 47 mm. There are notable exceptions, such as 1991, when 114 mm was measured for the year. Average monthly rainfall data utilised in the design are summarised in Table 23-49.

Table 23-49: Average monthly rainfall (mm)

Month	Somaïr site (1981-2011)	Arlit (1972-2005)
January	0.0	0.0
February	0.0	0.0
March	0.2	0.0
April	0.0	0.0
May	2.0	1.4
June	5.4	5.7
July	9.4	14.5
August	21.2	24.0
September	6.5	6.0
October	1.1	0.3
November	0.0	0.0
December	0.0	0.0
Average Annual	**45.7**	**52.5**

Daily evaporation data have been measured at the Somaïr site (from 1981-2005), with a recorded average of 12.3 mm/day, and 4,498 mm/year. The average daily and monthly evaporation data to be utilised in the design are summarised in Table 23-50.

Table 23-50: Average evaporation data

Month	Evaporation	
	Average Daily (mm)	Average Monthly (mm)
January	8.19	253.9
February	10.00	280.0
March	12.50	375.0
April	14.28	428.4
May	15.28	473.7
June	14.96	448.8
July	14.01	434.3
August	12.51	387.8

September	13.42	402.6
October	13.16	408.0
November	10.38	311.4
December	9.08	281.5
Average Annual		**4498**

Storm data

Storm data for the maximum rainfall per 24-hour duration was obtained from site records at the Somaïr site from 1977 through to 2005. The largest recorded daily precipitation event was 43 mm and occurred in 1991. An estimate was performed using the available storm recurrence intervals to determine storm depths for various return periods (Figure 23-68). Storm depths and frequencies are summarised in Table 23-51.

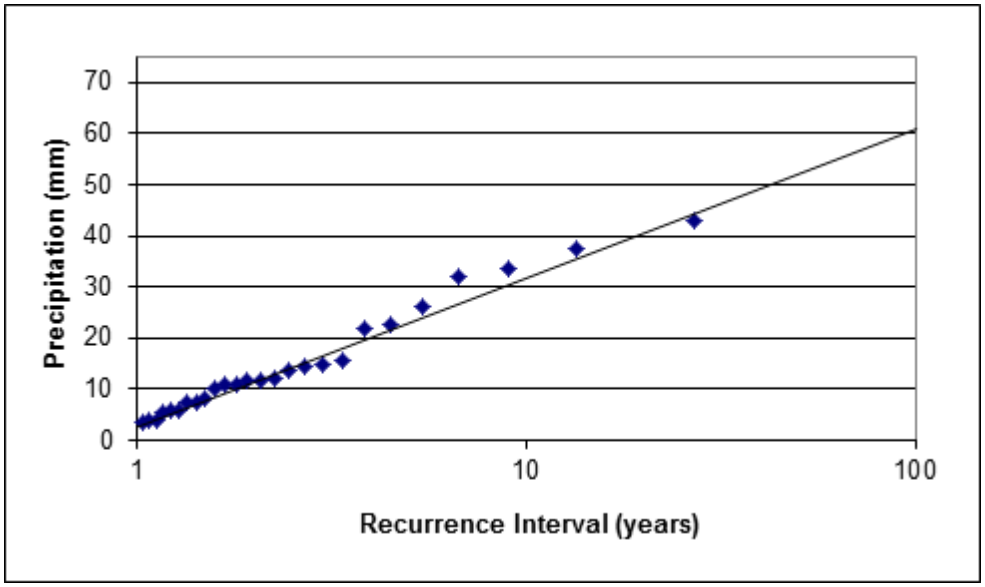


Figure 23-68: Storm size versus recurrence interval

Table 23-51: Design storms of 24-hour duration

Recurrence interval (Years)	Precipitation depth (mm)
5	25
10	35
25	44
50	55
100	61

Seismicity

SRK undertook a review of the United States Geological Services (USGS) website for seismic activities. The USGS website indicated that there has not been a recorded earthquake within 500 km of the site. As shown in Figure 23-69, the peak ground acceleration for a return period of 500 years (PGA) at the project area is negligible.

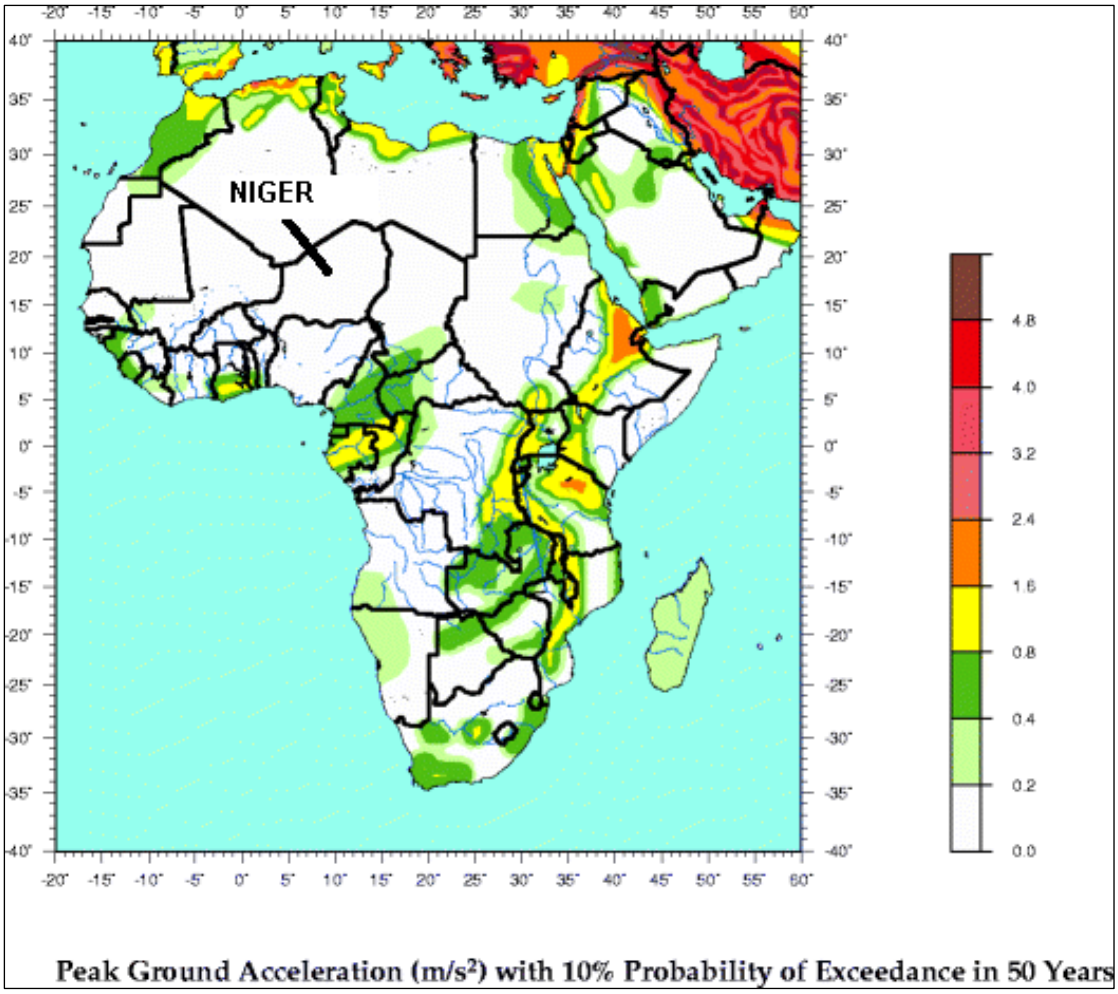


Figure 23-69: Seismic hazard map of Africa

Source: USGS (May, 2010)

23.5.3 Geotechnical Investigation and Testing

The field programme was undertaken during June and July 2012 by Legeni, a Niger-based geotechnical contractor. The aim of the programme was to determine the sub-surface conditions at the proposed TSF, heap leach pad (HLP) and plant sites. This included defining material thicknesses and classifying the soil and rock types. In particular, the thickness of the surface sands and gravels needed to be verified to assess depths and volumes of material that may need to be removed, in addition the thickness and quality of the clay layer.

The field investigation consisted of test pitting and disturbed sampling, as well as borehole drilling and sampling. Samples were also collected from the mud pit spoil piles associated with the exploration drillholes at the Miriam deposit. The number of test pits and boreholes carried out, as well as the sampling requirements for each, is presented in Table 23-52.

Table 23-52: Geotechnical site investigation

Task	No #	Requirements	Comment
Test pitting and sampling (disturbed)	20 TSF test pits 10 HLP test pits	Geotechnical logging/profiling of test pits across proposed TSF and leach pad sites. Profile to extent of excavator reach or refusal. Targeted depth >4 m Testing required on each sample: • Particle size distribution • Specific gravity determination • Atterberg limits testing • Proctor moisture /density relationship • Shear box testing (remoulded samples) • Permeability (remoulded samples)	Disturbed samples of low permeability materials (clay layers) were required. Testing will confirm compaction requirements as well as as-constructed conditions for use in seepage and stability assessments. Sample locations indicated on SRK Drawing SRK180-0100. Remoulded samples will need to be prepared to specified densities and moisture contents based on the results of the Proctor relationship.
Drilling and sampling (undisturbed)	11 TSF boreholes 6 HLP boreholes	Boreholes across proposed TSF and leach pad sites. Testing required on each soil sample: • Particle size distribution • Specific gravity determination • Atterberg limits • Moisture content • Triaxial testing (permeability and shear strength) Testing required on each rock sample: • Specific gravity determination • Moisture content • Unconfined compressive strength (UCS) testing • Laboratory point load testing (PLT)	Undisturbed soil samples of low permeability materials (clay layers) were required. Testing will confirm in-situ foundation conditions for use in seepage and stability assessments. Undisturbed rock samples were also required to assess the foundation rock strength. Sample locations indicated on SRK Drawing SRK180-0100.
Miram deposit sampling (disturbed)	21 samples	Sampling of the spoil piles associated with the drillholes on the Miriam deposit. Testing required on each sample • Particle size distribution • Specific gravity determination • Atterberg limits testing • Proctor moisture/density relationship • Permeability (remoulded samples)	Disturbed samples of low permeability materials (clay layers) were required. Testing will confirm compaction requirements as well as as-constructed conditions for use in seepage and stability assessments. Sample locations will correspond to Miriam deposit drillholes. Remoulded samples will need to be prepared to specified densities and moisture contents based on the results of the Proctor relationship.

Geotechnical investigation results

The samples have been classified in accordance with the Unified Soils Classification System (USCS) (ASTM D2487 - 2006) using the results of the particle size distribution and Atterberg limits testing. The USCS classification of the materials along with a geotechnical interpretation of the laboratory results for each site is provided below. The geotechnical investigation, including borehole locations, is presented in Figure 23-70.

TSF / Plant sites

The upper stratigraphy at the planned TSF area and plant sites consist of a clayey soil, usually a clayey sand (SC) or a clayey silt of low plasticity (CL-ML), ranging from 1 m up to 5 m in depth The clayey soil appears to be thickest in the north east corner of the TSF footprint. Underlaying the clayey soil is either sandstone or siltstone, which is sometimes alternating with low-plasticity silt layers (ML). There is also a downwards dipping trend for the argillite layer running from north west to south east (along geotechnical cross-sections 1-1, 2-2 and 4-4 as shown in Figure 23-70 which may be an indicator of possible seepage path.

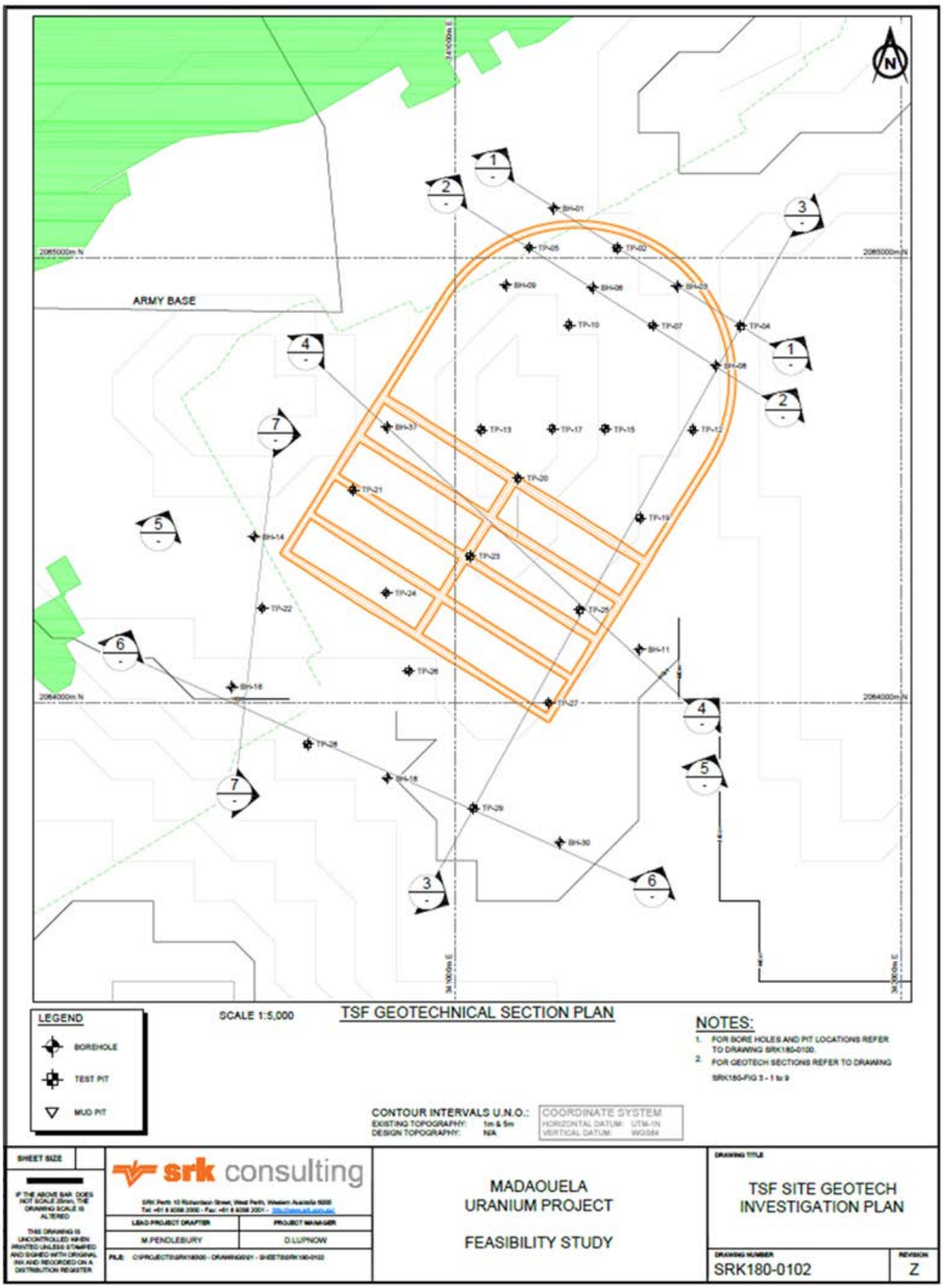


Figure 23-70: TSF Site Geotech Investigation Plan

HLP sites

At the planned HLP sites, the test pits indicate a layer of sand (SM or SC), although sometimes low-plasticity clay (CL), to a depth of 0.5 to 0.6 m. Immediately underlying the soil layer is sandstone. In some areas (TP 32 and TP 39), there is clay up to 1.3 m deep. From the borehole results, clay was found in BH 31 and BH 32 at HLP, extending to 3.6 m in depth. For the remaining boreholes, sandstone was present from the surface and extended to the end of the hole, usually to 11 m.

Miriam deposit

Logging of the mud pit spoil piles showed a concentrated area of clayey material in the south east corner of the Miriam deposit. This soil was logged as a sandy, clayey silt (CL-ML) or a clayey, silty fine sand (SC-SM). The area of clayey material is shown in Figure 23-71. The photos of the mud pit spoil piles were also an indicator to the presence of clay, as the colour changed to a red / brown for clayey material. Outside of the clayey zone, the soils were either a sand or gravelly sand.

The area of this clayey zone is approximately 300,000 m², corresponding to a volume of clayey material available of 300,000 m³, assuming an average clay depth of 1 m. This should be a sufficient volume for liningthe dedicated HLP at a thickness of 300 mm.

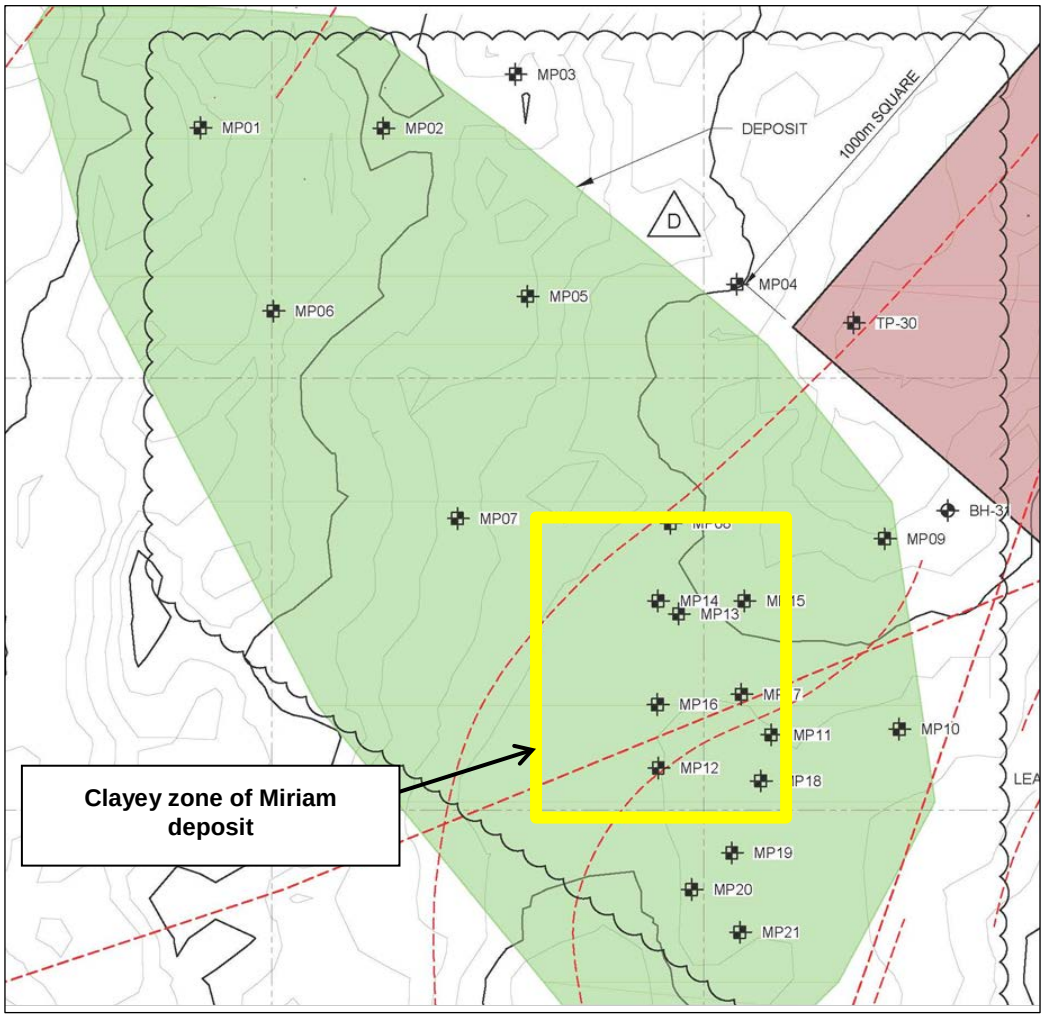


Figure 23-71: Clayey zone of the Miriam deposit

Laboratory testing results

As part of the geotechnical investigation, Legeni has taken a number of disturbed and undisturbed samples from the test pits and boreholes. SRK has reviewed the logs for the samples and layout of where they originated and has selected an initial sample set for laboratory testing. The samples have been chosen to give an overall representation of geotechnical parameters across the site. The results of the laboratory testing are summarised in the following sections.

Test pit samples

The laboratory results confirm the material types that were logged during the field programme, with the soils predominantly being SC or CL. It should be noted that at the HLP sites, the soils are all SC, whilst the soils at the TSF sites are a mixture of SC and CL. The plasticity index of the soils is, on average, lower at the HLP sites than at the TSF area. Particle size distribution is consistent for most samples, with most soils being sand with high to low fines content. The exception to this is samples TP 22, TP 02 (1.8 - 2.0 m), TP 17 and TP28. Sample TP 22 is poorly graded with over 90 % passing the No. 200 sieve, whilst the three (3) other samples are well graded with high gravel content between 30 - 45 %. The soil fines content is also greater at the TSF area (on average 49 % passing the No. 200 sieve) than at the HLP sites (on average 33 % passing the No. 200 sieve). The three (3) sedimentation tests that were carried out confirm that the fines content is of a clayey nature.

Borehole samples

The results confirm what was reported in the field logs, with clayey soils in the upper 1 - 2 m stratigraphy at the TSF area, and BH 31 and BH 32 showing the presence of clay in the boreholes at the HLP sites. The soils were either SC or CL, and all were of low plasticity. BH 16, 18, 31 and 32 all reported very large fines contents (in excess of 90 % passing the No. 200 sieve) and sedimentation tests confirmed the high clay fraction (20% or more). These samples with very high fines content were all classified as CL, whilst the remaining samples were classified as SC due to the relatively lower fines content. Sample BH 08 was an exception though, as fines content was approximately 58% and was classified as a CL.

Rock samples

Two (2) UCS test results were obtained, therefore correlation of the $Is_{(50)}$ value determined from point load testing to an equivalent UCS strength is subject to engineering interpretation. Using the UCS results from BH 18 and BH 35, the corresponding K conversion factors are 22 and 31 respectively. An average K factor of 27 has therefore been estimated. It is also assumed that the point load tests failed through the rock mass and were not affected by any existing defects.

On average, K factors for correlating PLT $Is_{(50)}$ to UCS strength are in the range of 20 to 25, this is in line with the results achieved in the geotechnical work. This ratio can vary from 15 to 50 though, depending on the rock type and rock properties, for example, anisotropic effects (ISRM, 1985). The equivalent UCS values applying a K factor of 27 are shown in Table 23-53. The equivalent UCS strengths range from 10 to 27 MPa, which indicates a weak strength rock (UCS in the range of 5 to 25 MPa). However, it must be noted that should the applied K factor be at the lower end of the scale, the equivalent UCS strength may be much lower.

Table 23-53: Equivalent UCS from point load test

Sample	Depth	Field Description	Location	Average $Is_{(50)}$ (MPa)	UCS (MPa)	K factor	Equivalent UCS (MPa)
BH 01	9.5 - 10.0 m	Siltstone	TSF	0.84	-	27	22.8
BH 09	5.7 - 6.3 m	Sandstone	TSF	1.01	-	27	27.4
BH 16	7.8 - 8.3 m	Sandstone	TSF	0.57	-	27	15.4
BH 18	18.5 - 19.0 m	Sandstone	TSF	0.99	21.5	27	26.7
BH 32	10.2 - 10.7 m	Sandstone	HLP 1	0.71	-	27	19.3
BH 35	2.4 - 2.9 m	Sandstone	HLP 2	0.35	11	27	9.5

Note: K Factor adopted based upon two UCS tests.

Mud pit samples

The laboratory testing results confirmed the presence of clayey material in the south east corner of the Miriam deposit. The soil samples are classified as either CL or SC. Although fines content is relatively high for all the samples (on average 53% is finer than No. 200 sieve), the sedimentation analysis shows a lower clay fraction (average clay fraction is 10%). As the plasticity index ranges from 9 to 11%, and with a relatively low liquid limit, the soil is still classified as CL.

23.5.4 Dry Stack Tailings Disposal

Facility design

SRK has developed a dry stacked TSF design, similar to the disposal methodology used at Somaïr (as shown in Figure 23-72) and Cominak. The tailings stacker will start discharging at the outer wall and gradually advance to the centre of the initial footprint (approximately a circle). The stacker will also rise as it advances to the final height of approximately 34 m above the natural ground level. Waste rock will be placed to facilitate trafficking on the tailings as required.

The main tailings conveyor tips onto the slewing stacker (Figure 23-73) that allows approximately 120 degrees deposition radius. The tailings will flow down from the stacker at the assumed 10% beach slope.

Figure 23-72: Main tailings conveyor inclined on tailings stack (active disposal at left, emergency at right)

Figure 23-73: Front view of main conveyor discharge and slewing stacker (note waste rock for trafficability)

The TSF has been based on the design criteria presented in Table 23-48. The proposed TSF location is shown in Figure 23-74; however, the final site selection is dependent on the location of the plant site and the portal.

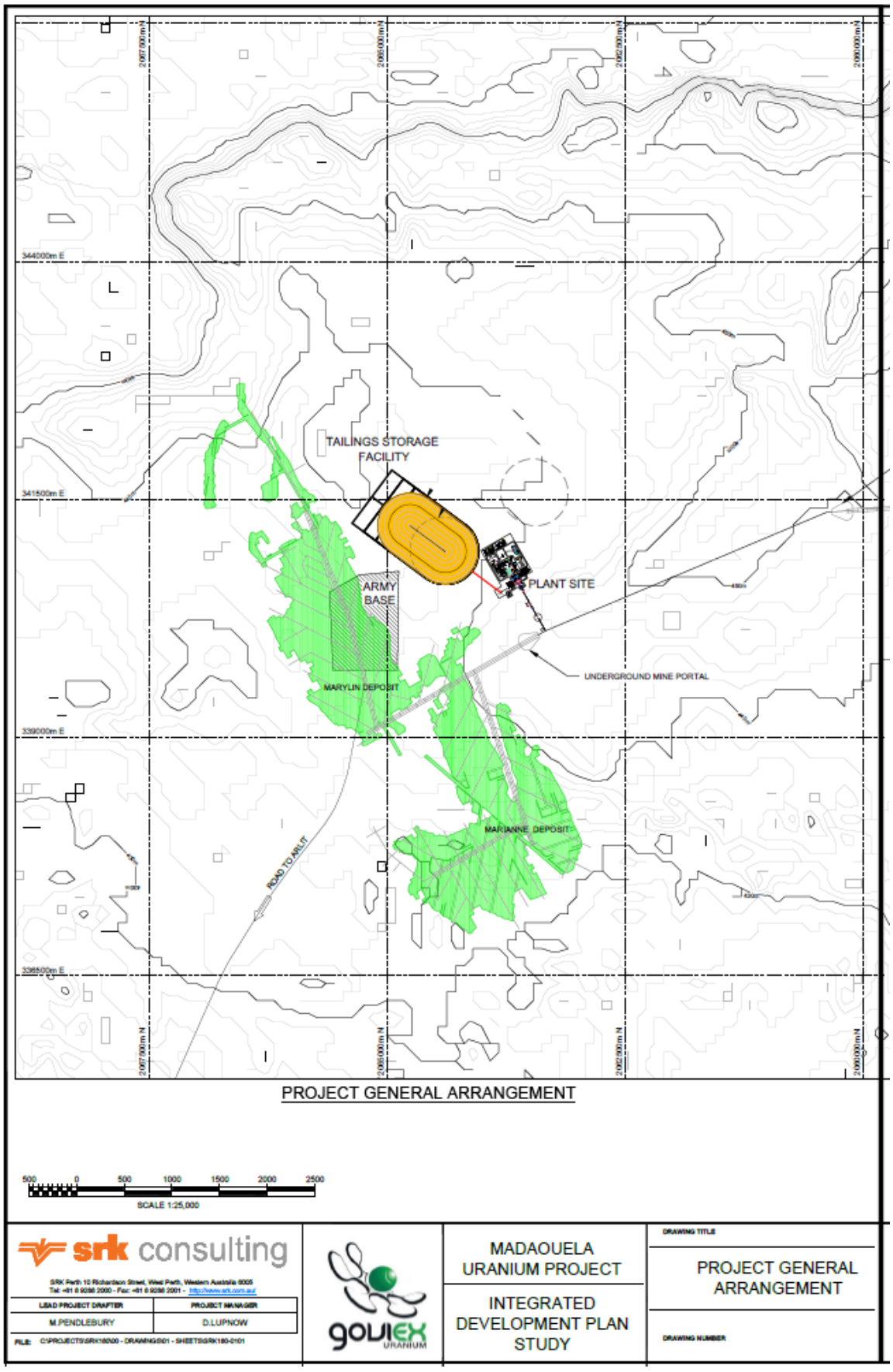


Figure 23-74: Project General Arrangement

The TSF consists of an initial tailings storage area of 450,000 m^2, and four (4) staged evaporation ponds.. The facility has been designed to store a total of 16.5 Mt, or 10.3 Mm^3 of tailings over 21 years, at a production rate of 3,350 tpd. The tailings will be stacked conically at a 10% beach slope angle for Years 1 through 5 to a maximum height of approximately 34 m and again years 7 to 10 and 10 to 15 m. For years 15 to 21, the tailings stack will not be raised any further, i.e. will maintain the maximum 34 m height, but will advance out over the evaporation ponds at a 10% beach slope, as shown in Figure 23-75.

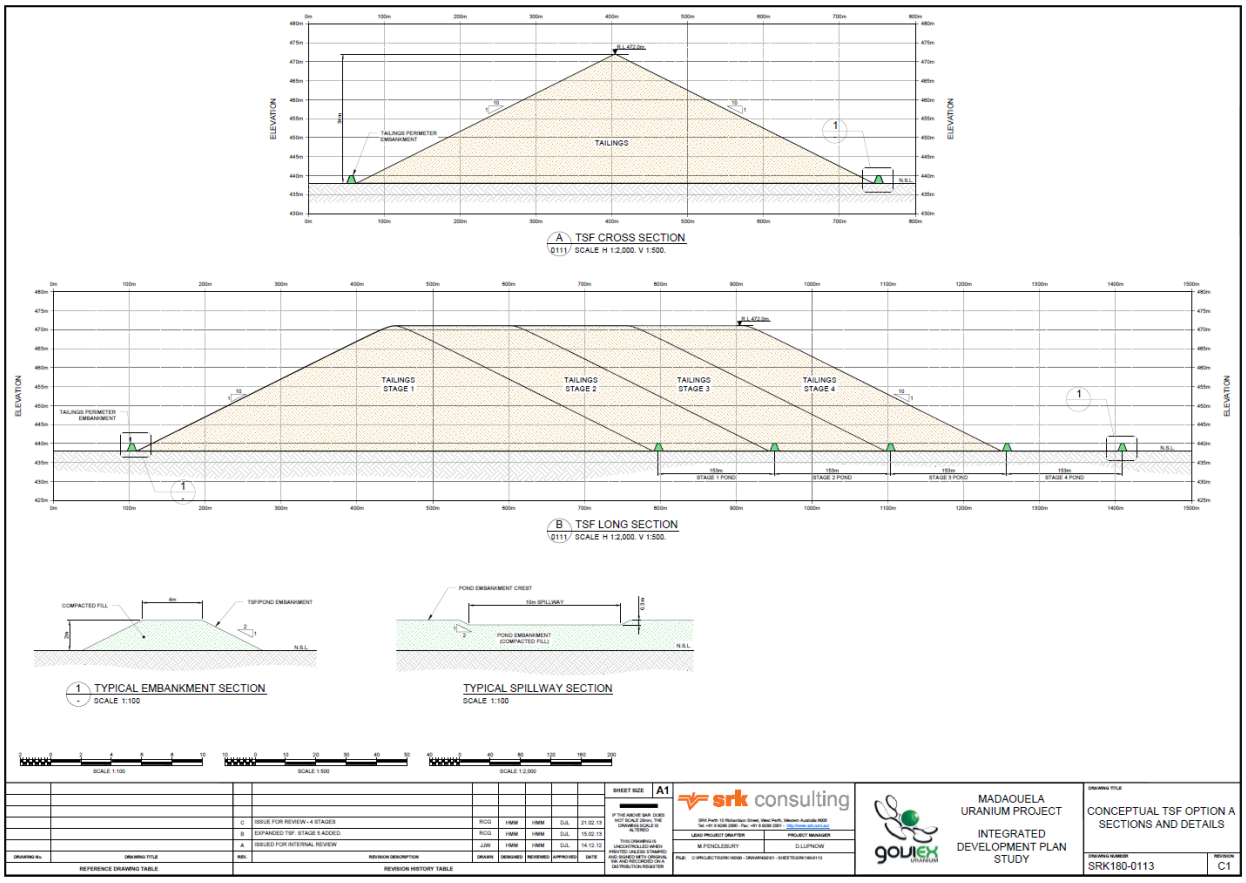


Figure 23-75: Conceptual TSF Section and Details

Foundation preparation

The foundation preparation is minimal as the project site is founded on relatively flat, unvegetated land.

The foundation preparation steps are as follows:

1 Remove loose sand and pebbles from the initial TSF footprint to depth of 50 mm.

2 Remove sufficient clay material from the base of the facility, to an approximate depth of 400 mm. The clay material should be stockpiled for use in the construction of the clay berm.

3 Work the base of facility including; ripping to 300 mm depth, moisture conditioning, and compacting to 95 % modified Proctor.

Containment berm construction

Containment berms will be constructed around the evaporation ponds and the TSF perimeter to facilitate the management of both tailings supernatant water and stormwater runoff. The 2 m-high containment berms will be constructed using waste rock lined with the stockpiled clay borrow material removed from the base of the facility. Details are shown on Figure 23-75.

The initial construction phase includes a containment berm surrounding the initial TSF footprint area and the evaporation ponds. The evaporation ponds have been divided into two (2) compartments to facilitate alternating use of ponds to allow operator flexibility. Each sub-pond has a spillway between the TSF footprint and the pond to allow any stormwater runoff that exceeds the TSF capacity to report to the ponds.

23.5.5 Tailings water balance

A monthly water balance was developed to model the TSF water requirements over the 21-year design life. The water balance utilised the precipitation and evaporation data and the design criteria presented in Section 23.5.1. While it is expected that the moisture content of the tailings may reduce as the tailings consolidates and dries, SRK has conservatively assumed that the final moisture content of the tailings is 20% w/w.

The TSF water balance considers all major inflows and outflows to / from the TSF, including; precipitation, tailings water, interstitial storage and evaporation.

Figure 23-76 show the total inflows and potential outflows to / from the TSF. This indicates that the potential outflows are consistently greater than the inflows, and that the evaporation rate is sufficient to counterbalance the cumulative inflows. While no TSF pond will be present on a regular basis, a limited volume of water may occasionally be available after high rainfall events.

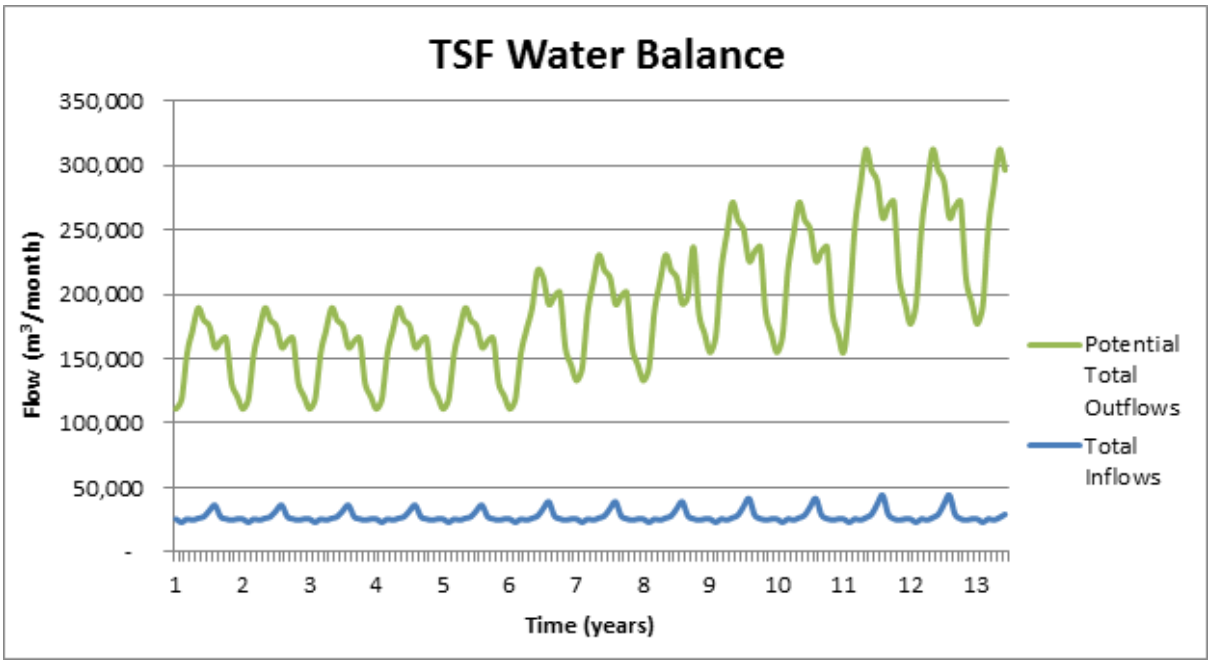


Figure 23-76: TSF water balance

Evaporation pond design

Pond sizing and layout

The evaporation ponds are sized to contain up to 35,560 kg/h (742 m^3/day) plant bleed water and the 24-hour, 1-in-100 year storm event of 61 mm, resulting in a total storage requirement of 54,000 m^3. The ponds are built in four, approximately 153 m x 695 m stages and are contained by 2 m-high bunds, with a minimum freeboard of 0.5 m. Wherever possible water will be reused to minimise water abstraction and water losses.

As with the main impoundment the foundation preparation will be the same as the main TSF.

Liner system

The evaporation ponds will be lined with a 1.5 mm-thick HDPE geomembrane liner.

Pond water balance

A separate water balance has been developed for the evaporation ponds. Both major inflows and outflows have been considered, including; precipitation and plant bleed water. The sole pond outflow is evaporation, which has been reduced by 30% to account for high saline content.

The potential evaporation rate is sufficient to counterbalance the cumulative inflows during the typical year; therefore, the water balance does not consider any spill over from the TSF.

Figure 23-77 shows the evaporation pond inflows, indicating that the majority of water to the evaporation ponds comes from the 35,560 kg/h plant bleed water. Figure 23-78 shows the total pond inflows, potential evaporation and the balance (inflows-outflows). This indicates that some water will be accumulated in the pond on a month-to-month basis, but the evaporation rate is sufficient to counterbalance the cumulative pond inflows during the average month.

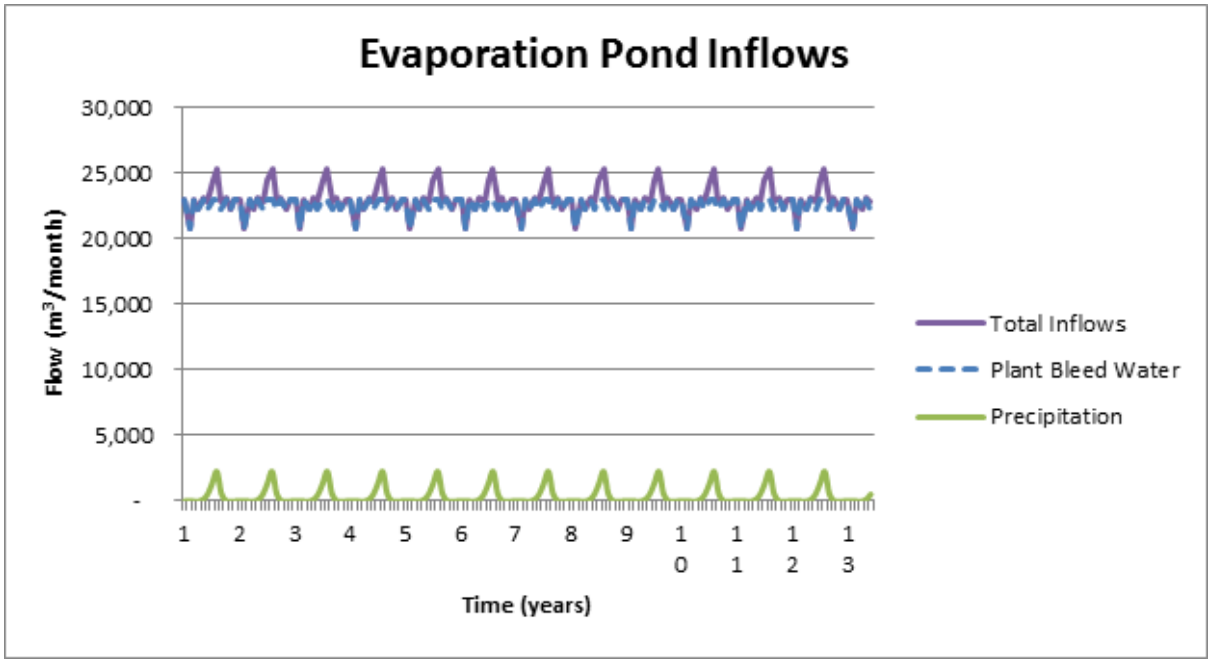


Figure 23-77: Evaporation pond inflows

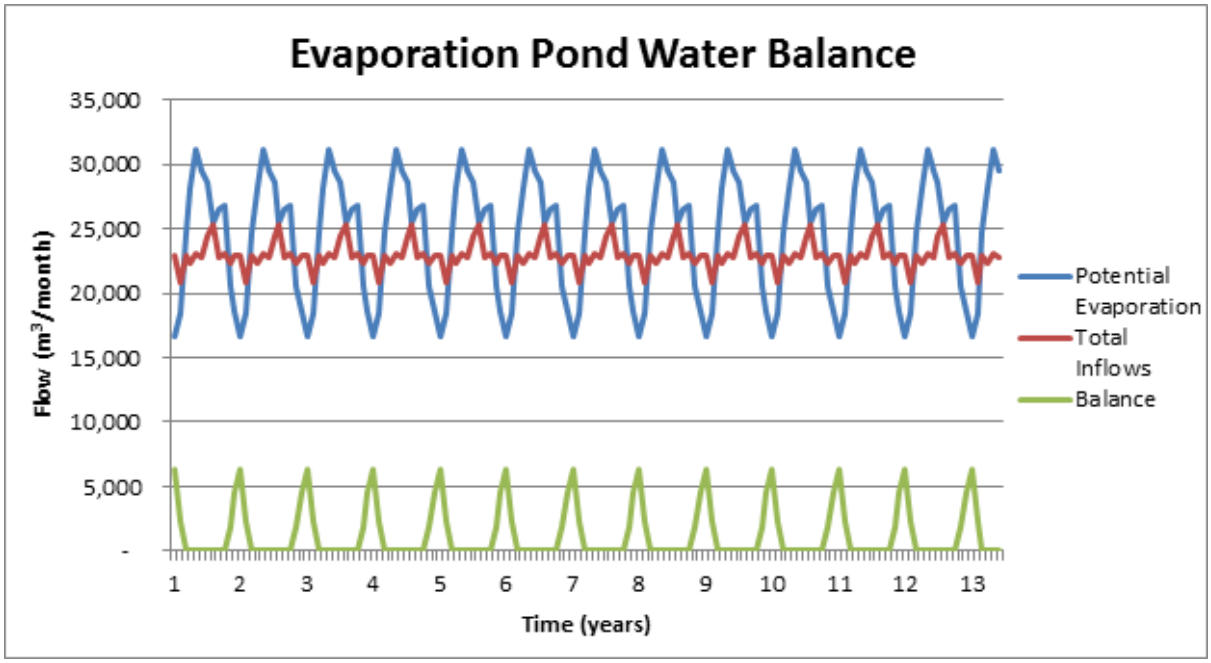


Figure 23-78: Evaporation pond water balance

23.5.6 Closure plan

TSF closure

SRK's conceptual closure plan proposes that the TSF is covered for closure. The cover will consist of two (2) layers – an inner mixed layer and an outer coarse rock layer. The inner mixed layer will consist of clay / low permeability material mixed with coarse sandy gravel. The purpose of this layer is to prevent oxygen ingress, radon emissions, and to act as a general barrier to radiation. This layer will be capped with a coarse rock layer to prevent wind and water erosion of the underlying finer material.

The containment berms will be re-graded onto the tailings such that the closed facility sheds all precipitation away from the facility. In order to reduce dust dispersion continuous reclamation of the tailings will be ongoing through placement of cover by ablation and ROS residue materials.

Evaporation ponds closure

The initial and intermediate evaporation ponds, and their precipitates, will be covered with tailings as the tailings stack advances.

SRK proposes that the final evaporation pond is covered for closure. Once any excess solution has evaporated, the exposed precipitates will be covered with a fill layer, potentially including some of the sorted rejects. The excess liner around the ponds will be folded back on top of the fill layer. This cover will be used to allow access and to form a landform that does not impound water. A closure cover system similar to that used on the tailings will then be implemented.

23.5.7 TSF

Capex

The results of the Capex cost estimate are presented in Table 23-54. The different production rates of ore must be considered when comparing costs. The following assumptions apply to the cost estimate provided below:

- Preliminary and general costs have been assumed as 25% of all capital costs excluding mechanical systems.
- Initial Capex costs include preliminary and general costs, site preparation, and excavation and earthworks for the Stage 1 TSF and evaporation ponds and drainage control for the Stage 1 evaporation ponds.
- Sustaining Capex including evaporation pond site preparation, excavation and earthworks, and liner costs are assigned at specific years for each expansion stage.
- Costs related to supplying and installing a filter plant and conveyor and stacker system are to be determined by Tenova for input into the cost estimate.

The results of the TSF Capex cost estimate are presented in Table 23-54.

Table 23-54: Summary of TSF Capex costs

	Schedule of Quantities for Madaouela - TSF Capex – Bid Cost Summary							
Item	**Description**	**Amount (USD)**						**% of Total**
		Year 0	**Year 5**	**Year 7**	**Year 10**	**Year 15**	**Subtotal**	
1	Preliminary and General	611,114	240,732	240,732	240,732	240,732	1,574,042	20
2	Site Preparation	1,342,631	326,480	326,480	326,480	326,480	2,648,551	35
3	Excavation and Earthworks	677,968	212,591	212,591	212,591	212,591	1,528,332	20
4	Mechanical Systems	Tenova to cost						N/A
5	Liners	423,858	423,858	423,858	423,858	423,858	2,119,290	25
Subtotal		3,055,571	1,203,661	1,203,661	1,203,661	1,203,661	**7,870,215**	100
% of Total		46	18	18	18	18	100	
Tonnes		16,494,000						
Cost per tonne		0.48						

Opex

Based on similar projects, the TSF is expected to require six full time labour employees, a part time maintenance foreman and a supervisor. The results of the Opex cost estimate are presented in Table 23-55. Opex costs include annual labour, electrical and flocculent costs.

Table 23-55: Summary of TSF Opex costs (per annum)

Item	**Description**	**Total (USD)**
1	Tailings Conveying	614,085
2	Labour Cost	420,080
Opex per Annum		**1,034,165**
Total		**21,717,458**
Cost per ton		**1.32**

TSF cost estimate summary

A summary of the TSF cost estimate is shown in Table 23-56. The TSF cost estimates do not include the cost of the filter plant, the conveyor and stacker system, or the processing costs.

Table 23-56: TSF cost estimate summary

Initial Capex	Sustained Capex	Opex	Total	Capex per Tonne	Opex per Tonne	Cost per Tonne (USD)
3,055,571	4,814,644	21,717,458	29,587,673	0.48	1.32	1.80

23.5.8 Conclusions and Recommendations

Risks and opportunities

- Tailings dewatering – SRK has assumed that tailings can be produced at an 80% solids content (w/w) and achieve a slope of 10%. Should a different solids content or slope be realised, the volumetric calculations for the TSF will have to be revised.
- Plant bleed water – SRK has assumed constant plant bleed water of 35,560 kg/h with a specific gravity (SG) of 1.15. Should the plant bleed water vary significantly, the evaporation ponds will need to be re-sized.
- Earthworks quantities – preliminary estimates of earthworks required to achieve a sufficient pad grade have been calculated based on SRTM topographic survey. The quantities could vary if the on-site topography is less/more suitable to the required pad grade.
- Haulage distances – haulage distances have been assumed for the quantity estimates. These are particularly significant for the on-off pad spent ore storage costs. Should these distances vary from those currently estimated, these costs will vary proportionally.
- Pad construction – pad fill is assumed to be undertaken by the mining fleet and rates have been applied accordingly. Should pad construction be required prior to mobilisation of the mining fleet, and a civil contractor is required to undertake the works, the expected rates could be significantly higher.

Conclusions

SRK has developed preliminary designs for a dry stack TSF provided high-level cost estimates. Based on the design criteria, the designs presented are technically feasible and will provide tailored solutions to the project requirements. Cost estimates have been provided and are intended to be used as an input to the Madaouela Project cost model.

24 INTERPRETATION AND CONCLUSIONS (ITEM 25)

SRK's interpretations of the geology, mineral resources, and scoping level study of conceptual mining and processing options for the Madaouela Uranium Project are as follows:

SRK has completed technical studies of pre-feasibility confidence for the project. The Project development plan envisions an average 2.69 Mlb per year U_3O_8 yellowcake production rate over a twenty one year mine life, with a 93.7 % ultimate recovery of uranium. In addition it assumes recovery of on average approximately 592 t of MoO_2 per annum at a sales price of USD11.00/lb, which revenue is used to offset processing costs. The base case project economics for this project at a long-term uranium price of USD 70 /lb U_3O_8 are positive, and indicate an after-tax NPV of USD 339 M (at 8 % discount rate) with an IRR of 23.5 % and a total life of mine ("LoM") net free cash of USD 1,126 M. Initial capital costs are estimated at USD 359 M, LoM capital costs at USD 676 M, and cash operating costs of USD 31.49 /lb U_3O_8 including royalties and by-product credits. It is noted that Mo has not been included in the Mineral Resource model and hence is not considered to be at the same level of confidence as the U grades.

The sensitivity of project value (NPV) to uranium price is +USD 99 M for a +10 % change and –USD 109 M for a -10 % change on the base price of USD 70 /lb U_3O_8. Sensitivity of project value (NPV) to operating costs and capital cost are broadly similar at around ±USD 41 M for a ± 10 % change. Sensitivity to every ±10% change in molybdenum grade is ±USD 8 M.

SRK recommends that GoviEx continue to move the Madaouela Uranium Project forward with continued exploration drilling in the vicinity of Marianne northwest, MSNE, Mariyvone and northwest Miriam. During further work it is essential to undertake additional technical studies to assist in the evaluation of preferred mining and processing options that would provide inputs to a feasibility study of the merits for developing a uranium mine.

GoviEx geologists are confident that the exploration potential on all tenements has the possibility to significantly increase the current resources of the Madaouela Uranium Project. GoviEx exploration drilling has identified additional targets including La Banane that is included in the resource statement and potential new targets at Marianne northwest, MSNE, Maryvone and northwest Miriam as well as additional targets not as yet drilled in Mad I, Agal, Era and Mad IV.

SRK concurs that the entire property position warrants the added exploration effort and does indeed have the potential to significantly increase the current project resources and thereby improve the potential economic viability of the Madaouela Uranium Project; however, SRK cautions that an exploration potential cannot be relied upon until further drilling and sampling is done to properly assess that potential.

The Madaouela Uranium project is sufficiently attractive from a technical and economic perspective that it justifies pursuit by GoviEx toward further resource definition and the steps to project development.

25 RECOMMENDATIONS (ITEM 26)

SRK recommends that GoviEx continue the process of moving the Madaouela Uranium Project forward, into feasibility study with additional technical studies.

Studies still required include;

- Optimization of mining equipment and schedules
- Detailed process design
- Pilot plant design, ablation and operation including ROS and solvent extraction options

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28 GLOSSARY

28.1 Mineral Resources and Reserves

28.1.1 Mineral Resources

Unless otherwise noted, the mineral resources and mineral reserves have been classified according to the "CIM Standards on Mineral Resources and Reserves: Definitions and Guidelines" (December 11, 2005). Accordingly, the Resources have been classified as Measured, Indicated or Inferred, the Reserves have been classified as Proven, and Probable based on the Measured and Indicated Resources as defined below.

A Mineral Resource is a concentration or occurrence of natural, solid, inorganic or fossilized organic material in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge.

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 geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes.

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 a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes that are spaced closely enough for geological and grade continuity to be reasonably assumed.

A 'Measured Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes that are spaced closely enough to confirm both geological and grade continuity.

28.1.2 Mineral Reserves

A Mineral Reserve is the economically mineable part of a Measured or Indicated Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, and economic and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified. A Mineral Reserve includes diluting materials and allowances for losses that may occur when the material is mined.

A 'Probable Mineral Reserve' is the economically mineable part of an Indicated, and in some circumstances a Measured Mineral Resource demonstrated by at least a Preliminary

Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified.

A 'Proven Mineral Reserve' is the economically mineable part of a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction is justified.

28.2 Glossary

Table 28-1: Glossary

Term	Definition
Assay:	The chemical analysis of mineral samples to determine the metal content.
Capital Expenditure:	All other expenditures not classified as operating costs.
Composite:	Combining more than one sample result to give an average result over a larger distance.
Concentrate:	A metal-rich product resulting from a mineral enrichment process such as gravity concentration or flotation, in which most of the desired mineral has been separated from the waste material in the ore.
Crushing:	Initial process of reducing ore particle size to render it more amenable for further processing.
Cut-off Grade ("CoG"):	The grade of mineralized rock, which determines as to whether or not it is economic to recover its metal content by further concentration.
Dilution:	Waste, which is unavoidably mined with ore.
Dip:	Angle of inclination of a geological feature/rock from the horizontal.
Fault:	The surface of a fracture along which movement has occurred.
Footwall:	The underlying side of an orebody or stope.
Gangue:	Non-valuable components of the ore.
Grade ("G"):	The measure of concentration of uranium within mineralized rock.
Hangingwall:	The overlying side of an orebody or slope.
Haulage:	A horizontal underground excavation which is used to transport mined ore.
Hydrocyclone:	A process whereby material is graded according to size by exploiting centrifugal forces of particulate materials.
Igneous:	Primary crystalline rock formed by the solidification of magma.
ICP-MS	Inductively coupled plasma – mass spectrometer; standard analytical technique
Kriging:	An interpolation method of assigning values from samples to blocks that minimizes the estimation error.
Level:	Horizontal tunnel the primary purpose is the transportation of personnel and materials.
Lithological:	Geological description pertaining to different rock types.
LoM Plans:	Life-of-Mine plans.
LRP:	Long Range Plan.
Material Properties:	Mine properties.
Milling:	A general term used to describe the process in which the ore is crushed and ground and subjected to physical or chemical treatment to extract the valuable metals to a concentrate or finished product.
Mineral/Mining Lease:	A lease area for which mineral rights are held.
Mining Assets:	The Material Properties and Significant Exploration Properties.
Ongoing Capital:	Capital estimates of a routine nature, which is necessary for sustaining operations.
Ore Reserve:	See Mineral Reserve.

Term	Definition
Pillar:	Rock left behind to help support the excavations in an underground mine.
RoM:	Run-of-Mine.
Sedimentary:	Pertaining to rocks formed by the accumulation of sediments, formed by the erosion of other rocks.
Shaft:	An opening cut downwards from the surface for transporting personnel, equipment, supplies, ore and waste.
Sill:	A thin, tabular, horizontal to sub-horizontal body of igneous rock formed by the injection of magma into planar zones of weakness.
Smelting:	A high temperature pyrometallurgical operation conducted in a furnace, in which the valuable metal is collected to a molten matte or doré phase and separated from the gangue components that accumulate in a less dense molten slag phase.
Stope:	Underground void created by mining.
Stratigraphy:	The study of stratified rocks in terms of time and space.
Strike:	Direction of line formed by the intersection of strata surfaces with the horizontal plane, always perpendicular to the dip direction.
Sulfide:	A 549ulphur bearing mineral.
Tailings:	Finely ground waste rock from which valuable minerals or metals have been extracted.
Thickening:	The process of concentrating solid particles in suspension.
Total Expenditure:	All expenditures including those of an operating and capital nature.
Uniform Conditioning ("UC")	A method for estimating the recoverable resource (tonnes and grade) which can be extracted as a series of small selective mining scale blocks ("SMU's") above a defined cut off grade from a larger ordinary kriged block
Uranium units	1.0 per mil = 1000 ppm = 0.10 % eU. And 0.1000 % eU = 0.1179 % eU_3O_8
Variogram:	A statistical representation of the characteristics (usually grade).

28.3 Abbreviations

The metric system has been used throughout this report unless otherwise stated. All currency is in U.S. dollars. Market prices are reported in USD per troy oz of gold and silver. Tonnes are metric of 1,000 kg, or 2,204.6 lbs. The following abbreviations may be used in this report.

Table 28-2: Abbreviations

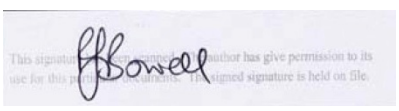
Abbreviation	Unit or Term
%	percent
A	ampere
AA	atomic absorption
A/m^2	amperes per square meter
ANFO	ammonium nitrate fuel oil
Ag	silver
Au	gold
AuEq	gold equivalent grade
°C	degrees Centigrade
CCD	counter-current decantation
cfm	cubic feet per minute
CIL	carbon-in-leach
CoG	Cut-off-Grade
cm	centimeter
cm^2	square centimeter

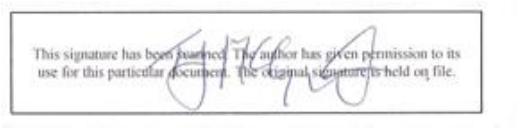
Abbreviation	Unit or Term
cm^3	cubic centimeter
cfm	cubic feet per minute
ConfC	confidence code
Crec	core recovery
CSS	closed-side setting
CPS CTW	counts per second calculated true width
°	degree (degrees)
dia.	Diameter
€	Eruo
eU	Equivalent uranium assay value; determined radiometrically
eU_3O_8	Equivalent U_3O_8; determined radiometrically
EIS	Environmental Impact Statement
EMP	Environmental Management Plan
FA	fire assay
ft	foot (feet)
ft^2	square foot (feet)
ft^3	cubic foot (feet)
G&A	General and Administrative project costs
g	gram
gal	gallon
g-mol	gram-mole
gpm	gallons per minute
gpt	grams per tonne
ha	hectares
HDPE	Height Density Polyethylene
hp	horsepower
HTW	horizontal true width
ICP	induced couple plasma
ID^2	inverse-distance squared
ID^3	inverse-distance cubed
IFC	International Finance Corporation
ILS	Intermediate Leach Solution
IX	
kA	kiloamperes
kg	kilograms
kg/m^3	Kilograms per cubic metre
kg/t eU	Kilograms per tonne of equivalent uranium metal
km	kilometer
km^2	square kilometer
koz	thousand troy ounce
kt	thousand tonnes
ktpa	Kilotonnes per annum
ktpd	thousand tonnes per day
ktpy	thousand tonnes per year
kV	kilovolt

Abbreviation	Unit or Term
kW	kilowatt
kWh	kilowatt-hour
kWh/t	kilowatt-hour per metric tonne
L	liter
Lps	liters per second
lb	pound
LHD	Long-Haul Dump truck
LLDDP	Linear Low Density Polyethylene Plastic
LOI	Loss On Ignition
LoM	Life-of-Mine
m	meter
m^2	square meter
m^3	cubic meter
M lcm	Million loose cubic metres
m/month	Metres per month
masl	meters above sea level
MARN	Ministry of the Environment and Natural Resources
MDA	Mine Development Associates
mg/L	milligrams/liter
Mlb	million pounds
mm	millimeter
mm^2	square millimeter
mm^3	cubic millimeter
MME	Mine & Mill Engineering
Moz	million troy ounces
Mt	million tonnes
MTW	measured true width
MW	million watts
m_{vert}/m_{hor}	Vertical metres per horizontal metre
m.y.	million years
NGO	non-governmental organization
NI 43-101	Canadian National Instrument 43-101
OSC	Ontario Securities Commission
oz	troy ounce
%	percent
PLC	Programmable Logic Controller
PLS	Pregnant Leach Solution
PMF	probable maximum flood
ppb	parts per billion
ppm	parts per million
QA/QC	Quality Assurance/Quality Control
RC	rotary circulation drilling
RoM	Run-of-Mine
RQD	Rock Quality Description
SEC	U.S. Securities & Exchange Commission

Abbreviation	Unit or Term
s	second
SG	specific gravity
SPT	standard penetration testing
st	short ton (2,000 pounds)
SX	
t	tonne (metric ton) (2,204.6 pounds)
t eU	Tonnes of equivalent uranium metal
t/doh	Tonnes per direct operating hour
tph	tonnes per hour
tpd	tonnes per day
tpy	tonnes per year
TSF	tailings storage facility
TSP	total suspended particulates
t_{waste}:t_{RoM}	Tonnes of waste per tonne of run-of-mine
µ	micron or microns
U	uranium
U_3O_8	Uranium expressed as an oxide; common units by which uranium is sold
USD/kg	US dollars per kilogram
USD/kg U	US dollars per kilogram of equivalent uranium
USD/lb U_3O_8	US dollars per pound of U_3O_8
USD/t	US dollars per tonne
USD/t_{metal}	US dollars per tonne of uranium metal
USD/t_{RoM}	US dollars per tonne of run-of-mine
USDk	Thousand US dollars
USDm	Million US dollars
eU_3O_8	Equivalent Uranium as determined by gamma log derivations
V	vanadium
V_2O_5	Vanadium expressed as an oxide; common units by which vanadium is sold
VFD	variable frequency drive
W	watt
XRD	x-ray diffraction
yr	year

For and on behalf of SRK Consulting (UK) Limited





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