NI 43-101 Preliminary Economic Assessment for the Saza-Makongolosi Gold Project, Tanzania

Prepared by

srk consulting

Authors:

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Date of Report: 12 September 2012

Effective Date: 12 September 2012

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Helio Resource Corporation

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Qualified Persons (QP)	**Qualified Persons (QP)**

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Certificate of Qualified Person

a)	I Duncan Pratt am a Senior Consultant (Mining) with SRK Consulting (Australasia) Pty Ltd with a business address at Level 8, 365 Queen Street, Melbourne, Victoria, 3000 Australia.
b)	This certificate applies to the technical report entitled, "Preliminary Economic Assessment for the Saza-Makongolosi Gold Project, Tanzania*"*, dated 12 September 2012 (the "Technical Report").
c)	I am a graduate of New South Wales University, Australia. In 2002, I obtained an Honours degree in Mining Engineering. I have practiced my profession continuously since 2003. I am a member in good standing of the Australian Institute of Mining and Metallurgy and am a Chartered Professional (Mining). I have worked as a Mining Engineer for a total of 10 years since my graduation. My relevant experience is over 10 years' of experience in both consultancy and production roles, including 6 years as open pit mining engineer 4 years as a consultant. I have worked on a variety of different geological environments and commodities, ranging from precious metals, uranium through to iron ore, and have worked on mine feasibility, mine planning and expansion studies, as well as audits. I am a 'Qualified Person' for purposes of National Instrument 43-101 (the "Instrument").
d)	I have personally inspected the the Saza-Makongolosi Gold Project site on 12 – 15 May 2012.
e)	I am responsible for Chapters 14 - 26 of the report, including the Reserve section of the Technical Report, including the Competent Person's Statement accompanying the Estimate.
f)	I am independent of Helio Resource Corporation as defined by Section 1.4 of the Instrument.
g)	I have no prior involvement with the Property that is the subject of the Technical Report.
h)	I have read the Instrument, and the Technical Report has been prepared in compliance with the Instrument. As of the date of this certificate, to the best of my knowledge, information and belief, this Independent Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading;
i)	I consent to the use of this report and our name for public filing any Provincial regulatory authority. Dated this 12th day of September 2012.
	"Original document dated, signed and sealed by Duncan Pratt, BEng *Hons (Mining)*  Senior Consultant (Mining) SRK Consulting (Australasia) Pty Ltd

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Consent of Author

To: British Columbia Securities Commission
Alberta Securities Commission
Ontario Securities Commission
TSX Venture Exchange
Helio Resource Corporation

I, Duncan Pratt, of SRK Consulting (Australasia) Pty Ltd, am author of the technical report entitled, "Preliminary Economic Assessment for the Saza-Makongolosi Gold Project, Tanzania ", dated 12 September 2012 and prepared for Helio Resource Corporation (the "Technical Report").

I do hereby consent to the public filing of the Technical Report and to the use of extracts from and summaries of the Technical Report in the news release of Helio Resource Corporation dated 13 September 2011 (the "News Release").

I confirm that I have read the News Release and that the disclosure in the News Release fairly and accurately represents the information in the Technical Report.

Dated effective 12 September 2012.

Duncan Pratt, *BEng Hons (Mining)*
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Certificate of Qualified Person

a)	I Robin Simpson am a Principal Consultant (Resource Geology) with SRK Consulting (Australasia) Pty Ltd with a business address at 10 Richardson Street, West Perth, WA 6005, Australia.
b)	This certificate applies to the technical report entitled, “Preliminary Economic Assessment for the Saza-Makongolosi Gold Project, Tanzania*”,* dated 12 September 2012 (the “Technical Report”). I am the author of the technical report titled “Mineral Resource Estimate Update
c)	I am a graduate of the University of Canterbury (BSc (Hons) in geology, 1995) and the University of Leeds (MSc in Geostatistics, 2004). I am a Member of the Australian Institute of Geoscientists (member number 3156). I have practised my profession continuously since graduation. My experience relevant to the subject matter of the technical report includes employment as a geologist, from 1996 until 2003, on several Australian gold mining operations and gold exploration projects. Since commencing work with SRK in 2005, I have also carried out consulting work for various gold projects located in Australia, Indonesia, China, Brazil, Zimbabwe and Ghana. By means of my education, membership of a professional association, and relevant experience, I fulfil the requirements to be “Qualified Person” for the purposes of National Instrument 43-101.
d)	I have carried out one personal inspection of the Saza-Makongolosi Gold Project, from 3 to 5 September, 2011.
e)	I am responsible for all items of the technical report.
f)	I am independent of the issuer as described in Section 1.5 of National Instrument 43-101.
g)	Before being engaged as an independent consultant by Helio Resource Corporation, I had no prior involvement with the Saza-Makongolosi Gold Project.
h)	I have read the Instrument, and the Technical Report has been prepared in compliance with the Instrument. As of the date of this certificate, to the best of my knowledge, information and belief, this Independent Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading;
i)	I consent to the use of this report and our name for public filing any Provincial regulatory authority. Dated this 12th day of September 2012.
	“Original document dated, signed and sealed by ***Robin Simpson****,* BSc (Hons), MSc, MAIG Principal Consultant (Resource Geology) SRK Consulting (Australasia) Pty Ltd

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Consent of Author

To: British Columbia Securities Commission
Alberta Securities Commission
Ontario Securities Commission
TSX Venture Exchange
Helio Resource Corporation

I, Robin Simpson, Bsc (Hons), MSc, MAIG, of SRK Consulting (Australasia) Pty Ltd, am a contributing author of the technical report entitled, "Preliminary Economic Assessment for the Saza-Makongolosi Gold Project, Tanzania ", dated 12 September 2012 and prepared for Helio Resource Corporation (the "Technical Report").

I do hereby consent to the public filing of the Technical Report and to the use of extracts from and summaries of the Technical Report in the news release of Helio Resource Corporation dated 13 September 2011 (the "News Release").

I confirm that I have read the News Release and that the disclosure in the News Release fairly and accurately represents the information in the Technical Report.

Dated effective 12 September 2012.

RGSimpson

Robin Simpson, *BSc Hons, MSc, MAIG*
SRK Consulting (Australasia) Pty Ltd

Important Notice

This report was prepared as a National Instrument 43-101 Technical Report for Helio Resource Corporation ("Helio") by SRK Consulting (Australasia) Pty Ltd ("SRK"). The quality of information, conclusions and estimates contained herein is consistent with the level of effort involved in SRK's services, based on: i) information available at the time of preparation, ii) data supplied by outside sources and iii) the assumptions, conditions and qualifications set forth in this report. This report is intended for use by Helio subject to the terms and conditions of its contract with SRK and relevant securities legislation. The contract permits Helio to file this report as a Technical Report with Canadian securities regulatory authorities pursuant to National Instrument 43-101, Standards of Disclosure for Mineral Projects. Except for the purposes legislated under provincial securities law, any other uses of this report by any third party is at that party's sole risk. The responsibility for this disclosure remains with Helio. The user of this document should ensure that this is the most recent Technical Report for the property as it is not valid if a new Technical Report has been issued.

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Executive Summary

ITEM 1 of Form 43-101F1

Introduction

SRK Consulting (Australia) (Pty) Ltd (“SRK”) was appointed by Helio Resource Corp (“Helio”) of Vancouver, Canada, to compile a Technical Report (“TR”) to support the results of a Resource Report disclosed publically by Helio on 14 February 2012 regarding the Saza Makongolosi Gold Project (“SMP”), located in the Mbeya Region of Tanzania.

Helio through its wholly owned subsidiary BAFEX Tanzania LTD (“BTL”) either holds or is in the process of formalising the acquisition of the five Project Licenses (“PL’s”) that make up the project area.

This TR has been prepared according to the requirements of the *National Instrument 43-101 Standards of Disclosure for Mineral Projects* (“NI43-101”) of the Canadian Securities Administrators and Form 43-101F1 *Technical Report* (the “Form”).

The achievability of the production schedule on which the Preliminary Economic Assessment (“PEA”) is based is neither warranted nor guaranteed by SRK. The production schedule is necessarily based on economic assumptions, many of which are beyond the control of Helio or SRK.

The PEA and production schedule evaluate Measured and Indicated Mineral Resources only. Inferred Mineral Resources have not been included. SRK notes this assessment is preliminary in nature and there is no certainty that the preliminary assessment will be realised. It should be noted that Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

The basis for the PEA and any qualifications / assumptions made by the Qualified Person (“QP”) includes:

- The financial analysis has been completed on a pre-tax basis;
- The process facility and other associated infrastructure is assumed to be constructed during year 0. Prestrip mining activities also take place during year 0;
- First ore is mined in Year 1;
- The process plant is commissioned during year 1, first ore in the plant occurs in Year 1;
- Sustaining capital is estimated at 5% of the initial capital requirements, applied annually;
- Capital cost contingencies are 15% of the total capital requirements; and
- Operating cost contingencies are 5% of the total mining, processing, tailings and environmental and social operating costs.

Note: SRK recommends caution with this level of contingency, however notes that this figure is broadly in line with other industry studies prepared at this stage of a project’s development.

Property Description and Location

The SMP is located in part of the Lupa Goldfield, which lies along the eastern edge of the Western Rift Valley close to Lake Rukwa. Mbeya, the capital of the Mbeya Region, is approximately 100 km southeast by road from the SMP.

The SMP covers approximately 238 km^2. Within this area, Helio has identified over 30 exploration targets. This Technical Report primarily concerns the targets where SRK considers there is sufficient information available to estimate Mineral Resources. BTL has named these prospects Porcupine, Kenge, Mbenge, Konokono and Tumbili.

Accessibility, Climate, Local Resources, Infrastructure and Physiography

In general the project area is flat, but a series of hills, the Ilunga range, occurs within the project area. The elevation ranges from around 900 m to 1,729 m.

A 33 kVA power line runs along the road from Mbeya to Chunya, then through the SMP to Mkwajuni, the nearby town where the Helio office is located. Mains electricity is available in Mkwajuni. All the major cell phone networks have pylons in Mkwajuni and Makongolosi, with an additional Vodacom tower situated on the peak of the western end of the Ilunga hills (outside of the SMP).

Exploration services and equipment are accessible through the road and rail networks of Tanzania.

The local workforce consists primarily of subsistence farmers and occasional independent artisanal miners. Tanzania has a rapidly expanding mining industry and a reasonably qualified workforce could be developed from other areas of the country.

No mining infrastructure is located on the SMP.

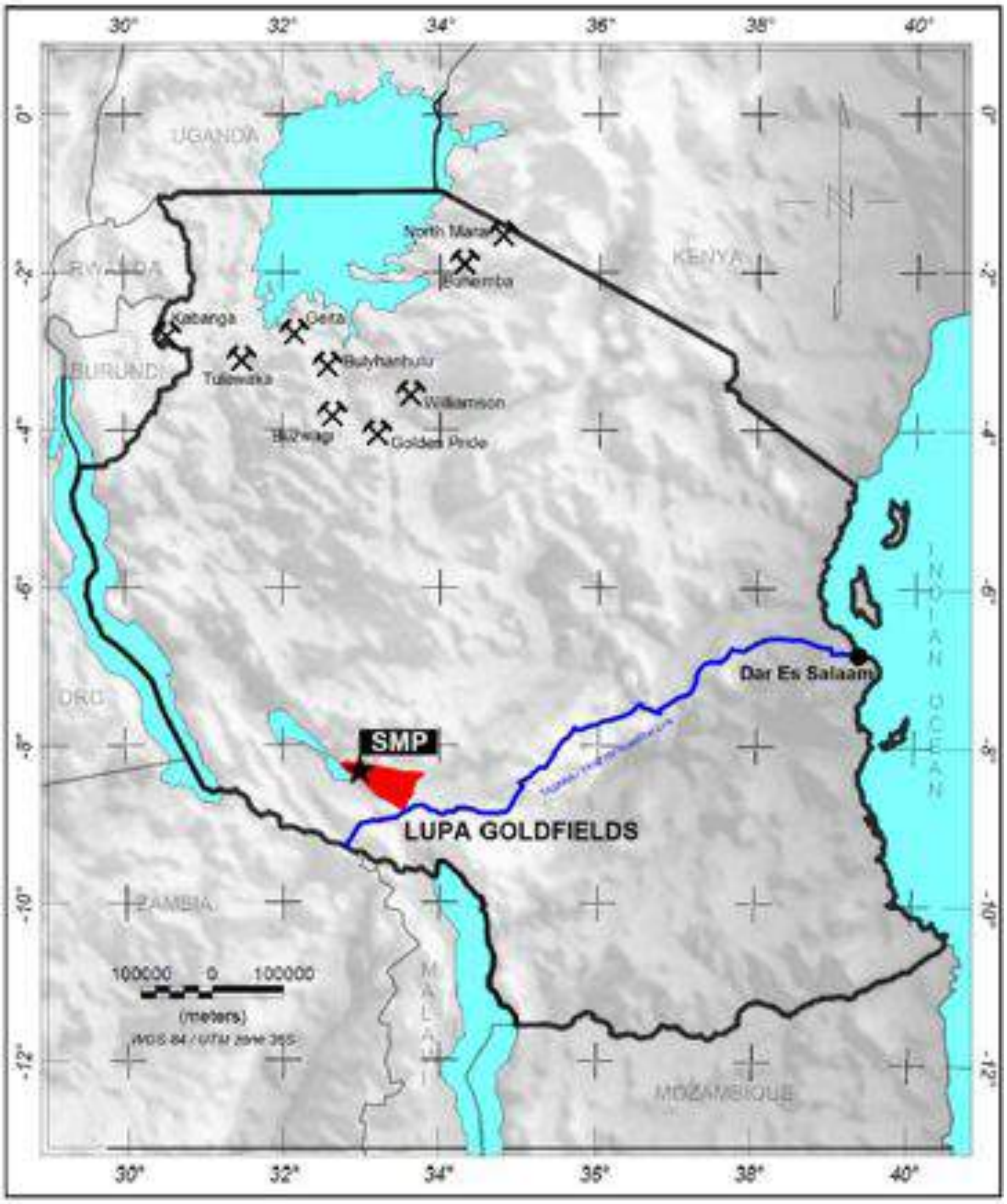

Figure ES-1: Location Map (Tanzania)

History

Gold was discovered at the Lupa Goldfields in the early 1900s. The New Saza Mine came to be the largest mine on the goldfields and drew material from several workings that are now within the area of the SMP. Reported production from 1939 until the end of mining in 1956 was 270,770 oz of fine gold and 242,942 oz of fine silver from 1.1 million tonnes of ore (Harris 1962). Since the colonial era ended and Tanzania gained independence in 1961, three companies are known to have carried out exploration activities on the SMP area: Technoexport (1970 to 1974), Princess Resources (1995 to 1999) and Anglogold (1997 to 1999).

Helio has access to some of the information collected from the historic exploration programmes. This data is used to assist exploration targeting, but are not considered to be of sufficient quality to be used for Mineral Resource estimation.

Geological Setting and Mineralisation

The Lupa Goldfield is situated in the southwestern part of the Tanzanian Craton, within the Lower Proterozoic mobile belt of the 1.8 Ga Ubendian System. Lithologies comprise granitic, intermediate and mafic intrusive rocks together with ferruginous quartzites.

Several prominent structural trends are observed in the Lupa Goldfield. The dominant Saza Shear Zone trends ENE. A strong WNW to NW trend is seen in outcrop and satellite imagery together with the ENE trending structures. The regional foliations are associated with major dextral shear zones.

Two distinct granites dominate the igneous suite that underlies the SMP area. The Ilunga Granite is observed extensively in the northern half of the SMP. The Saza Granite occurs in the southern portion of the prospect area.

Gold mineralisation occurs in both the Ilunga and Saza Granites. The mineralisation can be described broadly as shear zone-hosted orogenic or intrusion-related gold systems. Mineralisation is dominantly associated with "Saza-parallel" (070°) and "Kenge-parallel" (120°) shear zones.

Mineralisation at the SMP is relatively simple, comprising of pyrite (generally less than 1% by volume) with minor chalcopyrite and molybdenite plus occasional scheelite and galena. Gold occurs as free gold and occasionally as tellurides. Mineralisation is associated with quartz veining, silicification, sericitisation, haematisation (demagnetisation) and occasionally chloritisation.

Exploration

Helio began exploration activities on the SMP in 2006 and has conducted exploration programmes on the SMP every year since then. The work done by Helio includes:

- Regional and detailed soil geochemistry;
- Induced Polarisation ("IP") and magnetic geophysical surveys;
- Airborne magnetic and radiometric geophysical surveys;
- Diamond and Reverse Circulation ("RC") drilling;
- Mapping;
- Metallurgical testing; and
- Structural studies of the controls on mineralisation.

The total drilling done by Helio on the SMP since 2006 comprises 365 diamond holes for 64,646 m and 516 RC holes for 47,036 m.

There are 58,598 primary assays in Helio's database. The SMP samples were assayed by African Assay Laboratories, which is accredited to ISO/IEC 17025 standard and is a member of the SGS Group and located in Mwanza, northern Tanzania. The assay method was 50 g fire assay.

In the qualified person's opinion, the sampling collection, preparation, security and analytical procedures used by Helio meet generally accepted industry best practices. These procedures are therefore consistent with generating data of a quality suitable for Mineral Resource estimation.

Data Verifications

Robin Simpson, the qualified person for the Mineral Resource estimate included in this Technical Report, visited the SMP and then African Assay Laboratories facilities in Mwanza, from 3 to 5 September 2011. During the site visit Mr Simpson inspected drillcore, drill sites and exposures of the mineralised targets. Mr Simpson was given access to Helio's filing system and verified information in the drillhole database against primary sources such as logging sheets and assay certificates.

Subsequent to the site visit, SRK carried out thorough validation checks on the drillhole database, including statistical analysis and 3D visualisation of the drillholes. SRK also reviewed the results from analytical quality assurance and quality control (QA/QC) samples submitted by Helio with their primary samples. Helio's QA/QC program includes Certified Reference Materials, blanks and duplicates.

After carrying out the verification measures described above, the qualified person is confident that the database is suitable to be used for Mineral Resource estimation.

Mineral Resource Estimates

The Mineral Resources presented in Table ES-1 and Table ES-2 was first released on 10 February 2012. The Mineral Resources have been reported according to the guidelines of the CIM Standing Committee and are therefore considered by SRK to be compliant with NI43-101.

Table ES-1: 10 February 2012 Mineral Resource Estimate for Porcupine

Cut-off grade (g/t)	Resource Category	Mean Au grade (g/t)[1]	Tonnage (Mt)[2]	Metal Au (koz)[3]
0.3	Measured	1.35	12.3	530
0.3	Indicated	1.16	3.1	120
0.3	Measured + Indicated	1.31	15.4	650
0.3	Inferred	0.85	3.6	100
0.5	Measured	1.35	12.3	530
0.5	Indicated	1.16	3.1	120
0.5	Measured + Indicated	1.31	15.4	650
0.5	Inferred	0.89	3.3	90
0.7	Measured	1.41	11.3	510
0.7	Indicated	1.19	2.9	110
0.7	Measured + Indicated	1.37	14.3	630
0.7	Inferred	1.01	2.3	70
0.9	Measured	1.61	8.6	440
0.9	Indicated	1.32	2.2	90
0.9	Measured + Indicated	1.55	10.8	530
0.9	Inferred	1.15	1.4	50

1: Rounded to two decimal places

2: Rounded to nearest 0.1 Mt

3: Rounded to nearest 10 koz

Table ES-2 summarises the 10 February 2012 Mineral Resource Estimate for Kenge and Mbenge combined at four different cut-off grades.

Table ES-2: 10 February 2012 Mineral Resource Estimate for Kenge and Mbenge

Cut-off grade (g/t)	Class	Mean Au grade (g/t)[1]	Tonnage (Mt)[2]	Metal Au (koz)[3]
0.3	Measured	1.51	2.6	120
0.3	Indicated	1.22	6.3	250
0.3	Measured + Indicated	1.30	8.9	370
0.3	Inferred	1.07	3.2	110
0.5	Measured	1.51	2.6	120
0.5	Indicated	1.25	6.1	250
0.5	Measured + Indicated	1.33	8.7	370
0.5	Inferred	1.28	2.5	100
0.7	Measured	1.55	2.4	120
0.7	Indicated	1.36	5.2	230
0.7	Measured + Indicated	1.42	7.7	350
0.7	Inferred	1.45	2.0	90
0.9	Measured	1.59	2.3	120
0.9	Indicated	1.49	4.2	200
0.9	Measured + Indicated	1.53	6.6	320
0.9	Inferred	1.55	1.7	90

1: Rounded to two decimal places

2: Rounded to nearest 0.1 Mt

3: Rounded to nearest 10 koz

The deposits Konokono and Tumbili contained only Inferred material and as such, no economic assessment has been performed on these deposits.

Mineral Reserve Estimates

No Mineral Reserves have been quoted as part of this Preliminary Economic Assessment.

Mining Methods

For the PEA, it is assumed that a conventional open pit operation including drill and blast, followed by truck and shovel activities will be employed.

An earthmoving contractor is planned to be employed and it is expected a standard excavator and haul truck mining fleet will be utilised along with supporting auxiliary equipment (motor grader, water truck etc.). For scheduling purposes and expected fleet numbers, a 190 t excavator and 100 t off-highway diesel haul trucks have been used.

Drilling and blasting is planned to be performed on 10 m benches for Porcupine and Mbenge and 12.5 m benches for Kenge. These dimensions match the block size in the geological block model. Due to the expected selective mining that will be required for ore mining, load and haul will be performed on full bench heights for waste movement and half bench height for ore material.

The Porcupine pit is scheduled to be developed first with the process facility to be constructed adjacent to this pit. This will minimise the haulage requirements during the early years of the project. As Kenge and Mbenge are brought into production, a small fleet of road trucks will utilised to haul ore from the respective pit rim stockpiles to the processing facility adjacent to the Porcupine pit. The payload for the road trucks will nominally be 20 t.

The operation is scheduled to run for 9 years which includes an initial year of prestrip. During this initial year, it is expected all required infrastructure will be constructed onsite. The operation will process 1.6 million tonnes per annum (Mtpa) with a maximum total material movement of 14.5 Mtpa (ore and waste tonnes). Throughout the project approximately 94 people will be employed as part of the mining operation.

The project is to use proven technology, with no requirement for untried or untested technology.

Recovery Methods

SGS Lakefield Research Limited ("SGS") in Ontario, Canada conducted a program of preliminary metallurgical testwork to determine the processing characteristics of the Porcupine and Kenge mineralised material. Results from the Kenge study were published in August 2008 and followed by results from Porcupine in August 2009. The tests included head grade analysis, mineralogical evaluation, comminution testwork, gravity separation, flotation, cyanidation (of whole ore, gravity tailing and flotation concentrate) and preliminary environmental testing.

Both testwork programmes indicated amenability to conventional gravity and cyanidation gold recovery techniques. A value of 94.5% gold recovery has been used for this study. For reporting purposes, this figure has been rounded to 95%.

Project Infrastructure

Key infrastructure is planned to be constructed nearby to the Porcupine deposit, including the process facility, the tailings storage facility and the mobile equipment workshop and stores. This site layout is to facilitate a short haul during the initial years of the project.

A 33 kV power line runs through the property and it is expected that power will be drawn from the national grid. Water management infrastructure (dams, bore holes, pumps) has been allocated in the initial capital requirements.

Market Studies and Contracts

No market studies have been completed as part of this Preliminary Economic Assessment.

Environmental Studies, Permitting and Social Impacts

The company has developed a good relationship with local council and government agencies.

BTL conducted baseline community studies for a Community Support Action Plan (CSAP) on the villages of Saza, Mkwajuni and Patamela in January 2012.

This report has identified the key work that needs to be completed as part of the Environmental Impact Statement (EIS), to be completed in the future.

Capital and Operating Costs

The estimated capital requirements are USD143.4M. A 15% contingency amount has also been included. This figure is based on the company constructing a process facility, site infrastructure (including a tailings storage facility) and the mobilisation of a contractor mining fleet to site. Note:

SRK recommends caution with this level of contingency, however notes that this figure is broadly in line with other industry studies of this development.

Table ES-3 summarises the initial capital cost estimates used for the PEA.

Table ES-3: Summary of Capital Cost Estimates

Item	Project Capital (USDM)
Processing Facility	57.3
Tailings Disposal Facility	9.4
Laboratory	0.5
Mobile Equipment Workshop and Stores	3.5
Power Supply	3.5
Roads and Access	1.0
General Buildings	2.2
Fuel Storage and Distribution	0.5
Communications	0.5
Dewatering Infrastructure	0.5
General Site facilities upgrade	3.0
Contractor Mobilisation	5.0
Contractor Demobilisation	5.0
Sustaining Capital	32.8
Subtotal	**124.7**
Contingency (15%)	18.7
Total	**143.4**

The average unit mining cost over the life of the project is USD2.29/t (ore and waste). This value includes localised labour rates, equipment maintenance and a fuel price of $0.75/litre.

The total cash cost per ounce for the project is USD807/oz. on a pre-tax basis.

Table ES-4 summarises the operating cost estimates over the life of the project.

Table ES-4: Summary of Operating Cost Estimates

Description	Total (USDM)	Unit Cost (USD / t $_{processed}$)
Mining	161.6	14.44
Processing	115.3	10.30
Tailings Disposal	3.9	0.35
Admin (includes environmental)	56.0	5.00
Total Cash Operating Cost (before Contingency)	336.8	30.09
Contingency (@ 5%)	16.8	1.50
Royalty	34.9	3.12
Total Cash Operating Cost	**388.5**	**34.71**

Note: SRK recommends caution with this level of contingency, however notes that this figure is broadly in line with other industry studies of this development.

Economic Analysis

The PEA and production schedule evaluate Measured and Indicated Mineral Resources only. Inferred Mineral Resources have not been included. SRK notes this assessment is preliminary in nature and there is no certainty that the preliminary assessment will be realised. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

SRK notes that the technical work was focussed on recovering Measured and Indicated ounces.

Using a gold price of $1,450 / oz and a discount rate of 8%, the project reports the following:

- NPV of USD85.6M; and
- IRR of 24%.

Table ES-5 details the preliminary economic analysis performed.

Table ES-5: Economic Analysis

	Units	Total	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
Gold	USD / oz		1,450	1,450	1,450	1,450	1,450	1,450	1,450	1,450	1,450
Production											
Mining											
Ore Tonnes	t	11,193,026	120,980	863,298	1,519,154	1,465,715	1,807,184	1,741,963	1,626,295	1,325,148	723,289
Waste Tonnes	t	73,420,723	8,551,520	9,174,202	6,715,846	13,134,285	10,967,816	9,208,037	5,693,705	6,274,706	3,700,606
Stockpile Overhaul	t	559,720	0	0	80,846	103,432	0	0	0	274,852	100,590
Mining Ore Grade	g/t		1.06	1.15	1.66	1.50	1.30	1.37	1.51	1.41	1.32
Processing											
Processing Capacity	t	12,000,000	0	800,000	1,600,000	1,600,000	1,600,000	1,600,000	1,600,000	1,600,000	1,600,000
Processed Tonnes	t	11,193,026	0	800,000	1,600,000	1,569,147	1,600,000	1,600,000	1,600,000	1,600,000	823,879
Processed Ore Grade	g/t		0.00	1.15	1.63	1.47	1.30	1.37	1.51	1.40	1.32
Gold metal produced	oz	509,688	0	29,556	83,897	74,353	66,849	70,443	77,765	71,908	34,916
Gold recovery to Dore	%	94.5%	0	94.5%	94.5%	94.5%	94.5%	94.5%	94.5%	94.5%	94.5%
Gold in Dore	oz	481,655	0	27,930	79,282	70,264	63,173	66,569	73,488	67,954	32,995
Subtotal Paid Metal Value	USD M	698	0.0	40.5	115.0	101.9	91.6	96.5	106.6	98.5	47.8

Capital Costs											
Processing Plant	USD M	57.29	57.3	0	0	0	0	0	0	0	0
Site Infrastructure	USD M	24.63	24.6	0	0	0	0	0	0	0	0
Contractor Mobilisation	USD M	5	5.0	0	0	0	0	0	0	0	0
Contractor Demobilisation	USD M	5	0.0	0	0	0	0	0	0	0	5
Sustaining Capital	USD M	32.8	0.0	4.1	4.1	4.1	4.1	4.1	4.1	4.1	4.1
Contingency	USD M	18.7	13.0	0.6	0.6	0.6	0.6	0.6	0.6	0.6	1.4
Subtotal Capital Costs	USD M	143.4	100.0	4.7	4.7	4.7	4.7	4.7	4.7	4.7	10.5

Operating Costs											
Mining Cost	USD M	161.6	15.9	18.1	16.4	23.2	22.2	19.9	15.7	17.0	13.1
Processing Cost	USD M	115.3	0	8.2	16.6	16.2	16.4	16.5	16.5	16.5	8.5
Tailings Disposal	USD M	3.9	0	0.3	0.6	0.5	0.6	0.6	0.6	0.6	0.3
Administration	USD M	56.0	0	4.0	8.0	7.8	8.0	8.0	8.0	8.0	4.1
Subtotal pre-contingency	USD M	336.8	15.9	30.5	41.5	47.8	47.2	45.0	40.8	42.0	26.0
Contingency	USD M	16.8	0.8	1.5	2.1	2.4	2.4	2.2	2.0	2.1	1.3
Subtotal Operating Costs	USD M	354	17	32	44	50	50	47	43	44	27
Subtotal Operating Costs	($/t)		1.93	3.19	5.29	3.44	3.88	4.31	5.86	5.80	6.18
Subtotal Operating Costs	($/t ore)		138.29	37.13	28.69	34.24	27.44	27.10	26.36	33.28	37.78

Operating Profit (before Depreciation)	USD M	344.8	-16.7	8.4	71.4	51.7	42.0	49.3	63.7	54.4	20.5

Royalty	0	5%	5%	5%	5%	5%	5%	5%	5%	5%	5%
Royalty Paid	USD M	34.9	0.0	2.0	5.7	5.1	4.6	4.8	5.3	4.9	2.4

Pretax Cashflow	USD M	166.5	-116.7	1.7	60.9	41.9	32.7	39.8	53.7	44.8	7.7
Discount Rate	8%										

Cumulative Cashflow	USD M	0	-116.7	-115.0	-54.1	-12.2	20.6	60.3	114.0	158.8	166.5

Net Present Value	85.6	USD M
Internal Rate of Return	24%	

Adjacent Properties

Shanta Gold Ltd is in the process of constructing the New Luika Gold Mine in license properties adjacent to the SMP. Shanta have announced that the commissioning of the New Luika Gold Mine will be by the end of the second quarter of 2012 (Shanta news release March 29, 2012).

The first gold has been poured at this site 31 August, 2012.

Conclusions

Geology

The total Mineral Resource estimate for all SMP targets, at a 0.5 g/t cut-off and with an effective date of 10 February 2012, is:

- Combined Measured and Indicated Mineral Resources of 24.1 Mt @ 1.32 g/t for 1,020,000 ounces Au;
- Inferred Mineral Resources of 7.3 Mt @ 1.05 g/t for 250,000 ounces Au;
- This SRK estimate supersedes the previous Mineral Resource estimate, prepared by Golder Associates (UK) Ltd, with an effective data 30 November 2010. The totals for the previous estimate, also at a 0.5 g/t cut-off, were:
  - Combined Measured and Indicated Mineral Resources of 10.8 Mt @ 1.43 g/t for 500,000 ounces Au; and
  - Inferred Mineral Resources of 7.1 Mt @ 1.19 g/t for 270,000 ounces Au.

The main reason for the increase in Measured and Indicated tonnes and metal since the previous Mineral Resource estimate is the additional drilling Helio has done in 2011, particularly at the Porcupine target. A secondary reason for the increases is that SRK's interpretation of the mineralised zone at Kenge was a larger volume than the previous interpretation.

Considerable exploration potential for defining additional SMP Mineral Resource remains, both at the targets where Mineral Resources have already been estimated and at the 25 or so targets where as yet there is insufficient information for estimating Mineral Resources.

The Porcupine Main zone is open at depth. Closer-spaced drilling of some of the secondary mineralised structures around the Main zone (in particular the Quill zone) may add to the Mineral Resources defined for Porcupine.

The Kenge and Mbenge domains are also open at depth.

The Konokono target is currently covered by lines of RC drilling spaced 300 m apart. Around one of these lines several diamond holes have been drilled, which have made it possible to define an Inferred Mineral Resource. Tumbili is similar to Konokono: covered by lines of widely-spaced RC drilling (200 m) apart and around one of these lines four diamond holes have been drilled. An Inferred Mineral Resource was also defined for Tumbili. Further infill drilling at Konokono and Tumbili may lead to additional Mineral Resources being defined.

The Gap target, about 2 km northeast of Porcupine, has about 800 m of strike length that is covered by lines of RC and diamond drilling spaced 100 m apart. Further drilling at Gap may make it possible to estimate an initial Mineral Resource for this target.

Only Porcupine, Kenge and Mbenge have been included in the mine plan for this PEA.

Project

Exploration, block model interpretation and study work completed to date, including optimisation, mine design and mine scheduling indicate that the Project is potentially economically viable.

A year has been allocated for process facility and other associated infrastructure development, including prestrip. Nine (9) years of ore production at approximately 1.6 Mt per annum is planned to follow, based on the current Resource Statement. The Project is anticipated to recover approximately 480 k oz. gold over the life of the Project.

A preliminary high level cashflow financial model has been developed. Using a gold price of $1,450 / oz and a discount rate of 8%, the project reports the following:

- NPV of USD85.6M; and
- IRR of 24%.

Engineering and estimating has been undertaken to define the project scope and develop cost estimates, sufficient to support a Preliminary Economic Model (+/- 50%).

Recommendations

Based on the technical work completed to date, SRK recommends the following:

- Development of a metallurgical flowsheet to increase the understanding level of the processing requirements, including costs and recoveries;
- Development of a tailings disposal system, including costs and infrastructure requirements;
- Condemnation drilling of process facility and key infrastructure sites;
- Inclusion of a geotechnical drilling program to understand the geotechnical constraints for each of the deposits;
- Consider a geotechnical investigation for key infrastructure, including process facility, mobile equipment workshop and tailings storage facility;
- Development of an environmental management program, including baseline studies to understand the impact mining operations will have on the local environment. This will also assist in early identification of any environmental restrictions;
- Engage Tanzanian (or nearby) earthmoving contractors to better understand the local cost of mining and productivities;
- Update the optimisation process and mine design process based on the revised values from the further investigation (processing, mining costs etc.);
- Continued engagement with local landholders and resident of the local towns and villages;
- Continued engagement with government authorities relating to securing connection to the high voltage national power grid; and
- Assessment of ground and surface water volumes and qualities on site in conjunction with process and mine requirements, to understand the water balance of the site.

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2 Introduction and Terms of Reference

The Saza Makongolosi Project (SMP) is a gold exploration project, located approximately 90 km from the town of Mbeya in South West Tanzania. Helio Resource ("Helio") through its wholly owned subsidiary BAFEX Tanzania Ltd ("BTL") either holds or is in the process of formalising the acquisition of the five Project Licenses ("PL's") that make up the project area.

In March 2012, Helio commissioned SRK Consulting Australia Pty Ltd ("SRK") to visit the property and prepare a Preliminary Economic Assessment (PEA) document for the SMP, compliant with Canadian Guidelines National Instrument 43-101. The work was to be based on a Resource Statement prepared by SRK in 2012. The services were rendered between March 2012 to September 2012, leading to the preparation of the PEA reported herein that was disclosed publically by Helio in a news release on 7 September 2012.

This Technical Report documents a Mineral Resource statement for the SMP prepared by SRK. It was prepared following the guidelines of the Canadian Securities Administrators' National Instrument 43-101 and Form 43-101F1. The Mineral Resource statement reported herein was prepared in conformity with generally accepted CIM "Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines."

This Technical Report summarises the technical information available on the SMP and demonstrates that the SMP clearly qualifies as an "Advanced Exploration Property" as defined by the Toronto Stock Exchange.

2.1 Scope of Work

The scope of work, as defined in a letter of engagement executed on 2 April 2012 between Helio and SRK includes the preparation of a Preliminary Economic Assessment ("PEA") in compliance with National Instrument 43-101 and Form 43-101F1 guidelines. The PEA is to be based on a Resource Statement prepared by SRK in 2012.

This work involved the assessment of the following aspects of this project (in addition to the work already completed as part of the Mineral Resource Statement):

- Optimisation of block model;
- Pit and waste dump design;
- Mine schedule;
- Site infrastructure;
- Environmental;
- Operating and capital estimates; and
- Recommendations for additional work.

2.2 Work Programme

The Mineral Resource statement reported herein is a collaborative effort between Helio and SRK personnel. The exploration database was compiled and maintained by Helio and was audited by SRK. The geological model and outlines for the gold mineralisation were constructed by SRK. In the opinion of SRK, the geological model is a reasonable representation of the distribution of the targeted mineralisation at the current level of sampling. The geostatistical analysis, variography and grade models were completed by SRK during December 2011, January 2012 and February 2012. The Mineral Resource statement reported herein was disclosed publicly in a news release dated 14 February 2012.

The Mineral Resource statement reported herein was prepared in conformity with generally accepted CIM "Exploration Best Practices" and "Estimation of Mineral Resource and Mineral Reserves Best Practices" guidelines. This Technical Report was prepared following the guidelines of the Canadian Securities Administrators National Instrument 43-101 and Form 43-101F1.

The Technical Report was compiled in Melbourne from March 2012 to September 2012.

2.3 Basis of Technical Report

This report is based on information collected by SRK during a site visit performed between 12 May 2012 and 18 May 2012 and on additional information provided by Helio throughout the course of SRK's investigations. Other information was obtained from the public domain. SRK has no reason to doubt the reliability of the information provided by Helio. This Technical Report is based on the following sources of information:

- Discussions with Helio personnel;
- Inspection of the Saza Makongolosi Project area, including outcrop and drill core;
- Review of exploration data collected by Helio; and
- Additional information from public domain sources.

2.4 Qualifications of SRK and SRK Team

The SRK Group comprises over 1,400 professionals, offering expertise in a wide range of resource engineering disciplines. The SRK Group's independence is ensured by the fact that it holds no equity in any project and that its ownership rests solely with its staff. This fact permits SRK to provide its clients with conflict-free and objective recommendations on crucial judgment issues. SRK has a demonstrated track record in undertaking independent assessments of Mineral Resources and Mineral Reserves, project evaluations and audits, Technical Reports and independent feasibility evaluations to bankable standards on behalf of exploration and mining companies and financial institutions worldwide. The SRK Group has also worked with a large number of major international mining companies and their projects, providing mining industry consultancy service inputs.

The Resource Evaluation work and the compilation of this Technical Report was completed by Robin Simpson, MAIG (membership number 3156). By virtue of his education, membership of a recognised professional association and relevant work experience, Robin Simpson is an independent Qualified Person as this term is defined by National Instrument 43-101.

The PEA work of this Technical Report was compiled by Duncan Pratt, CP (Mining) MAusIMM (membership number 303501). By virtue of his education, membership of a recognised professional association and relevant work experience, Duncan Pratt is an independent Qualified Person as this term is defined by National Instrument 43-101.

Mike Warren, BSc (Mining Eng), MBA, FAusIMM, FAICD, Corporate Consultant with SRK, reviewed drafts of this Technical Report and provided Peer Review prior to their delivery to Helio as per SRK internal quality management procedures. Mike Warren did not visit the project.

2.5 Site Visit

In accordance with National Instrument 43-101 guidelines, Duncan Pratt visited the SMP on 12-18 May 2012 accompanied by Mike Ashley of Helio.

In accordance with National Instrument 43-101 guidelines, Robin Simpson visited the SMP on 3-5 September 2011 accompanied by Mike Ashley of Helio.

The purpose of the site visit by Duncan Pratt was to review the current site infrastructure, topography, likely locations of key developments on site (pit, process facility and waste dump).

Duncan's site visit was also aimed at understanding the logistical requirements to access site and the level of infrastructure in surrounding towns.

The purpose of the site visit by Robin Simpson was to review the digitalisation of the exploration database and validation procedures, review exploration procedures, define geological modelling procedures, examine drill core, interview project personnel and to collect all relevant information for the preparation of a revised Mineral Resource model and the compilation of a Technical Report.

Robin's site visit also aimed at investigating the geological and structural controls on the distribution of the gold mineralisation in order to aid the construction of 3D gold mineralisation domains.

SRK was given full access to relevant data and conducted interviews of Helio personnel to obtain information on the past exploration work, to understand procedures used to collect, record, store and analyse historical and current exploration data.

2.6 Declaration

SRK's opinion contained herein and effective **12 September, 2012**, is based on information collected by SRK throughout the course of SRK's investigations, which in turn reflect various technical and economic conditions at the time of writing. Given the nature of the mining business, these conditions can change significantly over relatively short periods of time. Consequently, actual results may be significantly more or less favourable.

This report may include technical information that requires subsequent calculations to derive sub-totals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, SRK does not consider them to be material.

SRK is not an insider, associate or an affiliate of Helio and neither SRK nor any affiliate has acted as advisor to Helio, its subsidiaries or its affiliates in connection with this project. The results of the technical review by SRK are not dependent on any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings between SRK and Helio.

3 Reliance on Other Experts

SRK has not performed an independent verification of land title and tenure. SRK did not verify the legality of any underlying agreement(s) that may exist concerning the permits or other agreement(s) between third parties, but have relied on the information provided by Helio, outlined in Section 4 of this report.

Helio has informed SRK that there are no known litigations potentially affecting the SMP and furthermore that there are no known environmental, socio-political, marketing or taxation issues that may materially affect the project.

SRK has relied on the metallurgical testwork reports supplied by SGS Ltd (Appendices D, E and F) and is satisfied that Section 12 of this report is an accurate representation of the information contained in those reports.

4 Property Description and Location

4.1 Location and Area of Property

The United Republic of Tanzania is located in East Africa, approximately between 1˚ to 12˚ south of the Equator and 4˚ and 10˚ east of the Prime Meridian. The SMP gold project (centred at approximately 8° 20’ S, 33° 05’ E) covers an area of approximately 238 km^2 (23,800 Ha) and is located in the Mbeya Region of Tanzania, 100 km by road north west of the regional capital, Mbeya, which itself lies some 760 km by road south-west of Dar es Salaam, the main port in Tanzania (Figure 4-1).

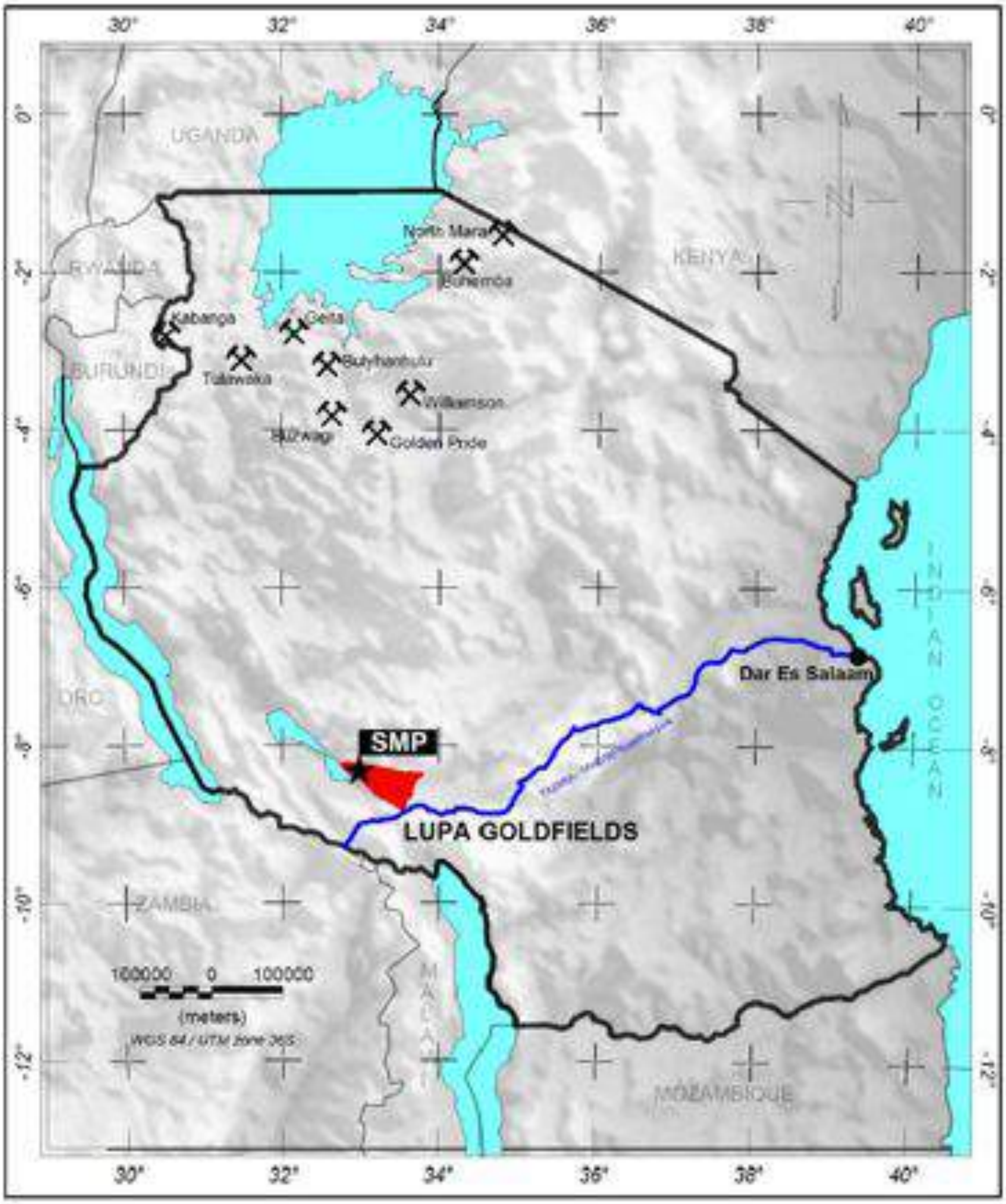

Figure 4-1: Location of the SMP within Tanzania

4.2 Mineral Tenure

The SMP area was initially defined by the license boundaries of 5 PLs which were acquired in transactions with Thorntree Minerals Limited (TTML) and Dhahabu Resources and Mining Company Limited (DREMCO). Helio is able to earn 100% interest in each PL subject to a 2% royalty, which can be reduced to 1% by a payment of USD1,000,000 by Helio to the JV partner. Figure 4-2 shows the configuration of the PLs which currently make up the SMP, Table 4-1 gives the PL numbers, ownership and current status. Note that the DREMCO licenses have been approved for transfer to Helio by the directors of DREMCO; the application for this transfer has been lodged with the Tanzanian Ministry of Energy and Minerals.

Table 4-1: Status of Prospecting Licenses

Informal Name	Owner	PL Number	Expiry Date	Application	Status
Gap	DREMCO	2963/2004	December 2013	n/a	Transfer of ownership to BTL underway
Kwaheri	DREMCO	2964/2004	December 2013	n/a	Transfer of ownership to BTL underway
Ilunga	DREMCO	2965/2004	December 2013	n/a	Transfer of ownership to BTL underway
Saza	BTL	2580/2004	June 2013	n/a	current
Gap North	BTL	7097/2011	November 2020	n/a	Current
Saza East	BTL	7143/2011	August 2020	n/a	Current
Makongolosi North	BTL	5990/2009	June 2016	n/a	Current
Saza South	BTL	4963/2008	March 2015	n/a	Current
Saza West	TTML	5326/2008	October 2015	n/a	Current
Mkwajuni North	TTML	7710/2012	February 2021	n/a	Current
Maleza	BTL	Not issued	-	HQ-P24588	Application with Ministry
Kwaheri East	BTL	Not issued	-	HQ-P21156	Application with Ministry

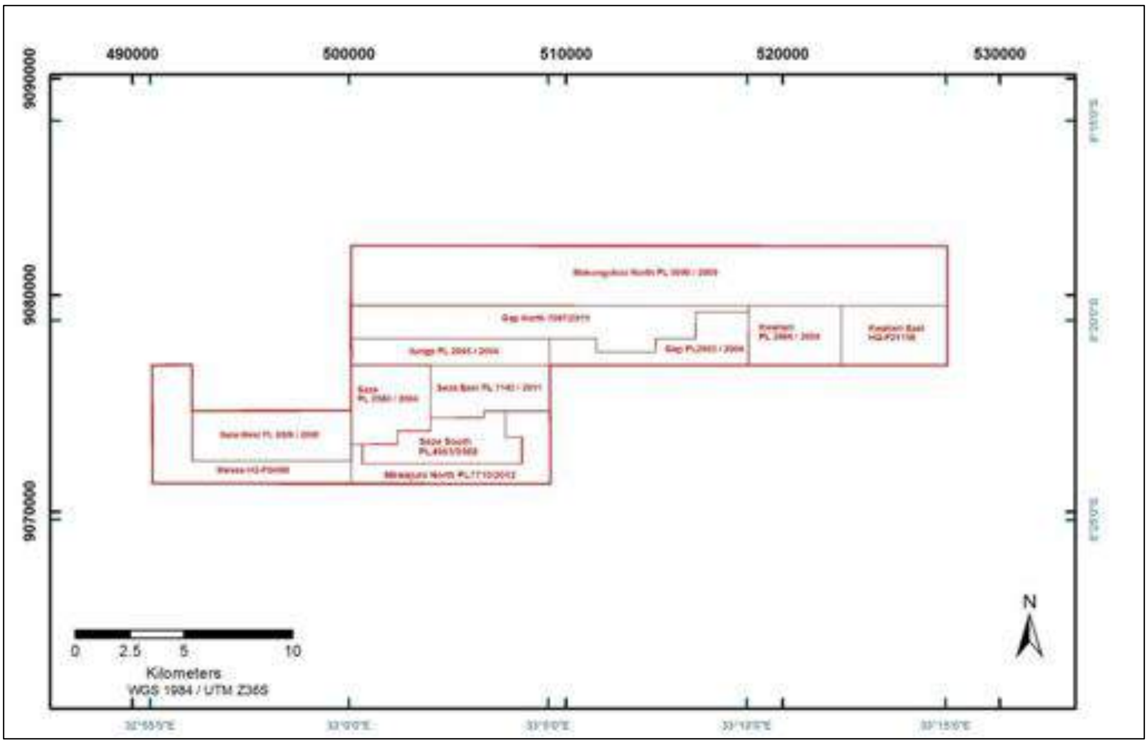

Figure 4-2: Map of SMP Licenses

4.3 Location of Mineralised Zones

Helio has identified over 30 targets (Figure 4-3) within the area covered by the SMP licenses (Figure 4-2). This report mostly concerns the targets where SRK considers there is sufficient information available to estimate Mineral Resources. These prospects are Kenge, Mbenge, Porcupine, Konokono and Tumbili.

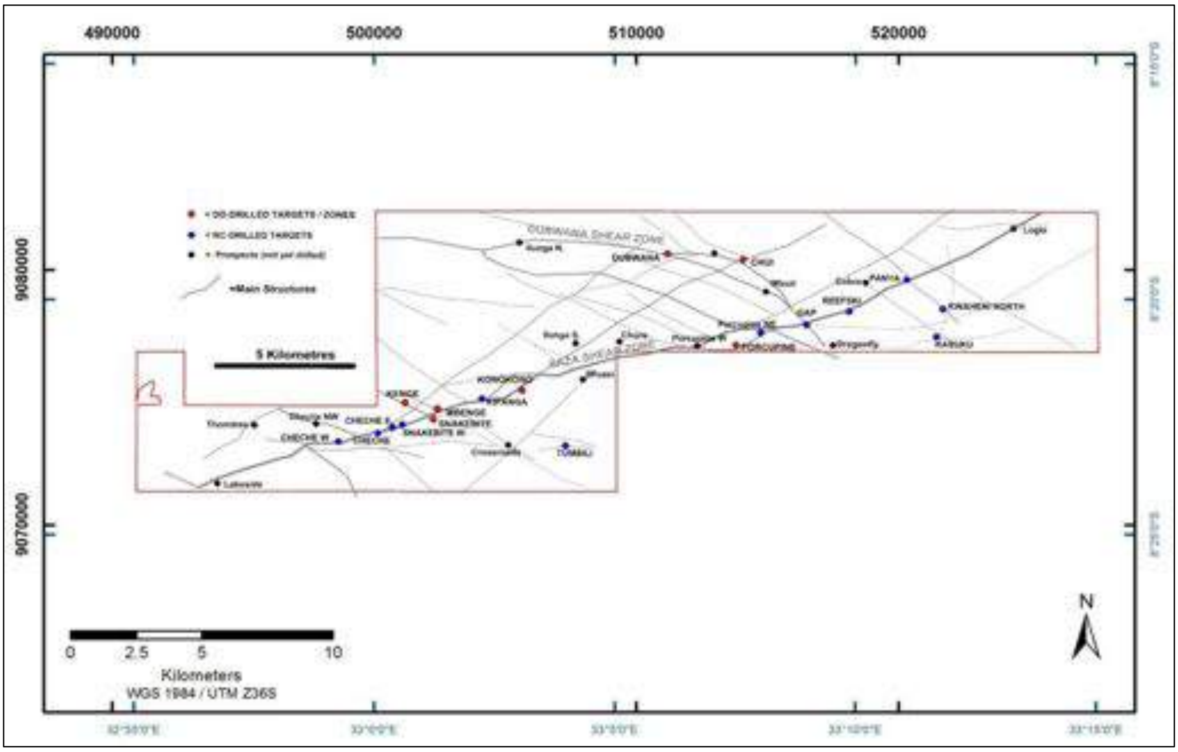

Figure 4-3: Map of prospects within the project area

4.4 Environmental Liabilities and Permits

SRK is not aware of any environmental issues or liabilities on the project and has no reason to doubt that all proper permits required to conduct exploration activities on the property have been obtained.

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

5.1 Topography, Elevation and Vegetation

The SMP is located on the Lupa Block, which lies along the eastern edge of the Western Rift Valley close to Lake Rukwa. In general the project area is flat, but a series of hills, the Ilunga range, occurs within the project area. The elevation ranges from around 900 m to 1729 m (Figure 5-1).

Vegetation throughout the area tends to be of the Miombo or Brachystegia-type woodland with occasional areas of thorn scrub. Moderate to intense deforestation for fuel and farming has occurred over much of the SMP and the surrounding countryside.

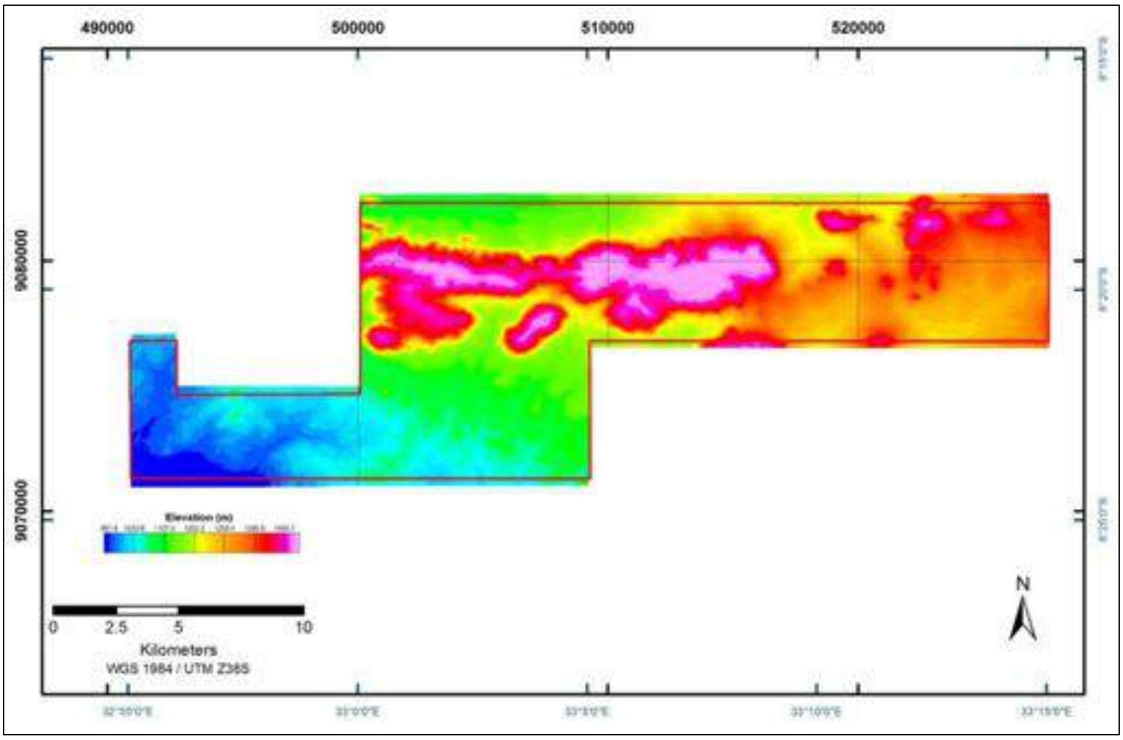

Figure 5-1: Topography within the SMP

5.2 Accessibility

Mbeya, the capital of Mbeya Region, is approximately 100 km southeast by road from the SMP. Mbeya is located on the main TAZARA railway and TANZAM highway, both of which link Dar es Salaam with Zambia and is thus a main hub for communications between southern and eastern Africa. There is a grass-strip airport which supports daily scheduled flights services to and from Dar es Salaam. Songea International Airport (100 km from the SMP, 20 km south-west of Mbeya) is expected to open in the second half of 2012.

The SMP is accessed by a dirt road from Mbeya. The journey between Mbeya and SMP requires approximately two to two and a half hours. At the time of writing sections of the dirt road are being tarred and it is anticipated that travel times and communications will be improved as a result.

The Regional Capital of Chunya is approximately one hour's drive east from the SMP. Chunya also has a grass airstrip and the road to Mbeya from Chunya is also currently being tarred.

5.3 Local Resources and Infrastructure

A 33 kVA power line runs along the road from Mbeya to Chunya, then through the SMP to Mkwajuni, where the Helio camp is located. Mains electricity is available on site. All the major cell phone networks have pylons in Mkwajuni and Makongolosi, with an additional Vodacom tower situated on the peak of the western end of the Ilunga hills (outside of the SMP).

Exploration services and equipment are accessible through the road and rail networks of Tanzania.

The local workforce consists primarily of subsistence farmers and occasional independent artisanal miners. Tanzania has a rapidly expanding mining industry and a reasonably qualified workforce could be developed from other areas of the country.

No mining infrastructure is located on the SMP. Shanta Gold Ltd is in the process of constructing the New Luika Gold Mine in license properties adjacent to the SMP. Shanta have announced that the commissioning of the New Luika Gold Mine will be by the end of the second quarter of 2012 (Shanta news release March 29, 2012).

The first gold has been poured at this site 31 August, 2012.

Helio established an exploration camp in Mkwajuni (some 5 km south of the SMP area) during early 2006. As the project has developed this camp has been expanded and improved. The camp is based in a renovated National Bank of Commerce building set within a large secure compound. The building provides an excellent large office area and living quarters for staff and the compound is large enough to allow for core logging, storage and cutting areas, as well as secure sample storage, diesel storage facilities and vehicle maintenance areas.

Small-scale fly camps are regularly mobilised to more remote areas to facilitate more efficient working schedules.

5.4 Climate

Tanzania's climate is sub-tropical; however, the climatic variation between the different regions of the country is significant; mountainous regions and coastal areas in particular are subject to significantly more rain than the lowlands and high plateau areas. Major rainfall is limited from November to April. Systematic weather monitoring stations are rare, the closest to the SMP being at Mbeya airstrip. However, given its notably higher altitude compared to the SMP and its proximity to a major mountain range, this location is significantly cooler and wetter than the project area. Since November 2007 Helio has collected rainfall and temperature figures at the Mkwajuni office.

Figure 5-2 displays the average monthly (daytime) temperature and rainfall (24 hours) recorded between November 2007 and December 2011.

Since commencing work on the SMP in 2006, Helio has not found that climatic conditions have ever significantly limited exploration activities.

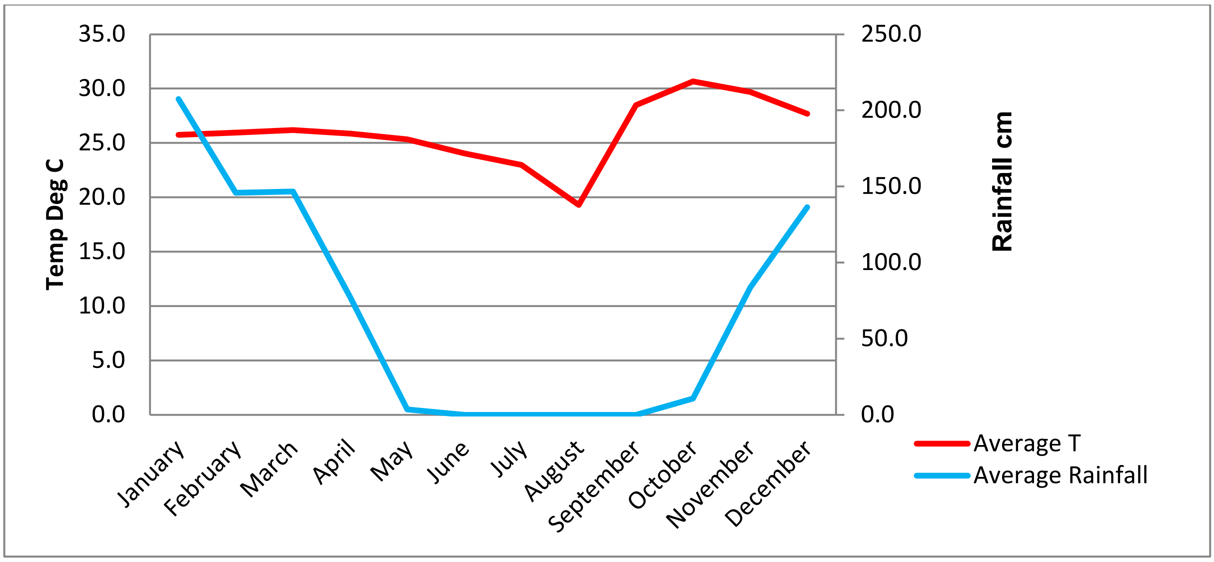

Figure 5-2: Temperature and rainfall averages, November 2007 to December 2011

Temperature readings are taken four times per day; at 7 am, 11 am, 3 pm and 7 pm. Rainfall is recorded as a total over a 24 hour period; 7 am to 7 am. Figure 5-3 and Figure 5-4 display temperature and rainfall data for 2008 to 2011. Due to staffing levels continuous recoding of these data is sporadic over December and January. Unreliable temperature data were collected between August and November 2011 (due to low battery power in the digital thermometer), therefore this data have been excluded from the dataset.

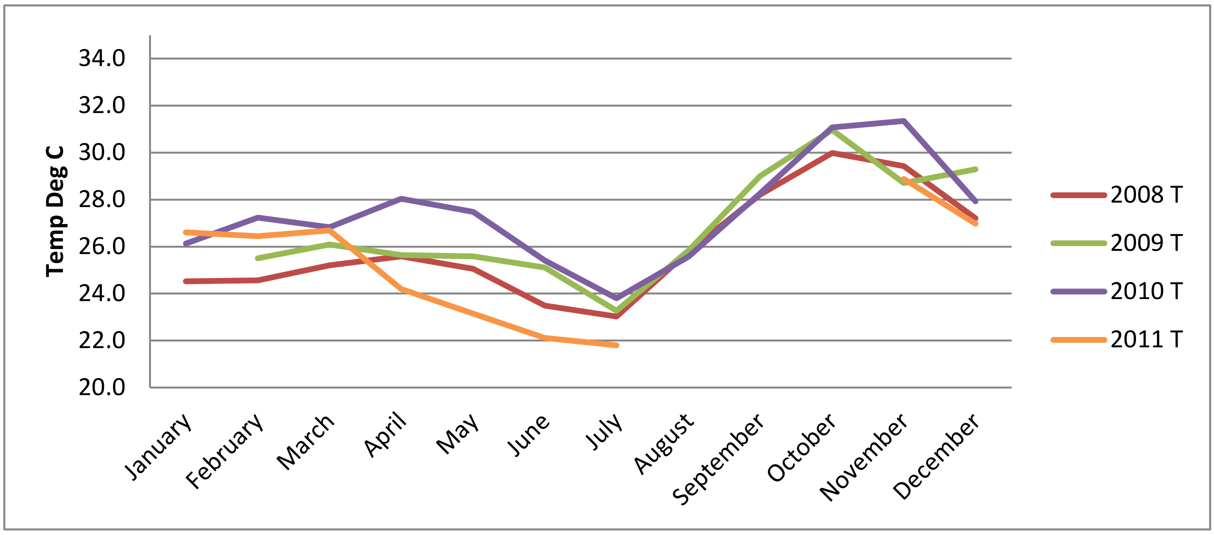

Figure 5-3: Recorded Temperature

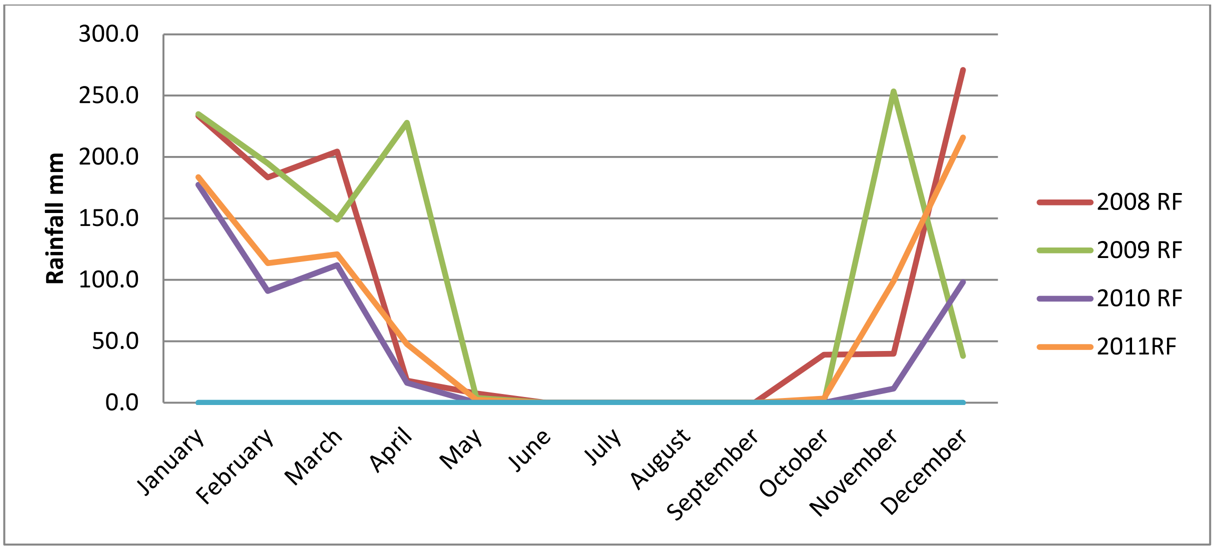

Figure 5-4: Recorded Rainfall

5.5 Physiography

Figure 5-5 shows the typical terrain at the site. The topography is gently sloping with light vegetation.

Figure 5-5: Typical Landscape in the Project Area.

Within the leases held by Helio, local artisanal mining activities take place intermittently. Figure 5-6 shows the remnants of these mining activities near the crest of Elizabeth Hill (near the Kenge mineralisation).

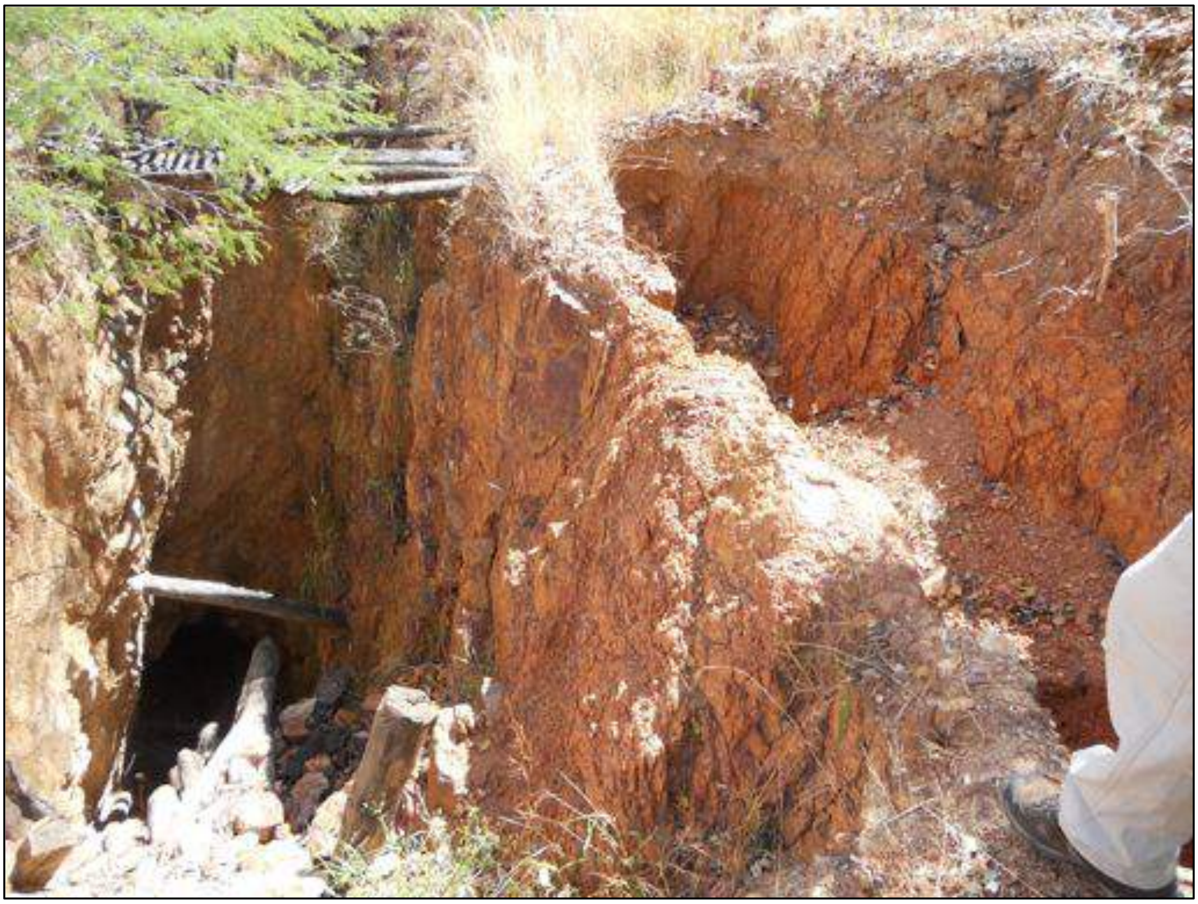

Figure 5-6: Artisanal mining activities at Kenge

6 History

6.1 Discovery and Historical Production

Gold was discovered in the early 1900s and the Lupa Goldfield began production, as an alluvial field, in 1922 (Teale & Oates 1946). Systematic mining started in 1935-6 by East African Goldfields at the Saza Mine (Gallagher 1936), which was subsequently continued by New Saza Mines Ltd between 1939 and 1956. The New Saza Mine was the largest mine on the goldfield and drew material from the Saza Mine Shafts #1 and #2, Luika Mine, Blacktree, Winter and Razorback mines. Reported production between 1939 and 1956 was 270,770 oz. of fine gold and 242,942 oz. of fine silver from 1.1 million tonnes of ore (Harris 1962).

Other small scale colonial era mines were exploited in the area, including Kwaheri, Gap and Nkatano; however, production is not clearly reported for these mines (Smith & Sango 2000).

Figure 6-1 shows historic mines located within the SMP area.

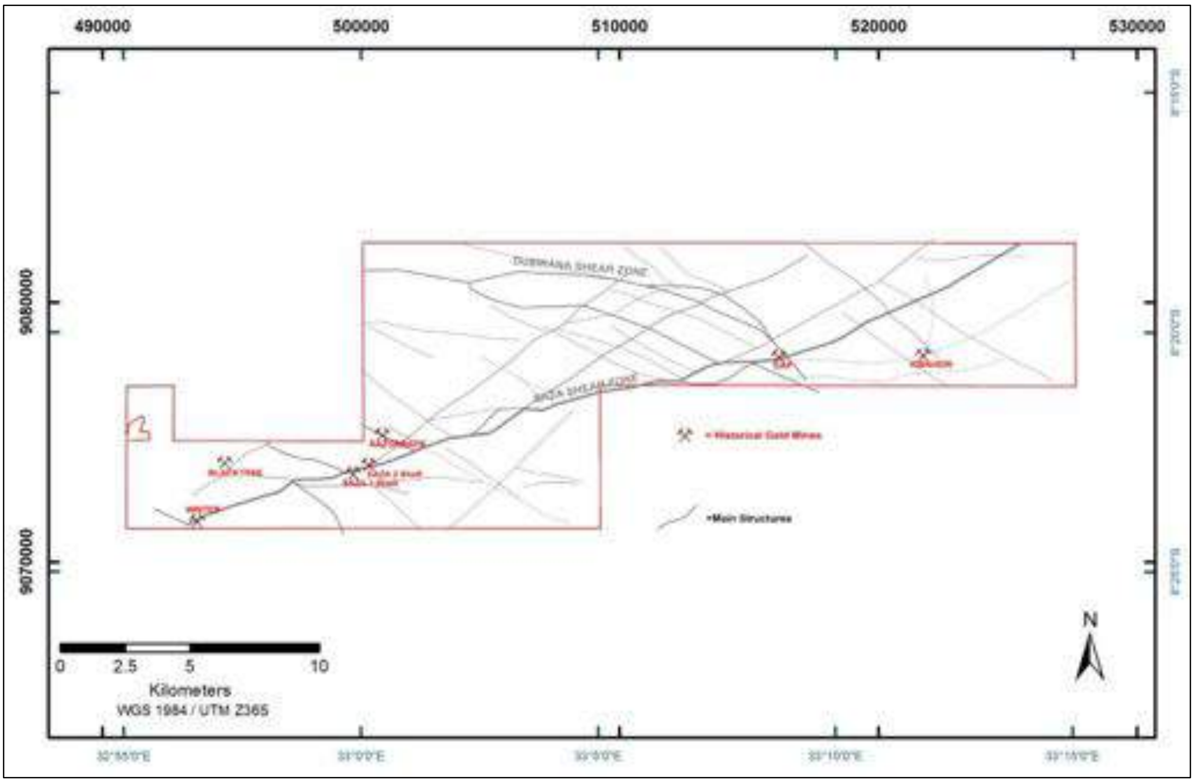

Figure 6-1: Location of historic mines within the SMP area

6.2 Post-Independence Exploration

Since Tanzania gained independence in 1961 several exploration programmes have been conducted over the areas that now make up the SMP area.

Table 6-1 gives a brief summary of historic exploration in the area details from these campaigns are given in the subsequent sections.

Table 6-1: Historic exploration in the SMP area

Company	Exploration Period
Technoexport	1970 – 1974
Princess Resources	1995 – 1999
Anglogold	1997 - 1999

6.2.1 Technoexport 1970-1974 (Luena et al 1974)

The Soviet-Tanzanian Agreement of 1969 provided for Technoexport to render Tanzania technical assistance in geological investigations including detailed prospecting for gold in the Lupa Goldfields. This is thought to be the first systematic prospecting and evaluation of the Lupa. Table 6-2 lists the total work conducted by Technoexport across the entire Goldfield.

Table 6-2: Work Conducted on the Lupa Goldfield by Technoexport between 1970 and 1974

Type of Work	Unit	Total Amount
Prospecting Traverses	km	450
Geochemical Survey	Sample	1014
Heavy Concentrates	Sample	7050
Channel Sampling	Sample	2550
Drilling		
For Reef Gold	Metres	5636
For Alluvial Gold	Metres	1525
Trenching	Cubic Metres	4800
Pitting (including cross cuts)	Metres	3150
Geophysical Survey		
Magnetics	Stations	12742
Electrical Profiling	Stations	17504
Vertical Electrical Sounding	Stations	344

Technoexport reported reserves of reef gold across the Lupa Goldfields to be 33,988 kg (~1 Moz), which could be increased with detailed exploration of several deposits, including Saza. Most of the reserves were reported to be in the North Western part of the goldfield and specifically noted that the closely localized deposits around the Gap mine and Nkutano (Helio targets Gap and Reefski respectively) were sufficient to operate a reduction plant, with additional ore added from Saza. The report; however, recommended further exploration of the area.

Information from Luena et al 1974 has been used by Helio for initial regional target generation when Helio first began work on the SMP. The whereabouts of core generated by Technoexport is unknown.

6.2.2 Princess Resources / CSA Africa 1995-1999 (Henderson & Lewis: various CSA Quarterly report)

Princess Resources held five PLs in the Makongolosi area in the mid to late 1990's, including ones that correspond to Helio's Ilunga and Kwaheri licences.

Information on this work is fragmented; however, it is clear the following activities took place:

- Remote sensing interpretation;
- Structural analysis;
- Geological mapping;
- Regional and detail soil sampling;
- Rock chip and trench sampling; and
- RC and Diamond drilling.

Discussions with the inhabitants of the village which was built out of the remains of the camp that CSA operated out of led to the discovery of a large pile of core which had been emptied out of its trays – none of this material is useful. A number of RC chip trays were recovered, however, many were damaged and missing material and the holes which had the best grade were absent.

In Helio's initial exploration over areas previously explored by Princess / CSA it became apparent that there were major issues with CSA's sampling methodology as well as accuracy and precision of their ability to locate their samples and drillholes. Therefore the work carried out by Princess / CSA has not been used to assist in the formulation of Helio's exploration strategy, other than in general terms.

6.2.3 Anglogold 1997-1999 (Smith & Sango February and December 2000)

Anglogold Exploration Tanzanian Limited worked across eleven PL's in the Lupa Goldfields. These PLs were owned by two separate companies, Tanganyika Gold Limited (TGL) and Dhahabu Exploration and Mining. Anglogold entered in to separate JV's with both companies whereby their subsidiary, Anmercosa Services (Eastern Africa) Limited, managed all exploration across all licenses. Exploration across all the PLs was conducted between September 1997 and October 1999.

Tanganyika Gold JV

Of the nine licenses held by TGL only five have direct links to the SMP. The licence areas worked on included those on Helio's Gap, Saza and Saza West licences. Listed below is all the work conducted by Anmercosa during the JV.

- Interpretation of regional airborne geophysical and Landsat image data;
- Integrated structural analysis;
- Landform and regolith mapping; and

Sampling, trenching and drilling displayed in Table 6-3.

Table 6-3: Work Conducted by Anmercosa between 1997 and 1999 across the 9 PLs belonging to Tanganyika Gold Limited

Type of Work	Unit	Total Amount
Regional Soil Grid	Sample	5693
Detailed Soil Grid	Sample	1578
Rock Grabs	Sample	191
Trenching	Metres	1080
Trench Samples	Sample	570
RAB Drilling	Metres	5239
RC Drilling	Metres	649
Diamond Drilling	Metres	949.5

Anmercosa identified two areas of interest for detailed exploration: the Stockwork Zone and the Saza Mine (Helio targets Konokono and Cheche respectively). Anmercosa concluded that during their exploration no significant gold mineralisation worthy of follow up was identified. The JV was terminated in 2000.

Dhahabu Exploration and Mining JV

The two PLs held by Dhahabu contained the Razorback and Gap mines. Listed below is all the work conducted by Anmercosa during the JV:

- Interpretation of regional airborne geophysical and Landsat image data;
- Integrated structural analysis;
- Landform and regolith mapping; and
- Sampling, trenching and drilling displayed in Table 6-4.

Table 6-4: Work Conducted by Anmercosa between 1997 and 1999 across the 2 PLs belonging to Dhahabu Exploration and Mining

Type of Work	Unit	Total Amount
Regional Soil Grid	Sample	427
Stream Sediment sampling	Sample	32
Rock Grabs	Sample	74
Trenching	Metres	726
Trench Samples	Sample	367
RAB Drilling	Metres	4513

Anmercosa targeted the Saza mine (Helio Cheche and Kenge targets) for detailed exploration and concluded that during their exploration no significant gold mineralisation worthy of follow up was identified. The JV was terminated in 2000.

Anmercosa's data that has been obtained by Helio has been digitized and where possible ground truthed. The data is in relatively good order however it has not been used by Helio for any other purpose than to carry out initial regional target generation when Helio first began work on the SMP. The whereabouts of core generated by Anmercosa is unknown.

7 Geological Setting and Mineralisation

7.1 Regional Geology

The Lupa Goldfield is situated at the southwestern part of the Tanzanian Craton, within the Lower Proterozoic mobile belt of the Ubendian System (Figure 7-1). Lithologies comprise granitic, intermediate and mafic intrusive rocks together with ferrunginous quartzites. The Lupa Goldfield is bounded to the south and west by the WNW trending Rukwa Faults, to the east by the NE trending Usangu Rift Faults and to the north by the ESE trending Northern /boundary Fault.

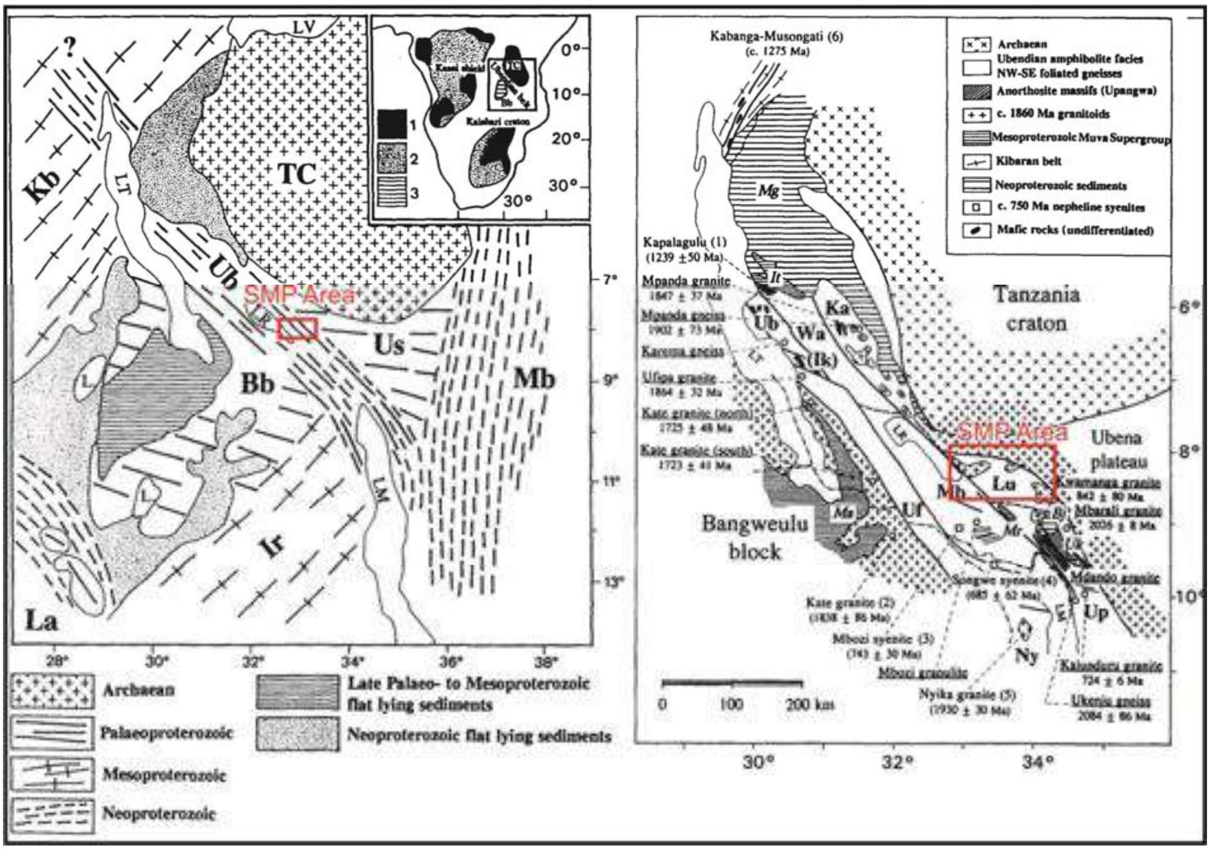

Figure 7-1: Regional setting of the Ubendian Belt and the SMP area (Lenoir et al, 1995)

The ferruginous quartzites are banded quartz-magnetite rocks, which are interpreted to have been banded iron formations and are presumed to be the oldest formation to occur (Smith and Sango, 2000).

There are a number of stages of granite intrusion. The earliest phase is often sheared and mylonitized. The intrusion of diorites appears to have been roughly coeval, however, intrusive relationships are difficult to establish at some locations.

The Ilunga granite, which comprises the Ilunga Hills, forms a prominent ridge. It is a hypidiomorphic (distinctly crystalline in nature) granite, tending towards alkali composition. The Saza granite is a true granite or granodiorite and also is hypidiomorphic in texture. Basic intrusive rocks of dioritic to gabbroic compositions are common in the Lupa Goldfield. Dolerite dykes are the youngest intrusive event.

Several prominent structural trends are observed in the Lupa Goldfield. A strong WNW to NW trend is seen in outcrop and satellite imagery. This foliation is associated with major dextral shear zones. Many of the WNW structures show NW-SE splays, which link adjacent shears and are common near lithological contacts.

Prominent ENE- to E-striking structures occur as shear and mylonite zones. These zones appear to be dextral, although a later sinistral displacement has also been reported. In some places these structures cut and displace the WNW structures, suggesting the ENE- to E-striking structures are younger.

The Saza Shear Zone is one of the well-known ENE-striking structures. It is over 35 km long and hosts most of the known significant gold mineralisation in the western part of the Lupa Goldfield.

NNW-striking structures are also known in the goldfield, in most cases they are not extensive and are often bounded by WNW-striking shear zones. NE-striking shear zones are also common and dextrally displace the WNW-striking shear zones.

All structures have had long histories of re-activation, the latest period occurring during Cenozoic rifting.

7.2 Property Geology

The project is located on the western margin of the Lupa Goldfield and the southwestern corner of the property covers a small portion of the Rukwa Trough. The lithologies of the project are mostly part of a bimodal igneous suite with minor volcanics. Recent sediments are present in the small area that lies within the Rukwa Trough.

The igneous suite is dominated by granite and granodiorite. Two distinct granites are observed:

- Saza Granite- a post-tectonic hornblende-biotite granite with coarse grained quartz and feldspar, in places grading to a granodioritic composition; and
- Ilunga Granite: A medium grained, leucocratic, alkali granite, thought to have been intruded towards the end of the Ubendian Orogeny.

The Ilunga granite is observed extensively in the northern half of the project. It outcrops prominently and is the primary constituent of the E to W trending Ilunga Hills and of the smaller hills in the east of the license area. The Saza Granite is observed extensively in the southern portion of the prospect area. Gold mineralisation occurs in both the Ilunga and Saza Granites.

Significant diorite / gabbro bodies are observed in the southern portion of the project, but contacts are often mutually intrusive and therefore relationships between the mafic and felsic lithologies can be difficult to ascertain.

Except when the rocks are sheared, primary igneous textures are preserved. Lower greenschist facies metamorphism is common and there is little evidence for a major thermal event.

The Saza Shear zone is the dominant structure in the area (Figure 7-2), striking 070° and traceable for over 35 km within the project. The New Saza Mine's 1 and 2 Shafts, as well as the Gap and Winter mines, are all located at various points on the Saza Shear.

Two prominent structure sets are found in the project, both of which are associated with mineralisation. As such, they have been given names which reflect the nature of structures where they were first identified (the Saza Shear Zone and the Kenge Shear Zone):

- Saza-parallel (striking 070°); and
- Kenge-parallel (striking 120°).

The significance of the intersections of these two structural orientations is becoming increasingly apparent. For example, the main zone of the Porcupine Target lies on an intersection of Saza-parallel and Kenge-parallel structures.

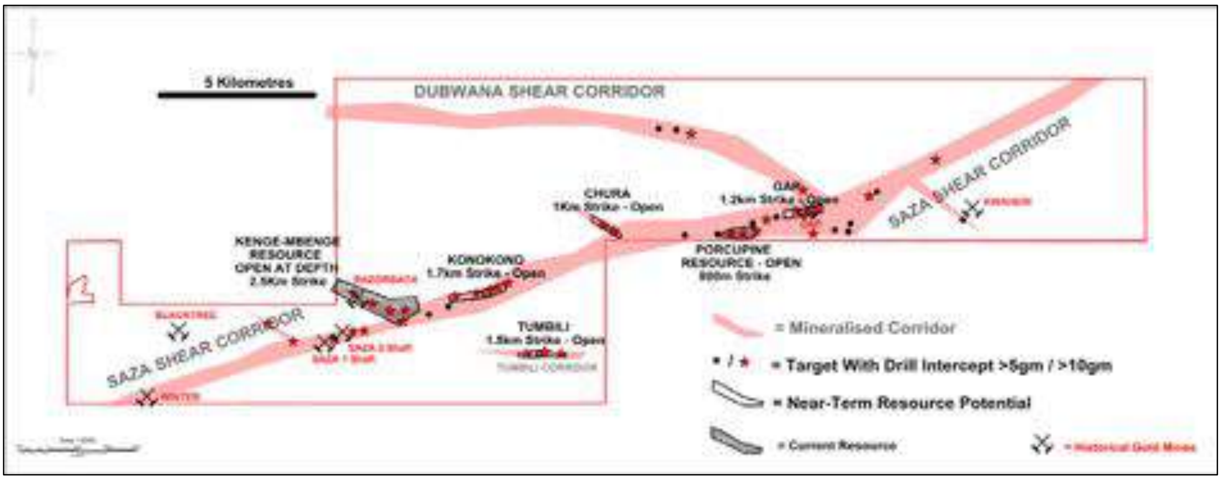

Figure 7-2: The Saza and Dubwana Shear Zones in relation to targets, historical workings and current Mineral Resources

Source: Helio

7.3 Mineralisation

Gold in the Lupa Goldfield is observed to be to be associated with shear zone mineralisation, whether this mineralisation style is orogenic or intrusion-related is still unclear. Mineralisation is widespread across the SMP and varies in size from metre- to kilometre-scale deposits.

Mineralisation at the SMP is relatively simple, comprising of pyrite (generally less than 1% by volume) with minor chalcopyrite and molybdenite plus occasional scheelite and galena. Gold occurs as free gold and occasionally as tellurides. Mineralisation is associated with quartz veining, silicification, sericitisation, haematisation (demagnetisation) and occasionally chloritisation. Minor brecciation is important in localising mineralisation at Porcupine and mylonitisation is dominant at the Kenge Main Zone.

The two deposits which are the main focus of this report, Kenge and Porcupine, display the characteristics described in the following sections of this report.

7.3.1 Kenge

Mineralisation is focused on a series of massive anastamosing quartz veins which are emplaced in a shear zone that trends 120˚ and follows the contact between Saza Granite and granodiorite bodies. Gold mineralisation is observed up to 40 m in thickness and is associated with the vein quartz and the mylonitized Saza Granite and granodiorite wall rocks.

The Kenge Target has a strike length in excess of 2000 m and is made up of five zones: North West, Main, South East, Snake bite and Mbenge.

7.3.2 Porcupine

A sheeted vein quartz system within the Ilunga Granite at the intersection of two structures (070˚ and 120˚). Gold mineralisation is observed up to 90 m in thickness in the vein quartz and the altered host granite.

7.3.3 Konokono and Tumbili

Drilling on the Konokono and Tumbili targets is much more restricted than on Porcupine and Kenge, with the majority of holes being drilled using an RC rig. The mineralisation observed at these targets corresponds broadly with the Porcupine system, specific mineralisation characteristics will be formulated as these targets are developed over time.

7.4 Re-Os Dating

Helio has conducted Re-Os dating on molybdenite at the Kenge and Porcupine deposits. Kenge has recorded an age of around 1.93 Ga and Porcupine of 1.88 Ga. As such, the mineralisation at the SMP is more similar to deposits of the Birrimian in West Africa (e.g. Chirano and Ahafo deposits) than those elsewhere in Tanzania, which are Archaean in age.

8 Deposit Types

The gold deposits within the project area can be broadly described as shear zone-hosted orogenic or intrusion-related gold systems. Mineralisation is dominantly associated with Saza-parallel (070°) and Kenge-parallel (120°) shear zones. These directions follow the trends of the Palaeoproterozoic Ubendian and Usagaran belts.

Grantham (1932) suggests the Saza Granite is the parent of at least some of the gold reefs in the area, whereas Gallagher (1932) suggests that there is a genetic relationship between mineralisation and alaskite (leucogranite) which is based mostly on observations of the association of mineralisation with leucogranite at the Luika Mine (not within the Project). Within the project, leucogranite is a minor constituent of the lithological suite. There is little evidence to suggest a major thermal event. Work conducted by Helio on the provenance of the mineralizing fluids favours an intrusion-related origin.

Rhenium – Osmidium ("Re-Os") dating carried out by Helio on molydenite at the SMP deposits indicates an age of around 1.88 to 1.93 Ga. As such, the mineralisation at the SMP is similar in age to the Birimian deposits of West Africa (e.g. Chirano and Ahafo deposits). Gold deposits elsewhere in Tanzania are usually Archaean in age.

The deposit characteristics of the Kenge-Mbenge and Porcupine targets are as follows:

8.1 Kenge

- Target hosted within a ductile deformation regime;
- Mineralisation is focussed along a 120° trending shear zone (which follows the contact of a granitoid);
- Gold is hosted by pyrite-bearing quartz veins and mylonitized wall rock which has locally undergone intense sulphidation and sericitisation;
- Quartz veins range from 10 cm to 10 m in thickness;
- Drilled intersections of the main mineralised structure up to 40 m thick; and
- Alteration is dominated by sericite-chlorite-carbonate, which is characteristic of low temperature and pressure hydrothermal systems.

8.2 Porcupine

- Target hosted within a brittle deformation regime;
- Mineralisation is focussed by the intersection of Kenge-parallel and Saza-parallel shear zones within the Ilunga Granite;
- Gold mineralisation is associated with a sheeted vein quartz, quartz/pyrite and pyrite fracture system;
- Quartz veins range from 0.5 cm to 2 m in thickness;
- Mineralised structures are up to 90 m thick; and
- Alteration is dominated by sericite-chlorite-carbonate, which is characteristic of low temperature and pressure hydrothermal systems.

8.3 Konokono and Tumbili

Drilling on the Konokono and Tumbili targets is much more restricted than on Porcupine and Kenge, with the majority of holes being drilled using an RC rig. The mineralisation observed at these targets corresponds broadly with the Porcupine system, however in both cases a number of different granites are observed and significantly more mafic material is present. Specific deposit characteristics will be formulated as these targets are developed over time.

9 Exploration

Helio began exploration operations on the Saza PL (PL2580/2004) in April 2006. In the last quarter of 2006 the Gap, Kwaheri and Ilunga PLs (2963/2004, 2964/2004 and 2965/2004 respectively) were added to the SMP and initial field work was conducted. In October 2008 the Saza West PL (5326/2008) was added to the project and the SMP attained its current dimensions.

Since the beginning of exploration activities in 2006 a number of geophysical, geochemical, drilling and remote sensing exercises have been conducted across the SMP. In addition to work conducted by Helio there have been a number of occasions where Helio has used contractors to assist. Exploration conducted on the SMP since 2006 is summarised in the following tables:

Table 9-1: Regional soil geochemistry

Table 9-2: Detailed soil geochemistry

Table 9-3: Geophysical Surveys

Table 9-4: Airborne Magnetic and Radiometric geophysical surveys

Table 9-5: Summary of holes drilled by Project License and year

Table 9-6: Drillholes and metres by year

Table 9-7: Metallurgical testing

Table 9-8: Studies by consultants

In addition to the work listed in the tables below, Helio has also conducted mapping exercises of varying complexities over many areas and specific targets within the SMP at all stages of the project to date. During this field work a total of **548 rock samples** have been collected and analysed.

Table 9-1: Regional soil geochemistry

PL	Year	Samples	Specifications
Gap (2963/2004)	2007	866	250 m x 250 m offset grid
Kwaheri (2964/2004)	2007	865	250 m x 250 m offset grid
Ilunga (2965/2004)	2007	843	250 m x 250 m offset grid
Saza (2580/2004)	2007	865	250 m x 250 m offset grid
Saza West (5326/2008)	2008	565	250 m x 250 m offset grid

Note that the number of samples includes duplicate samples and Certified Reference Material inserted for QA/QC.

Table 9-2: Detailed soil geochemistry

PL	Year	Target	Samples	Specifications
Gap (2963/2004)	2007	Dubwana	245	50 m x 100 m grid
Kwaheri (2964/2004)	2007	Panya	258	25 m x 100 m grid
Saza South (4963/2008)	2008	Tumbili	1,087	25 m x 100 m grid
Combined Programme: Saza East (7143/2011) Ilunga (2965/2004) Saza (2580/2004) Saza South (4963/2008)	2011	Saza East and surrounds	 2880 1358 503 288	25m x 200m grid

Note that the number of samples includes duplicate samples and Certified Reference Material inserted for QA/QC

Table 9-3: Geophysical Surveys

PL	Year	Method	Line km
Saza (2580/2004)	2006	IP – Gradient Array and Pole-Dipole. Magnetics	130
Saza (2580/2004)	2007	IP – Gradient Array. Infill lines	8
Saza (2580/2004)	2007	Magnetics	50
Gap (2963/2004)	2007	IP – Gradient Array	78
Kwaheri (2964/2004)	2007	IP – Gradient Array	130
Ilunga (2965/2004)	2007	IP – Gradient Array	50

Table 9-4: Airborne Magnetic and Radiometric geophysical surveys

Year	Line km	Specifications
2007	1130	200 m line spacing, 20-30 m elevation (terrain dependant)
2009	5290	50 m line spacing, 20-30 m elevation (terrain dependant)

Note that in 2007 the SMP consisted of the Saza, Gap, Kwaheri and Ilunga License, whereas in 2009 it also included Saza West. The airborne surveys covered all license areas which were operated by Helio at the time of flying.

Table 9-5: Summary of holes drilled by Project License and year

	2006		2007		2008		2009		2010		2011		Total
	RC	DD	RC	DD	RC	DD	RC	DD	RC	DD	RC	DD	PL
Saza (2580/2004)	33	6	88	67		112	12	2	17			11	348
Kwaheri (2964/2004)			3				23		26				52
Gap (2963/2004)			6		20	41	100	20	31	31		50	299
Ilunga (2965/2004)												16	16
Makongolosi North (5990/2009)								5	33				38
Saza South (4963/2008)									16	4	36		56
Saza West (5326/2008)							25		10				35
Saza East (7143/2011)											37		37
Total Year	**33**	**6**	**97**	**67**	**20**	**153**	**160**	**27**	**133**	**35**	**73**	**77**	**881**

Table 9-6: Drillholes and metres by year

Year	RC holes	RC metres	DD holes	DD metres	Total holes	Total metres
2006	33	3,138	6	1,027.55	39	4,165.55
2007	97	8,531	67	9,485.82	164	18,016.82
2008	20	1,621	153	27,177.85	173	28,798.85
2009	160	15,049	27	8,945.45	187	23,994.45
2010	133	12,136	35	7,809.2	168	19,945.20
2011	73	6561	77	10,200.17	150	16,761.17
Total	**516**	**47,036**	**365**	**64,646.04**	**881**	**111,682.04**

Table 9-7: Metallurgical testing

PL	Job Number	Drill Material	Date of Report
Saza (2580/2004)	11940-001	50 kg Composite of SZD011, 013 & 021	August 2008
Saza (2580/2004)	11940-002	Remaining material from job 11940-001 (SZD011, 013 & 021)	May 2009
Gap (2963/2004)	11940-003	50 kg of material from GPD004	August 2009

Refer to Section 13 for a discussion of the metallurgical testing.

Table 9-8: Studies by consultants

PL	Contractor	Work Conducted	Date
Saza (2580/2004)	Dave Coller	Study of structural controls of Au mineralisation in the Kenge target	April 2008
Gap (2963/2004)	SRK Consulting	Study of structural controls of Au mineralisation in the Porcupine target	January 2010
All SMP	Impel Geoscience	Study of structural controls of Au mineralisation across the SMP	July 2010
All SMP	Golder Associates	Mineral Resource estimation	November 2010
All SMP	SRK Consulting	Review of Mineral Resource estimation	January 2011

10 Drilling

The collar locations of all drillholes in the SMP database are shown in Figure 10-1. In Helio's database, the primary coordinate system used is UTM Zone 36S, datum WGS 84.

The drillholes are listed in Appendix A and the relevant intervals from these holes used for estimating the Mineral Resources are in Appendix B.

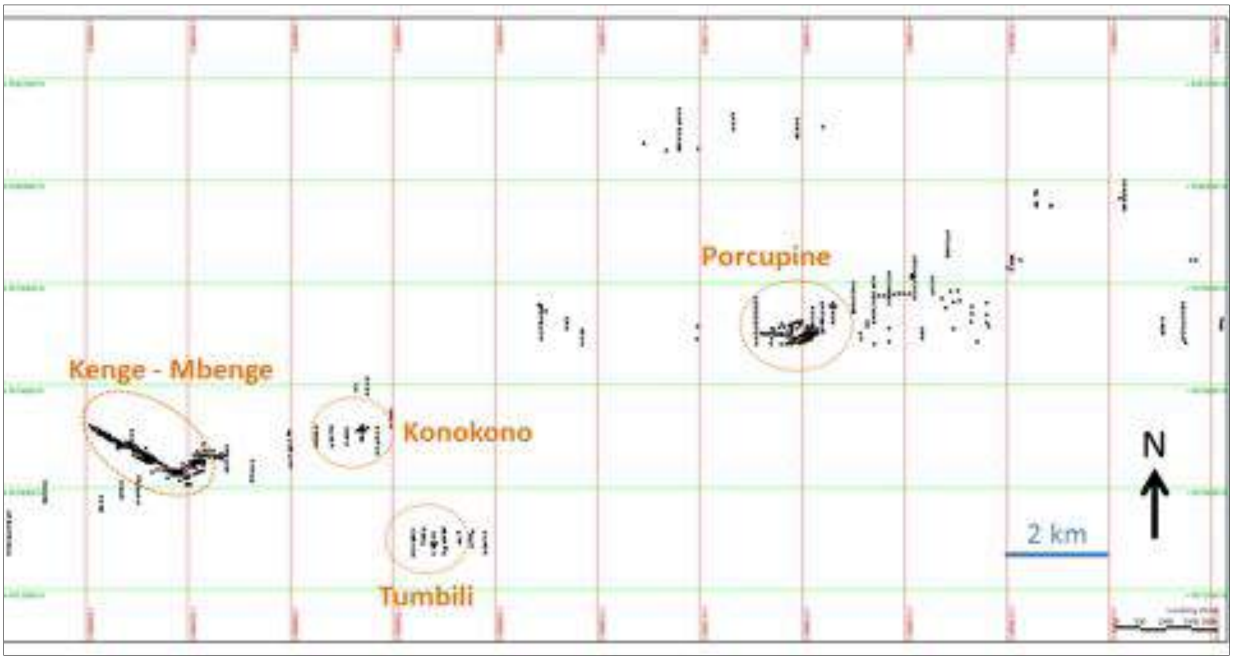

Figure 10-1: SMP drillhole collars

Helio has undertaken twelve drilling campaigns during exploration on the SMP.

Table 10-1 displays statistics of each campaign. Helio has utilised diamond core and reverse circulation drilling (DD and RC respectively) across the SMP, using the two methods for specific aspects of exploration.

RC drilling is mainly utilised as a first pass exploration tool: once a target is identified through geochemical, geophysical or mapping work, an RC rig will be used to drill exploratory drill fences, which allows the Company to quickly test large areas of ground at moderate expense.

Diamond core drilling is used mainly to follow up any discoveries made by RC drilling. The DD rig will in most cases replicate the original RC hole to confirm the original drilling and assess the degree of upgrade in assay figures which is frequently seen when comparing RC to DD grades. If the repeated hole confirms the discovery in the RC hole, diamond drilling will then be used to drill a grid pattern around the discovery hole to assess the strike and dip extensions of the discovery. If a discovery is significant then a drill plan will be designed on a local grid to regulate the drilling in the area.

On occasion access and availability constraints will result in RC and DD rigs swapping roles so as to enable first pass and detailed drilling in areas which are off limits to the machines that would ordinarily do the job.

Table 10-1: Drilling programmes on the SMP to December 2011

Year	Company	Rig	Holes	Metres
2006	Major Drilling Tanzania	KL150		3,138
	Geo-logical Drilling Ltd	Longyear 38		1,027.55
2007	Stanley Mining Services Ltd	UDR650		8,531
	Geo-logical Drilling Ltd	Longyear 38 Longyear 44		9,485.82
2008	Capital Drilling Tanzania Ltd	KL600		1,621
	Geo-logical Drilling Ltd	2 x Longyear 38's Longyear 44 2 x Goldenbears		27,177.85
2009	Tandrill Ltd	Smith Capital 10RSH		15,049
	Geo-logical Drilling Ltd	Goldenbear		8,945.45
2010	Tandrill Ltd	Smith Capital 10RSH		12,136
	Geo-logical Drilling Ltd	Longyear 38 Goldenbear		7,809.20
2011	Layne Drilling Ltd	Smith Capital Hotline		6561
	Geo-logical Drilling Ltd	2 x Longyear 38		10,200.17
Total			**881**	**111,682.04**

Of the 881 holes drilled, all but 51 were surveyed using digital downhole survey tools. Major Drilling did not survey their work, neither did Capital Drilling (NB two of the holes drilled by Major were subsequently surveyed by Geo-Logical in order to replicate discovery holes). The following tools were used by the other drilling contractors:

- Geological Drilling: Reflex;
- Stanley Mining Services: Flex-it;
- Tandrill: Reflex; and
- Layne: Flex-it.

Of the 365 diamond drillholes 86 were drilled using orientation equipment, 16 using Ezi-Mark tools, 70 with a Reflex Act tool. All holes were located during drilling using a standard handheld GPS. Once the rig vacated the site a Differential GPS unit is used to record an accurate and precise set of X, Y and Z coordinates.

10.1 Reverse Circulation Drilling

Helio has prepared a manual of standard operating procedures (Helio, 2010). This section summarises the procedures relevant to RC drilling and sampling.

10.1.1 Positioning of RC Drillholes

The vast majority of RC drilling conducted by Helio takes the form of fence drilling. Once a target is identified a line of RC holes is planned, the holes are positioned so that there is an overlap at the top and bottom of each hole. Figure 10-2 illustrates how an RC fence allows for precise quantities of drilling to be planned and executed whilst ensuring anomalous zones (picked out in blue) are sampled regardless of the spatial extent of the zone compared to the length of the hole.

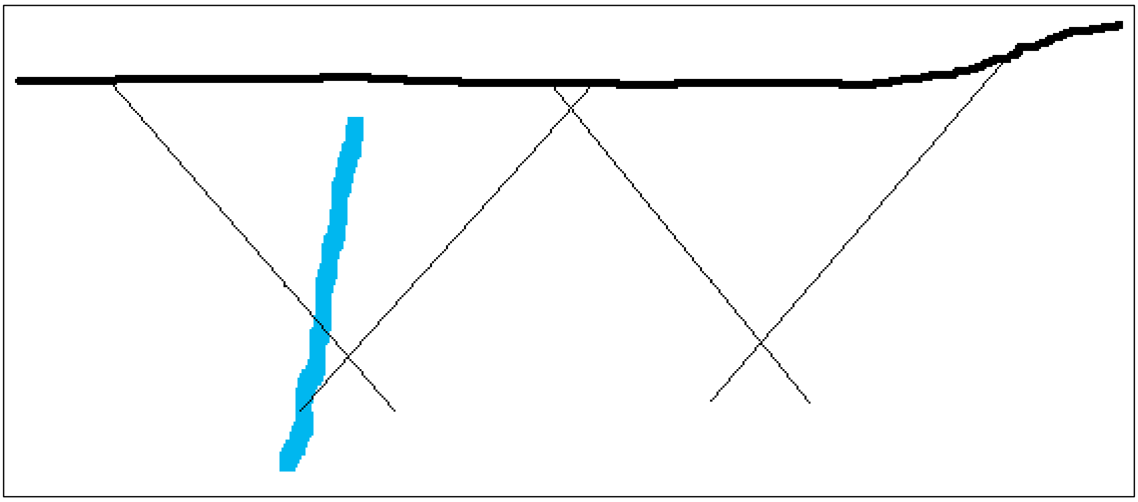

Figure 10-2: Illustration of an RC drill fence

Drillholes in black, anomalous zone in blue

In rare instances there may be factors which require an RC rig to drill single holes, in which case the target is thoroughly reviewed and a cross section drawn up to ensure the hole is drilled to sufficient depth to intersect any postulated zones of mineralisation.

10.1.2 RC Drilling Procedures

The rock chip material produced by an RC rig is collected directly from the machine's cyclone by a Helio employee, with the material collected for each metre depth of the hole in specially prepared and marked rice sacks. To minimise downhole smearing of anomalies the driller is instructed to lift the rod string slightly and blow out the hole after every metre is drilled. Once removed from the cyclone the sample weight is recorded.

The drilling contractor is required to clean out the cyclone at least between holes, or at the end of the day. If significant water is intersected in the hole the cyclone is cleaned more regularly.

The sampling / logging area is located up wind of the machine to minimise contamination from dust released during the drilling process. Except where there are small amounts of recovered material or where the sample is wet (in which case pipe sampling is used), all material is passed through a three-tier riffle splitter to homogenize the sample and reduce it to a suitable size. Depending on the size of the bit used to drill the hole, the remaining 1/8th portion of recovered material results in a sample size of 2-4 kg.

In order to maximise the detail of sampling in each hole whilst at the same time minimizing the cost of sample transportation and analysis, RC holes are composited into 2 m samples. Usually, 2 m composite samples are made up from consecutive sub-samples, which are homogenized through the splitter. This results in two samples being collected: a 2 m composite laboratory sample; and a 2 m composite reference sample which are stored by Helio.

To reduce the possibility of cross contamination between samples, a compressed air gun using the HP take-off from the rigs compressor is used to clean the splitter between samples. Where this is not possible, stringent (water-free) efforts to clean the splitter are made to reduce the possibility of cross contamination.

Lithological logging is conducted using washed chips which are subsequently stored by the metre in a chip tray. Lithological observations are recorded using prescribed forms and standard lithological codes. Helio has modified the Australia Geological Survey Organization (AGSO) drill codes to give a standard Helio Code for each metre drilled. In addition to the chip tray which is stored in the Mkwajuni office upon completion, a chip pad of material is created for each hole: drill chips (washed in a sieve) from each 1 m drill sample are piled next to the dust from the same sample in the order in which they are drilled. A photograph of the chip pad is taken in uniform lighting which will give a record of both the solid and powdered colours of the material drilled (Figure 10-3). Should the production rate of the rig surpass the sampling rate then all logging exercises are suspended to ensure that priority is given to the sampling procedure.

Figure 10-3: RC chip pad

Magnetic susceptibility is measured from the initial 1 m rice sack sample where possible, if the speed of drilling is such that this operation must be suspended then the measurements are taken from the 2 m composite reference sample.

RC holes are routinely surveyed to record azimuth and dip. Holes will be surveyed at approximately 15 m depth to confirm that the rig is set up correctly a survey at the base of the hole is also taken to confirm the path of the hole. In holes longer than 100 m a third survey is taken midway down the hole.

10.2 Diamond Drilling

Helio has prepared a manual of standard operating procedures (Helio, 2010). This section summarises the procedures relevant to DD drilling and sampling.

10.2.1 Hole Planning

Diamond drilling is usually conducted as a follow up to RC drilling: once a target has been confirmed by analysis of RC material a DD rig will be mobilized to redrill the RC hole to replicate the results. The majority of DD holes drilled are part of a plan which is devised around the known and postulated extents of mineralised areas. Should a target show good potential for strike and dip extent a 'mine grid' is generated, usually on 25 m x 25 m centres which allows for systematic drill location on a variety of scales.

In rare instances a DD rig may be used as a first pass exploration tool, in which case fence drilling programmes (like those described for RC drilling) may be under taken. Alternatively a DD rig might be used to test new targets in which case the target area is thoroughly investigated and a cross section drawn up to ensure the hole is drilled to sufficient depth to intersect any postulated zones of mineralisation.

10.2.2 Downhole Surveys

Diamond core holes are routinely surveyed to record azimuth and dip. Holes will be surveyed at the top of the hole at the start of drilling to confirm that the setup of the rig is correct, a survey is taken every 50 m downhole and a final survey is taken at the base of the hole to confirm the path of the hole.

10.2.3 Core Processing and Logging

Once recovered core has been reconstructed and cleaned, it is placed in a core tray with a core block introduced to the core string after each drill run. All core boxes are labelled with the hole number, box number and metres (from and to). Should a core orientation tool be used on a hole, the reorientation and marking of the bottom line on the core is completed prior to the core being inserted into the core tray. Any artificial core breaks made by the drillers are clearly marked so as to give an indication of fractures not to be included in RQD measurements.

Once the core is cleaned, reconstructed, marked and in the core tray, it is stacked at the drill site and regularly removed to the core processing site. Core is transported in a metal frame which holds the core trays safely in place. Each tray is covered with a thick layer of foam padding to stop the core from moving during transportation.

On arrival at the core processing site a summary log is completed by the geologist prior to detailed geological logging and core processing. The main purposes of the summary logging are to monitor the progress of the drilling, make a rapid assessment of how the actual rock types intersected by the hole compare against the predicted lithologies and mineralisation and be the basis for progress reports to Helio management.

The core is received at the processing site by geotechnicians and given a second clean and reconstruction. Where orientated core is being handled the bottom of core mark is extended to its fullest extent and compared up and downhole to the next bottom of core mark. Where core is un-orientated or orientations are not possible, an arbitrary line is drawn on the core in a different colour. The lines on the core (orientated or un-orientated) are used as cutting guides when the core is split for sampling.

Using the core blocks inserted at the bottom of each core run, the core is metre marked to aid with logging and sampling. At this stage the core is photographed dry and wet so a record of the core in its original state is created.

Once the core has been fully prepared and photographed a number of different logs and processes are recorded:

Lithological Log

All holes are subject to a comprehensive lithological log. The log records depth, lithology, contacts, structure, alteration, veining and mineralisation. Logging of core is done using prescribed forms and standard lithological codes. Helio's logging codes are based on the standard codes used by the Australian Geological Survey Organisation (AGSO).

Structural Log

If core is orientated α, β and γ angles are measured for contacts, fractures, veins, foliations, lineations and any other structure found in the core.

Sample Log, Preparation and Dispatch

Sampling regimes differ depending on the knowledge of a target. If the target has not been drilled or the understanding of the target is poor then the entire hole is sampled. If there is a strong understanding of mineralisation controls at the target, then zones of interest are sampled continuously; the area has a bracket of sampling around it appropriate to the size of the zone. Areas believed to be barren are sampled at least once per box.

Once samples are defined the core is split, the samples bagged and dispatched to the laboratory (see Item 11: Sample Preparation, Analyses and Security).

Core Recovery and RQD Log

Core recovery percentage is determined by comparing the measured length (ML) of the core between two core blocks and dividing it by the indicated length (IL) noted by the driller on the end of run core block (ML/IL x 100 = core recovery %).

The quality of the core is defined as the percentage of core recovered during drilling, counting only those pieces of intact rock over 100 mm long.

Magnetic susceptibility Log

Magnetic susceptibility readings are taken at each metre on the metre. Additional readings are taken over anomalous zones where these do not coincide with the default 1 m spacing.

Specific Gravity Log

The SG of each sample taken is measured (unless the sample would not survive immersion in water). This is usually completed once the core has been split and the sample dispatched to the laboratory. Each piece of half core in the sample has a number written on it to aid with reconstructing the core once the SG is measured. The sample is weighed in air and again in water and the following calculation used to calculate the SG of the sample.[1]

[1] SG = weight in air / (weight in air – weight in water)

10.3 Relationship of Drilling to the Orientation and True Thickness of Mineralisation

Drillholes are generally planned to have an azimuth that is perpendicular to the strike of the mineralised zone. This azimuth is usually 030° for Kenge, 360° for Mbenge, 340° for Porcupine and 360° and 180° for Konokono and Tumbili. Figure 10-4, Figure 10-6, Figure 10-8 and Figure 10-9 show the drilling pattern in relation to the wireframes SRK modelled to constrain grade estimation in the mineralised zones.

Drillhole dips range from moderate to vertical, but are generally planned to intersect the steeply-dipping mineralisation at a high angle. SRK estimated that, where the drillholes intersect mineralisation, the true thickness of the mineralised zone is almost always in the range of 70% to 95% of the intersection length. The cross sections in Figure 10-5 and Figure 10-7 show the typical relationships between the orientations of the drillholes and the mineralised zones. These sections are also typical of the depth extent of the Kenge-Mbenge and Porcupine drilling coverage.

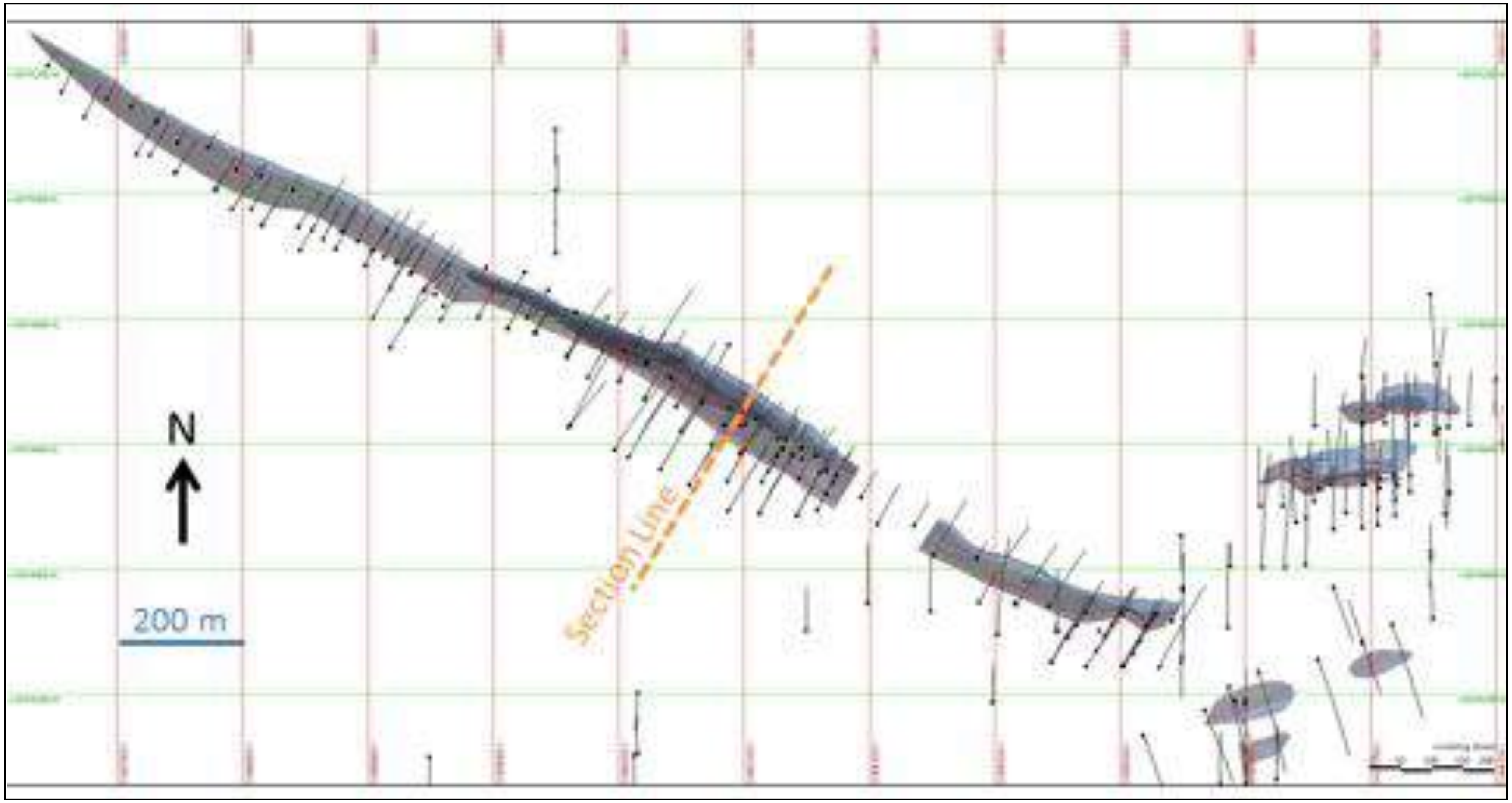

Figure 10-4: Map of Kenge-Mbenge drillholes and interpretation of mineralised domain

Figure 10-5: Kenge cross section, view towards azimuth 120°

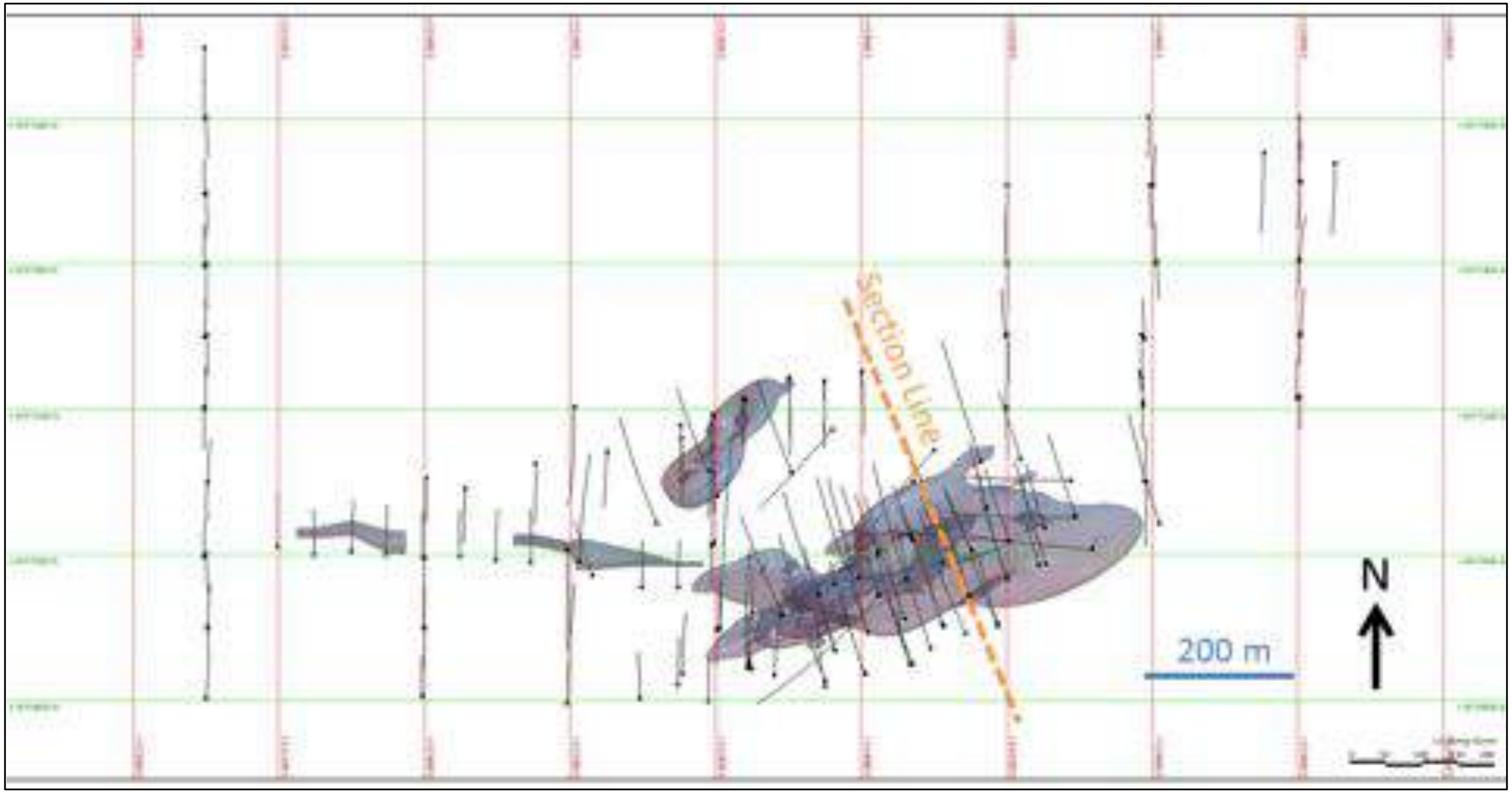

Figure 10-6: Map of Porcupine drillholes and interpretation of mineralised domain

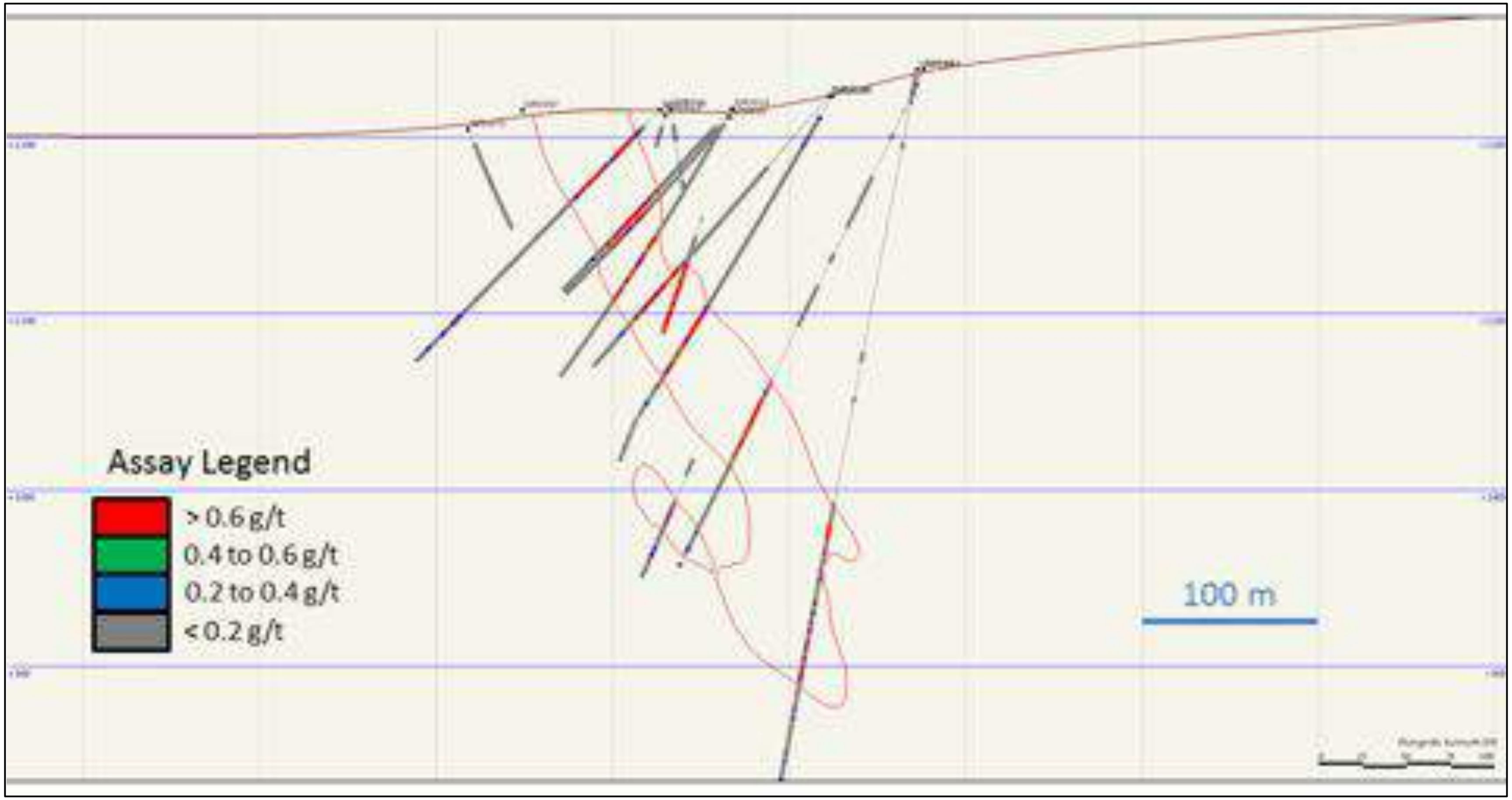

Figure 10-7: Porcupine cross section, view towards azimuth 070°

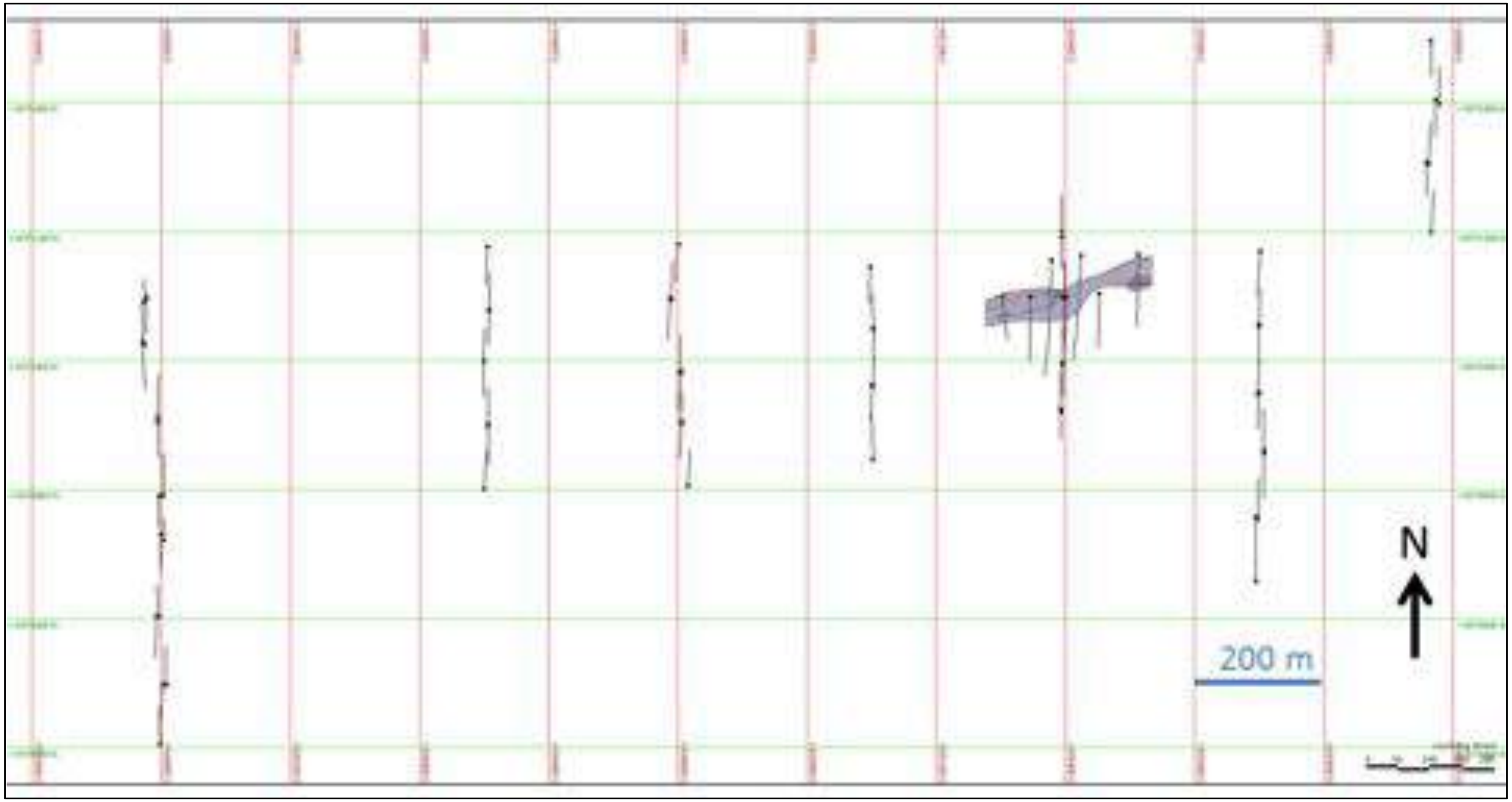

Figure 10-8: Map of Konokono drillholes and interpretation of mineralised domain

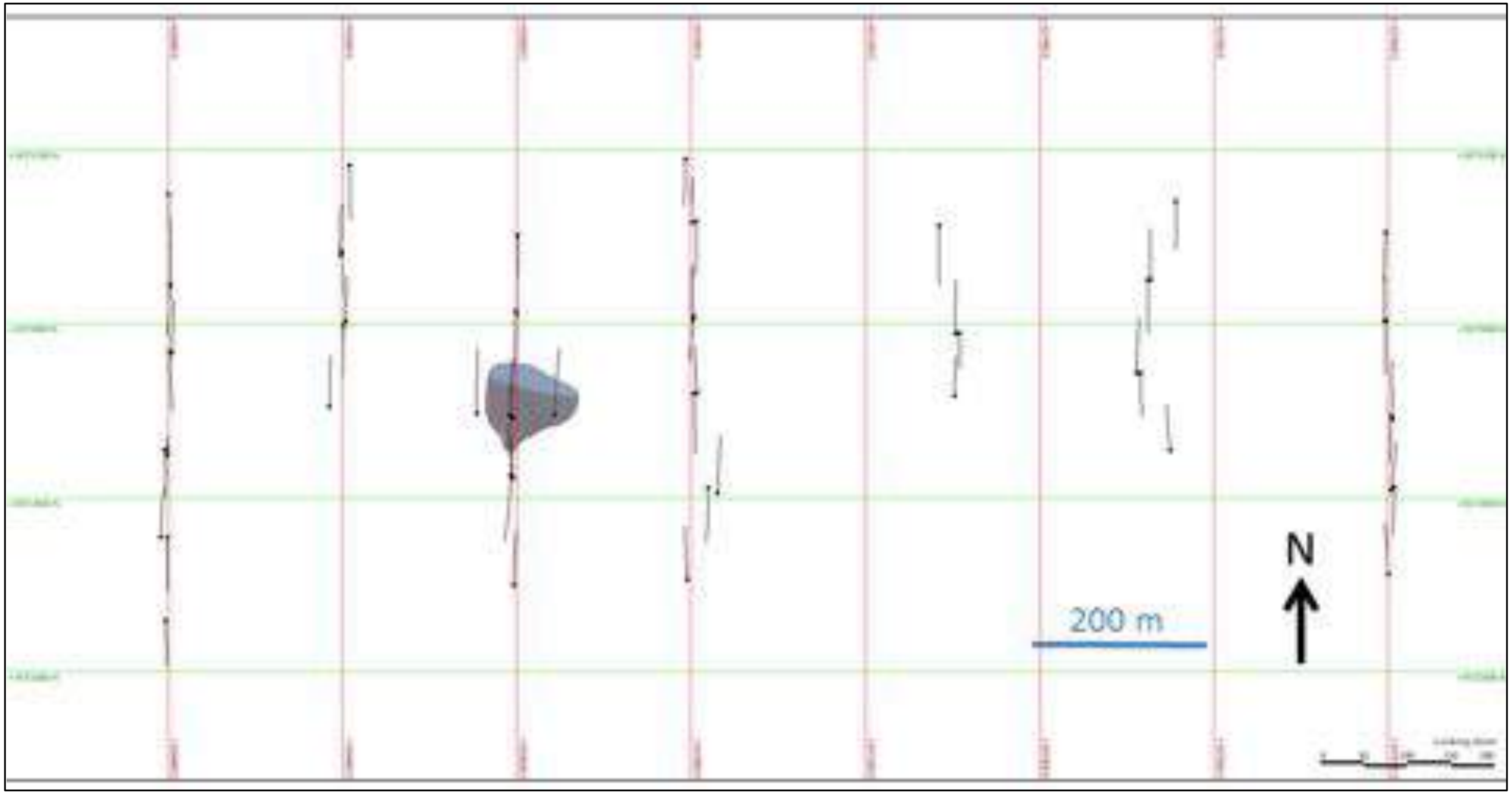

Figure 10-9: Map of Tumbili drillholes and interpretation of mineralised domain

11 Sample Preparation, Analyses and Security

11.1 General

Prior to arrival at the work site, all Helio employees involved in taking any type of samples for laboratory analysis are required to remove any metal rings and bracelets or any item of clothing or jewellery which has potential to bias analysis.

Samples submitted to any laboratories are given sample numbers along with instructions for preparation and analysis. No information is transmitted which would allow a laboratory to geographically locate the sample, or to aid in the identification of duplicate, Certified Reference Material (CRM) or blank samples.

CRM is obtained from Geostats Pty Ltd of Perth, Australia. Blank material is created by Helio using stored reference RC material which is known to have null values and not be proximal to areas of known mineralisation. Numerous null-grade reference samples are homogenised and multiple random samples of the resulting blank are sent for analysis to confirm the Au content is zero before the blank is introduced into general usage.

11.2 Soil Sampling

Soil geochemistry has been used as a regional and targeted exploration tool within the SMP. Regional soil sampling is conducted on a 250 m x 250 m offset grid. Where large areas have returned good soil geochemistry results, detailed soil sampling grids have been carried out with associated mapping and rock sampling. Detailed soil grids have been conducted at 25 m x 100 m, 50 m x 100 m and 25 m x 200 m offset grids.

The entire SMP has been covered by regional soil sampling, a total of 4,004 samples were collected. 6,619 soil samples have been collected (Table 9-1 and Table 9-2) on detailed soil grids over the Panya, Dubwana and Tumbili targets, as well as over the entire area of the Saza East PL and its surrounds. (Note: these figures include QA/QC samples).

Samples are collected on a pre-determined grid, sample sites are moved only if the site was in a stream or river bed, or if it was directly on top of outcropping rock. Should the sample site not be suitable the closest suitable site is identified and sampled and the new coordinates are noted. Prior to the commencement of the sampling programme duplicate and CRM samples are added alternately every 25 samples to the sequence to assist in Helio's QA/QC regime.

Each soil sample was collected from a hand-dug pit to the soil-rock interface or, where this was not reached, a depth of 50 cm. Soil from the base of the pit is sieved and approximately 100-150 g of the -250 µm fraction is retained for analysis. Should a sample be damp when excavated, a 3 kg bulk sample is collected this material is then dried and sieved to collect the sample. The -250 µm fraction is placed in a wire sealable 'kraft' sample packet, which in turn is enclosed in a plastic zip-lock bag and boxed for shipment.

Helio submits soil samples to Acme Laboratories in Vancouver, Canada for 36 element aqua regia digestion ICP-MS analysis, Group 1DX, which has a lower limit of detection of 0.5 ppb for Au.

11.3 Rock Sampling

Rock sampling is conducted in one of two ways: Channel sampling and grab sampling. Of these two methods, grab sampling has been used for the vast majority of rock sampling conducted to date on the SMP.

A channel sample is conducted by collecting a continuous set of rock chips across a specified length of outcropping rock. This is a difficult operation to achieve without introducing a bias to the results without the use of a motorised channel sampler. For this reason channel sampling is not frequently conducted by Helio.

Grab samples are generally a collection of rocks from an outcrop, but in regional work grabs may be composites of rocks occurring within 10-20m of a sample locality.

Samples are double bagged securely in durable plastic bags labelled inside and out with the sample number. A sample tag is also included in the bag, which is secured using cable ties.

Samples are submitted for fire assay with AAS finish to African Assay Laboratories (Tanzania) Limited, Mwanza, Tanzania which is accredited to ISO/IEC 17025 and is part of the SGS Group.

11.4 Reverse Circulation Sampling

When an RC hole is drilled the entire length of the hole is sampled. Whilst planning the sampling sequence the geologist will use a pre-printed form which has designated samples which will be CRM, blank material or duplicate samples. In a 54 sample sequence, two CRM, two duplicate and one blank sample will be inserted. As well as the pre-printed sample sheet a waterproof 3-tag sample book is used; one tag is inserted inside the sample bag that is sent to the laboratory, one inside the reference sample bag and the third tag remains in the book as a record of the sample sequence.

Material is collected from the cyclone in a plastic-lined polyweave ‘rice sack’ which has metre numbers marked on it. The rice sack is secured to the cyclone using a length of rubber bungee cord. The driller indicates to the Helio employee manning the cyclone when a metre has been completed and the Helio employee then removes the rice sack from the cyclone. The driller will then lift the drill string from the base of the hole and blow the hole out to reduce any downhole smearing caused by residual heavy minerals in the hole. Once the hole is cleared the bag for the next interval is attached and drilling recommences.

The cyclone is cleaned between holes and whenever an obvious build-up of material is observed, especially if the sample is damp or wet.

The rice sack containing the recovered material is weighed and then is moved away from the rig to the sampling and logging area, which is located upwind of the machine. Recovered material usually weighs 30 to 40 kg depending on the diameter of the drill bit. Weighing the recovered material not only gives an indication of the density of the material recovered, but also serves as a check to ensure the driller is measuring metres drilled accurately and correct sample return is being produced.

Recovered material is homogenised and reduced to a usable quantity by being processed in a 3-tier riffle splitter (Figure 11-1). The one-eighth split is retained, the other seven-eighths are returned to the rice sack which is then removed from the sample area and put in consecutive order with the material drilled from the previous metre sampled.

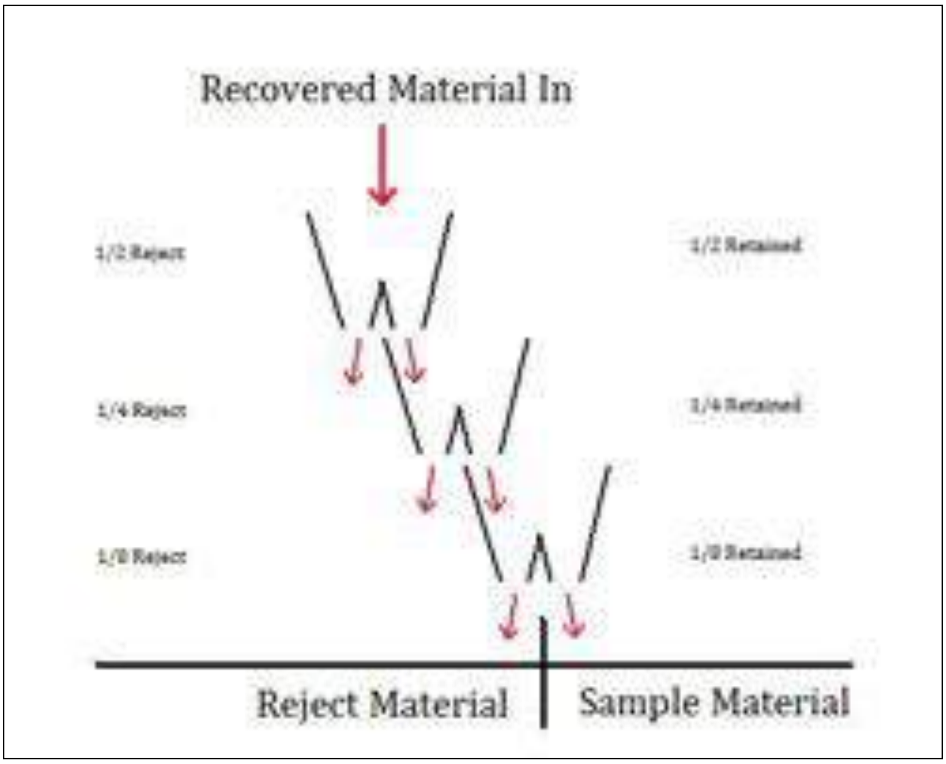

Figure 11-1: Riffle Splitting

Prior to the drilling of the hole the geologist in charge of drilling will fill out a sample sheet and sample tag book for the hole. Each sample is a composite of two consecutive metres, therefore each sample will have two sample bags prepared, one of which is marked as a reference sample. Once the drilled material for the first metre of the sample has been split the retained material is placed in the first sample bag. When the second metre of the sample has been split and placed in the reference sample bag both sets of material are passed through a single split, thus homogenising the two separate metres into one single sample. The two splits of this 2 m composite sample are then returned to the two sample bags, the sample which is being submitted to the laboratory is always taken from the same side of the splitter. Sample tickets are added and the bags securely sealed. Once 10 samples (20 m of drilling) have been collected they are placed inside marked rice sacks. The samples are sent for analysis in the next sample shipment, the material marked as reference is stored by Helio as insurance against accident or loss of the original sample, or for further analytical work in the future.

Occasionally, water is encountered in RC holes. If water ingress is minimal, then the hole can continue, but if significant water is encountered, the hole is abandoned to reduce the potential for downhole contamination. Where material is recovered wet, pipe sampling is used so as not to contaminate the riffle splitter. After mixing and homogenising the material comprising the sample, the rice sack containing the sample is laid flat on the floor and its contents are evened out. A PVC pipe with an internal diameter of 45 mm is inserted into the bag as shown in Figure 11-2 to obtain a sample of the metre drilled.

Figure 11-2: Pipe sampling procedure

Once a hole is completed and the geologist on site is satisfied that all sampling has been completed, the remaining drill material is disposed of. Reference samples are retained and stockpiled for future reference.

In total 26,117 samples have been generated from RC drilling. This figure includes CRM, blank and duplicate samples, as well as samples which have been resubmitted to laboratories for umpiring. Samples from RC drill holes are submitted for fire assay with AAS finish to African Assay Laboratories (Tanzania) Limited, Mwanza, Tanzania which is part of the SGS Group. Screen fire assay and ICP work is conducted on selected samples at Acme Laboratories in Vancouver, Canada and Genalysis Laboratory Services, Perth, Australia respectively.

11.5 Diamond Core Sampling

Once DD core has been returned to the processing site it is cleaned, marked, photographed and has geotechnical and lithological logging conducted on it. When planning the sampling sequence the geologist will use a pre-printed form which has designated samples which will be CRM, blank material or duplicate samples. In a 54 sample sequence two CRM, two duplicate and one blank sample will be inserted. As well as the pre-printed sample sheet a waterproof 3-tag sample book is used; one tag is stuck inside the core tray to indicate the sample specifications, one is inserted inside the sample bag that is sent to the laboratory and the third tag remains in the book as a record of the sample sequence.

Sampling of core is conducted using the following conventions:

1 Mineralised zones are sampled continuously, with the sampling also extending some way into the hanging wall and footwall. The mineralised zones can be identified by the presence of alteration (hematite, sericite or chlorite), veining, structural deformation and sulphidisation (disseminated pyrite).

2 Areas believed to be barren are sampled at least once per box (seven metres or so). Barren zones are usually unaltered with no pyrite.

3 Samples are taken from the metres (or half metres) mark cut. Sampling is not conducted according to lithology. Sample intervals are cut perpendicular to the axis of the core.

4 HQ core is sampled at 2 m, 1 m or 0.5 m intervals and NQ core is sampled at 2 m or 1 m intervals. In general, zones identified as mineralised are sampled at shorter intervals.

5 When a sample cuts across change in core size the length of each type of core in the sample is recorded.

6 Areas of extreme core loss where there is insufficient sample to submit to the laboratory are composited into the nearest appropriate sample.

7 A sample is always taken at the end of the hole.

8 Holes drilled on new targets are sampled in their entirety until written instruction from Chief Operations Officer specifies otherwise.

Point 3 was adopted in 2008 core sampled prior to this was sampled according to lithology.

On average, core recovery from the SMP is approximately 95%.

Once the geologist has identified the areas to be sampled, sample tags are inserted into the core box at the start of each sample and secured in place with a sticker which has the interval marked on it. The core itself is marked with the sample number and the half which is to be submitted to the laboratory is clearly marked using a grease pencil. Core is split using CorStore core splitters and Almonte automated core cutting machines. After cutting through any interval with visible gold, a cleaning block will be cut afterwards, to avoid any potential contamination of the following piece of core. The core is split along the bottom-of-hole line when the core is orientated or along the arbitrary centre of core line where it is not. Once split the core is returned to the core tray and once all core that requires cutting in the tray is split, the core tray is removed from the cutting room and on to the sampling benches.

The same half of the core is submitted to the laboratory for all samples. Core is double bagged in durable plastic sample bags along with a sample ticket. Prior to packaging for transportation, the samples are photographed for reference.

In total 39,006 samples have been generated with diamond drilling. This figure includes CRM, blank and duplicate samples, as well as samples which have been resubmitted to laboratories for umpiring. Samples from diamond drill holes are submitted to African Assay Laboratories (Tanzania) Limited, Mwanza, Tanzania which is part of the SGS Group for fire assay with AAS finish. Screen fire assay and ICP work is conducted on selected samples at Acme Laboratories in Vancouver, Canada and Genalysis Laboratory Services, Perth, Australia respectively.

11.6 Sample Storage and Dispatch

The collection and processing of all samples prior to dispatch to laboratory is conducted by Helio employees. Duplicates, CRM and blank material are inserted in to all sample sequences before dispatch to laboratory. All sampling is divided into batches; one batch is an entire drillhole, collection of related rock samples or collection of related soil samples. Samples are submitted using a standardized laboratory submission form which lists the sample numbers, type of material and analysis required and batch number.

After collection the samples are stockpiled in a designated covered area within the core processing area which is a fenced and gated area inside the secure Helio office compound in Mkwajuni. The compound is patrolled 24 hours by guards from the Security Group Tanzania Limited. Access to the core processing area is restricted to Helio employees.

All drilling samples are are transported from site to Africa Assay Laboratories ("AAL") in Mwanza in a secure truck provided by Kanji Lalji Limited. Individual sample bags are double bagged inside polyweave 'rice sacks' and a photograph of each hole is taken as a record of what is dispatched. Samples arriving at AAL are checked in to the laboratory against the laboratory submission form provided both electronically to the laboratory and by hard copy which accompanies the samples. AAL provides Helio with sample reconciliation data which lists samples received, as well as additional or missing samples if such situations arise.

Samples which are sent to laboratories outside of Tanzania must be examined by the Madini (Ministry of Energy and Minerals of the United Republic of Tanzania) and cleared for exportation. Upon clearance the samples are securely packaged and secured inside the transportation container with an official wax seal, the samples are then dispatched to their destination via courier.

11.7 Laboratory Procedures

11.7.1 African Assay Laboratories (AAL)

AAL are based in Mwanza, northern Tanzania and are part of the SGS Group. AAL's Mwanza is accredited to ISO/IEC 17025 standard by the South African National Accreditation System (SANAS). The laboratory also participates in numerous formal proficiency testing and round robin reference material certification programmes. AAL applies internal quality control procedures by inserting Certified Reference Materials and duplicates into submitted sample sequences. The results from the duplicates are included in the assays reported to Helio.

Samples are weighed on receipt, recorded and reported. RC and DD material is dried in trays, crushed to a nominal 2 mm using a jaw crusher and cone crusher, then approximately 1 kg is split using a Jones type riffle splitter. Rejected material is retained in the original bag. The split is pulverised in a chrome steel bowl to a nominal 75 µm. A 50 g sub-sample is taken for assay, with the pulverised residue retained in a plastic bag.

The 50 g sub-sample is fused with a litharge based flux in a ceramic crucible, the resulting glass bead is dissolved in aqua regia and the quantity of gold in the sample is determined by flame AAS. The detection limits of this analysis are 0.01 ppm to 100 ppm.

Rejected course and pulped material is returned to the Helio's Mkwajuni office compound in returning sample trucks. It is catalogued and stored for later resampling. Once all work has been completed on the samples and at least six months has elapsed, permission is sought from Helio's Chief Operations Officer to dispose of unwanted material. Samples that are part of mineralised zones are retained.

11.7.2 Acme Laboratories

Acme Laboratories Vancouver attained ISO 90001 accreditation in 1996 and has maintained its registration in good standing since then. Work is ongoing to attain ISO 17025:2005 accreditation.

Helio uses Acme for ICP-MS analysis of soils and pulp materials.

Soils are dried at 60˚C to minimise the loss of volatile element and are screened to -180 µm. Preparation of soils is conducted in a specific part of the laboratory which is exclusive to soils, till and sediment. Aqua regia is used to digest a 30 g analyte of the sample and ICP-MS is used to determine the values for 36 elements.

Acme applies internal quality control procedures by inserting Certified Reference Materials and duplicates into submitted sample sequences.

11.8 Quality Assurance and Quality Control

For drilling programmes, Helio inserts standards at a planned rate of 1 in 25 samples, blanks at a planned rate of 1 in 50 and duplicates at a planned rate of 1 in 25. The actual rates of insertion are close to these targets. RC duplicates are collected as a secondary split at the drill rig as the sample is passed through the riffle splitter. Diamond core duplicates were collected after the initial crushing stage at AAL Mwanza, as directed by Helio. These duplicates are therefore not submitted blind, contrary to generally accepted methods for the submission of QA/QC samples, but this compromise was deemed necessary as Helio do not have their own sample preparation equipment and are reluctant to sacrifice the other half of the core.

For soil sampling, standards (including blanks) are inserted at the rate of between 1 in 25 to 50 samples and duplicates at a rate of between 1 in 40 to 70 samples.

The results of SRK's analysis of the QA/QC samples are discussed in Item 12.3.

11.9 SRK Comments

In the opinion of SRK the sampling preparation, security and analytical procedures used by Helio are consistent with generally accepted industry best practices and are therefore adequate.

12 Data Verification

12.1 Site Visit

Robin Simpson from SRK visited the SMP site from the 3 to 4 of September, 2011. No drilling rigs were active during this visit as the diamond rigs had recently finished their campaign and departed and the RC rig was on site but awaiting repairs.

SRK inspected Helio's drillcore. A selection of complete holes and intersections, some requested by SRK and others suggested by Helio, were set out in Helio's compound. SRK compared the core against the logging and assay records. SRK was satisfied that the information recorded in the geological logs was a good representation of the core and that the assay results were consistent with observable zones of alteration and mineralisation.

SRK also walked over several of the targets within the SMP, including Kenge-Mbenge and Porcupine. During these tours SRK examined old drilling sites and collars and verified the locations of these holes against maps of the drilling. Apart from the collars, there is little to inspect on the surface at Porcupine. The Kenge Shear Zone though coincides with a prominent ridge and SRK viewed old workings along this ridge which have exposed zones of shearing, veining and alteration.

SRK had access to the filing system in Helio's compound and extracted several of the original logging sheets and assay certificates, for the purposes of comparing these records against the spreadsheets that made up the copy of the drillhole database given to SRK. No inconsistencies were found. A similar but much more extensive check of the assay certificates has been done before by Golder as part of their site visit in preparation for the previous Mineral Resource estimate (Harrison, 2011). Golder checked 29,422 assays (100% of the database at that time). There were a few hundred minor data entry errors where certificate values of <0.01 had been entered into the database as 0.01 or 0.05; these were replaced with 0.005. Otherwise the assay data were found to be quite clean.

Following the visit to the SMP, Robin Simpson from SRK, accompanied by Mike Ashley from Helio, travelled to the SGS assay laboratory in Mwanza. During this visit SRK was able to observe all stages of sample preparation and analysis and discuss the processing of Helio's samples with laboratory personnel.

12.2 Database Checks

Helio delivered the drilling data to SRK as spreadsheets. SRK imported the data from the spreadsheets into Leapfrog™, Gemcom Surpac™ and Isatis™ software for statistical analysis and 3D visualisation. During the importing process, these programmes carried out a number of validation checks, such as testing for duplicate intervals, overlapping intervals and inconsistent naming of drillholes between different tables.

Visualising the holes in 3D software such as Leapfrog™ and Gemcom Surpac™ was also an important validation tool, to check for such things as collars that plot well above (or below) the topography and holes with improbably abrupt changes of dip or azimuth.

As SRK proceeded with data processing and analysis, some minor additional errors were found, such as non-numeric and negative values in the assay table and inconsistencies in the way certified reference materials were named.

Where database problems were identified, SRK notified Helio and corrected spreadsheets were issued. SRK is satisfied that the final version of the database, issued by Helio on 20 December 2011 and used for preparing the Mineral Resource estimate, contains no critical errors.

12.3 Data from Analytical Quality Control Samples

SRK reviewed the QA/QC results stored in Helio's database: assays done on Certified Reference Material, blanks and field duplicates.

12.3.1 Certified Reference Material

Helio inserts standards at a planned rate of 1 in 25 samples. The standards Helio uses are sourced from Geostats Pty Ltd in Perth (Australia) and have certified values ranging from 0.24 g/t to 48.53 g/t gold. During every sampling campaign, several different standards have been active at once, so the assay table in Helio's database contains results from 13 different standards with at least five analyses each.

The results from the standards are summarised in Table 12-1; charts for the individual standards are in Appendix C. In total, the 13 main standards have 2,210 analyses (compared to 58,598 primary assays in the database).

Table 12-1: Analyses of SMP standards

Standard	Number of analyses	Mean of analyses	Certified Value	Upper Control: Certified Value plus 2 × Certified Std Dev	Lower Control: Certified Value minus 2 × Certified Std Dev	Number of analyses > Upper Control	Number of analyses <Lower Control	Percent of analyses outside controls
G302-2	371	2.54	2.50	2.78	2.22	15	8	6.2%
G303-8	81	0.26	0.26	0.32	0.20	4	2	7.4%
G306-1	318	0.81	0.41	0.47	0.35	15	11	8.2%
G306-4	318	20.48	21.57	23.13	20.01	4	26	9.4%
G307-3	9	0.24	0.24	0.28	0.20	0	0	0.0%
G310-10	9	50.29	48.53	51.87	45.19	0	0	0.0%
G310-4	6	0.44	0.43	0.49	0.37	1	0	16.7%
G399-2	58	1.46	1.46	1.64	1.28	2	2	6.9%
G901-7	295	1.48	1.52	1.64	1.40	2	11	4.4%
G901-9	40	0.68	0.69	0.77	0.61	1	3	10.0%
G902-1	285	0.56	0.39	0.47	0.31	0	1	0.4%
G998-6	71	0.83	0.80	0.92	0.68	5	3	11.3%
G999-4	349	2.96	3.02	3.36	2.68	5	9	4.0%

Of the 2,210 analyses of standards, about 40 (approximately 2%) plot as obvious outliers. Almost always the outlier values correspond to the grade of a different standard – strong evidence that most of the outliers are occurring due to mislabelling between standards. Outliers that are probably due to mislabelling of standard G306-4 are particularly prominent. This standard has a certified value of 21.57 g/t: outliers close to this value show up in the charts for G902-1, G306-1 and G302-2.

If the analyses conform to a normal distribution with the certified mean and standard deviation, then about 95% of the analyses should be expected to be within two standard deviations of the mean. For each standard, the column on the far right of Table 12-1 gives the percentage of analyses that fall outside two certified standard deviations from the mean. If the assumed mislabelled standards were removed from the dataset, then the variability of the remaining data would be a good match to the expected precision (given the standard deviations quoted in the certifications).

For most standards, the analyses are spread reasonably evenly both above and below the certified mean grade, so give no cause for concern about significant bias. The most asymmetric spreads of data occur for the two highest grade standards. The analyses of G310-10 appear to be centred around 50.0 g/t (compared to the certified value of 48.53 g/t) and the analyses of G306-4 appear to be centred around 21.0 g/t (compared to the certified value of 21.57 g/t). These deviations may be occurring because the assay process at Mwanza is calibrated for lower grades. Uncertainties of this magnitude, occurring at the very high grade ranges, are unlikely to have a material effect on the selection of material above any reasonable choice of cut-off grade for the SMP.

12.3.2 Blanks

Helio inserts blank samples at a planned rate of 1 in 50. Blanks samples, with an expected grade of zero, are submitted to ensure that there is no contamination between samples during the sample preparation and analysis. There are 1,337 analyses of blank samples recorded in Helio's database (compared to 58,598 primary assays). The results from the blanks are presented in a chart in Appendix A.

Of the blank analyses, 21 (1.6%) returned a grade of at least 0.1 g/t, 11 (0.8%) returned grades above 0.5 g/t and 4 (0.3%) returned grades greater than 1 g/t. SRK strongly suspects that many of these anomalously high assays are due to sample swaps between blanks and standards and regular assays, rather than actual contamination. The highest grade returned from a blank analysis (21.0 g/t from sample 03431) is a good match to the certified value of standard G306-4 (21.57 g/t).

SRK's conclusion is that the occurrences of mislabelling or contamination revealed by the blank samples are infrequent enough to not have a material effect on the Mineral Resource estimate.

12.3.3 Duplicates

Helio inserted duplicates at a planned rate of 1 in 25 samples. There are 2,392 duplicate assays in Helio's database. The results from analysis of duplicates will be a function of the sampling methods and mineralisation style, therefore SRK filtered and grouped the duplicate data by deposit and drilling type. Summary statistics are given in Table 12-2 to Table 12-5. Scatter plots and Half Absolute Relative Difference (HARD) plots for the duplicates are in Appendix C.

SRK's conclusion, after reviewing the statistics and the plots from the duplicates, is that there is an acceptable level of precision and repeatability at the stage of the sampling process at which the duplicates were taken.

Table 12-2: Summary statistics for DD duplicates from Kenge and Mbenge

	Original	Duplicate
Count	479	479
Mean	0.62	0.61
Std Dev	2.52	2.40
Minimum	0.005	0.005
Maximum	40.4	38.9
Coefficient of Linear Correlation:	0.987	

Table 12-3: Summary statistics for RC duplicates from Kenge and Mbenge

	Original	Duplicate
Count	155	155
Mean	0.12	0.12
Std Dev	0.41	0.40
Minimum	0.005	0.005
Maximum	3.23	3.15
Coefficient of Linear Correlation:	0.948	

Table 12-4: Summary statistics for DD duplicates from Porcupine, Konokono and Tumbili

	Original	Duplicate
Count	681	681
Mean	0.40	0.39
Std Dev	1.71	1.69
Minimum	0.005	0.005
Maximum	26.3	27.2
Coefficient of Linear Correlation:	0.965	

Table 12-5: Summary statistics for RC duplicates from Porcupine, Konokono and Tumbili

	Original	Duplicate
Count	372	372
Mean	0.06	0.06
Std Dev	0.20	0.21
Minimum	0.005	0.005
Maximum	2.03	2.09
Coefficient of Linear Correlation:	0.870	

12.4 SRK Comments

After carrying out the verification measures described above, the qualified person is confident that the database is suitable to be used for Mineral Resource estimation.

13 Mineral Processing and Metallurgical Testing

A program of preliminary metallurgical testwork was conducted on behalf of Helio by SGS Lakefield Research Limited ("SGS") in Ontario, Canada to determine the processing characteristics of the Porcupine and Kenge mineralised material and to develop a preliminary process flowsheet. Results from the Kenge study were published in August 2008 and followed by results from Porcupine in August 2009. The tests included head grade analysis, mineralogical evaluation, comminution testwork, gravity separation, flotation, cyanidation (of whole ore, gravity tailing and flotation concentrate) and preliminary environmental testing. Both testwork programmes indicated amenability to conventional gravity and cyanidation gold recovery techniques. A follow-up cursory heap leach amenability study was conducted in May 2009 by SGS on the Kenge mineralised material. Full SGS reports on the testwork can be found in Appendices C, D and E.

Summary results of this testwork are reported below.

13.1 Kenge Optimum Circuit Responses (% Au Recoveries) (see Appendix E)

- 95.6% by Gravity Separation + Gravity Tailing Flotation;
- 95.6% by Whole Ore Flotation;
- 94.5% by Gravity Separation + Gravity Tailing Cyanidation;
- 93.3% by Gravity Separation + Flotation Concentrate Cyanidation;
- 92.5% by Whole Ore Cyanidation;
- 34.7% by Gravity Separation;
- Bond ball mill work index of 15 (metric) -- "intermediate hardness";
- No preg-robbing activity detected;
- Low cyanide consumption; and
- Tailings should be non-acid generating and free from environmentally deleterious elements.

13.2 Porcupine Optimum Circuit Responses (% Au Recoveries) (see Appendix E)

- 94.8% by Whole Ore Flotation;
- 93.4% by Gravity Separation + Gravity Tailing Flotation;
- 91.9% by Gravity Separation + Flotation Concentrate Cyanidation;
- 89.1% by Gravity Separation + Gravity Tailing Cyanidation;
- 88.9% by Whole Ore Cyanidation;
- 22.0% by Gravity Separation;
- Bond ball mill work index of 15.7 (metric) - "moderately hard";
- No preg-robbing activity detected;
- Low cyanide consumption; and
- Tailings should be non-acid generating and free from environmentally deleterious elements.

13.3 Metallurgical Sample Selection

A 50 kg test sample was composited using coarse reject material from mineralised drillcore from the Kenge target. Material for the test sample was sourced from three diamond drillholes (SZD011 from Kenge SE Zone and SZD013 and SZD021 from Kenge Main Zone). Helio composited the test sample to have a weighted average head grade of 3.05 g/t Au on the basis of previously reported assaying. Screened metallics tests by SGS indicated an average head grade for the test sample of 3.6 g/t Au.

A 50 kg composite test sample taken from the counterpart half-core from diamond drillhole GPD4 was used in the porcupine testwork. The intercept chosen assayed 3.3 g/t Au over 49.63 m from 52.76 m, including 0.6 m at 33.2 g/t Au from 66.1 m and 1.8 m at 39.1 g/t Au from 69.4 m. Head grade analysis of the bulk sample from GPD4 conducted by SGS indicated that the sample graded 2.4 g/t Au.

Samples were created for the flotation and cyanidation testwork from the gravity tailings of the initial gravity concentration testing. Whole ore samples were also used in the testwork.

13.4 Mineralogical Evaluation

Mineralogical evaluations of the samples by polished section and XRD (x-ray diffraction) identified that pyrite was the major sulphide present while minor amounts of chalcopyrite and galena were observed in the Kenge mineralised material and minor amounts of chalcopyrite, covellite and chalcocite were observed in the Porcupine mineralised material. The results also support internal petrological, mineralogical and analytical studies indicating that mineralised material from the Porcupine and Kenge targets has a simple mineralogy.

13.5 Mineral Processing Testwork

13.5.1 Comminution

Comminution testing using standard Bond ball mill work index tests concluded indices of 15 (metric), considered to be of “intermediate hardness” and 15.7 (metric), considered to be of “moderate hardness” for the Kenge and Porcupine mineralised material respectively. The implication of these indicated characteristics on milling energy and maintenance costs is that they will not be particularly onerous.

13.5.2 Gravity Separation

Gravity separation analysis was conducted on samples within a grind size range of 105 to133 µm (P80) for the Porcupine ore and a grind size range of 92 to 126 µm (P80) for the Kenge material.

The initial Kenge tests indicate that gold recoveries up to 34.7% can be achieved by conventional gravity separation techniques. In a similar test scenario gravity separation of the Porcupine material indicated gold recoveries up to 22.0%. The high recovery rates observed in both tests suggest that inclusion of a gravity circuit in plant design and future testwork would be an obvious step towards optimizing recovery.

13.5.3 Flotation

In both series of gravity tailing flotation and whole ore flotation tests, high recoveries were observed. The Kenge testwork demonstrated an “excellent” response to gold recovery by flotation. Processing by flotation of gravity tailings produced gold recoveries from 93.9% (92 µm -P80) to 95.6% (75 µm -P80). Whole ore flotation tests concluded gold recoveries from 93.4% (125 µm -P80) to 95.6% (100 µm -P80).

For Porcupine, gold recoveries for the combined gravity and flotation process ranged from 91.6% (133 µm -P80) to 93.4% (105 µm -P80). Similarly high recoveries were also observed during whole ore flotation testing ranging from 90.5% (256 µm -P80) to 94.8% (61 µm -P80).

In the Kenge testwork minimal grind size to gold recovery variations were noted. In the Porcupine ore a potential relationship between finer grinding and increased gold recovery was established although further testwork is required to confirm this and even so, recoveries of over 90% at grind sizes of 256 µm are particularly encouraging.

13.5.4 Cyanidation

Standard bottle roll testing was used for the cyanidation testwork. The grind sizes tested for the Kenge material ranged from 126 µm to 52 µm (P80). Gold recoveries for cyanidation of the gravity tailings ranged from 89.9% (126 µm -P80) to 94.5% (59 µm -P80). Whole ore cyanidation tests concluded gold recoveries from 86.7% (96 µm -P80) to 92.5% (58 µm -P_{80}).

The grind sizes tested for the Porcupine material ranged from 174 µm to 79 µm (P_{80}) for cyanidation of the gravity tailings however, a coarser feed was used to assess the response of whole ore samples to cyanidation, from 406 µm to 75 µm (P_{80}). Gold recoveries for cyanidation of the gravity tailings ranged from 82.0% (174 µm -P_{80}) to 89.1% (79 µm -P_{80}). Whole ore cyanidation tests concluded gold recoveries from 70.3% (406 µm -P_{80}) to 88.9% (75 µm -P_{80}).

Both the Kenge and Porcupine material showed a trend towards increased gold recovery with finer grind sizes in both whole ore and gravity tailing tests.

Carbon-In-Leach (CIL) tests on single samples of gravity tailings for both the Kenge and Porcupine material showed no increased gold recovery therefore no preg-robbing activity is expected in either ore. It is also notable that both tests concluded the material has a low cyanide consumption, in the region of 0.04 - 0.11 kg/t for Kenge and 0.11 – 0.62 kg/t for Porcupine.

Regrinding followed by cyanidation of the floatation concentrate was conducted to determine the influence of regrinding on gold recovery. For both Kenge and Porcupine significant increases in recovery were recorded.

Further testwork was undertaken in May 2009 by SGS Lakefield on material retained from the previous 2008 Kenge study to assess amenability to heap leaching. The testwork focused on bottle roll cyanidation testing of coarse material. The material used represented the coarsest material retained by SGS and ranged in size from 1.7 mm to 3.35 mm. Results from this phase of testing indicated that gold recoveries in the order of 70 % are possible, coupled with low reagent consumptions.

13.6 Environmental implications

Various acid generation tests and broad spectrum ICP testing of the ore suggest that tailings should be non-acid generating and free from environmentally deleterious elements.

13.7 Further Work Planned

Further metallurgical testwork is planned in order to optimise grinding sizes and flotation flowsheet configurations, together with studies of the potential amenability of Porcupine ore to heap leaching. Further tests to include samples representing a wider spectrum of mineralisation across the Porcupine target are also planned. This will assist in the analysis of plant design weighting in respect to the proportion of different ores introduced into the final processing circuit.

14 Mineral Resource Estimates

14.1 Introduction

This section describes the resource estimation methodology and summarizes the key assumptions considered by SRK. In the opinion of SRK, the Resource Evaluation reported herein is a reasonable representation of the global gold Mineral Resources found in the SMP at the current level of sampling. The Mineral Resources have been estimated in conformity with generally accepted CIM "Estimation of Mineral Resource and Mineral Reserves Best Practices" guidelines and are reported in accordance with the Canadian Securities Administrators' National Instrument 43-101. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resource will be converted into Mineral Reserve.

The Mineral Resource Statement presented herein represents the second Mineral Resource Evaluation prepared for the SMP in accordance with the Canadian Securities Administrators' National Instrument 43-101. The previous Mineral Resource estimate was prepared by Golder Associates (UK) Ltd and announced on November 30, 2010.

The database used to estimate Mineral Resources was verified by SRK. SRK is of the opinion that the current drilling information is sufficiently reliable to interpret with confidence the boundaries for gold mineralisation and that the assay data are sufficiently reliable to support Mineral Resource estimation.

Leapfrog™ and Gemcom Surpac™ software were used to construct the geological solids, prepare assay data for geostatistical analysis, construct the block model and tabulate Mineral Resources. Isatis™ was used for geostatistical analysis, variography and to estimate metal grades.

The Mineral Resource estimate concerns five of the identified targets within the SMP: Kenge, Mbenge, Porcupine, Konokono and Tumbili. Each of these deposits was estimated using unique parameters.

The coordinate system used for modelling was the same as the primary coordinate system stored in Helio's drillhole database: UTM Zone 36S, datum WGS 84.

14.2 Topography

The topographic surface used to constrain the geological domains and block models was constructed by SRK from the Differential GPS surveys of the drillhole collars. SRK modelled the surface using the "Topography" module in Leapfrog™, with the resolution set to 10 m (i.e. vertices in the topography wireframe were generated approximately every 10 m in the x and y dimensions). The "offset to points" function in Leapfrog™ was enabled, to ensure that vertices in the wireframe exactly coincided with the available surveys of collar points.

14.3 Weathering Domains

No weathering surfaces were modelled. The depth to which oxidisation and weathering has penetrated is usually limited to the first two metres or less. Therefore, at the scale used for modelling, weathering domains were not necessary to constrain grade estimation or control how densities were assigned.

14.4 Modelling of the Mineralised Domains

14.4.1 Kenge

The mineralised domains for Kenge were modelled primarily from geological logging. SRK compared the spatial distribution of the geology and assays and noted that gold mineralisation closely corresponds to the following lithology codes used by Helio: Vein Quartz (VNQZ), Chlorite Schist (CSHT), Chlorite Sericite Schist (CSST), Sericite Schist (SSHT). SRK used Leapfrog™ to extract points at the first and last occurrences of any of these four codes downhole. Some minor manual adjustments to the points were made in Gemcom Surpac™, based on the assays.

The points were imported back into Leapfrog and the automated wireframing functions in Leapfrog were used to generate two approximately parallel surfaces that become the hanging wall and footwall contacts of the Kenge lode. Perimeters to constrain the extrapolation of the lode along strike and down dip were digitized manually in Gemcom Surpac™. The final lode shape was defined by the intersection of the hanging wall and footwall surfaces with these perimeters and the topography surface.

In detail, the Kenge mineralisation domain is made up of three separate shapes: Kenge Hanging Wall, Kenge Footwall and Kenge Southeast (Figure 14-1). The geometry of the Hanging Wall and Footwall domains suggests that these were once a single unit, which has since been cut by a reverse fault dipping shallowly to the north. All three domains strike approximately 120° and dip approximately 60 to 70° southwest.

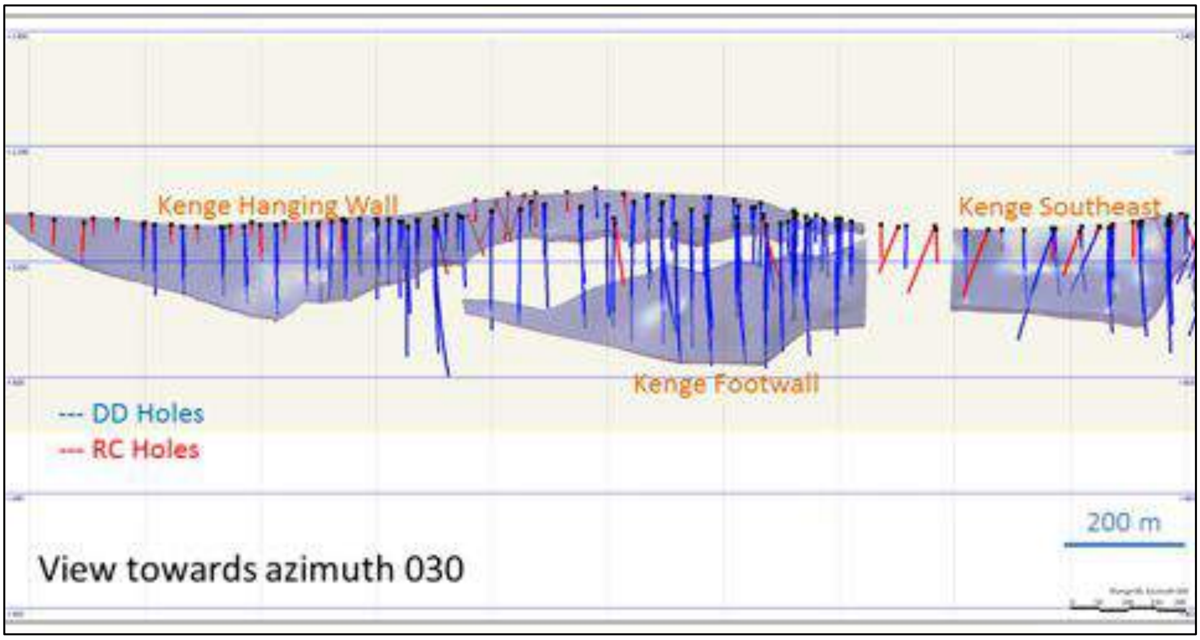

Figure 14-1: Long-section view of Kenge mineralised domains

14.4.2 Mbenge

The Mbenge target is on the east side and adjacent to Kenge. For the statement of SMP Mineral Resources, Helio asked SRK to group Kenge and Mbenge together as one deposit.

SRK modelled Mbenge in Leapfrog™ and defined the limits of mineralisation by 3D contouring of grade shells. The assays were composited to 2 m, the threshold for contouring was set at 0.3 g/t and a trend was applied so that the contouring followed the overall orientation of mineralisation continuity. For the main part of Mbenge, this trend was set by a plane dipping 77° towards 180 and for the southern part of Mbenge the trend was set by a plane dipping 66° towards 340.

After cutting with the topography and filtering out small shapes based on single-hole intersections, the final domain model for Mbenge has six domains (Figure 14-2): three shapes in the main (steeply south-dipping) part of Mbenge and three shapes in the southern (steeply NNE-dipping) part.

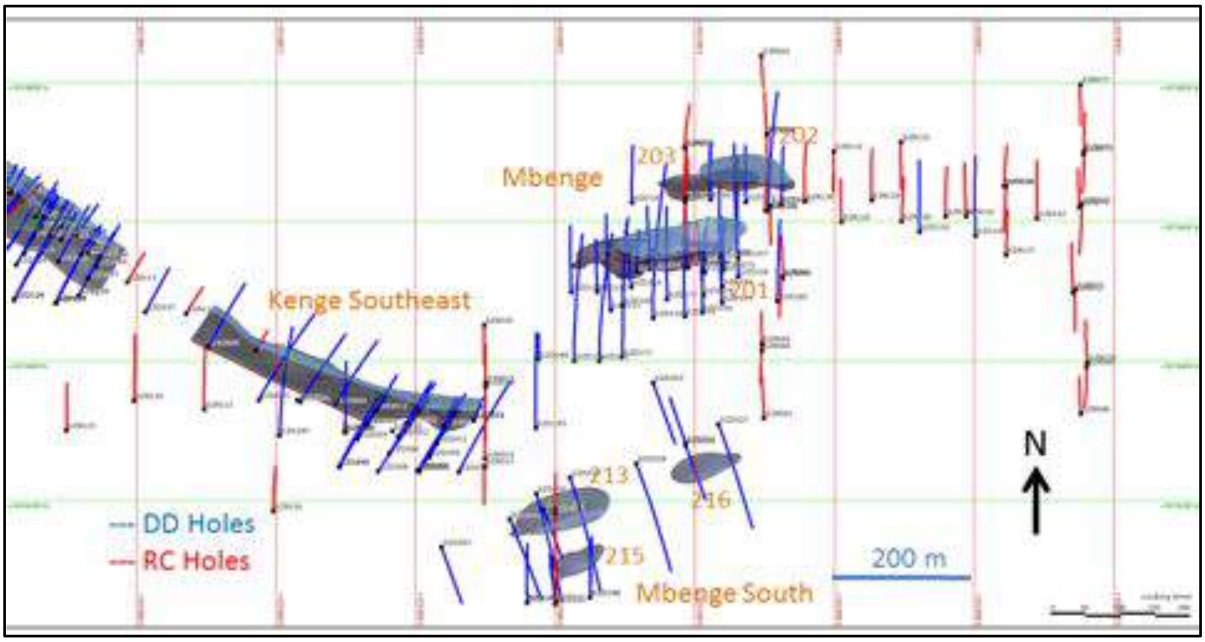

Figure 14-2: Plan view of Mbenge mineralised domains

14.4.3 Porcupine

Three domains of mineralisation have been identified at Porcupine (Figure 14-3), each with different orientations of mineralisation continuity. The main Porcupine domain dips steeply to the SSE and plunges shallowly east. Quill dips steeply NW, with no obvious plunge. Porcupine Northwest is made up of two shapes, which strike E-W and are subvertical.

SRK modelled Porcupine Main and Quill in Leapfrog™ and defined the limits of mineralisation by 3D contouring of grade shells. The assays were composited to 2 m, the threshold for contouring was set at 0.3 g/t and a trend was applied so that the contouring followed the overall orientation of mineralisation continuity. For Porcupine Main, this trend was set by a plane dipping 63° towards 148, with a pitch angle of 30° from the east. For Quill, the trend was set by a plane dipping 70° towards 315.

The Porcupine Northwest shapes were constructed in Gemcom Surpac™ by manually digitising mineralisation perimeters on sections and then triangulating these perimeter strings into solids models. A 0.3 g/t threshold was used for defining mineralisation.

All the Porcupine shapes were cut with the topography to create the final domain model. For Porcupine Main and Quill, spurious small shapes generated by the contouring process were filtered out, leaving only a single large shape for each of these domains.

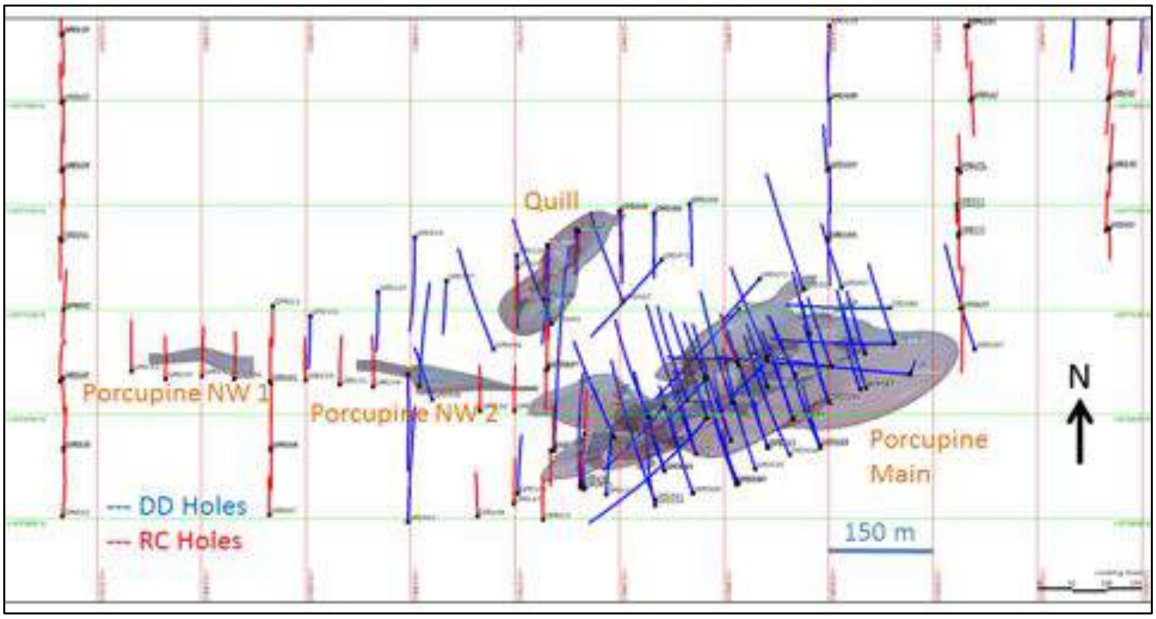

Figure 14-3: Plan view of Porcupine mineralised domains

14.4.4 Konokono

SRK modelled Konokono in Leapfrog™ and defined the limits of mineralisation by 3D contouring of a grade shell. The assays were composited to 2 m, the threshold for contouring was set at 0.3 g/t and a trend was applied so that the contouring followed the overall orientation of mineralisation continuity. This trend was set by a plane dipping 60° towards 345 and with a horizontal plunge.

Spurious small shapes generated by the contouring process were filtered out, leaving only a single large shape for the Konokono domain. This shape was cut with the topography and trimmed to limit extrapolation along strike to no further than 20 m beyond the outer drilling lines. Figure 14-4 shows the final domain model.

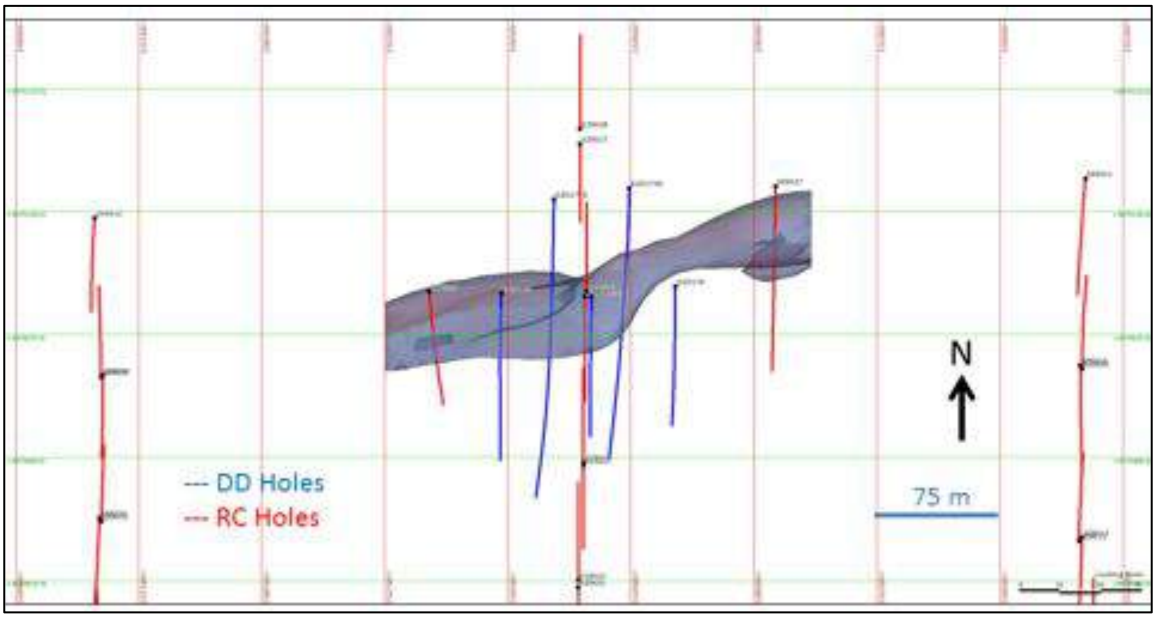

Figure 14-4: Plan view of Konokono mineralised domain

14.4.5 Tumbili

SRK modelled Tumbili in Leapfrog™ and defined the limits of mineralisation by 3D contouring of a grade shell. The assays were composited to 2 m, the threshold for contouring was set at 0.3 g/t and a trend was applied so that the contouring followed the overall orientation of mineralisation continuity. This trend was set by a plane dipping 55° towards 190 and with a vertical plunge.

Spurious small shapes generated by the contouring process were filtered out, leaving only a single large shape for the Tumbili domain. The final domain was built by cutting this shape with the topography.

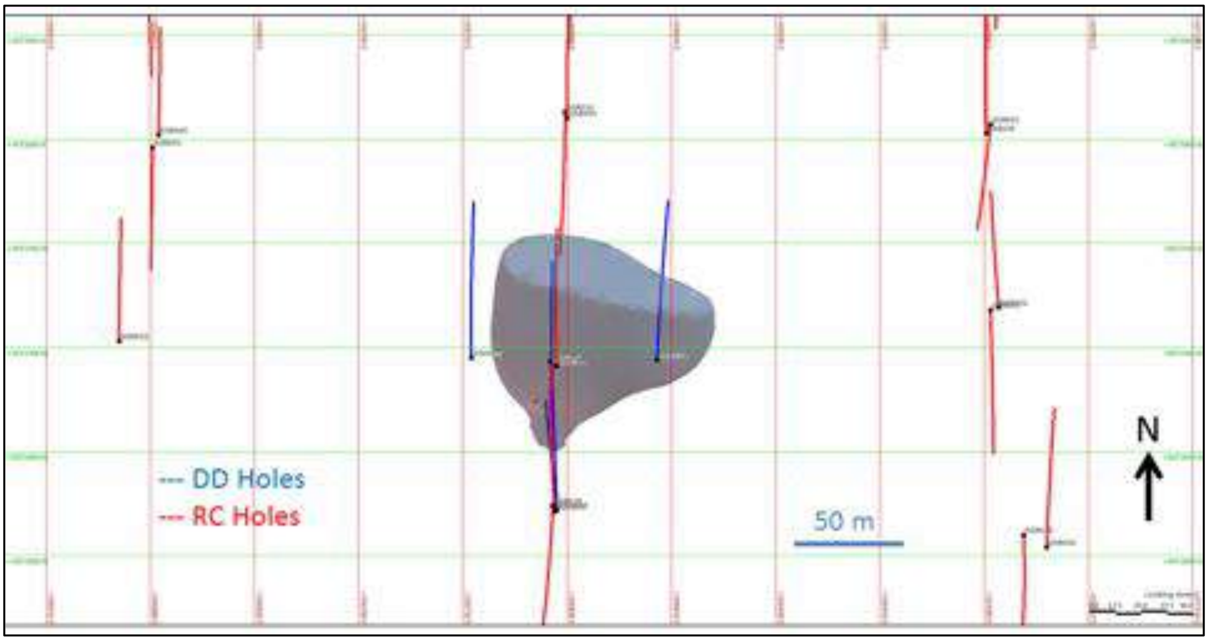

Figure 14-5: Plan view of Tumbili mineralised domain

14.5 Compositing

14.5.1 Raw Sample Lengths

In all domains, the majority of samples are from diamond drilling. Most diamond holes have been sampled using a fixed 1 m sample length. Some of the earlier diamond holes drill by Helio were sampled according to lithology boundaries, but on average these sample lengths are about 1 m too. RC drilling is generally sampled on 2 m intervals.

14.5.2 Kenge

The true thicknesses of the Kenge domains are generally narrow (averaging around 12 m) compared to the strike and dip extents (hundreds of metres). SRK chose 2D Ordinary Kriging as the most effective method for estimating grades within these wireframes. Two-dimension estimation has the advantage of avoiding the difficulties of finding a reasonable composite length and block size that will fit the thin dimension of the domain and 2D estimation also limits the complications that minor undulations of a thin domain shape can cause for variogram modelling and grade estimation.

The inherent assumption with 2D estimation is that there will be no mining selectivity in the thin dimension (approximately the y-dimension for Kenge). This assumption seems reasonable to SRK, given how narrow the wireframe is and how erratically grades are distributed within single intersections through the Kenge lodes (i.e. there are no clearly defined internal zones of higher or lower grades, parallel to the contacts.

To prepare for 2D estimation, each drillhole intersection through the mineralisation domain needs to be composited to a single point with two key attributes:

1 True thickness (calculated from the length of the intersection and the orientation of the drillhole relative to the lode); and

2 Gold accumulation (product of true thickness and mean gold grade).

For calculating the true thicknesses, SRK assumed a constant overall dip for Kenge of 65° towards 210.

Unsampled intervals within the Kenge domain are rare and contribute <1% to the combined length of all Kenge intersections. For the purposes of the compositing, such intervals were treated as missing values, not zero values.

Applying a 2D estimation can be complicated by the presence of incomplete intersections in the database – drillholes that either start or finish in mineralisation. Only one such hole was present at Kenge (SZD145, which finishes in mineralisation). SRK chose to use this intersection in the estimation, with no further changes, because nearby drillholes imply that SZD145 is reasonably representative of the full intersection length.

14.5.3 Mbenge, Porcupine, Konokono and Tumbili

Unlike Kenge, the shapes of the other domains were not suitable for 2D estimation, so the samples in these domains were composited for 3D Ordinary Kriging. A 2 m length was used for Mbenge and a 5 m length was used for Porcupine, Konokono and Tumbili. The shorter length for Mbenge was necessary to produce enough composites for variogram modelling to be viable. The 5 m length for Porcupine was chosen because experimental variograms from these composites were better structured and easier to fit models to than the more variable grades from a shorter composite length.

Compositing started and finished at the domain boundaries. Compositing was done in Gemcom Surpac™ and the "best fit" compositing option was chosen, which allows minor variations from the fixed composite length, in order to avoid creating short residual composites at the end of intersections.

Unsampled intervals within the mineralised domains are rare and contribute <1% to the combined length of all intersections. For the purposes of the compositing, such intervals were treated as missing values, not zero values.

14.6 Statistical Analysis

Summary statistics for the composite grades are given in Table 14-1. Outlier values to be restrained during estimation were identified from analysing the histograms of the composite grades (Appendix C) and 3D visualisation of where the highest grades occur.

For almost all domains, the mean and variance of the composite values decrease with declustering. Declustering cell sizes and the declustered values are in Table 14-2.

Table 14-1: Composite lengths, summary statistics and top cuts

Deposit	Domain Code	Variable	Composite Length	Number of composites in Domain	Minimum	Maximum	Threshold for restraining	Number of composites restrained and their values	Mean (No top cut)	Std Dev (No top cut)	Mean (With top cut)	Std Dev (With top cut)
Kenge Footwall	101	Au Accumulation	2D	44	0.01	124.52	60	1 (124.52)	17.81	20.83	16.34	14.61
Kenge Footwall	101	Thickness	2D	44	0.61	40.40	None		11.40	7.28		
Kenge Hanging Wall	102	Au Accumulation	2D	86	0.01	66.76	None		13.43	15.29		
Kenge Hanging Wall	102	Thickness	2D	86	1.15	37.80	None		11.16	6.29		
Kenge SE	103	Au Accumulation	2D	24	0.31	72.10	None		18.02	18.22		
Kenge SE	103	Thickness	2D	24	2.87	42.14	None		18.79	10.06		
Mbenge	201	Au	2 m	248	0.02	18.02	6	4 (18.02, 7.78, 7.67, 7.29)	1.63	1.87	1.56	1.52
Mbenge	202	Au	2 m	47	0.00	6.18	6	1 (6.18)	1.02	1.69	1.02	1.68
Mbenge	203	Au	2 m	22	0.16	4.19	None		1.31	1.10		
Mbenge South	213	Au	2 m	21	0.14	8.86	6	1 (8.86)	2.17	2.19	2.03	1.83
Mbenge South	215	Au	2 m	32	0.00	26.02	None		1.95	5.67		
Mbenge South	216	Au	2 m	8	0.06	11.23	6	1 (11.23)	2.22	3.58	1.57	2.01
Porcupine	300	Au	5 m	647	0.00	23.42	15	2 (23.42, 18.19)	1.40	2.05	1.38	1.90
Quill	310	Au	5 m	67	0.01	11.33	5	3 (11.33, 5.38, 5.56)	0.97	1.70	0.86	1.19
Porcupine NW	321	Au	5 m	12	0.12	0.89	None		0.51	0.22		
Porcupine NW	322	Au	5 m	26	0.03	1.79	None		0.58	0.48		
Konokono	400	Au	5 m	32	0.05	6.79	None		1.11	1.54		
Tumbili	500	Au	5 m	12	0.19	2.26	None		0.93	0.65		

Table 14-2: Declustered mean and standard deviation

Deposit	Domain Code	Variable	Composite Length	Number of composites in Domain	Threshold for restraining	Mean (With top cut)	Std Dev (with top cut)	Declustering Cell Size (m)	Mean (Declustered and top cut)	Std Dev (Declustered and top cut)
Kenge Footwall	101	Au Accumulation	2D	44	60	16.34	14.61	50 x 50	14.16	13.70
Kenge Footwall	101	Thickness	2D	44	None	11.40	7.28	50 x 50	10.31	6.59
Kenge Hanging Wall	102	Au Accumulation	2D	86	None	13.43	15.29	50 x 50	10.00	13.22
Kenge Hanging Wall	102	Thickness	2D	86	None	11.16	6.29	50 x 50	9.65	5.89
Kenge SE	103	Au Accumulation	2D	24	None	18.02	18.22	50 x 50	14.54	15.72
Kenge SE	103	Thickness	2D	24	None	18.79	10.06	50 x 50	17.99	10.74
Mbenge	201	Au	2 m	248	6	1.56	1.52	20 x 10 x 20	1.30	1.40
Mbenge	202	Au	2 m	47	6	1.02	1.68	20 x 10 x 20	1.01	1.59
Mbenge	203	Au	2 m	22	None	1.31	1.10	20 x 10 x 20	1.18	1.03
Mbenge South	213	Au	2 m	21	6	2.03	1.83	20 x 10 x 20	1.86	1.74
Mbenge South	215	Au	2 m	32	None	1.95	5.67	20 x 10 x 20	1.33	4.40
Mbenge South	216	Au	2 m	8	6	1.57	2.01	20 x 10 x 20	1.31	1.79
Porcupine	300	Au	5 m	647	15	1.38	1.90	30 x 30 x 5	1.34	1.91
Quill	310	Au	5 m	67	5	0.86	1.19	50 x 50 x 5	0.89	1.21
Porcupine NW	321	Au	5 m	12	None	0.51	0.22	50 x 50 x 5	0.51	0.22
Porcupine NW	322	Au	5 m	26	None	0.58	0.48	50 x 50 x 5	0.68	0.50
Konokono	400	Au	5 m	32	None	1.11	1.54	50 x 50 x 5	1.04	1.43
Tumbili	500	Au	5 m	12	None	0.93	0.65	50 x 50 x 5	0.93	0.65

14.6.1 RC versus Diamond Drilling Data

SRK considered whether or not there was a case for either excluding the RC data from the estimation, or modifying the RC assays in some way to make them compatible with the assays from diamond drilling.

Through the Kenge lodes, there are two twinned RC and diamond drilling intersections (Figure 14-6) and two other pairs that are close but possibly not close enough to be strictly considered as twins. The statistics for these pairs are in Table 14-3. The mean grade across all four pairs is not vastly different (1.55 g/t for RC versus 1.46 g/t for diamond drilling) and the number of data points available for this comparison is insufficient to confidently fit a modifying factor to convert RC to equivalent diamond drilling grades, even if the comparison was done using 2 m composites instead of entire intersections.

Discarding the RC data was considered inappropriate as well, because there are portions of the mineralised domains where the RC intersections are locally important to achieve sufficient data density for a good quality estimate; for example, the upper edge of the Kenge Hanging Wall domain (Figure 14-6).

SRK chose to use all the RC intersections, without modification, for the estimation. Estimated block grades that are significantly influenced by RC rather than DD composites would accordingly be assigned a lower confidence category during Mineral Resource classification.

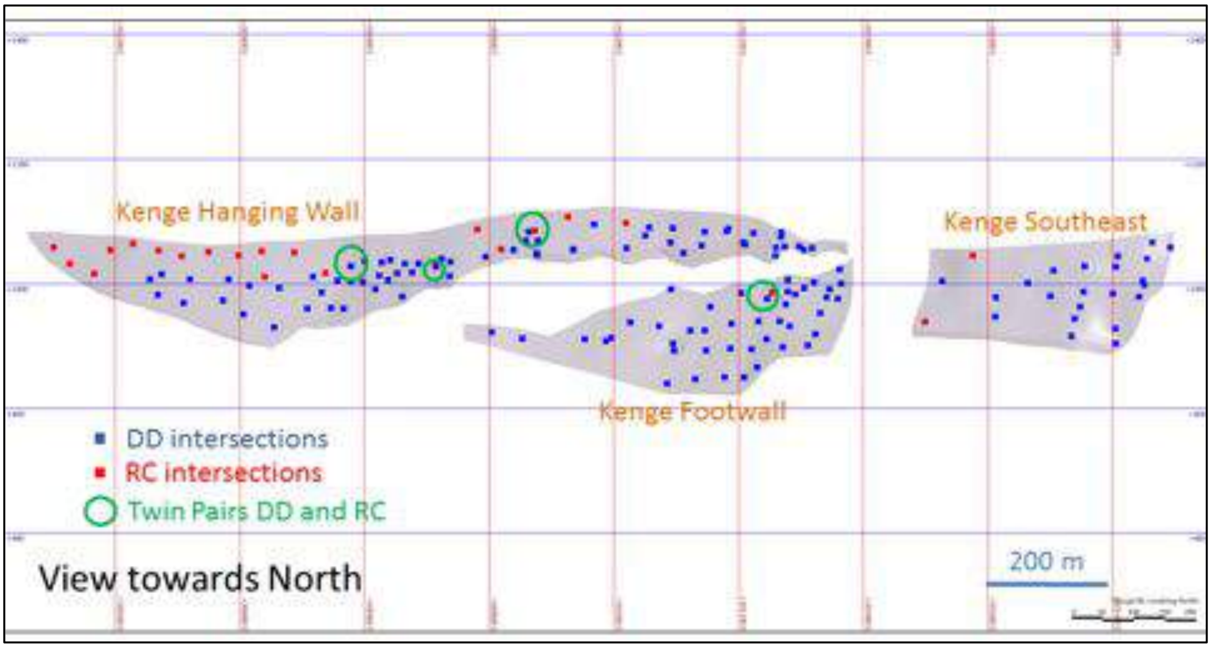

Figure 14-6: Long section showing location of diamond drilling and RC intersection centres through the Kenge domains

Table 14-3: Twinned RC and DD holes

RC Hole ID	DD Hole ID	Distance between intersection centers (m)	Length of RC intersection (m)	Mean grade of RC intersection (g/t)	Length of DD intersection (m)	Mean grade of DD intersection (g/t)
SZR011	SZD013	3.5	26.2	1.05	25.0	1.61
SZR111	SZD071	2.9	18.6	2.44	18.8	1.33
SZR098	SZD067	10.3	15.7	1.64	10.6	0.59
SZR010	SZD019	12.9	18.6	1.32	16.2	1.94
Mean of all four intersections			19.8	1.55	17.6	1.46

14.7 Variogram Modelling

Variograms were modelled in Isatis. The parameters of the variogram models are in Table 14-4.

Table 14-4: Declustered mean and standard deviation

Deposit	Domain Code	Variable	Composite Length	Number of composites in Domain	Threshold for restraining	Mean (With top cut)	Std Dev (with top cut)	Declustering Cell Size (m)	Mean (Declustered and top cut)	Std Dev (Declustered and top cut)
Kenge Footwall	101	Au Accumulation	2D	44	60	16.34	14.61	50 x 50	14.16	13.70
Kenge Footwall	101	Thickness	2D	44	None	11.40	7.28	50 x 50	10.31	6.59
Kenge Hanging Wall	102	Au Accumulation	2D	86	None	13.43	15.29	50 x 50	10.00	13.22
Kenge Hanging Wall	102	Thickness	2D	86	None	11.16	6.29	50 x 50	9.65	5.89
Kenge SE	103	Au Accumulation	2D	24	None	18.02	18.22	50 x 50	14.54	15.72
Kenge SE	103	Thickness	2D	24	None	18.79	10.06	50 x 50	17.99	10.74
Mbenge	201	Au	2 m	248	6	1.56	1.52	20 x 10 x 20	1.30	1.40
Mbenge	202	Au	2 m	47	6	1.02	1.68	20 x 10 x 20	1.01	1.59
Mbenge	203	Au	2 m	22	None	1.31	1.10	20 x 10 x 20	1.18	1.03
Mbenge South	213	Au	2 m	21	6	2.03	1.83	20 x 10 x 20	1.86	1.74
Mbenge South	215	Au	2 m	32	None	1.95	5.67	20 x 10 x 20	1.33	4.40
Mbenge South	216	Au	2 m	8	6	1.57	2.01	20 x 10 x 20	1.31	1.79
Porcupine	300	Au	5 m	647	15	1.38	1.90	30 x 30 x 5	1.34	1.91
Quill	310	Au	5 m	67	5	0.86	1.19	50 x 50 x 5	0.89	1.21
Porcupine NW	321	Au	5 m	12	None	0.51	0.22	50 x 50 x 5	0.51	0.22
Porcupine NW	322	Au	5 m	26	None	0.58	0.48	50 x 50 x 5	0.68	0.50
Konokono	400	Au	5 m	32	None	1.11	1.54	50 x 50 x 5	1.04	1.43
Tumbili	500	Au	5 m	12	None	0.93	0.65	50 x 50 x 5	0.93	0.65

Figures of the experimental variograms and the models fitted to them are in Appendix H.

For variogram modelling of the Kenge true thickness and accumulation, the set of 3D points at intersection centres was converted to 2D by ignoring the y values of the coordinates and setting the z value as the new y value.

The model fitted to the Kenge Hanging Wall domain was based on 86 2D composites. This model was applied to the Kenge Footwall and Kenge Southeast domains, where there were too few composites (44 and 24 respectively) for confidently fitting models.

The variogram model for the Porcupine Main domain was fitted via a Gaussian transform of the composite grades. For highly skewed data, such as the Porcupine composites, a Gaussian transform will often reveal structure during variogram modelling. Without a transform, the squared differences between a few high grade samples can obscure all other structure in the experimental variograms. The "Gaussian Anamorphosis Modelling" function in Isatis was used to transform the Porcupine composites to Gaussian space, the variogram model was fitted to the Gaussian data and then back-transformed to an equivalent model in raw space that would be suitable to be used for the kriging estimation. SRK found that there were no significant benefits from using a transform with the Kenge and Mbenge models, so these models were fitted directly to the raw composite grades.

In several domains there were too few composites for viable 3D variogram modelling. The Mbenge South variogram model was adopted from the Mbenge model, with orientation of the anisotropy adjusted to match the overall geometry of the mineralised domain. Similarly, the Porcupine Northwest variogram model was based on the Porcupine Main model, with an adjustment to the orientation of anisotropy. The nugget component for the Konokono and Tumbili models could be modelled from downhole variograms, but the structured components were assumed values, based on parameters from the Porcupine Main model.

14.8 Block Model and Grade Estimation

The framework for the block model was created in Gemcom Surpac™. Block sizes for each domain were chosen based on drill spacing and continuity of the data. These dimensions are in Table 14-5. Sub-blocking was used to improve the geometric precision of coding the block model with the mineralisation domain wireframes.

Block grades for all deposits were estimated in Isatis software by Ordinary Kriging. For the Kenge deposits, 2D Ordinary Kriging was used: the thickness and gold accumulation were estimated and then for each block the gold grade was derived from dividing the accumulation estimate by the thickness estimate. These 2D block grades for Kenge were then imported into a 3D block model framework in Gemcom Surpac™ for visualisation and reporting. For all other deposits, gold grades were estimated directly by 3D Ordinary Kriging. Kriging was done in a single pass for all domains.

Table 14-7 lists the kriging neighbourhood parameters. Anomalously high grades were controlled in the estimation by using grade and distance thresholds. Where a composite grade exceeded the grade and search distance threshold, the composite was top-cut to the grade threshold. Where a composite grade exceeded the grade threshold but not the distance threshold, the composite was included at its full value.

The dimensions of the search ellipsoids and the maximum number of composites used per block estimate were optimised by kriging neighbourhood analysis. During the optimisation, a key parameter of interest is the "slope of regression" – a theoretical value calculated for each kriging estimate, that represents the slope of linear regression between estimated and true block grades.

This choice of neighbourhood parameters is usually a compromise between maximising the slope of regression (by including more composites) whilst avoiding excessive negative kriging weights (by using fewer composites).

Table 14-5: Block sizes for used for modelling each deposit

Deposit	Block Dimensions (m)			Sub-block Dimensions (m)		
	X	Y	Z	X	Y	Z
Kenge	25	8	25	6.25	2	6.25
Mbenge	10	10	10	5	2.5	5
Porcupine	10	10	10	5	5	5
Konokono	20	20	20	5	5	5
Tumbili	20	20	20	5	5	5

Table 14-6: Parameters of the variogram models

Deposit	Variable	Nugget	Sill1	Sill2	Range1 U	Range1 V	Range1 W	Range2 U	Range2 V	Range2 W	Orientation U*	Orientation V*	Orientation W*
Kenge	Accumulation	40	60	75	80	80	n/a (2D)	300	300	n/a (2D)	Horizontal	Vertical	n/a (2D)
Kenge	Thickness	0	19.5	15	80	80	n/a (2D)	300	300	n/a (2D)	Horizontal	Vertical	n/a (2D)
Mbenge	Au grade	0.65	1.3		32	32	5				0 -> 090	-77 -> 180	-13 -> 000
Mbenge South	Au grade	0.65	1.3		32	32	5				0 -> 110	-66 -> 200	-24 -> 020
Porcupine Main	Au grade	1.6	1.24	0.79	125	40	10	125	100	20	-57 -> 187	-18 -> 067	-27 -> 328
Quill	Au grade	0.9	2		80	32	10				-70 -> 315	0 -> 225	-20 -> 045
Porcupine NW	Au grade	1.6	1.24	0.79	125	40	10	125	100	20	0 -> 090	-90 -> 000	0 -> 180
Konokono	Au grade	1.05	0.82	0.52	125	40	10	125	100	20	0 -> 255	-60 -> 345	-30 -> 165
Tumbili	Au grade	0.17	0.15	0.09	125	40	10	125	100	20	-55 -> 190	0 -> 100	-35 -> 010

Table 14-7: Kriging neighbourhood parameters

Deposit	Variable	Number of angular sectors	Optimum number of composites per sector	Minimum number of composites required for an estimate	Orientation of principal axes of the search ellipsoid			Search Distance (m)			Discretisation			Threshold for grade restraint	Distance for grade restraint (m)
					U*	V*	W*	U*	V*	W*	x	y	z		
Kenge	Accumulation	4	3	1	Horizontal	Vertical	n/a (2D)	250	250	n/a (2D)	8	8	1	60	15
Kenge	Thickness	4	3	1	Horizontal	Vertical	n/a (2D)	250	250	n/a (2D)	8	8	1		
Mbenge	Au grade	8	5	1	0 -> 090	77 -> 180	13 -> 000	70	70	20	8	8	8	6	10
Mbenge South	Au grade	8	5	1	0 -> 110	66 -> 200	24 -> 020	70	70	20	8	8	8	6	10
Porcupine Main	Au grade	8	8	5	57 -> 187	18 -> 067	27 -> 328	200	200	50	10	10	1	15	10
Quill	Au grade	8	8	1	70 -> 315	0 -> 225	20 -> 045	300	300	50	10	10	1	5	10
Porcupine NW	Au grade	8	8	1	0 -> 090	90 -> 000	0 -> 180	300	300	50	10	10	1		
Konokono	Au grade	8	8	5	0 -> 255	60 -> 345	30 -> 165	200	200	50	10	10	1		
Tumbili	Au grade	8	8	5	55 -> 190	0 -> 100	35 -> 010	200	200	50	10	10	1		

*U, V and W are, respectively, the major, semi-major and minor axes of the anisotropy ellipsoid. The orientations of these axes are given as: "dip -> azimuth" (negative dip is down).

14.9 Density

Dry bulk density values in the model were assigned based on the average of density measurements from each domain (Table 14-8), after excluding outliers. These factors were used to convert volumes in the block model into tonnages.

Table 14-8: Density values assigned to each domain

Deposit	Number of Density Measurements from Domain	Assigned Dry Bulk Density
Kenge Footwall	163	2.74
Kenge Hanging Wall	53	2.74
Kenge SE	124	2.74
Mbenge	87	2.71
Mbenge South	8	2.71 (assigned from Mbenge)
Porcupine Main	2760	2.63
Quill	72	2.58
Porcupine NW	82	2.54
Konokono	0	2.63 (assigned from Porcupine Main)
Tumbili	0	2.63 (assigned from Porcupine Main)

14.10 Mining Depletion

As discussed in Section 6.1, there are several old workings within the area of the SMP. Some of these workings are on the ridge that marks the Kenge target. SRK's judgement is that the volume extracted from these workings that potentially intersects the volume contained within the mineralised domains used for the Mineral Resource estimation is not a significant quantity and therefore no mining depletion needs to be applied to the statement of Mineral Resources.

14.11 Model Validation and Sensitivity

The block models were validated by visual checks against the drillholes and wireframed domains and statistical checks were completed against the raw samples and composited grades.

Summary statistics from the validation are in Table 14-9. For the larger domains, the block and composite grades are acceptably close. For some of the smaller domains, particularly those which contain intersections from five or fewer drillholes, there can be substantial differences between the mean grades from blocks and composites. SRK reviewed the estimates from the smaller domains and in all cases was satisfied that the apparently poor performance during the statistical validation checks was due to the difficulty of finding an effective declustering cell size, rather than flaws in the block model. Material in these smaller domains was always classified as Inferred.

For the Kenge domains and the Porcupine Main domain, statistical validation was done in more detail by means of swath plots (Appendix I). The composites file for the Kenge swath plots was a set of 2 m composites, instead of the 2D composites used for estimation.

Grade-tonnage relationships were plotted in order to examine the sensitivity of the models to various choices of cut-off grades. The grade-tonnage curves for Kenge and the Porcupine Main domain are in Figure 14-7 and Figure 14-8.

Table 14-9: Summary statistics for block grades versus declustered and top-cut composite grades

Deposit	Domain Code	Composite Length	Number of composites in Domain	Declustering Cell Size	Composites Mean (Declustered and top cut)	Composites Std Dev (Declustered and top cut)	Block Mean	Block Std Dev
Kenge Footwall	101	2D	44	50 x 50	1.42*	0.87*	1.44	0.45
Kenge Hanging Wall	102	2D	86	50 x 50	1.04*	0.81*	1.12	0.53
Kenge SE	103	2D	24	50 x 50	0.81*	0.77*	0.71	0.49
Mbenge	201	2 m	248	20 x 10 x 20	1.30	1.40	1.46	0.48
Mbenge	202	2 m	47	20 x 10 x 20	1.01	1.59	1.43	0.57
Mbenge	203	2 m	22	20 x 10 x 20	1.18	1.03	1.38	0.27
Mbenge South	213	2 m	21	20 x 10 x 20	1.86	1.74	1.91	0.18
Mbenge South	215	2 m	32	20 x 10 x 20	1.33	4.40	0.79	0.28
Mbenge South	216	2 m	8	20 x 10 x 20	1.31	1.79	1.42	0.28
Porcupine	300	5 m	647	30 x 30 x 5	1.34	1.91	1.28	0.64
Quill	310	5 m	67	50 x 50 x 5	0.89	1.21	0.84	0.3
Porcupine NW	321	5 m	12	50 x 50 x 5	0.51	0.22	0.52	0.04
Porcupine NW	322	5 m	26	50 x 50 x 5	0.68	0.50	0.56	0.13
Konokono	400	5 m	32	50 x 50 x 5	1.04	1.43	1.06	0.31
Tumbili	500	5 m	12	50 x 50 x 5	0.93	0.65	0.99	0.16

*For this declustering, the grades of the 2D Kenge composites were weighted by both the true thickness and the weight from the declustering cell.

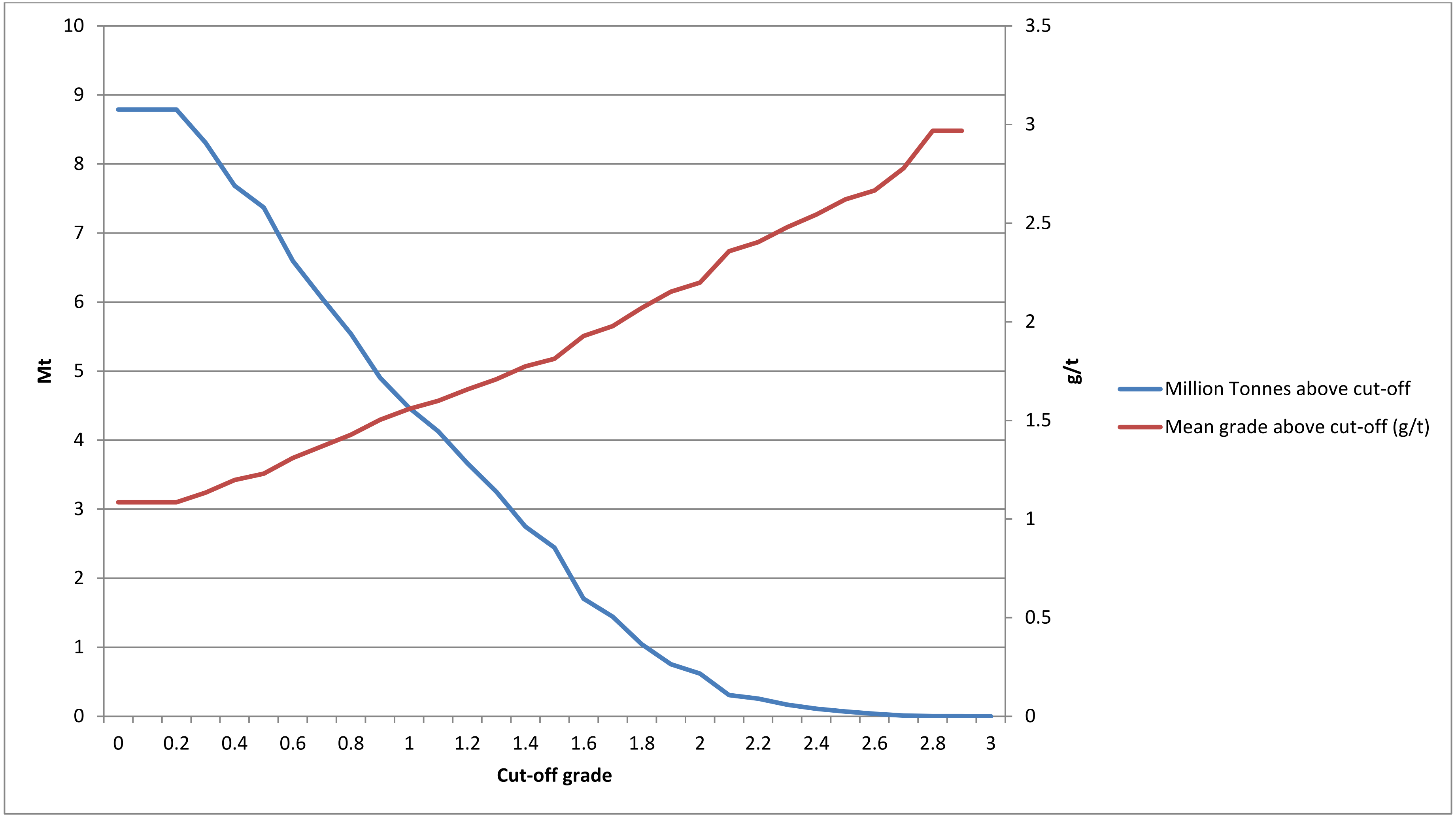

Figure 14-7: Grade-tonnage curves for combined Kenge domains

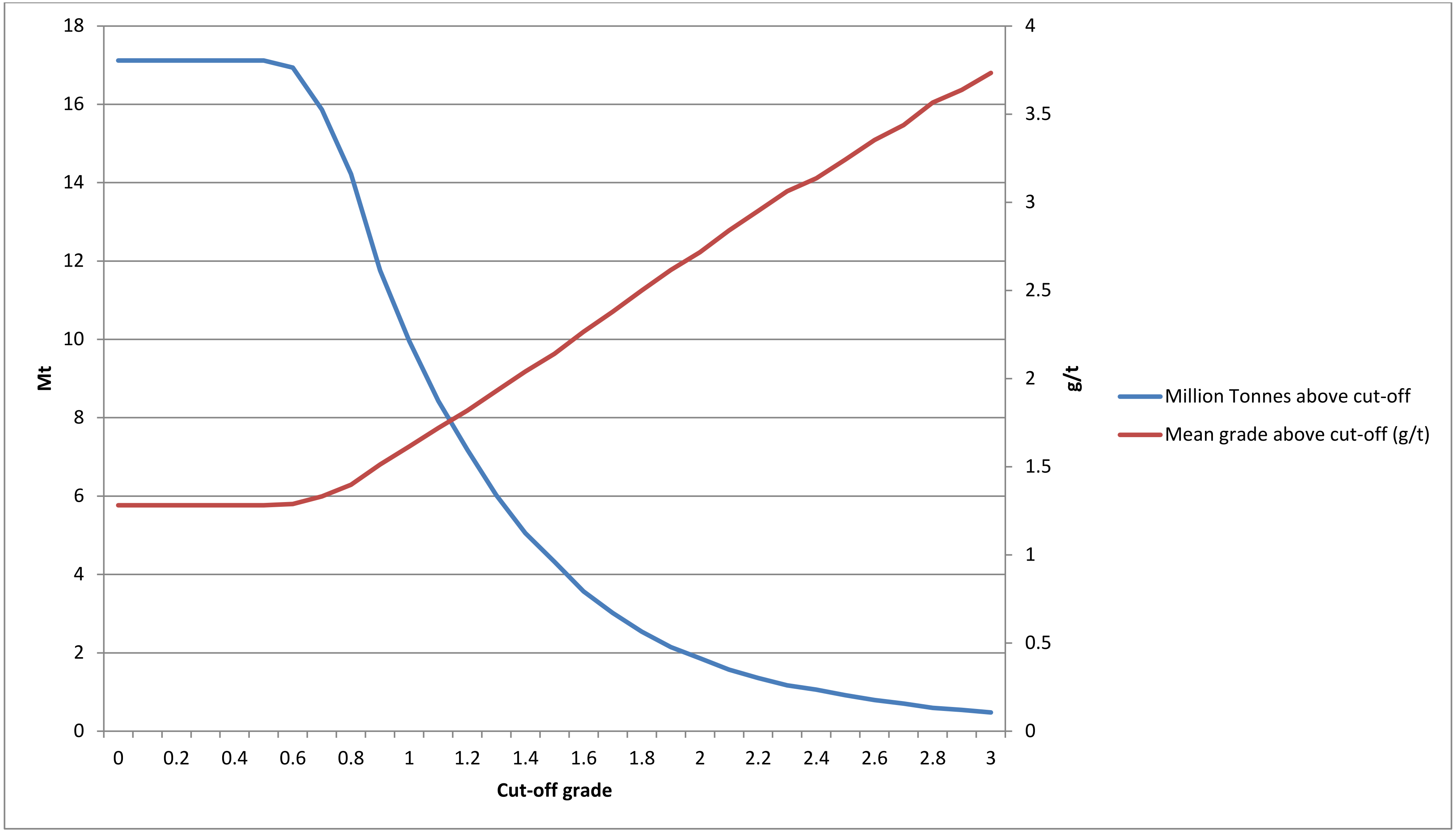

Figure 14-8: Grade-tonnage curves for Porcupine Main domain

14.12 Mineral Resource Classification

The three main elements considered during classification of the Mineral Resource were:

1 Confidence in the geological continuity of the mineralised structures;

2 The quality and quantity of the exploration data supporting the estimates; and

3 Geostatistical confidence in the tonnage and grade estimates.

The following sections discuss how these elements contributed to the SRK's classification decisions for each target.

14.12.1 Kenge

SRK has high confidence in the continuity of the Kenge mineralised structure. After reviewing kriging quality results generated during the estimation, SRK considered that an Indicated classification was appropriate for segments of the domain where drill intersection centres were spaced about 50 m apart. Segments with a wider spacing of intersection centres, or that were mostly defined from RC drilling, were assigned an Inferred classification. The Measured category was applied to segments with an average intersection spacing of about 30 m or less and where the interpretation and estimation did not substantially depend on information from RC holes.

The classification for Kenge is shown in approximately long section view in Figure 14-9. Segmenting the domain into different classification zones was generally done on a scale of 100 m or more, with easting lines as simple boundaries between categories, in order to avoid creating a complex patchwork of different classifications.

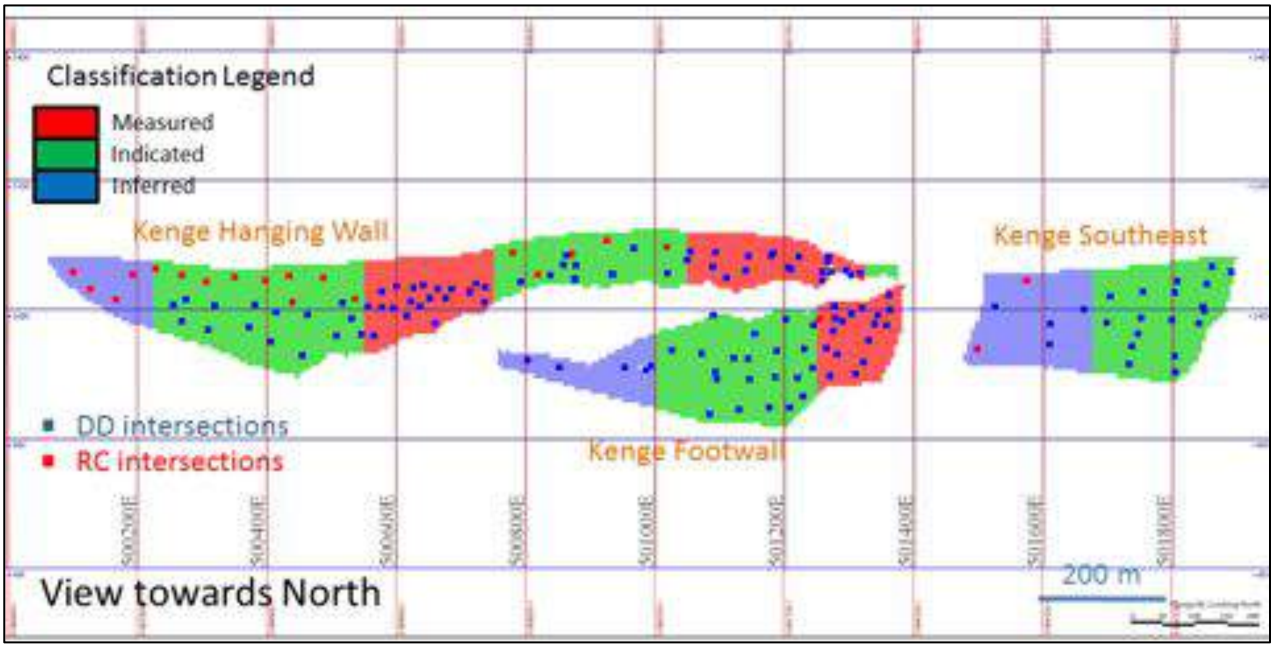

Figure 14-9: Kenge classification

14.12.2 Mbenge

The largest of the Mbenge domains (code 201) is defined from about 20 diamond drill intersections, spaced 25 to 50 m apart and contains 248 composites. The shape of the mineralised domain is more complex than the Kenge domains, but confidence in mineralisation continuity for this Mbenge domain is still reasonably good. The portion of this domain above 940 mRL was classified as Indicated. Below 940 mRL, an Inferred classification was applied, because the deeper part of the domain is modelled from more widely spaced drill intersections and incorporates some extrapolation of up to 40 m away from the drillholes.

The other Mbenge and Mbenge South domains were all classified as Inferred. These domains are each modelled from 8 or fewer drillholes and individually have too few composites for variogram modelling, so relied on assumed rather than fitted parameters for the estimation.

14.12.3 Porcupine

Most of the Porcupine Main domain is covered by diamond drilling at a spacing of about 30 m between intersection centres. Confidence in the geological continuity is high and the kriging quality parameters generated during the estimation stage also consistently meet high confidence thresholds. Therefore most of the Porcupine Main domain is classified as Measured (Figure 14-10). Near the margins of the Porcupine Main domain though, there are zones estimated from more widely spaced drilling and zones where grades have been extrapolated up to about 40 m beyond drillholes. SRK digitised perimeters to define Indicated and Inferred components for these zones of lower confidence.

The other Porcupine domains (Quill and Porcupine Northwest) were all classified as Inferred. These domains are defined and estimated from more widely spaced drilling than Porcupine Main, are more dependent on information from RC drilling and individually have too few composites for variogram modelling, so use assumed rather than fitted parameters for the estimation.

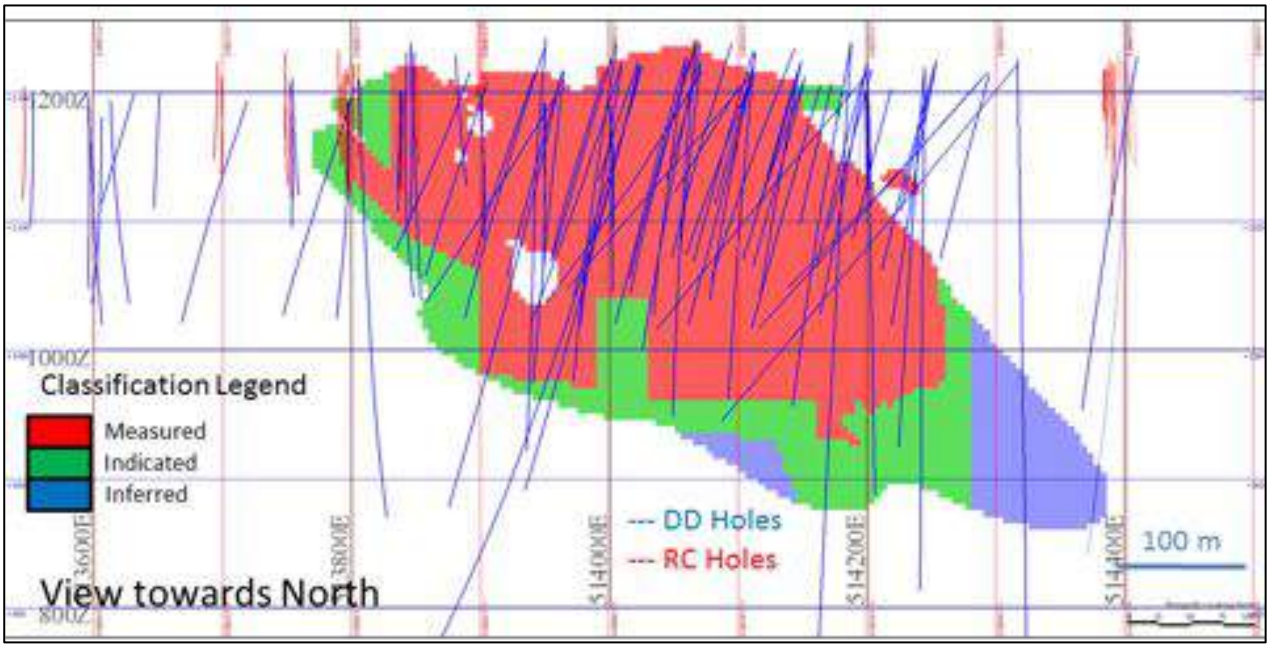

Figure 14-10: Porcupine Main classification

14.12.4 Konokono and Tumbili

Konokono and Tumbili were both classified as Inferred. These domains are much smaller than Kenge or Porcupine Main, so use assumed rather than fitted parameters for the estimation. Konokono is estimated from 32 composites and Tumbili has only 12.

14.13 Mineral Resource Statement

Table 14-10 to Table 14-14 summarise the results of the independent Mineral Resource Estimate for the SMP Gold Project. The effective date of this Mineral Resource Estimate is 10 February 2012.

Currently no economic evaluation has been undertaken by SRK for the SMP Gold Project and accordingly the results at multiple cut-off grades are reported here, all of which SRK considers meet the test of “a reasonable prospect of economic extraction”. The 0.5 g/t cut-off is the preferred scenario, based on SRK’s knowledge of similar deposits, analysis of grade-tonnage curves and the results from the optimisation work done by SRK (Section 14.14).

Helio has informed SRK that there are no known litigations potentially affecting the SMP and furthermore that there are no known environmental, socio-political, marketing or taxation issues that may materially affect the project.

Table 14-10: 10 February 2012 Mineral Resource Estimate for all SMP deposits combined

Cut-off	Class	Mean Au grade (g/t)[1]	Tonnage (Mt)[2]	Metal Au (koz)[3]
0.3	Measured	1.38	14.8	660
0.3	Indicated	1.20	9.5	370
0.3	Measured+Indicated	1.31	24.3	1,020
0.3	Inferred	0.97	8.3	260
0.5	Measured	1.38	14.8	660
0.5	Indicated	1.22	9.2	360
0.5	Measured+Indicated	1.32	24.1	1,020
0.5	Inferred	1.05	7.3	250
0.7	Measured	1.44	13.8	640
0.7	Indicated	1.30	8.2	340
0.7	Measured+Indicated	1.39	21.9	980
0.7	Inferred	1.18	5.7	220
0.9	Measured	1.60	10.9	560
0.9	Indicated	1.43	6.4	300
0.9	Measured+Indicated	1.54	17.3	860
0.9	Inferred	1.33	4.0	170

1: Rounded to two decimal places

2: Rounded to nearest 0.1 Mt

3: Rounded to nearest 10 koz

Table 14-11: February 10, 2012 Mineral Resource Estimate for Porcupine

Cut-off	Class	Mean Au grade (g/t)[1]	Tonnage (Mt)[2]	Metal Au (koz)[3]
0.3	Measured	1.35	12.3	530
0.3	Indicated	1.16	3.1	120
0.3	Measured+Indicated	1.31	15.4	650
0.3	Inferred	0.85	3.6	100
0.5	Measured	1.35	12.3	530
0.5	Indicated	1.16	3.1	120
0.5	Measured+Indicated	1.31	15.4	650
0.5	Inferred	0.89	3.3	90
0.7	Measured	1.41	11.3	510
0.7	Indicated	1.19	2.9	110
0.7	Measured+Indicated	1.37	14.3	630
0.7	Inferred	1.01	2.3	70
0.9	Measured	1.61	8.6	440
0.9	Indicated	1.32	2.2	90
0.9	Measured+Indicated	1.55	10.8	530
0.9	Inferred	1.15	1.4	50

Table 14-12: February 10, 2012 Mineral Resource Estimate for Kenge and Mbenge

Cut-off	Class	Mean Au grade (g/t)[1]	Tonnage (Mt)[2]	Metal Au (koz)[3]
0.3	Measured	1.51	2.6	120
0.3	Indicated	1.22	6.3	250
0.3	Measured+Indicated	1.30	8.9	370
0.3	Inferred	1.07	3.2	110
0.5	Measured	1.51	2.6	120
0.5	Indicated	1.25	6.1	250
0.5	Measured+Indicated	1.33	8.7	370
0.5	Inferred	1.28	2.5	100
0.7	Measured	1.55	2.4	120
0.7	Indicated	1.36	5.2	230
0.7	Measured+Indicated	1.42	7.7	350
0.7	Inferred	1.45	2.0	90
0.9	Measured	1.59	2.3	120
0.9	Indicated	1.49	4.2	200
0.9	Measured+Indicated	1.53	6.6	320
0.9	Inferred	1.55	1.7	90

Table 14-13: February 10, 2012 Mineral Resource Estimate for Konokono

Cut-off	Class	Mean Au grade (g/t)[1]	Tonnage (Mt)[2]	Metal Au (koz)[3]
0.3	Inferred	1.06	1.0	30
0.5	Inferred	1.06	1.0	30
0.7	Inferred	1.09	0.9	30
0.9	Inferred	1.25	0.6	20

Table 14-14: February 10, 2012 Mineral Resource Estimate for Tumbili

Cut-off	Class	Mean Au grade (g/t)[1]	Tonnage (Mt)[2]	Metal Au (koz)[3]
0.3	Inferred	0.99	0.5	10
0.5	Inferred	0.99	0.5	10
0.7	Inferred	0.99	0.5	10
0.9	Inferred	1.08	0.3	10

1: Rounded to two decimal places

2: Rounded to nearest 0.1 Mt

3: Rounded to nearest 10 koz

14.14 Pit Optimisation

Helio requested that SRK carry out a preliminary pit optimisation study on the results from the Mineral Resource Estimate. The purpose of this optimisation was to ensure that the Mineral Resources estimated for the SMP meet the test of reasonable prospect of economic extraction.

The block models from the Porcupine, Kenge and Mbenge estimations were imported into Whittle™ software and the Measured and Indicated components were optimised according to parameters supplied by Helio (Table 14-15). The optimisation work was done by Duncan Pratt, a Senior Consultant (Mining) with SRK.

The optimal pit shells are shown in Figure 14-11 and Figure 14-12 and the results from within the optimal pit shells defined by this study are summarised in Table 14-10. When these results are compared against the results at 0.5 g/t cut-offs in Table 14-11 and

Table 14-12, it is apparent that the optimisation is capturing just over 90% of the Measured and Indicated ounces in the Porcupine Mineral Resource estimate and almost 80% of the Measured and Indicated ounces in the Kenge and Mbenge Mineral Resource estimate. This outcome from the optimisation is evidence that the Mineral Resource estimates do meet the test of reasonable prospect of economic extraction and this outcome also supports the choice of 0.5 g/t as the favoured cut-off grade for reporting the Mineral Resources.

The results from the optimisation also have implications for future exploration. The pit shell for Porcupine (Figure 14-11) draws down close the base of the Measured and Indicated component of the Mineral Resource, implying that investing in further drilling to infill and extend the Porcupine coverage at depth would be worthwhile. For Kenge though, the pit shells (Figure 14-12) are often much shallower than the depth limit of the Measured and Indicated components, therefore deeper drilling would need to be justified on the grounds that such drilling will target zones that are higher grade than the overall Kenge mineralisation.

Table 14-15: Optimisation parameters

Item	Unit	Value
Gold Value		
Gold Price	US$/oz	1450.00
Off site costs	US$/oz	7.00
Royalties @ 3% of NSR	US$/oz	43.29
Net gold price	US$/oz	1399.71
Net gold price	US$/g	45.00
On Site Costs		
Mining Cost	US$/t rock	1.75
Milling Cost (for CIL)	US$/t ore	10.00
G&A	US$/t ore	5.00
Sustaining Capital Cost	US$/t ore	0.50
Sub-total Mill, G&A, Sust. Capital	US$/t ore	15.50
Process and Mining Losses		
Process Recovery of gold	%	95
Mining Dilution	%	10
Incremental Cut-off Grade (excl. mining)	g/t Au	0.4
Geotechnical Parameters		
Slope Angles (Overall)	Degrees	55

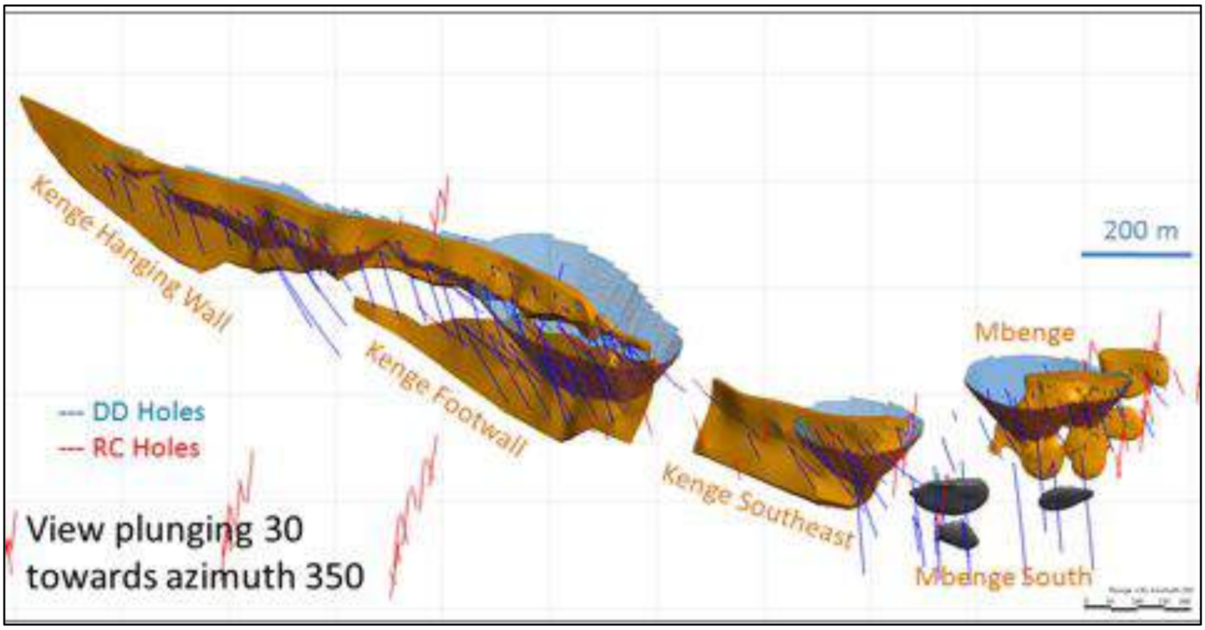

Figure 14-11: Oblique view showing optimal pit shells for Kenge and Mbenge in relation to the mineralised domains and drilling

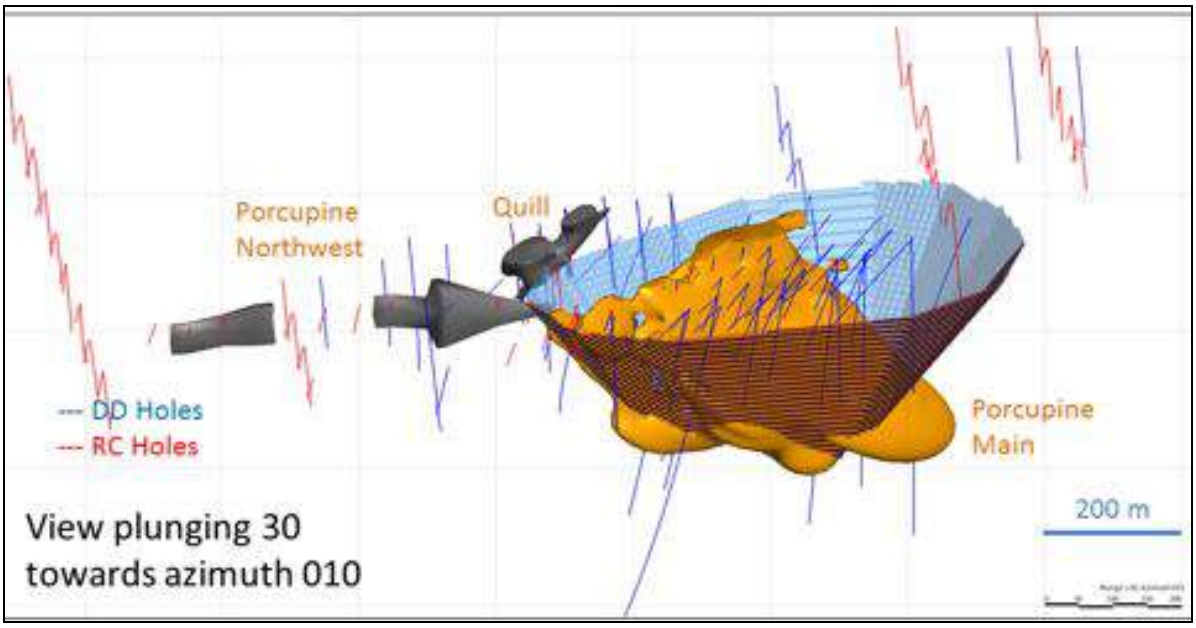

Figure 14-12: Oblique view showing optimal pit shell for Porcupine in relation to the mineralised domains and drilling

Table 14-16: Optimisation results

Deposit	Ore Tonnes (Mt)[1]	Waste Tonnes (Mt)[1]	Strip Ratio	Au Grade (g/t)[2]	Metal Au (koz)[3]
Porcupine	15.5	79.8	5.2	1.21	600
Kenge and Mbenge	7.0	33.6	4.8	1.30	290

1: Rounded to nearest 0.1 Mt

2: Rounded to two decimal places

3: Rounded to nearest 10 koz

14.15 Previous Mineral Resource Estimates

This Technical Report is the second Mineral Resource estimate to be published for the SMP. The first Mineral Resource estimate was prepared by Alexandra Harrison of Golder Associates (UK) Ltd. This initial estimate was announced on November 30, 2010 and is described in a Technical Report (Harrison, 2011) dated January 13, 2011. The detailed Mineral Resource statement from the previous estimate is in Appendix J.

Comparisons between the initial estimate and SRK's estimate are discussed below.

14.15.1 Porcupine

Helio carried out a substantial amount of drilling on Porcupine and adjacent targets during 2011 and the new information from this drilling has led to a substantial revision of the wireframed interpretations of the Porcupine domains. The new drilling has infilled and extended drilling coverage of the Porcupine Main domain in particular and therefore the Porcupine Measured and Indicated tonnes and metal in the new resource model have approximately doubled from the previous model.

14.15.2 Kenge and Mbenge

Helio has carried out no new drilling on the Kenge and Mbenge targets since the previous Mineral Resource estimate. Differences between the new model and the previous model are therefore due to changes to the geological interpretation and geostatistical modelling methodology, rather than differences in the dataset used. Overall, the results from the SRK model for Kenge have somewhat lower grade, higher tonnes and higher metal than the previous estimate. The main reasons for these differences are:

1 The mineralisation envelope SRK used to constrain the estimation was based primarily on lithology rather than grade. Where grades were used to define the envelope, a threshold of about 0.3 g/t was used. The previous model was constrained by envelopes defined at a nominal threshold of 0.5 g/t. Therefore there is a greater volume within the envelope modelled by SRK. SRK considers that a 0.5 g/t threshold for defining the mineralisation envelope is too close to the favoured cut-off grade for reporting Mineral Resources (also 0.5 g/t) and would be likely to bias the estimates high.

2 The Kenge domains were estimated by a 2D method, which inherently assumes that mining selectivity will not be possible in the direction perpendicular to the lode. The previous model estimated into a 3D block model, so when a cut-off grade was applied to the model, selectivity in the perpendicular direction will influence the results.

3 SRK's interpretation of the Kenge SE domain was extended further along strike, to the NW, than the previous interpretation. The additional material included by this extension though is generally lower grade though than the previous domain average grade.

4 Other notable changes compared to the previous model for Kenge are:

   - The use of larger composites (2D full intersection lengths, compared to 2 m lengths in the previous mode) meant that outliers were fewer and less extreme, so top-cutting or restricting the influence of high grades did not need to be so severe;
   - SRK considered that a Measured classification was justified for parts of the Kenge lodes where there was relatively dense sampling by diamond drilling (about 30 m spacing between intersection centres). The previous Kenge Mineral Resource was classified as no higher than Indicated; and
   - Since the previous Kenge Mineral Resource estimate, the collars of drillholes has been resurveyed by DGPS, hence increasing the confidence in the data.

14.15.3 Konokono and Tumbili

There were no Mineral Resources for Konokono and Tumbili in the previous estimate. The additional drilling done by Helio on these targets during 2011 has made it possible to define Mineral Resources from Konokono and Tumbili for the first time.

14.16 Recommendations for Conversion of Mineral Resources into Mineral Reserves

There has not been any conversion from Mineral Resources into Mineral Reserves for this PEA.

15 Mineral Reserve Estimates

A Mineral Reserve Statement has not been calculated or issued as part of this PEA.

16 Mining Methods

16.1 Introduction

For the PEA, it is assumed that a conventional open pit operation including drill and blast, followed by truck and shovel activities will be employed.

An earthmoving contractor is proposed to be employed and it is expected a standard excavator and haul truck mining fleet will be utilised along with supporting auxiliary equipment (motor grader, water truck etc.). For scheduling purposes and expected fleet numbers, a 190 t excavator and 100 t off-highway diesel haul trucks have been used.

Drilling and blasting are planned to be performed on 10m benches for Porcupine and Mbenge and 12.5 m benches for Kenge. This matches the block size in the geological block model. Due to the expected selective mining that will be required for ore mining, load and haul are planned to be performed on full bench heights for waste movement and half bench height for ore material.

The Porcupine pit is planned to be developed first with the process facility to be constructed adjacent to this pit. This will minimise the haulage requirements during the early years of the project. As Kenge and Mbenge are brought into production, a small fleet of road trucks are proposed to be utilised to haul ore from the respective pit rim stockpiles to the processing facility adjacent to the Porcupine pit. The payload for the road trucks will nominally be 20 t.

The project plans to use proven technology, with no requirement for untried or untested technology.

A Base Case scenario has been presented as well as a case representing upside potential, based on 55 degree wall angle for each pit.

16.2 Base Case

The operation is planned to run for 9 years which includes an initial year of prestrip. During this initial year, it is expected all required infrastructure will be constructed onsite. The operation will process 1.6 million tonnes per annum (Mtpa) with a maximum total material movement of 14.6 Mtpa (ore and waste tonnes). Throughout the project approximately 94 people are planned to be employed as part of the mining operation.

16.2.1 Pit Optimisation

Mineral Resource Models

All three Mineral Resource block model was imported into Whittle and verified against the original Mineral Resource block model (block model), created in Surpac. The Surpac block model subsequently was coded in preparation for optimisation. The verification process indicated no material changes to the block model tonnes and grade during the process of importing into Whittle.

Table 16-1: Block Model block sizes

	Kenge	Mbenge	Porcupine
X(m)	6.25	5	5
Y(m)	2	2.5	5
Z(m)	6.25	5	5

Optimisation Process

The process and parameters described below were used for the optimisation of all three deposits.

Topographic Data

The most recent fully validated topographic data was used during construction of the block model. This defined the blocks above the topographic surface as "air" blocks and below the surface as "rock".

Optimisation Constraints

The optimisation process is restricted to classifications of Indicated and Measured in accordance with the Canadian National Instrument guideline NI 43-101. For the purpose of the SMP optimisation, there were no production or processing limits used within Whittle and all material not classified as Measured or Indicated has been coded as waste.

Optimisation Parameters

The pit optimisation for all three deposits has been carried out using Whittle optimisation software (Whittle Version 4.4). Revenue, mining costs, processing values and other factors as described below were input to the Whittle software. These parameters are used to determine the optimum pit shell to be used as a guide for the preparation of open pit deigns. The parameters used for the optimisation discussed below and summarised in Table 16-2 have been supplied by the client.

Mining Dilution and Ore Losses

The block model as imported into Whittle was undiluted. The optimisation process included factors of 10% mining dilution and 100% ore recovery. These parameters were supplied by the client but considered by SRK to be reasonable.

When Whittle applies dilution, a grade of 0% Gold (Au) is used by default and is therefore conservative considering the likely diluting material may be marginally below the economic cut-off.

Discount Rate

The pit optimisation for all three deposits did not utilise a discounting factor. Inflation was not factored into the costs, which represent an indication of the "Current Prices" in the analysis.

The Lerchs-Grossmann algorithm (on which the Whittle software is based) produces a series of mathematically optimum pit shells directly linked to the Revenue Factor utilised if the maximum undiscounted cash flow is the selection criterion for optimisation. Most mining companies require a pit that produces the maximum Net Present Value (NPV) of the project, often assuming real discount rates of 10%. Where this is the case, the Whittle pit analysis does not produce the optimum pit shell because at the time of the optimisation process the exact year of mining for each block is not known. If a discount rate is applied, SRK considers this would unfairly penalise any deposit, as there is no functionality to account for inflation or differing commodity prices within the initial Whittle optimisation process.

Royalties

Royalties have been defined by the client. These were applied in the Whittle model as a selling cost. No other private royalties are included. A total of $50.29 / oz was applied as selling costs (including Royalties and Off-site costs).

Mining Costs

The base mining costs were originally supplied by the client. SRK reviewed the proposed costs and modified the input value based on prior experience with similar projects. SRK applied an incremental cost to account for the increased cost of mining at depth. SRK applied an increase of USD0.05 per 10 vertical metres increase in depth which is considered standard within the industry for optimisation purposes.

All material has been classified as Fresh material and a mining cost of $2.75 / tonne has been applied (for material at surface).

SRK understands the operating mining costs to include the following:

- Drilling and blasting;
- Loading;
- Hauling;
- All auxiliary mining activities;
- Dewatering; and
- Any required rehandling activities.

Processing Costs and Recoveries

The estimated processing cost for the SMP deposits is constant based on the deposits only containing Fresh material. The processing costs and recovery has been supplied by the client based on recoveries from metallurgical testing by SGS Lakefield Research Limited ("SGS") in Ontario, Canada. Processing costs have been based on similar processing scenarios.

- Fresh / Sulphide Material - $15.50 / ore tonne at 95% Au recovery.

This value includes processing cost, administration cost and sustaining capital cost for the optimisation (detailed in Table 16-2).

No additional surface transport costs have been applied to the SMP optimisations.

Parameter Summary

Table 16-2 summarises the optimisation parameters used.

Table 16-2: Optimisation Parameters (Base Case)

Parameter	Unit	Value
Mining Dilution	%	10
Mining Dilution Grade		0.00
Mining Recovery	%	100
Overall Slope Angle (Kenge)	(^{o})	48.5
Overall Slope Angle (Mbenge)	(^{o})	50.2
Overall Slope Angle (Porcupine)	(^{o})	47.5
Mining Cost	$ / t	2.75
Incremental Mining Cost (for depth)	$ / 10 m	0.05
Mining Rate	Mtpa	unlimited
Processing Rate	Mtpa	1.850
Process Recovery Au	%	95
Processing Costs Au	$ / t_{ore}	10.00
General and Administration	$ / t_{ore}	5.00

Parameter	Unit	Value
Sustaining Capital Cost	$ / t_{ore}	0.50
Gold (Au) Price	USD / oz	1,450
Gold Royalty	$ / oz	43.29
Off-site costs	$ / oz	7.00

Optimisation Process

To optimise the Kenge deposit, a series of nested pit shells have been calculated over a range of Revenue Factors (RFs). Each of the nested pit shells are generated based on the maximum undiscounted cash flow calculated for the applicable RF. The generated nested pit shells will increase in size as the RF and maximum undiscounted cash flow also increase.

To determine the optimum pit shell and for reporting purposes within Whittle, the reported cash flow for RF=1 has been used.

As part of the optimisation process, Whittle uses the pit tonnages from nested pits and calculates the cashflow based on RF=1. Therefore, nested pit shells generated for a RF less than 1 will have cash flows greater than those used to determine the physical nested pit shell. Nested pit shells generated at a RF greater than 1 will have cash flows less (even negative) than those used to determine the physical nested pit shell. This is because material is mined (in the larger pits) that is economic when the original RF is applied; however, when Revenue Factors greater than 1 are used, some material within the pit becomes uneconomic, thus reducing the cashflow of that pit shell.

Optimisation Results

Kenge

Subsequent to completion of the pit design and scheduling process described in the following sections, SRK discovered an error within the Whittle optimisation process whereby the mining cost function was not correctly applied for the Kenge deposit. Further investigation revealed that the result of this was that a larger pit was selected as the basis for the pit design work than if the Revenue Factor 1 pit was chosen. SRK has undertaken preliminary analysis and does not consider the effect material to the project value.

Table 16-3 tabulates the results of the optimisation process for Kenge.

Table 16-3: Kenge Optimisation Results (Base Case)

Pit	Cashflow	Ore Tonnes	Waste Tonnes	Au Grade
	($M)	(Mt)	(Mt)	(g/t)
30	87.2	3.0	7.9	1.30
31	87.3	3.0	7.9	1.29
32	87.4	3.0	8.1	1.28
33	87.5	3.1	8.2	1.27
34	87.7	3.2	8.7	1.26
35	87.7	3.2	8.9	1.25
36	87.7	3.3	9.2	1.25
37	87.7	3.3	9.4	1.24
38	87.6	3.4	9.7	1.23
39	87.6	3.4	10.0	1.22
40	84.9	4.1	20.5	1.26
41	84.1	4.3	22.4	1.25
42	84.1	4.3	22.6	1.25
43	83.8	4.3	23.0	1.25
44	82.9	4.5	24.5	1.25
45	82.6	4.5	24.8	1.24
46	82.1	4.6	25.4	1.24
47	81.6	4.6	26.1	1.24
48	81.0	4.6	26.8	1.24
49	80.1	4.7	27.8	1.24
50	79.1	4.8	28.9	1.24
51	79.0	4.8	29.0	1.24
52	78.5	4.8	29.5	1.24
53	77.4	4.9	30.6	1.24
54	76.0	5.0	31.9	1.24
55	75.4	5.0	32.2	1.23
56	74.7	5.0	32.9	1.23
57	74.4	5.1	33.1	1.23
58	74.0	5.1	33.3	1.23
59	73.2	5.1	33.9	1.23
60	70.8	5.2	35.7	1.22

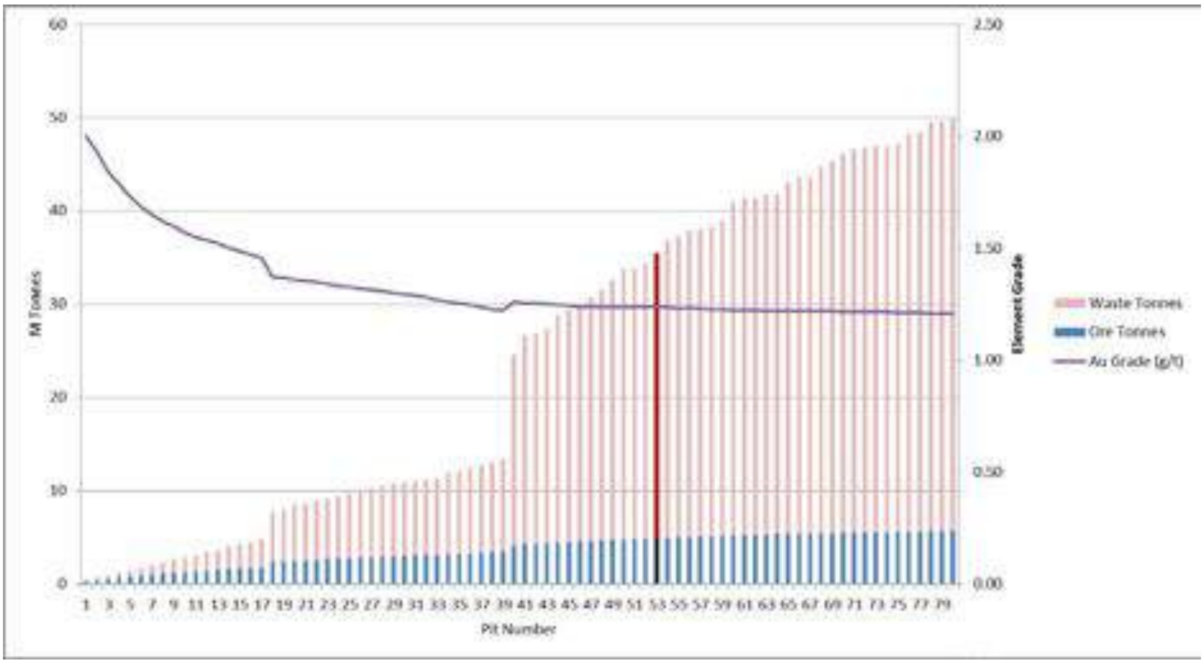

Figure 16-1: Kenge Optimisation Results – Au Grade

Figure 16-2: Kenge Optimisation Results – Cashflow

Mbenge

Table 16-4 tabulates the results of the optimisation process for Mbenge.

Table 16-4: Mbenge Optimisation Results (Base Case)

Pit	Cashflow	Ore Tonnes	Waste Tonnes	Au Grade
	($M)	(Mt)	(Mt)	(g/t)
21	59.3	1.6	3.6	1.47
22	59.3	1.6	3.6	1.47
23	60.2	1.6	4.0	1.47
24	60.2	1.6	4.1	1.47
25	60.5	1.7	4.3	1.46
26	60.7	1.7	4.5	1.46
27	60.7	1.7	4.5	1.46
28	61.0	1.7	4.7	1.46
29	61.0	1.7	4.8	1.46
30	61.1	1.7	4.9	1.46
31	61.3	1.8	5.4	1.45
32	61.3	1.8	5.5	1.45
33	61.3	1.8	5.6	1.45
34	61.1	1.8	5.9	1.44
35	61.1	1.8	6.0	1.44
36	61.0	1.8	6.0	1.44
37	61.0	1.8	6.1	1.44
38	61.0	1.8	6.1	1.44
39	61.0	1.8	6.1	1.44
40	61.0	1.8	6.1	1.44
41	60.7	1.9	6.4	1.43
42	60.4	1.9	6.7	1.43
43	60.4	1.9	6.7	1.43
44	60.2	1.9	6.8	1.43
45	60.1	1.9	6.8	1.43

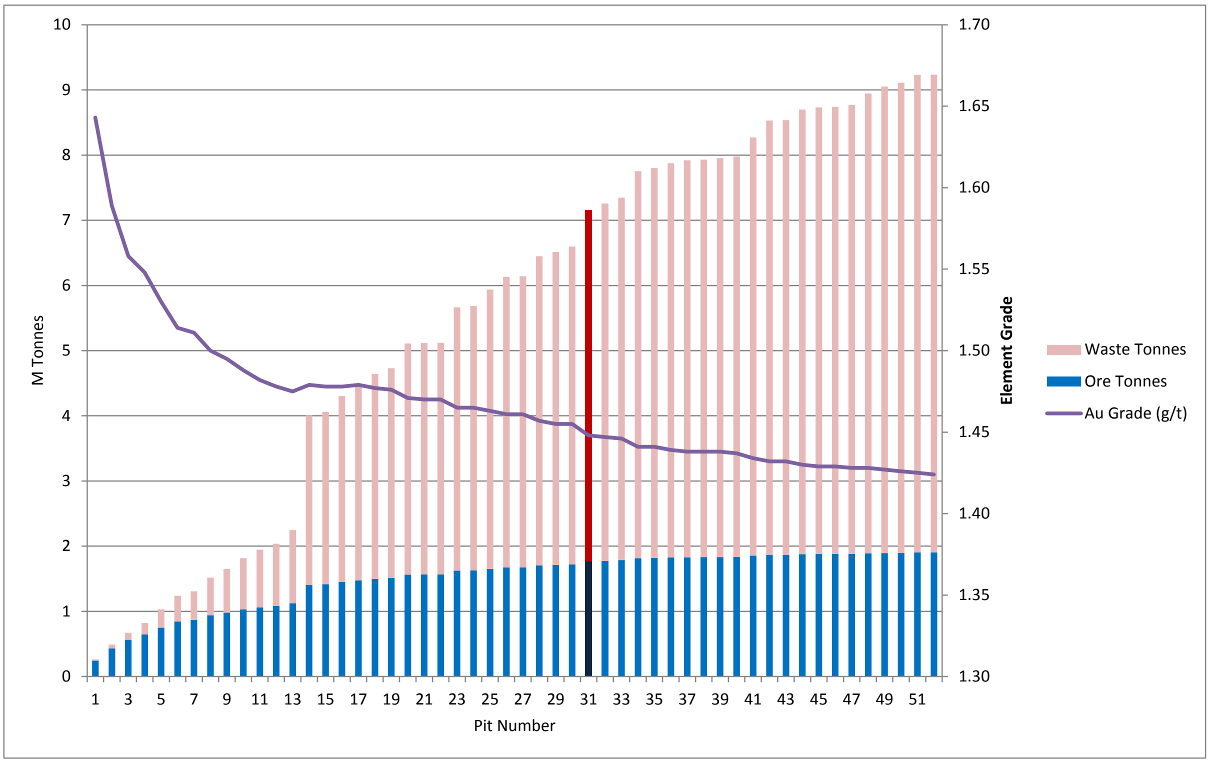

Figure 16-3: Mbenge Optimisation Results – Au Grade

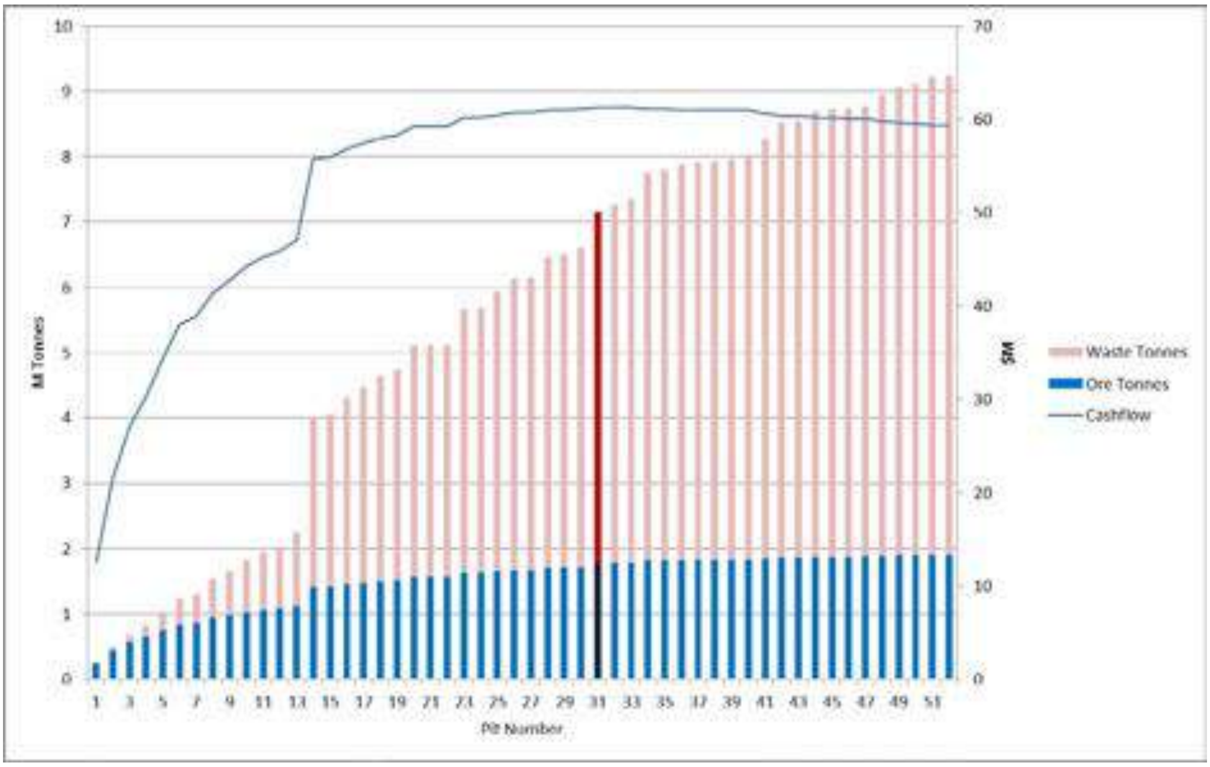

Figure 16-4: Mbenge Optimisation Results – Cashflow

Porcupine

Table 16-5 tabulates the results of the optimisation process for Porcupine.

Table 16-5: Porcupine Optimisation Results (Base Case)

Pit	Cashflow	Ore Tonnes	Waste Tonnes	Au Grade
	($M)	(Mt)	(Mt)	(g/t)
15	133.5	2.7	4.9	1.71
16	140.0	3.0	6.1	1.66
17	140.6	3.0	6.2	1.66
18	146.6	3.3	7.5	1.62
19	146.8	3.4	7.6	1.62
20	149.8	3.5	8.3	1.59
21	152.7	3.7	9.3	1.57
22	156.4	4.0	10.6	1.54
23	157.8	4.1	11.0	1.52
24	159.3	4.2	11.7	1.52
25	174.6	5.4	20.3	1.47
26	176.5	5.5	21.5	1.47
27	177.7	5.6	22.3	1.46
28	179.2	5.8	23.5	1.44
29	180.3	6.0	24.2	1.42
30	182.1	6.3	26.3	1.41
31	182.8	6.4	27.4	1.41
32	183.7	6.6	28.9	1.40
33	183.9	6.7	29.4	1.39
34	184.5	7.0	31.9	1.38
35	184.6	7.0	32.2	1.38
36	184.6	7.1	32.6	1.38
37	184.6	7.2	33.4	1.37
38	184.5	7.3	34.1	1.37
39	184.4	7.3	34.4	1.37
40	183.7	7.5	36.2	1.36
41	183.6	7.5	36.4	1.36
42	182.9	7.6	38.0	1.36
43	182.6	7.7	38.4	1.35
44	181.4	7.8	40.1	1.35
45	180.7	7.9	41.2	1.34

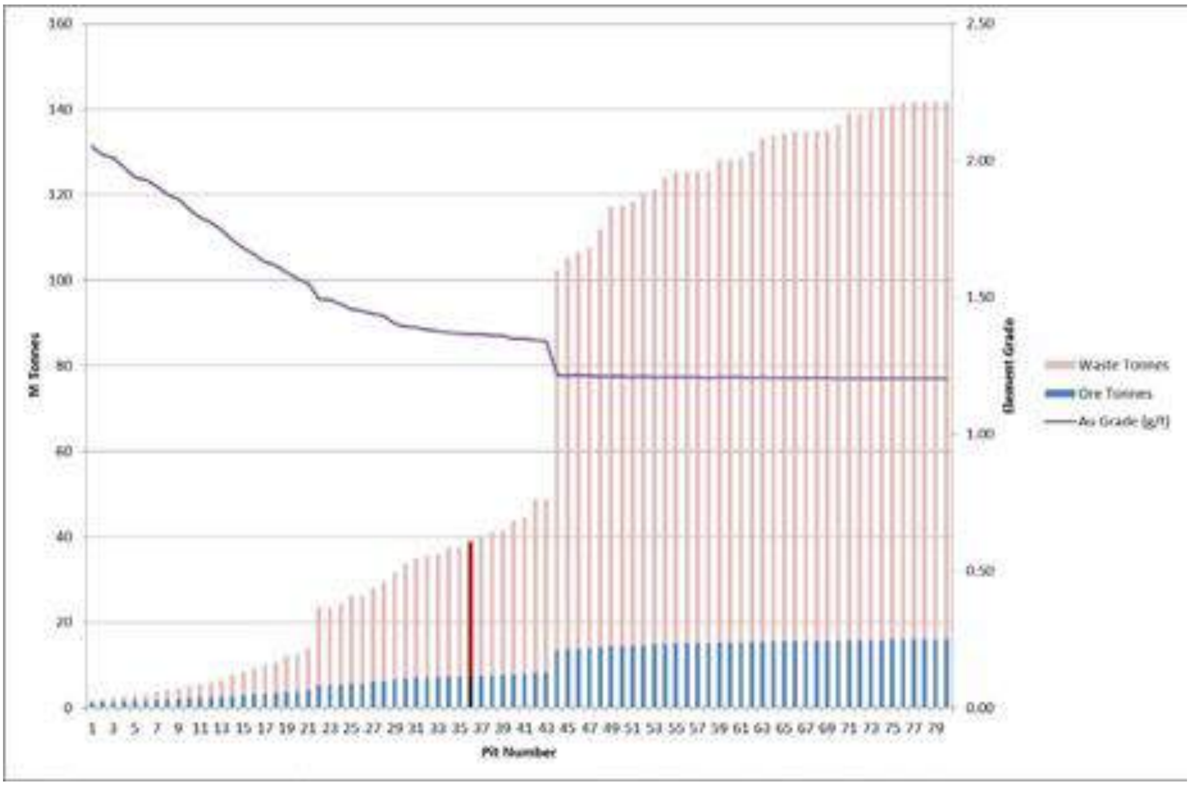

Figure 16-5: Porcupine Optimisation Results – Au Grade

Figure 16-6: Porcupine Optimisation Results – Cashflow

Optimisation Sensitivities

Kenge

As part of the optimisation process, the sensitivity of the Kenge deposit to gold price, mining cost and processing cost has been conducted and presented in Figure 16-7, Figure 16-8 and Figure 16-9.

Table 16-6 tabulates the Kenge sensitivities.

Table 16-6: Kenge Sensitivities (Base Case)

Scenario	Pit Shell	Ore Tonnes	Waste Tonnes	Strip Ratio	Au Grade
		(Mt)	(Mt)	(W:O)	(g/t)
Base Case	53	4.9	30.6	6.3	1.24
Au Price $1100/ oz	53	3.3	9.4	2.9	1.25
Au Price $1200/ oz	53	4.3	22.5	5.3	1.25
Au Price $1300/ oz	53	4.5	25.4	5.6	1.24
Base Case	53	4.9	30.6	6.3	1.24
Au Price $1600/ oz	53	5.2	35.7	6.8	1.22
Au Price $1700/ oz	53	5.3	36.9	6.9	1.22
Au Price $1800/ oz	53	5.5	39.8	7.3	1.21
Processing Cost * 50%	53	5.2	33.9	6.5	1.22
Processing Cost * 75%	53	5.1	32.4	6.4	1.23
Base Case	53	4.9	30.6	6.3	1.24
Processing Cost * 125%	53	4.8	29.3	6.2	1.25
Processing Cost * 150%	53	4.5	26.8	6.0	1.27
Mining Cost * 50%	53	5.9	49.1	8.4	1.21
Mining Cost * 75%	53	5.4	39.1	7.3	1.22
Base Case	53	4.9	30.6	6.3	1.24
Mining Cost * 125%	53	4.5	24.1	5.4	1.24
Mining Cost * 150%	53	3.5	10.0	2.8	1.21

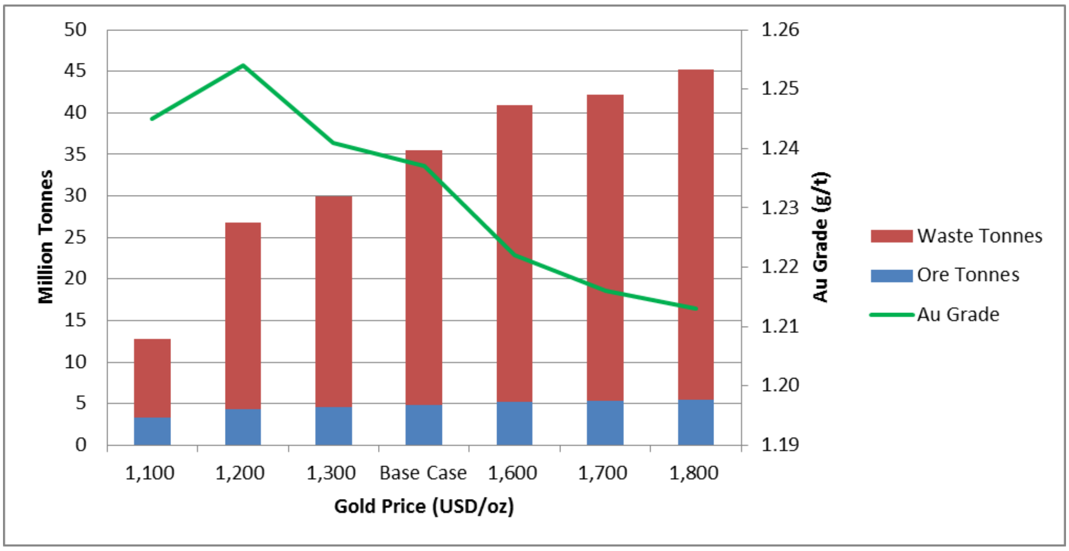

Figure 16-7: Kenge Gold Price Sensitivity

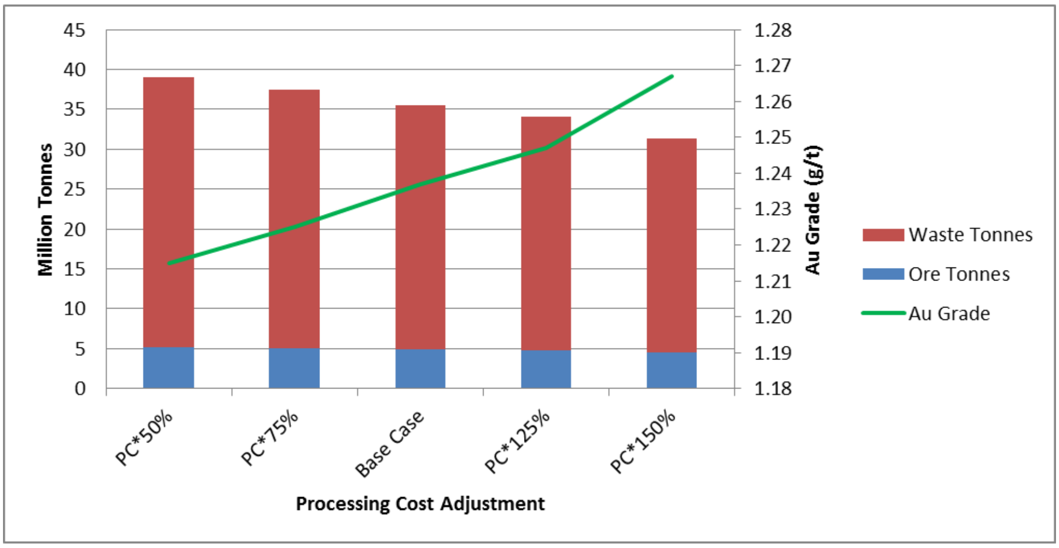

Figure 16-8: Kenge Processing Cost Sensitivity

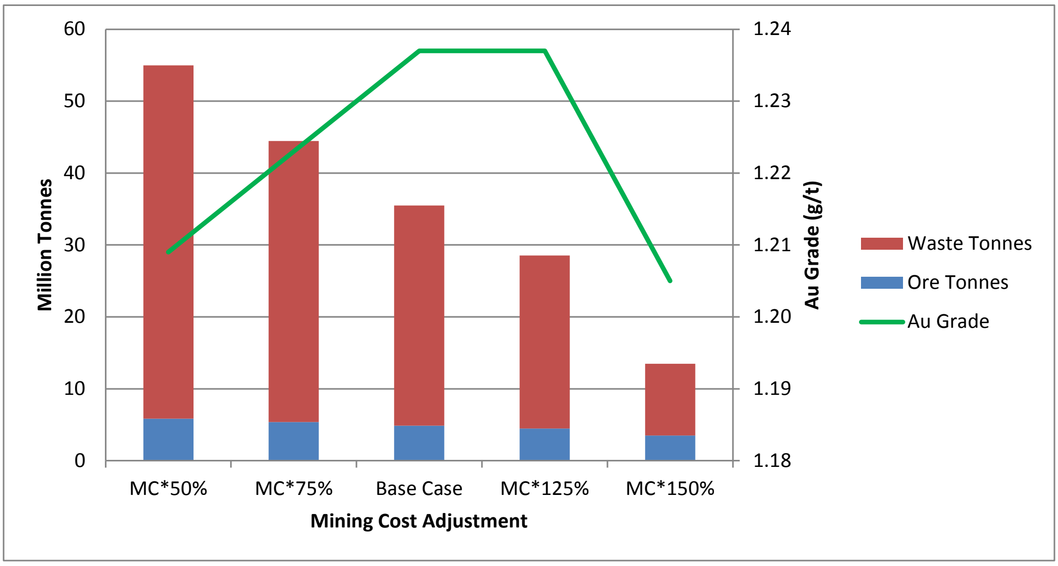

Figure 16-9: Kenge Mining Cost Sensitivity

Mbenge

As part of the optimisation process, the sensitivity of the Mbenge deposit to gold price, mining cost and processing cost has been conducted and presented in Figure 16-10, Figure 16-11 and Figure 16-12.

Table 16-7 tabulates the Mbenge sensitivities.

Table 16-7: Mbenge sensitivities (Base Case)

Scenario	Pit Shell	Ore Tonnes	Waste Tonnes	Strip Ratio	Au Grade
		(Mt)	(Mt)	(W:O)	(g/t)
Base Case	31	1.8	5.4	3.04	1.45
Au Price $1100/ oz	34	1.6	3.8	2.39	1.47
Au Price $1200/ oz	34	1.7	4.5	2.67	1.46
Au Price $1300/ oz	34	1.7	4.8	2.8	1.46
Base Case	31	1.8	5.4	3.04	1.45
Au Price $1600/ oz	30	1.8	5.6	3.11	1.45
Au Price $1700/ oz	29	1.8	6.0	3.31	1.44
Au Price $1800/ oz	28	1.8	6.1	3.33	1.44
Processing Cost * 50%	28	1.8	5.6	3.11	1.45
Processing Cost * 75%	30	1.8	5.5	3.08	1.45
Base Case	31	1.8	5.4	3.04	1.45
Processing Cost * 125%	34	1.8	5.4	3.04	1.45
Processing Cost * 150%	34	1.7	4.9	2.86	1.46
Mining Cost * 50%	33	1.9	6.8	3.63	1.43
Mining Cost * 75%	31	1.8	6.1	3.33	1.44
Base Case	31	1.8	5.4	3.04	1.45
Mining Cost * 125%	31	1.7	4.8	2.8	1.46
Mining Cost * 150%	33	1.6	4.0	2.48	1.46

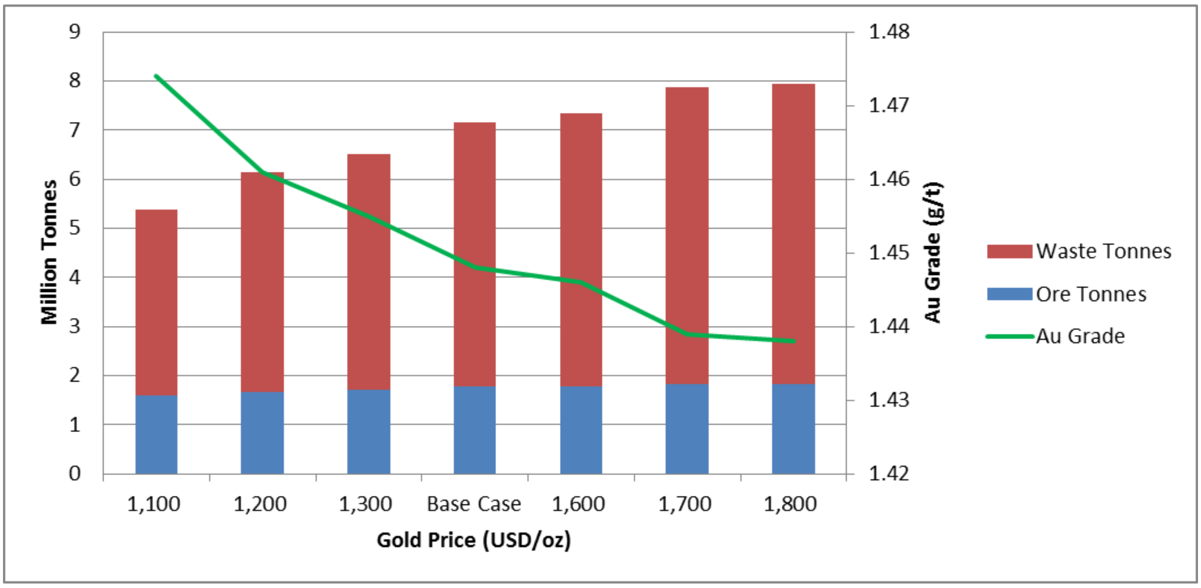

Figure 16-10: Mbenge Gold Price Sensitivity

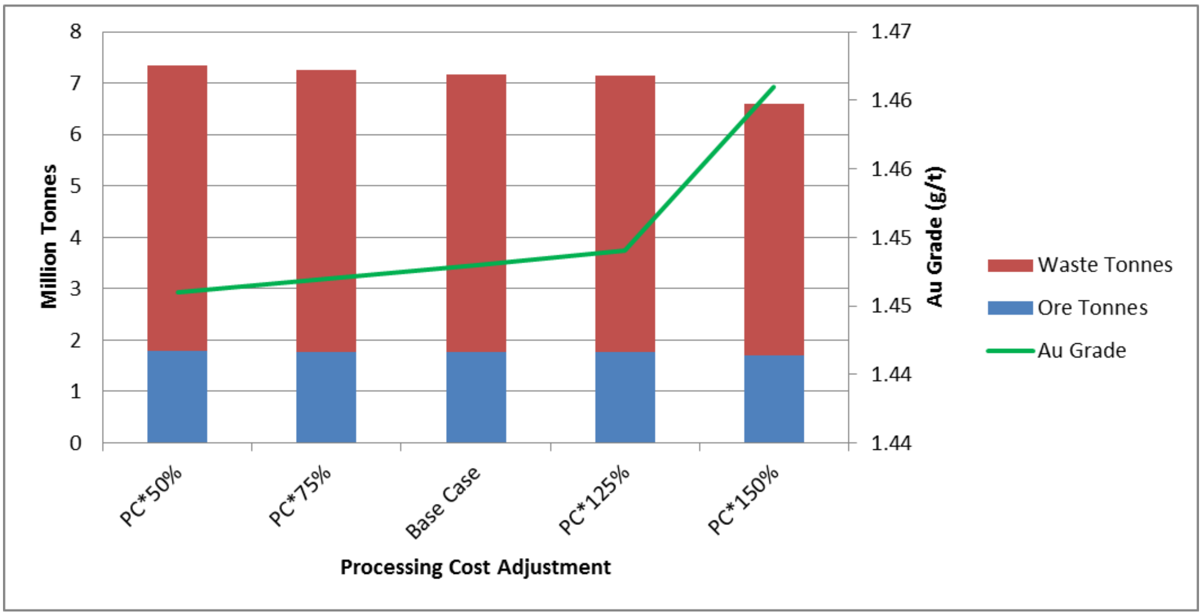

Figure 16-11: Mbenge Processing Cost Sensitivity

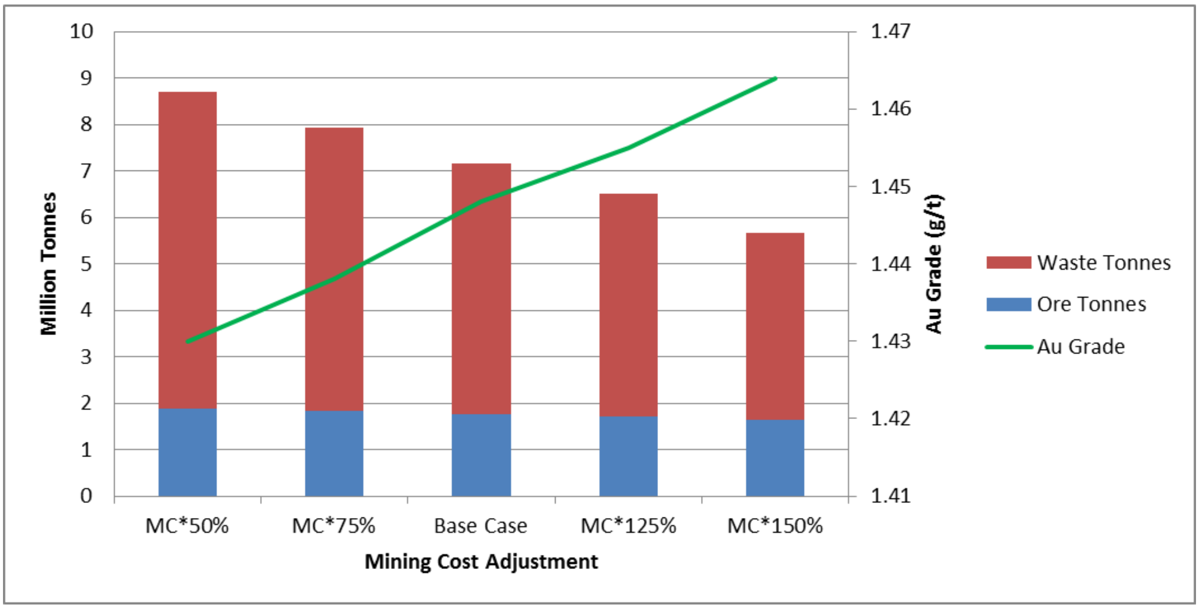

Figure 16-12: Mbenge Mining Cost Sensitivity

Porcupine

As part of the optimisation process, the sensitivity of the Porcupine deposit to gold price, mining cost and processing cost has been conducted and presented in Figure 16-13, Figure 16-14 and Figure 16-15.

Table 16-8 tabulates the Porcupine sensitivities.

Table 16-8: Porcupine Sensitivities (Base Case)

Scenario	Pit Shell	Ore Tonnes	Waste Tonnes	Strip Ratio	Au Grade
		(Mt)	(Mt)	(W:O)	(g/t)
Base Case	36	7.1	32.6	4.6	1.38
Au Price $1100/ oz	36	4.2	11.7	2.8	1.52
Au Price $1200/ oz	36	5.7	22.5	3.96	1.45
Au Price $1300/ oz	36	6.4	26.9	4.23	1.41
Base Case	36	7.1	32.6	4.6	1.38
Au Price $1600/ oz	36	7.6	37.5	4.95	1.36
Au Price $1700/ oz	36	7.9	41.1	5.18	1.35
Au Price $1800/ oz	36	12.9	92.6	7.17	1.21
Processing Cost * 50%	36	7.6	37.1	4.87	1.35
Processing Cost * 75%	36	7.3	34.4	4.71	1.37
Base Case	36	7.1	32.6	4.6	1.38
Processing Cost * 125%	36	6.9	31.3	4.56	1.39
Processing Cost * 150%	36	6.4	28.1	4.35	1.41
Mining Cost * 50%	36	14.7	119.3	8.11	1.21
Mining Cost * 75%	36	8.1	43.7	5.4	1.35
Base Case	36	7.1	32.6	4.6	1.38
Mining Cost * 125%	36	6.2	25.1	4.04	1.41
Mining Cost * 150%	36	5.4	20.0	3.69	1.45

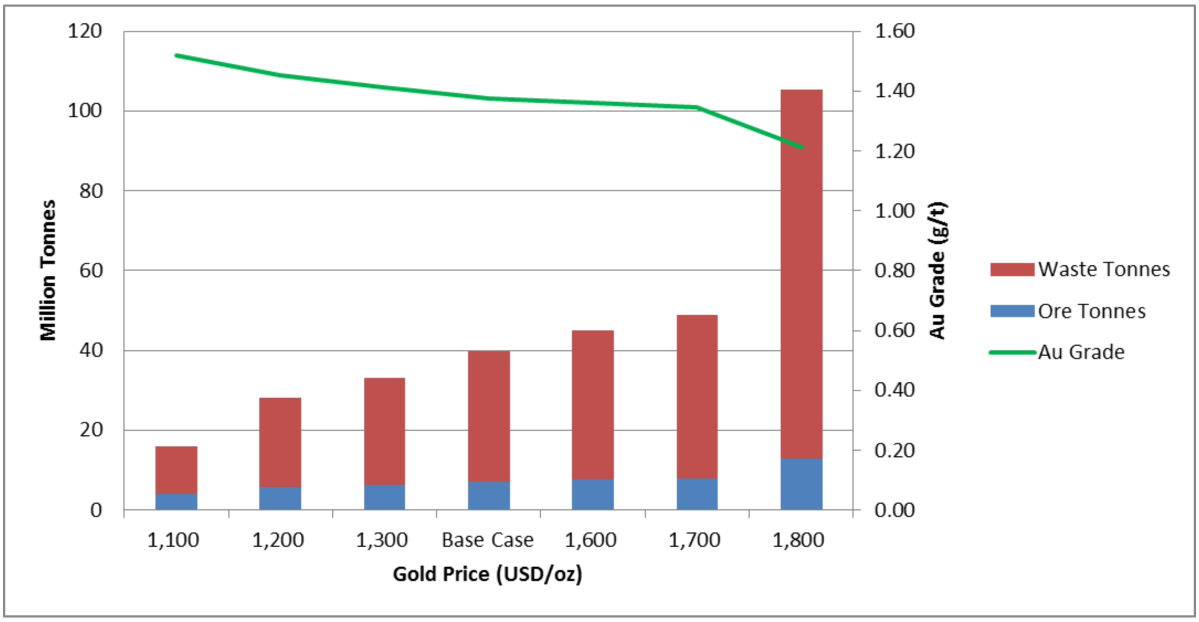

Figure 16-13: Porcupine Gold Price Sensitivity

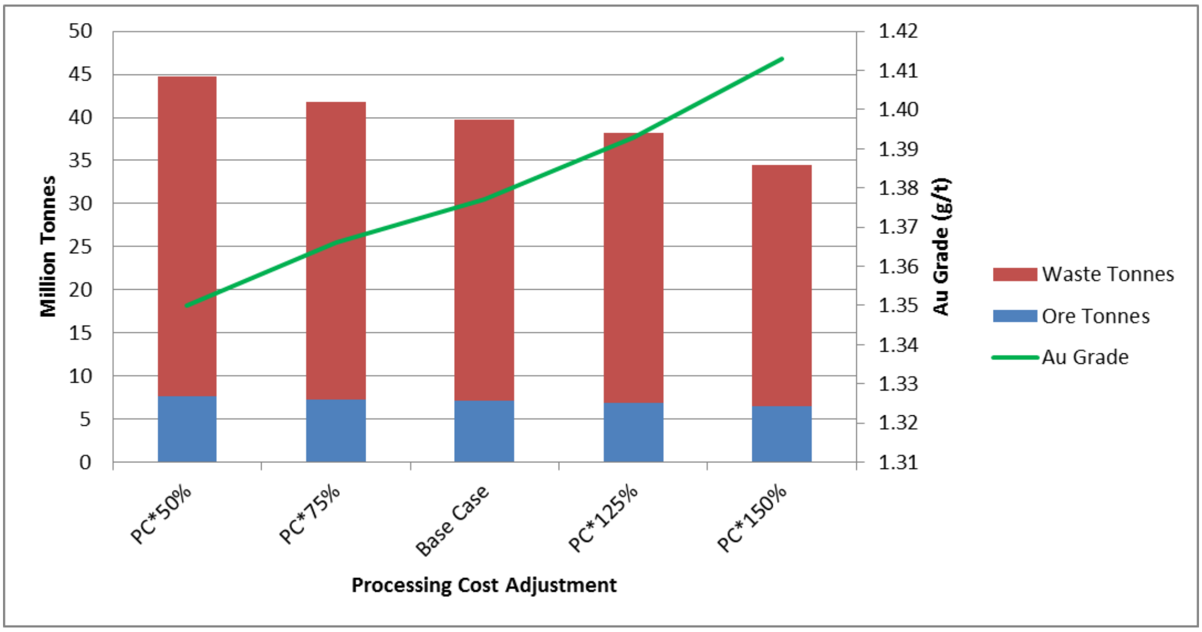

Figure 16-14: Porcupine Processing Cost Sensitivity

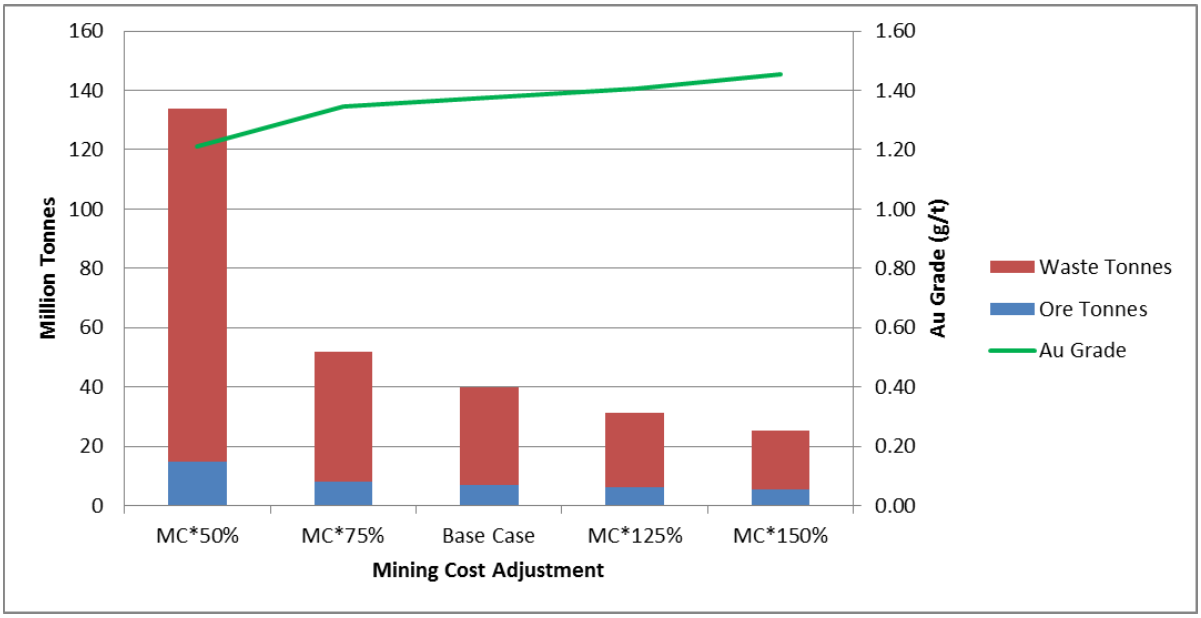

Figure 16-15: Porcupine Mining Cost Sensitivity

16.2.2 Mine Design

Design Parameters

Table 16-9 summarises the parameters used for the pit designs. Due to the lack of geotechnical data, a series of assumptions have been made.

Further, it has been assumed there are no structural influences on the design parameters (e.g. bedding, dip etc).

Table 16-9: Pit Design Parameters (Base Case)

Parameter	Units	Kenge	Mbenge	Porcupine
Material Type		Fresh	Fresh	Fresh
Maximum Ramp Width	(m)	25	25	25
Bench Height (Kenge)	(m)	12.5	10.0	10.0
Batter Angle	(^{o})	70	70	70
Berm Width	(m)	10.0	8.0	8.0
Number of ramps per wall		1.5	1.5	3.5

Design Results

Kenge

Figure 16-16 graphically shows the design conformance with the optimised shell.

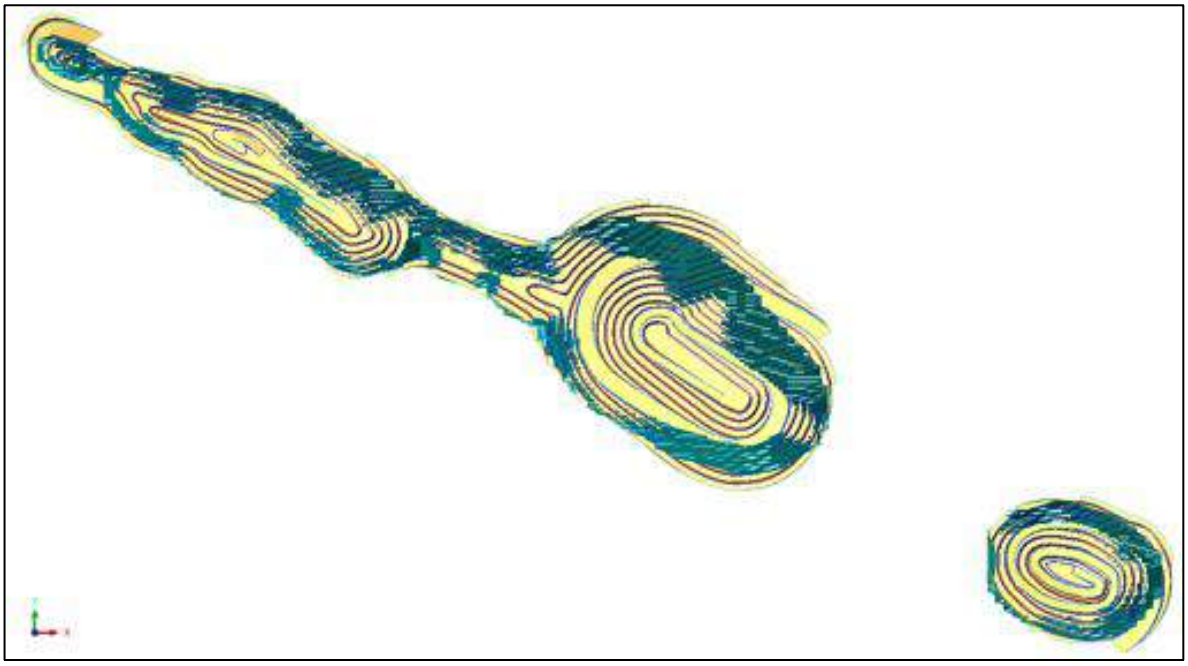

Figure 16-16: Kenge Mine Design (tan) conformance with Optimised Shell (teal)

Table 16-10 tabulates the designed pit conformance with the optimised shell.

Table 16-10: Kenge Designed Pit Conformance with Optimised Shell

	Whittle Tonnes	Designed Pit Tonnes	Conformance
Waste	31,057,172	31,503,531	101%
Indicated	2,270,989	1,589,628	70%
Measured	2,174,233	1,918,856	88%
Total	35,502,394	35,012,015	99%
Ounces	194,475	152,720	79%

Figure 16-17 and Figure 16-18 shows the Kenge pit design.

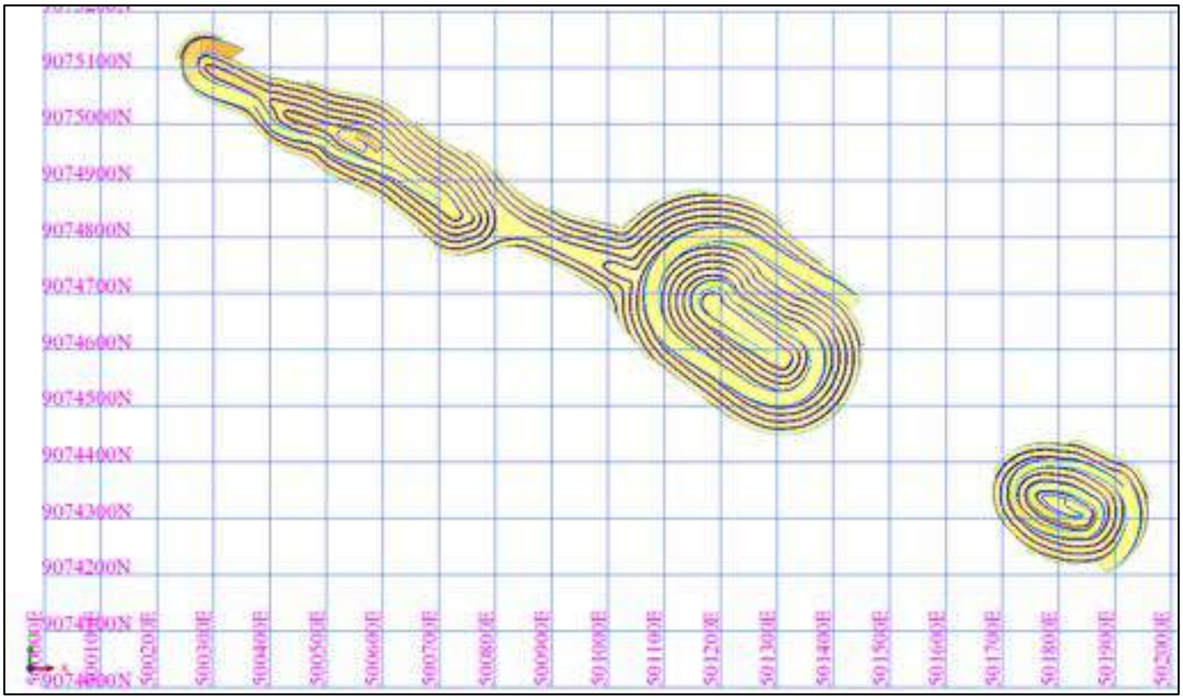

Figure 16-17: Kenge Mine Design – Plan View

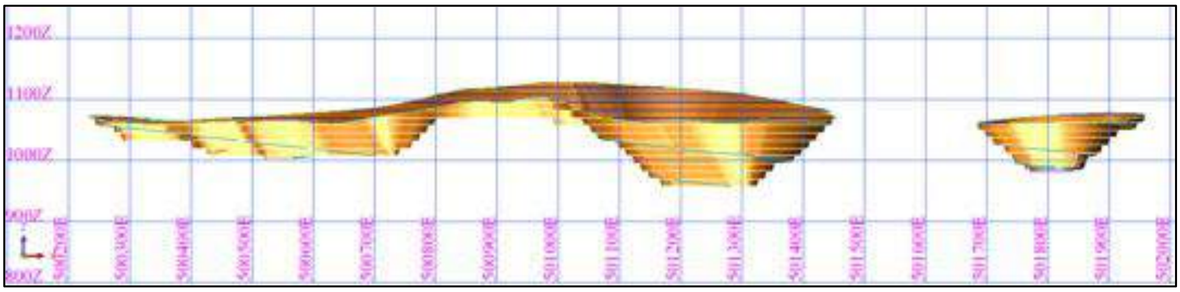

Figure 16-18: Kenge Mine Design – Looking North

Mbenge

Figure 16-19 graphically shows the design conformance with the optimised shell.

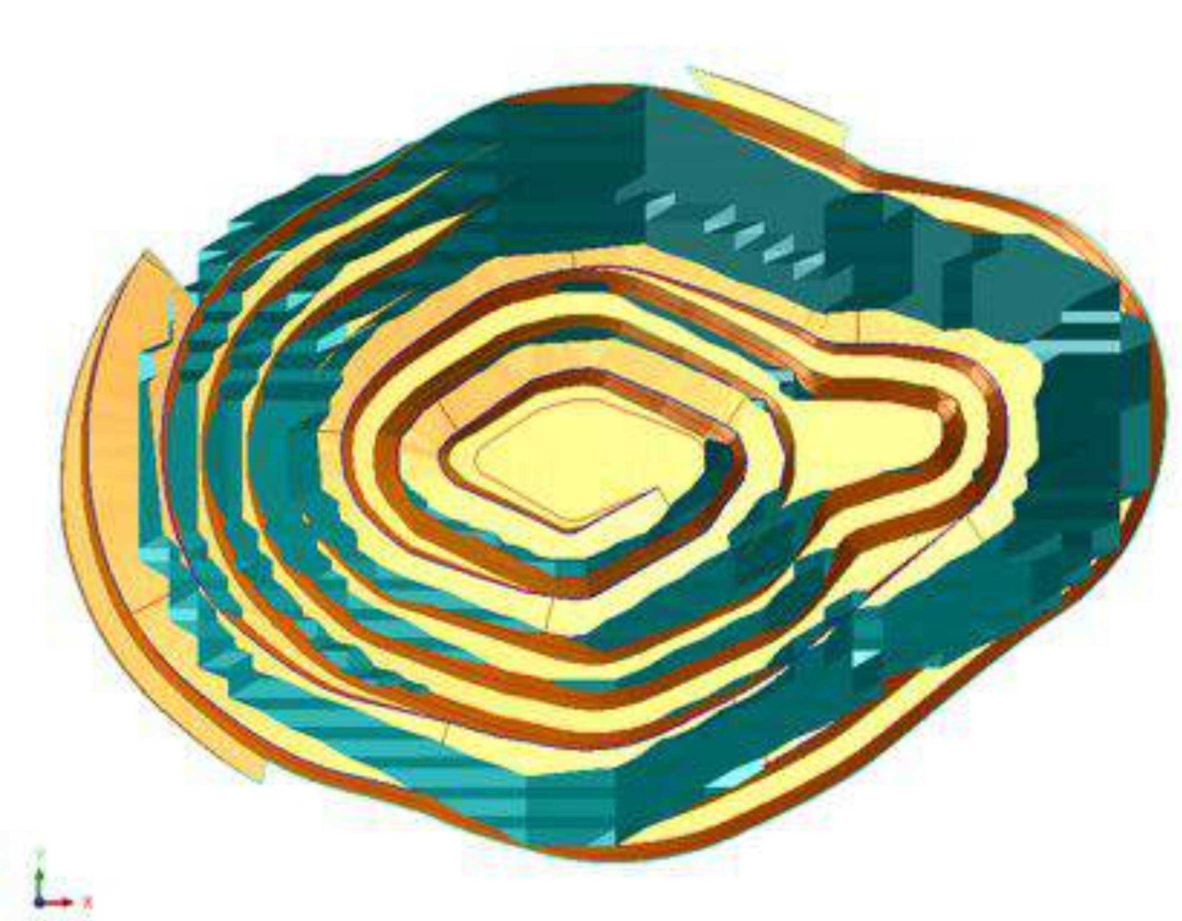

Figure 16-19: Mbenge Mine Design (tan) conformance with Optimised Shell (teal)

Table 16-11 tabulates the designed pit conformance with the optimised shell.

Table 16-11: Mbenge Designed Pit Conformance with Optimised Shell

	Optimised Tonnes	Designed Pit Tonnes	Conformance
Waste	6,522,801	7,163,038	110%
Indicated	1,681,894	1,503,711	89%
Total	8,204,695	8,666,749	106%
Ounces	84,896	77,836	92%

Figure 16-20 and Figure 16-21 shows the Mbenge pit design.

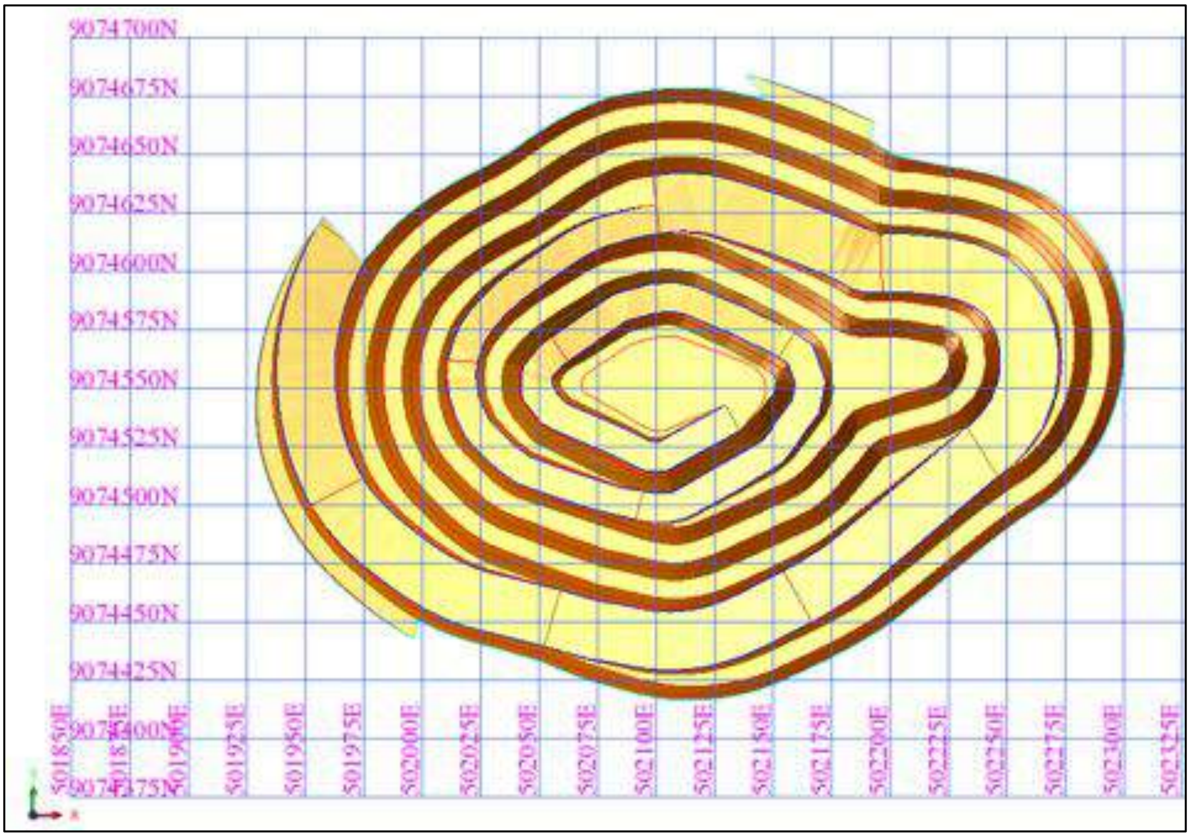

Figure 16-20: Mbenge Mine Design – Plan View

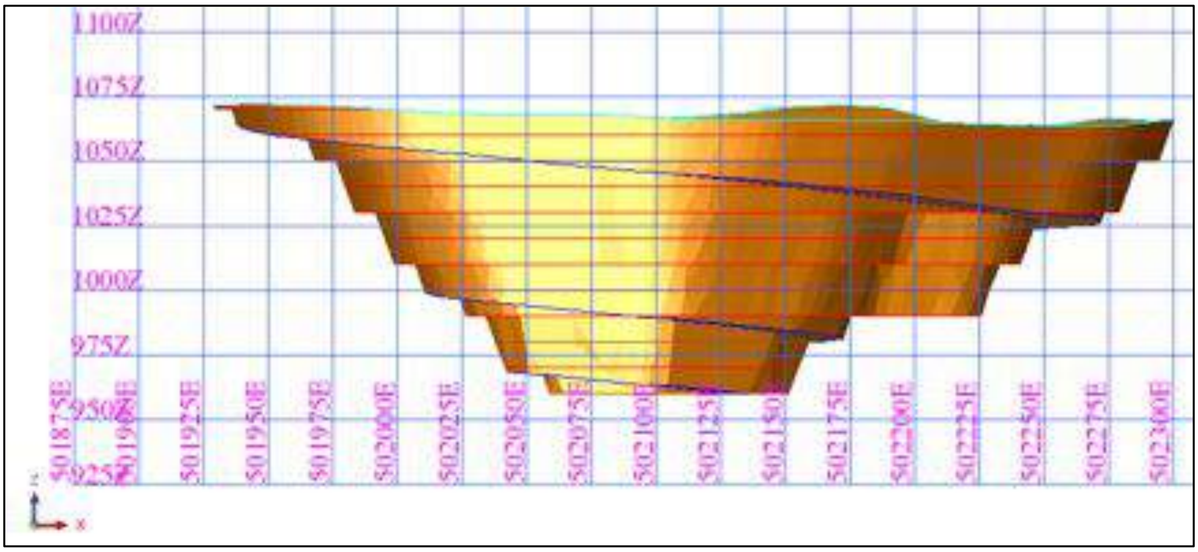

Figure 16-21: Mbenge Mine Design – Looking North

Porcupine

Figure 16-22 graphically shows the design conformance with the optimised shell.

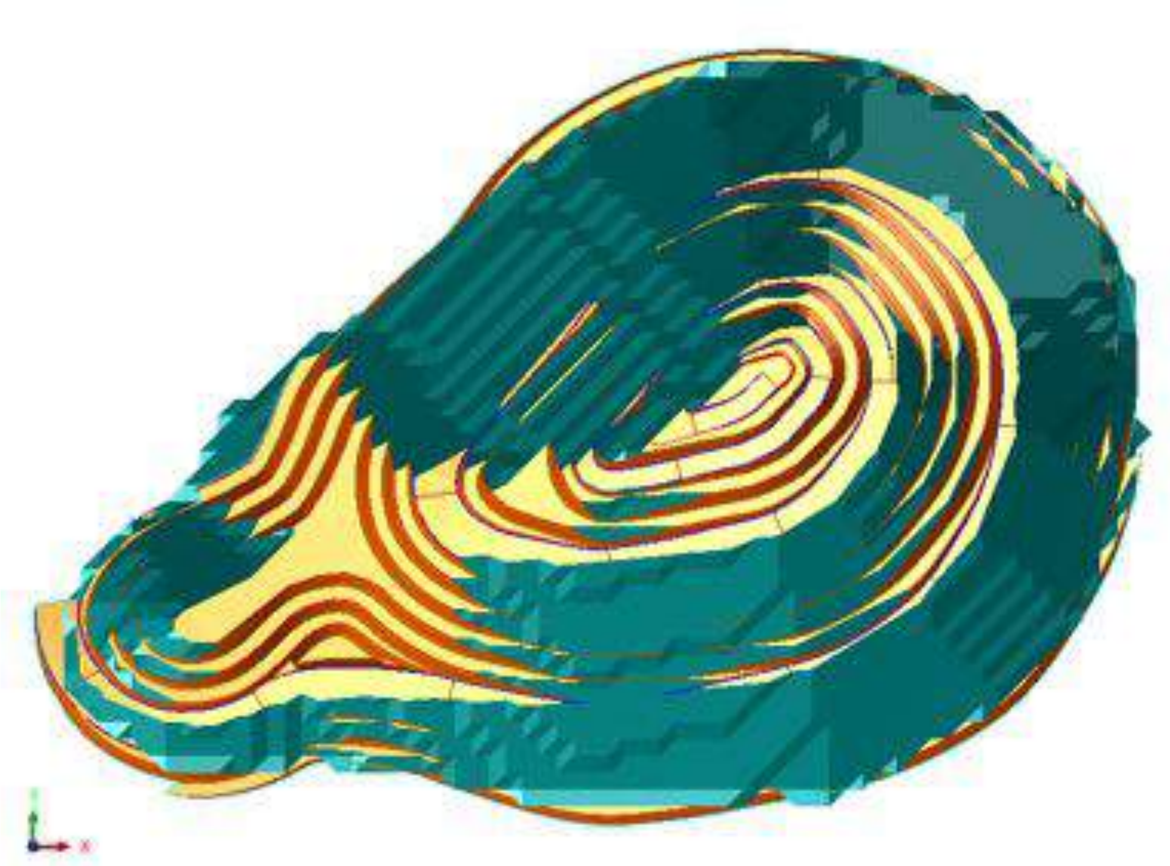

Figure 16-22: Porcupine Mine Design (tan) conformance with Optimised Shell (teal)

Table 16-12 tabulates the designed pit conformance with the optimised shell.

Table 16-12: Porcupine Designed Pit Conformance with Optimised Shell

	Optimised Tonnes	Designed Pit Tonnes	Conformance
Waste	31,742,363	34,754,088	109%
Indicated	157,800	164,375	104%
Measured	6,537,523	6,016,454	92%
Total	38,437,686	40,934,917	106%
Ounces	324,281	304,969	94%

Figure 16-23 and Figure 16-24 shows the Porcupine pit design.

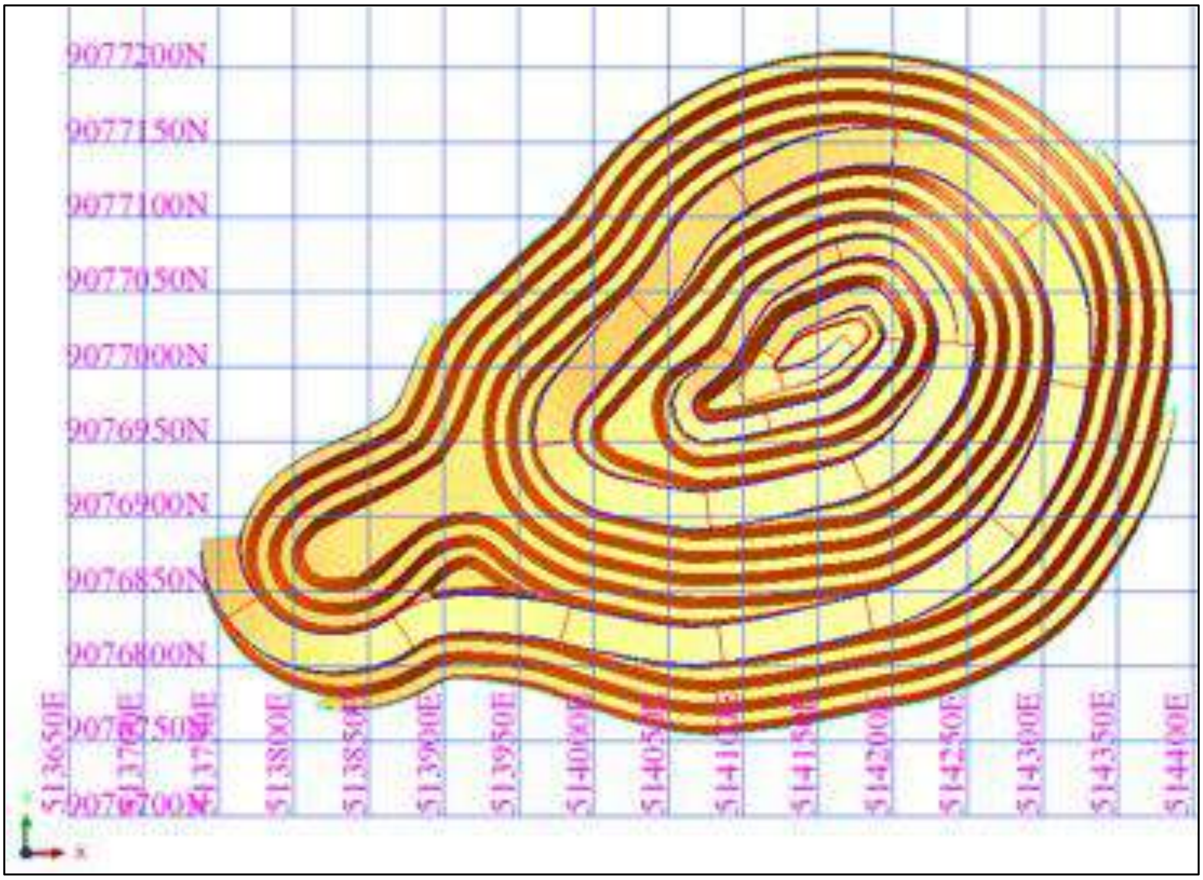

Figure 16-23: Porcupine Mine Design – Plan View

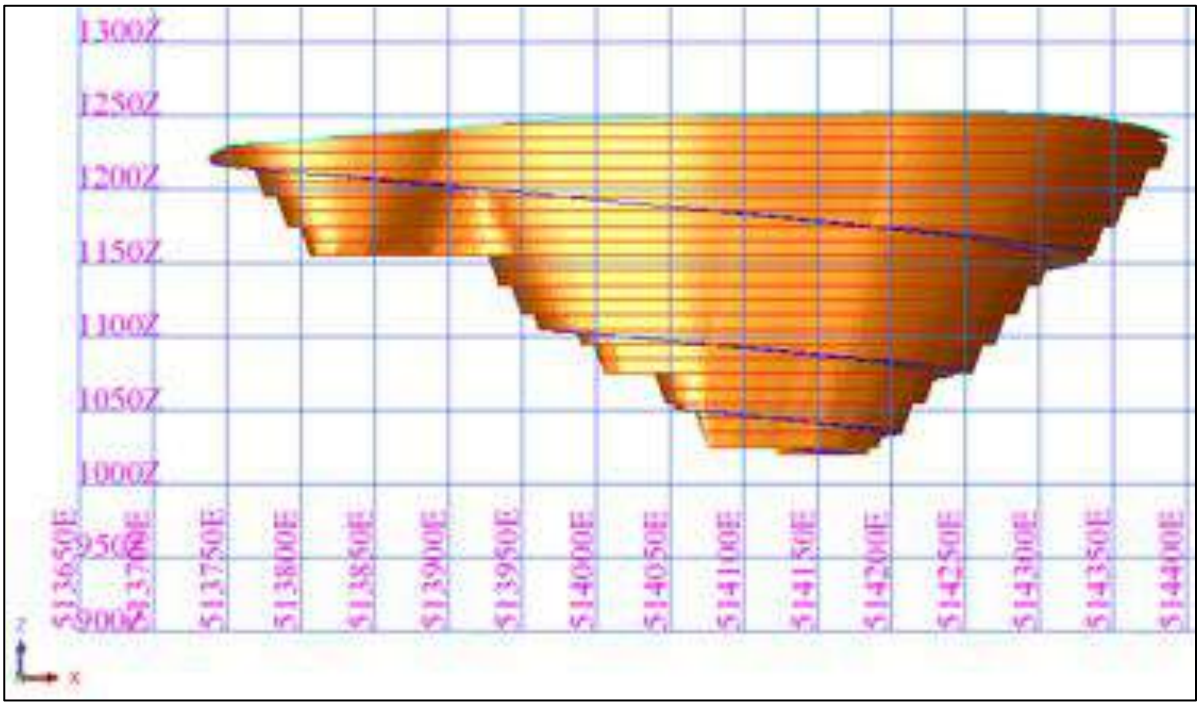

Figure 16-24: Porcupine Mine Design – Looking North

16.2.3 Waste Rock Dump Design

Two waste dump designs have been generated for the SMP deposits. The following criteria have been used as part of the design process for both dump designs:

- 250 m standoff from pit design crest;
- Ramp width of 25 m;
- Batter angle of 37 degrees;
- Berm width of 45.8 m; and
- Target buffer capacity between 5-15%.

The aim of the dump design was to minimise the footprint on the surrounding vegetation whilst meeting the storage requirements and adhering to the design criteria.

Kenge Waste Dump

A single waste dump has been designed capable of storing all waste material from both the Kenge and Mbenge deposits.

Table 16-13 shows the inputs used for calculating the volume requirements for the Kenge waste dump.

Table 16-13: Kenge Waste Dump (Base Case)

Pit	Units	Value
Kenge	Mm^3	13.1
Mbenge	Mm^3	3.2
Total	**Mm^3**	**16.3**
Swell Factor		1.3
Required Dump Capacity	Mm^3	21.2
Designed Dump Capacity	Mm^3	22.6
Buffer Capacity		6%
Approximate Dump Height	m	80

Figure 16-25 shows the waste dump with the Kenge and Mbenge pit designs.

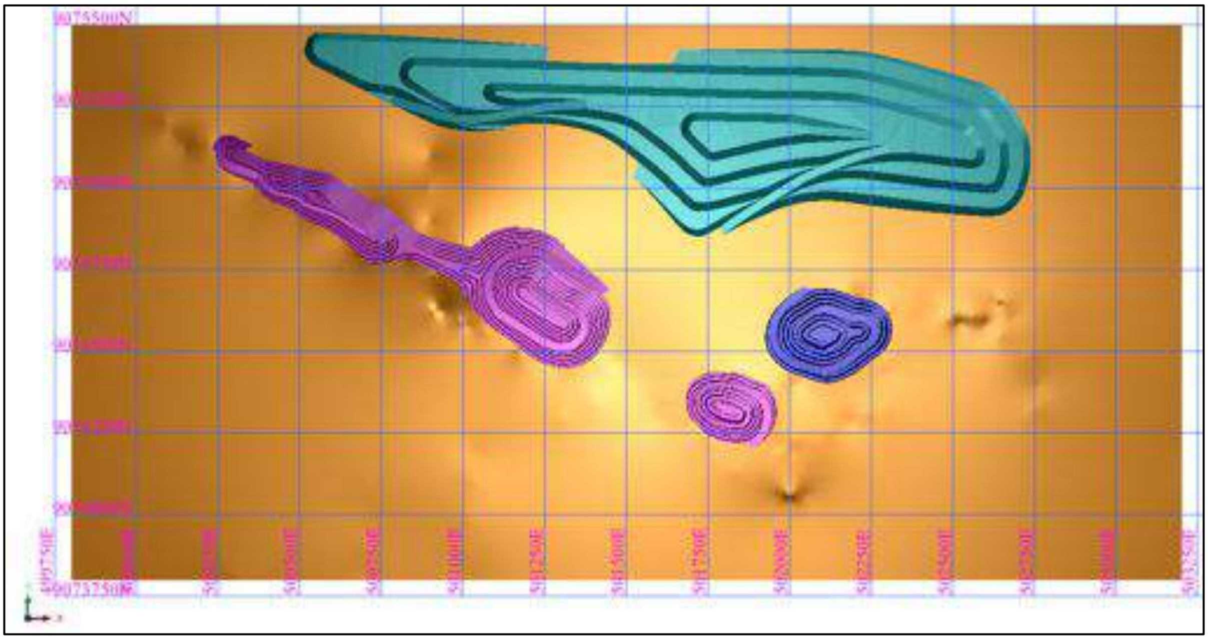

Figure 16-25: Kenge Waste Dump

The waste dump has been configured with a dual ramp access system to minimise the haulage requirements.

Porcupine Waste Dump

A single waste dump has been designed capable of storing all waste material from the Porcupine deposit.

Table 16-14 shows the inputs used for calculating the volume requirements for the Porcupine waste dump.

Table 16-14: Porcupine Waste Dump (Base Case)

Pit	Units	Value
Porcupine	Mm^3	15.5
Swell Factor		1.3
Required Dump Capacity	Mm^3	20.1
Design Dump Capacity	Mm^3	22.8
Buffer Capacity		13%
Approximate Dump Height	m	90

Figure 16-26 shows the waste dump with the Porcupine pit design.

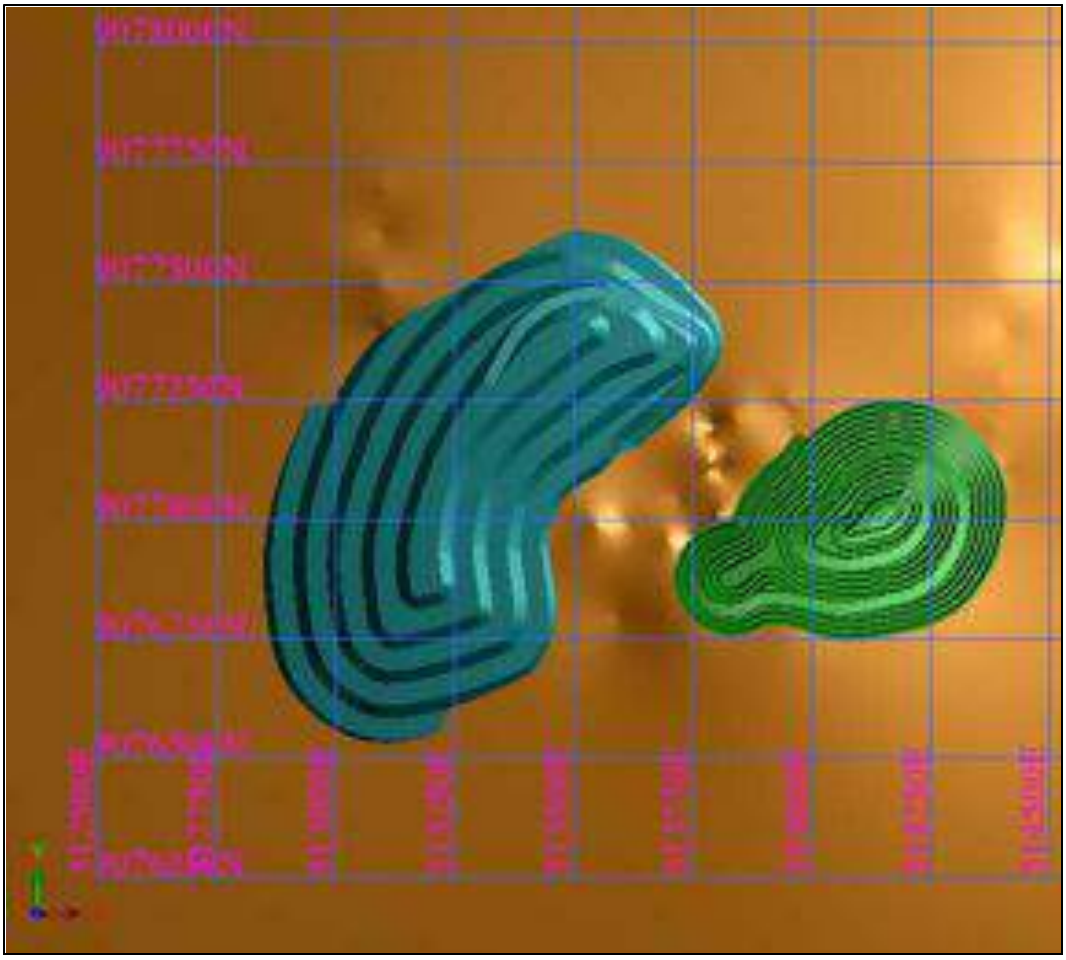

Figure 16-26: Porcupine Waste Dump

16.2.4 Mine Operations

All earthmoving operations (including drill and blast) on site are planned to be completed with an earthmoving contractor.

Drill and Blast Operations

Drill and blast activities will be undertaken on a standard 10 m bench height for Porcupine and Mbenge, with Kenge to be designed using a 12.5 m bench height. This is to match the standard mining unit within the geological block model.

Drillholes with a diameter of 6" for ore material and 9" for waste material is recommended where possible and a drill capable of drilling both these diameter holes is required.

The drill subdrill, burden and spacing will vary according to the material type, required powder factor and explosives to be used.

Load and Haul Operations

A single excavator has been planned for both ore and waste loading activities. The ore mining process is expected to be reasonably selective. This reflects the mineralisation style. To accommodate the required selectivity, it is envisaged a backhoe configuration will be used, in conjunction with geological grade control assistance.

A diesel mechanical drive 100 t truck is recommended as the base case haulage unit. This type of haulage unit has a proven performance record and is relatively easy to maintain in remote locations.

Where possible, it is expected that material will be dumped directly from the haulage unit into the process facility crushing system. Where this isn't possible, the trucks will dump ore at a stockpile which will later be fed into the crushing system by a front end loader.

For this PEA, it has been assumed that all waste material will be dumped ex-pit at the designed waste dump.

Other Mining Activities

The mining contractor will supply all mining auxiliary and support services for the operation. Examples of services include:

- Loading unit support;
- Haul road maintenance;
- Bench maintenance and development;
- Pit dewatering (including pipe movement, trenching, joining);
- Batter development;
- Operational drill pad preparation;
- Waste dump maintenance and development;
- Ore stockpile maintenance; and
- General on-site earthworks.

16.2.5 Mine Production

A target of approximately 75,000 oz per annum of gold was targeted for production at the request of Helio. SRK agreed this was a sensible number to target for the PEA. No hard maximum material movement limit was placed on the schedule.

The operation is planned to be mined using a contract mining fleet, hence the fleet size was variable to minimise costs, whilst ensuring that a practical mining fleet configuration could be maintained.

Key features of the schedule include:

- Pits are to be mined in a general sequence of Porcupine, then Kenge followed by Mbenge;
- Maximum of two pits open at a single point in time to ensure supervisory commitments are manageable;
- First year scheduled in quarters, followed by annual increments;
- No ore is processed during the first year of the mining operation. This allows for the construction of the processing facility to occur whilst the mine is being developed. Any ore mined during this period will be stockpiled for feed into the plant once the plant is commissioned and operating. There is not expected to be any deterioration in ore quality during this stockpiling period;
- 800 kt ore is planned to be fed during the first year of the process facility's operation, 1.6 Mt ore to be fed during the second year of operation which is the planned maximum feed rate for the process facility; and
- A Run of Mine ("ROM") stockpile is to be utilised to 'smooth' the production profile and feed rate to the process facility. The maximum size of the ROM stockpile is approximately 375 kt.

Table 16-15 and Figure 16-27 detail the mine production schedule.

Table 16-15: SMP combined Mine Production (Base Case)

Period	Q1 2014	Q2 2014	Q3 2014	Q4 2014	2015	2016	2017	2018
Ore Tonnes (Mt)	0	0	0	0.1	0.9	1.5	1.5	1.8
Waste Tonnes (Mt)	2.0	2.0	2.3	2.2	9.2	6.7	13.1	11.0
Total Tonnes (Mt)	2.0	2.0	2.3	2.3	10.0	8.2	14.6	12.8
Au Grade (g/t)	1.06	0.00	0.93	1.07	1.15	1.66	1.50	1.30

Period	2019	2020	2021	2022
Ore Tonnes (Mt)	1.7	1.6	1.3	0.7
Waste Tonnes (Mt)	9.2	5.7	6.3	3.7
Total Tonnes (Mt)	11.0	7.3	7.6	4.4
Au Grade (g/t)	1.37	1.51	1.41	1.32

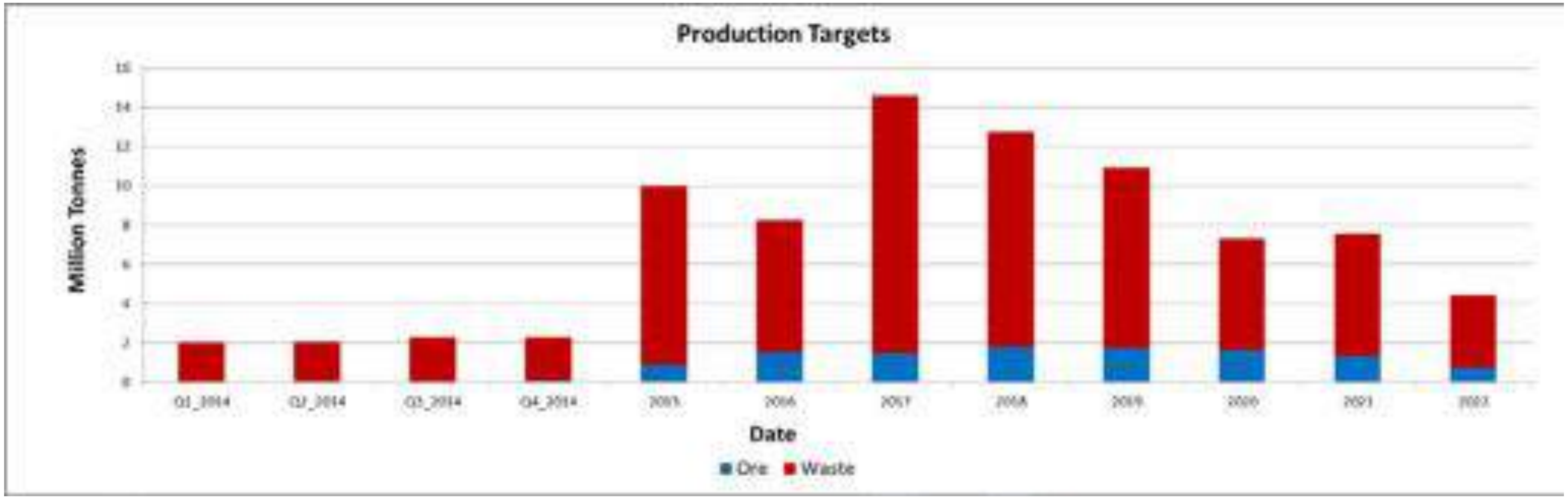

Figure 16-27: SMP Mine Production Schedule (Base Case)

Figure 16-28: SMP Mine Production by deposit (Base Case)

16.2.6 Fleet Requirements

A first principles approach was utilised to estimate the equipment requirements. The process involved measuring the required haul distances and calculating the dynamic productivities for each piece of equipment. To estimate the requirements for auxiliary equipment, experience from current operating mines of similar size and scale has been used.

Table 16-16 summarises the estimated equipment requirements.

Table 16-16: Equipment requirements (Base Case)

Mining Equipment	Type	Maximum Number
Primary Equipment		
Haul Truck	Cat 777 (100t)	11
Loaders	Ex 1900 (190 t machine)	3
Drills	6-9 inch diameter	2
Secondary Equipment	**(Auxiliary)**	
FEL – ROM ore pad	Cat 980 (260 kW engine)	1
Track Dozer	Cat D10 (634 kW engine)	1
Track Dozer	Cat D6 (433 kW engine)	1
Wheel Dozer	Cat 834 (674 kW engine)	1
Fuel Truck	~10,000 L capacity	1
Water Truck	Cat 785 (1082 kW engine)	1
Grader	Cat 16M (221 kW engine)	1
Tertiary Equipment	**(Auxiliary)**	
Pick-up trucks	4 x 4 troop carriers	10
Lighting plants		10
Service vehicles	(hi lift, forklift, IT24)	5

The equipment type nominated is indicative only. The specific equipment will be determined as part of further studies.

Figure 16-29 details the loading unit requirements by period.

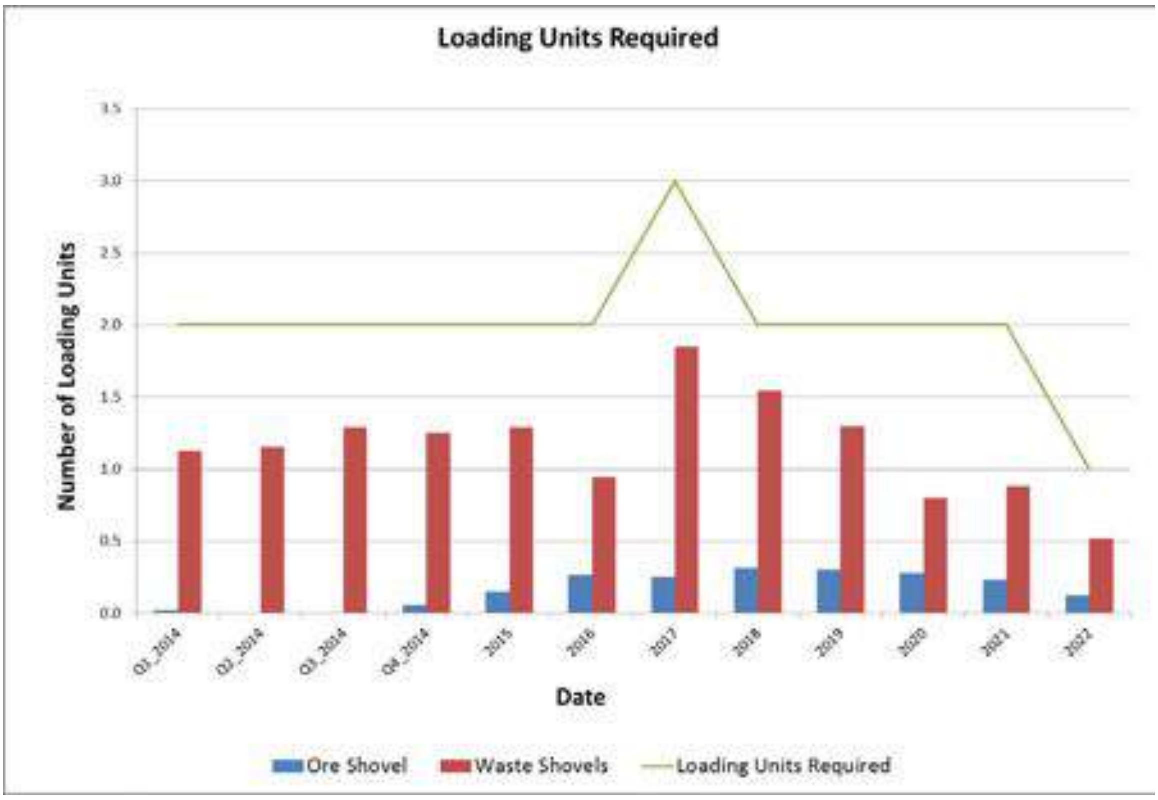

Figure 16-29: Loading Unit Requirements (Base Case)

Figure 16-30 details the trucking requirements by period.

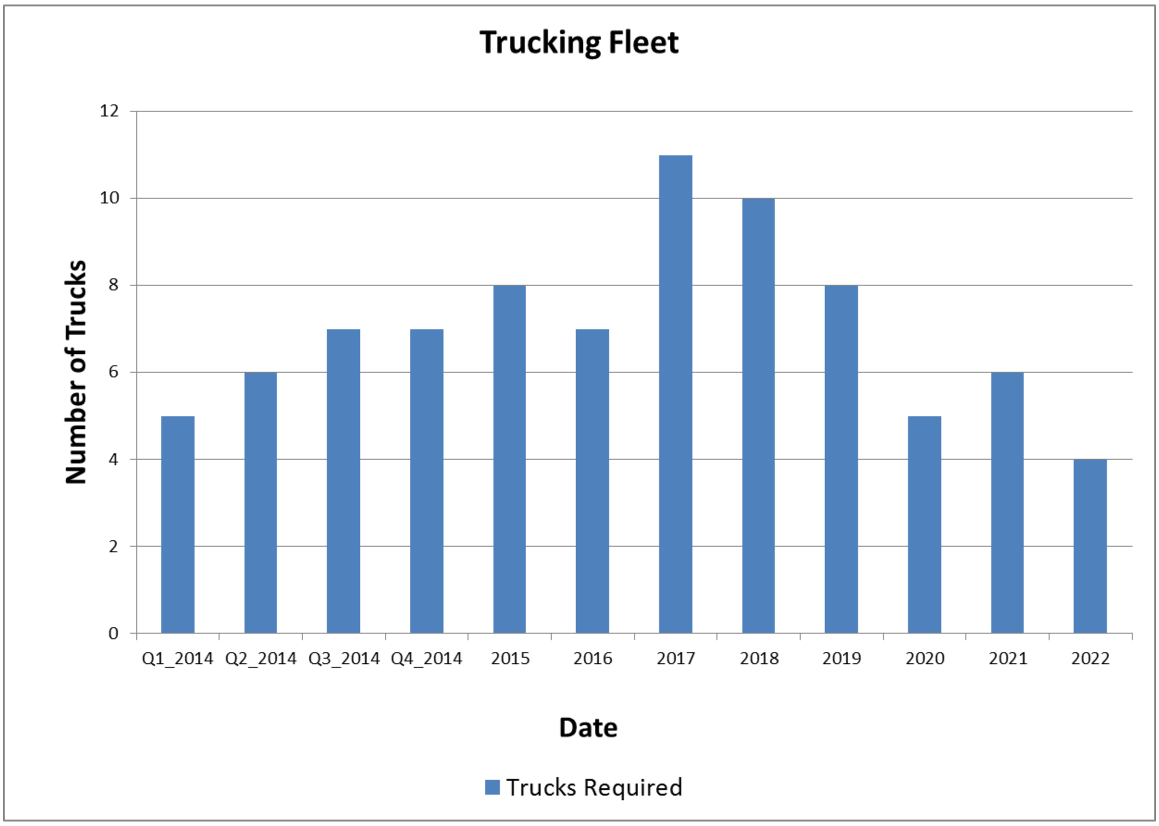

Figure 16-30: Truck Fleet Requirements (Base Case)

16.2.7 Labour Requirements

The site will be planned to operate 24 hours a day, 365 days a year.

The labour requirements have been developed using first principles and experience from other similarly sized operations. It is expected that the contractor will supply the workforce for the earthmoving activities. Nominally, it is expected the workforce will work a shift rotation of 2 weeks on 1 week off. This requires three operating shifts for the mine labour.

Figure 16-31 shows the estimated labour requirements for the mining operation throughout the life of the project.

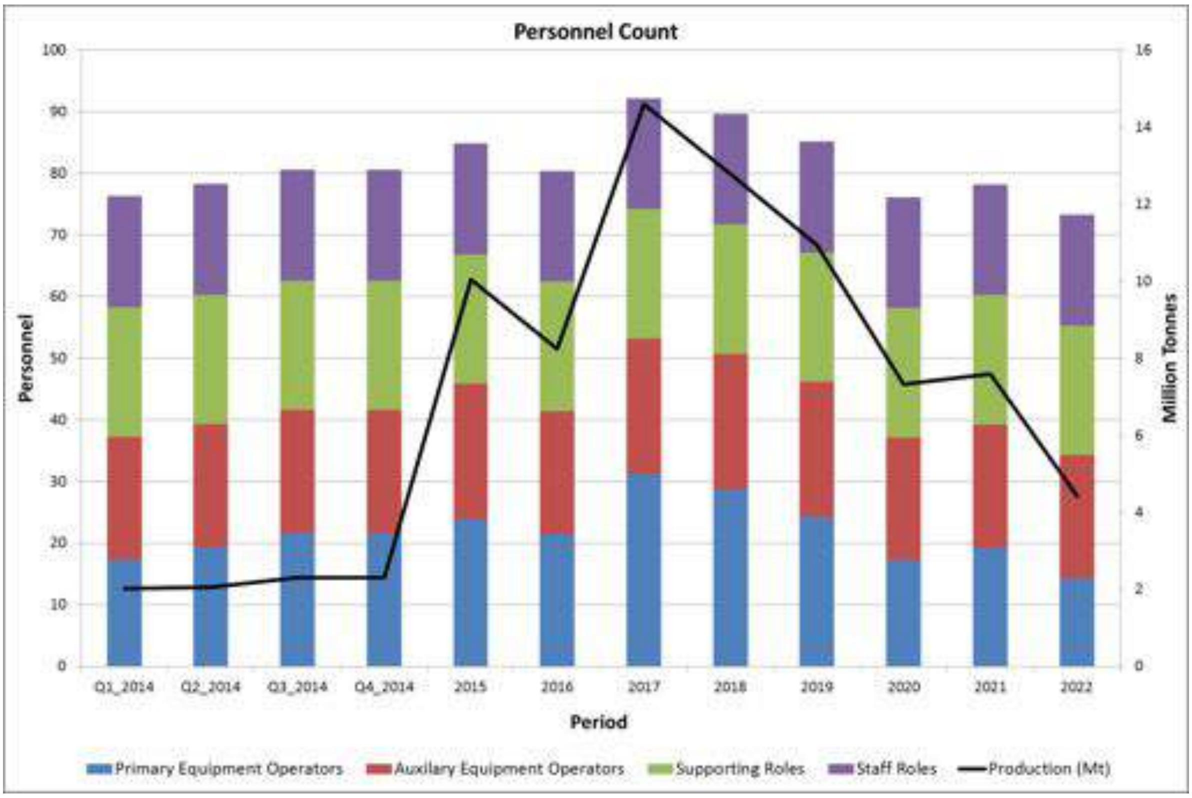

Figure 16-31: Mining Labour Requirements (Base Case)

16.2.8 Mining Cost Estimates

The estimated mining operating cost has been derived using first principles. The average mining cost for the life of the project is $2.29 / t total material mined, summarised in Table 16-17.

Table 16-17: Operating Cost Estimate (Base Case)

Parameter	Unit	Estimated Unit Cost
Equipment	USD / t	0.59
Fuel	USD / t	0.71
Over haulage	USD / t	0.08
Dewatering	USD / t	0.02
Labour	USD / t	0.31
Blasting	USD / t	0.20
Subtotal	**USD / t**	**1.91**
Contractor Premium (@20%)	USD / t	0.38
Total	**USD / t**	**2.29**

16.3 Upside Potential Case

SRK has prepared a scenario to understand the potential upside of the project, utilising wall angles of 55°.

The operation is planned to run for 10 years which includes an initial year of prestrip. During this initial year, it is expected all required infrastructure will be constructed onsite. The operation will process 2.4 million tonnes per annum (Mtpa) with a maximum total material movement of 17.4 Mtpa (ore and waste tonnes). Throughout the project approximately 109 people are planned to be employed as part of the mining operation.

Table 16-18 summarises the paramters utilised for the Upside Potential Case.

Table 16-18: Upside Potential Optimisation Parameters

Parameter	Unit	Value
Mining Dilution	%	10
Mining Dilution Grade		0.00
Mining Recovery	%	100
Overall Slope Angle (Kenge)	(o)	55
Overall Slope Angle (Mbenge)	(o)	55
Overall Slope Angle (Porcupine)	(o)	55
Mining Cost	$ / t	2.75
Incremental Mining Cost (for depth)	$ / 10 m	0.05
Mining Rate	Mtpa	unlimited
Processing Rate	Mtpa	1.850
Process Recovery Au	%	95
Processing Costs Au	$ / tore	10.00
General and Administration	$ / tore	5.00
Sustaining Capital Cost	$ / tore	0.50
Gold (Au) Price	USD / oz	1,450
Gold Royalty	$ / oz	43.29
Off-site costs	$ / oz	7.00
Discount Rate	%	10

SRK notes the only difference with the Base Case optimisation parameters is the overall slope angle.

16.3.1 Optimisation Results

Kenge

Table 16-19 tabulates the results of the optimisation process for Kenge.

Table 16-19: Kenge Optimisation Results (Upside Potential Case)

Pit	Cashflow ($M)	Ore Tonnes (Mt)	Waste Tonnes (Mt)	Au Grade (g/t)
21	85.8	2.6	5.3	1.36
22	87.0	2.6	5.8	1.35
23	87.3	2.7	5.9	1.34
24	87.7	2.7	6.0	1.34
25	88.4	2.8	6.3	1.33

Pit	Cashflow ($M)	Ore Tonnes (Mt)	Waste Tonnes (Mt)	Au Grade (g/t)
26	89.0	2.8	6.7	1.32
27	89.3	2.9	6.8	1.32
28	90.2	3.0	7.3	1.29
29	90.4	3.1	7.4	1.28
30	90.5	3.1	7.5	1.27
31	90.9	3.2	8.0	1.27
32	94.6	3.8	16.2	1.31
33	94.8	3.9	16.4	1.29
34	95.1	4.0	17.4	1.29
35	95.5	4.3	20.2	1.28
36	95.5	4.3	20.5	1.28
37	95.5	4.3	20.9	1.28
38	95.4	4.4	21.5	1.27
39	95.4	4.4	21.6	1.27
40	95.3	4.5	21.7	1.27
41	94.9	4.5	22.8	1.27
42	94.9	4.6	22.8	1.27
43	94.4	4.7	23.4	1.25
44	94.2	4.7	23.7	1.25

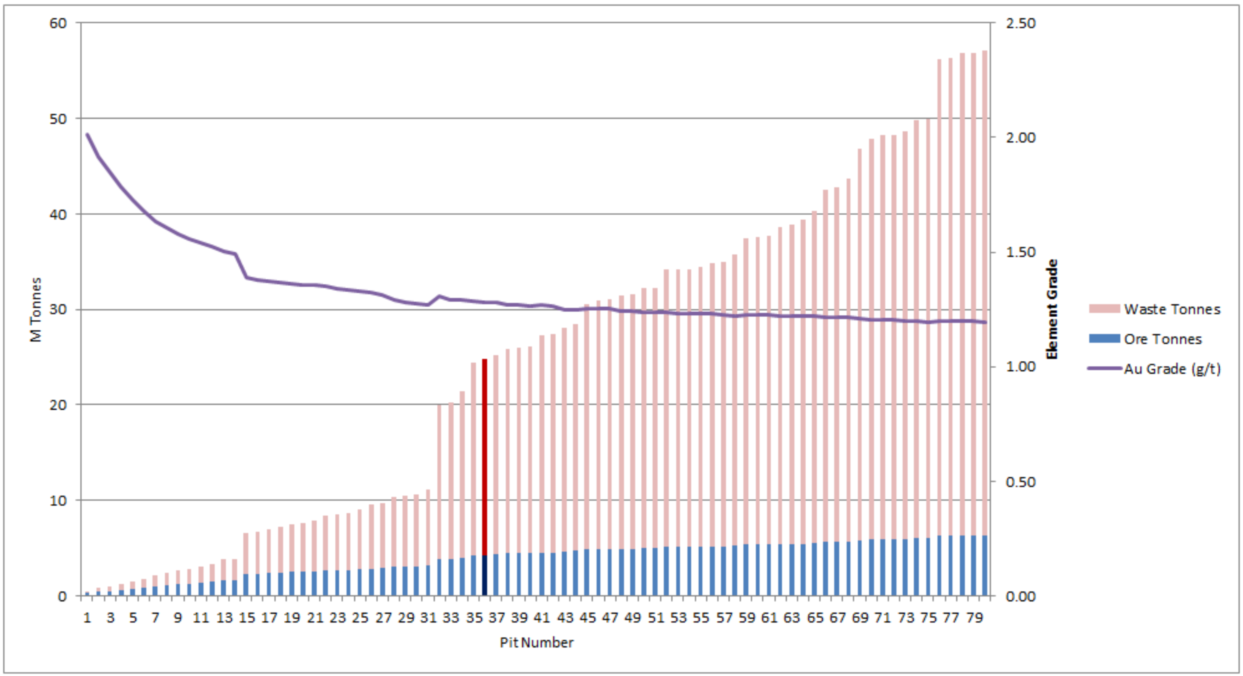

Figure 16-32: Kenge Optimisation Results – Au Grade

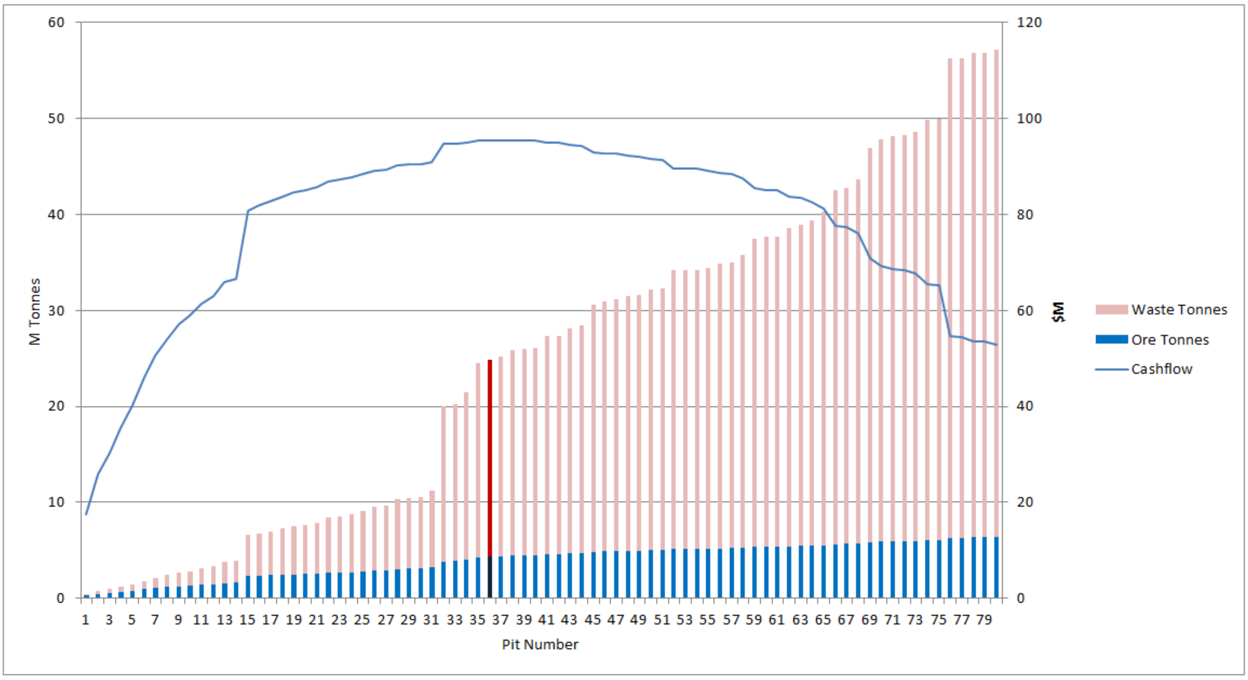

Figure 16-33: Kenge Optimisation Results – Cashflow

Mbenge

Table 16-20 tabulates the results of the optimisation process for Mbenge.

Table 16-20: Mbenge Optimisation Results (Upside Potential Case)

Pit	Cashflow	Ore Tonnes	Waste Tonnes	Au Grade
	($M)	(Mt)	(Mt)	(g/t)
17	65.5	1.7	3.1	1.46
18	65.6	1.7	3.1	1.46
19	65.6	1.7	3.1	1.46
20	66.2	1.7	3.3	1.45
21	66.3	1.7	3.3	1.45
22	66.4	1.7	3.3	1.45
23	66.5	1.8	3.4	1.45
24	67.0	1.8	3.7	1.44
25	67.1	1.8	3.7	1.44
26	67.1	1.8	3.8	1.44
27	67.1	1.8	3.8	1.44
28	67.2	1.8	3.8	1.44
29	67.2	1.8	3.8	1.44
30	67.2	1.8	3.9	1.44
31	67.2	1.8	3.9	1.44
32	67.2	1.8	3.9	1.44
33	67.2	1.8	3.9	1.44
34	67.1	1.8	3.9	1.44
35	66.8	1.9	4.4	1.43
36	66.8	1.9	4.4	1.43
37	66.8	1.9	4.5	1.43

Pit	Cashflow	Ore Tonnes	Waste Tonnes	Au Grade
38	66.6	1.9	4.7	1.42
39	66.6	1.9	4.7	1.42
40	66.5	1.9	4.8	1.42
41	66.5	1.9	4.8	1.42
42	66.3	1.9	4.9	1.42
43	66.3	1.9	4.9	1.42

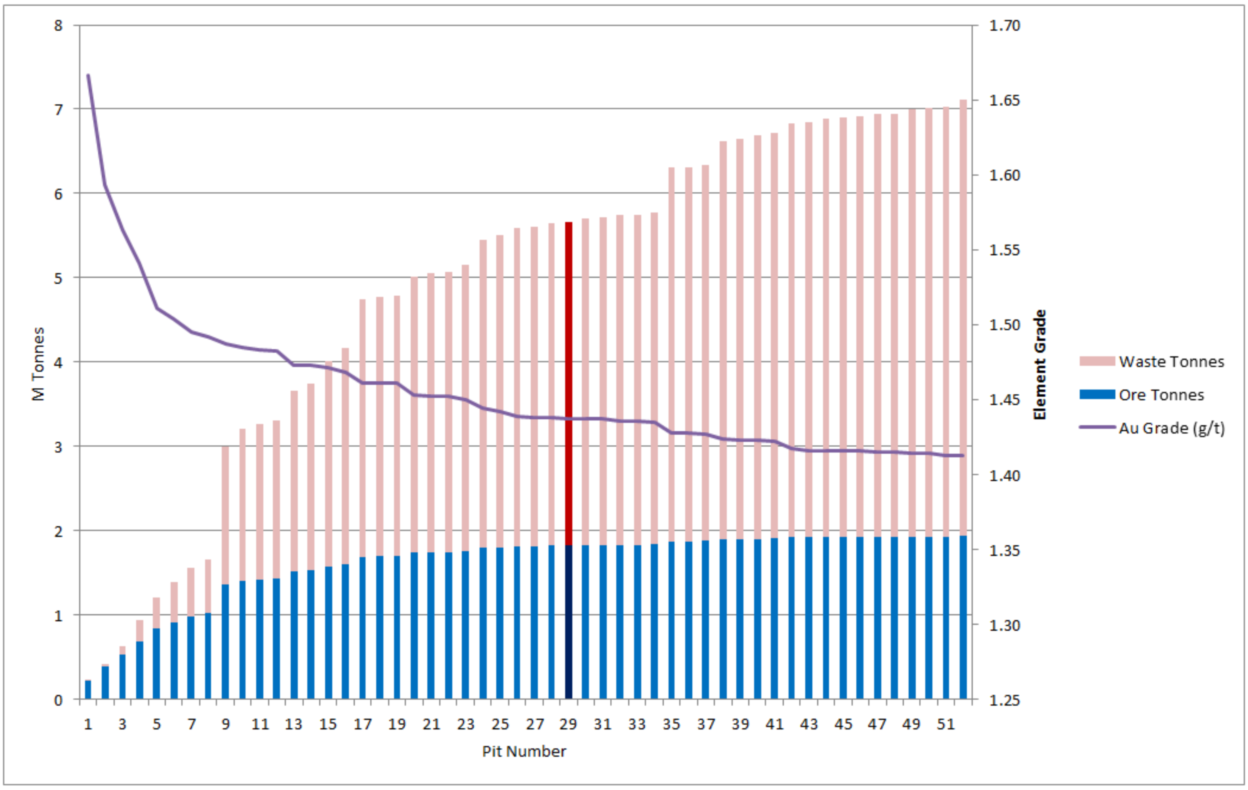

Figure 16-34: Mbenge Optimisation Results – Au Grade

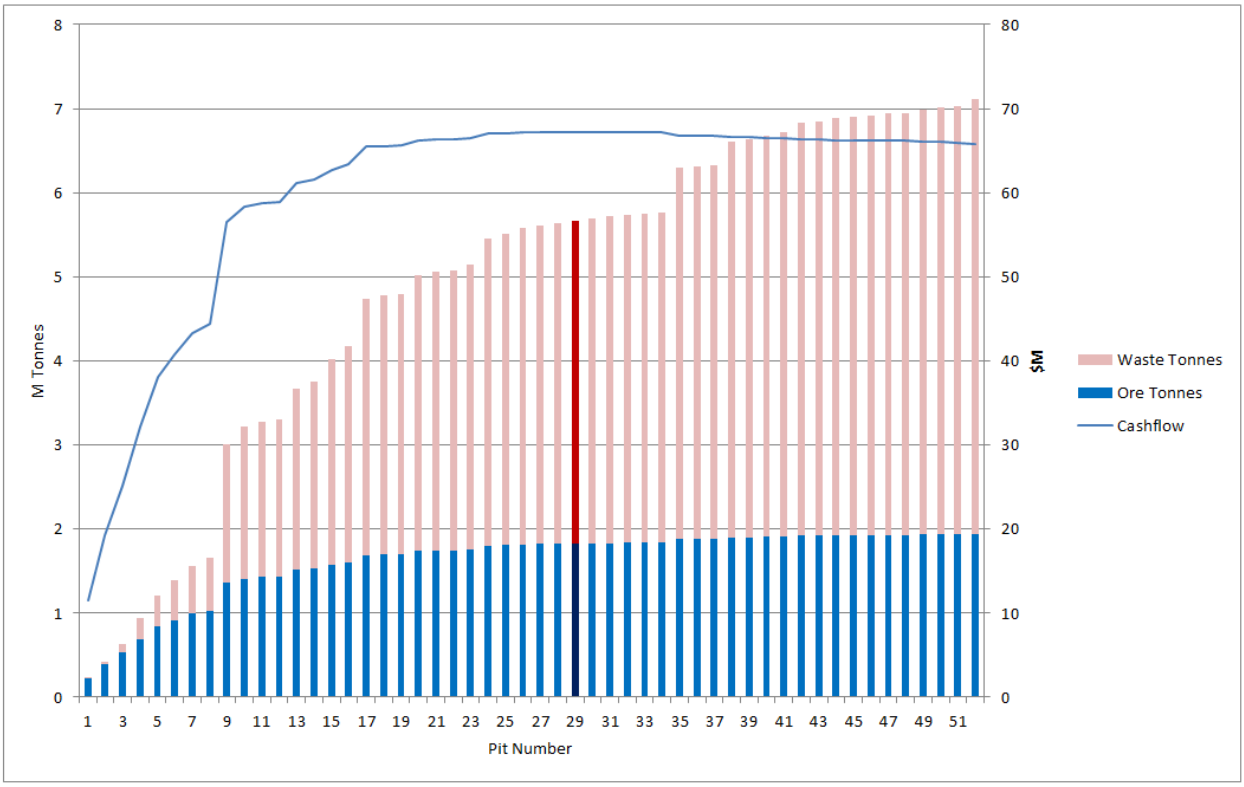

Figure 16-35: Mbenge Optimisation Results – Cashflow

Porcupine

Table 16-21 tabulates the results of the optimisation process for Porcupine.

Table 16-21: Porcupine Optimisation Results (Upside Potential Case)

Pit	Cashflow	Ore Tonnes	Waste Tonnes	Au Grade
	($M)	(Mt)	(Mt)	(g/t)
15	229.2	6.1	15.2	1.51
16	232.7	6.3	15.9	1.48
17	240.1	6.8	19.2	1.47
18	242.4	7.0	20.0	1.45
19	260.7	8.6	32.3	1.43
20	266.6	9.2	35.6	1.41
21	267.5	9.3	36.0	1.41
22	276.6	10.7	42.1	1.35
23	288.3	12.7	50.7	1.28
24	289.3	12.9	52.3	1.28
25	289.7	13.1	55.6	1.28
26	290.2	13.4	56.7	1.27
27	290.4	13.5	57.0	1.27
28	290.2	13.5	57.9	1.27
29	290.1	13.6	58.2	1.27
30	288.3	14.5	63.3	1.24
31	287.8	14.7	64.5	1.24

Pit	Cashflow	Ore Tonnes	Waste Tonnes	Au Grade
32	287.0	14.9	65.7	1.23
33	286.2	15.1	66.9	1.22
34	285.6	15.2	67.5	1.22
35	285.1	15.3	68.1	1.22
36	284.5	15.3	68.7	1.22
37	284.1	15.4	69.0	1.22
38	283.1	15.4	69.9	1.22
39	281.9	15.6	70.8	1.21
40	280.4	15.6	71.9	1.21
41	277.1	15.9	74.2	1.21
42	276.1	15.9	74.8	1.21
43	275.3	16.0	75.3	1.21
44	274.8	16.0	75.6	1.21
45	273.1	16.1	76.7	1.21
46	272.9	16.1	76.8	1.21
47	272.8	16.1	76.9	1.21
48	270.8	16.2	78.0	1.21

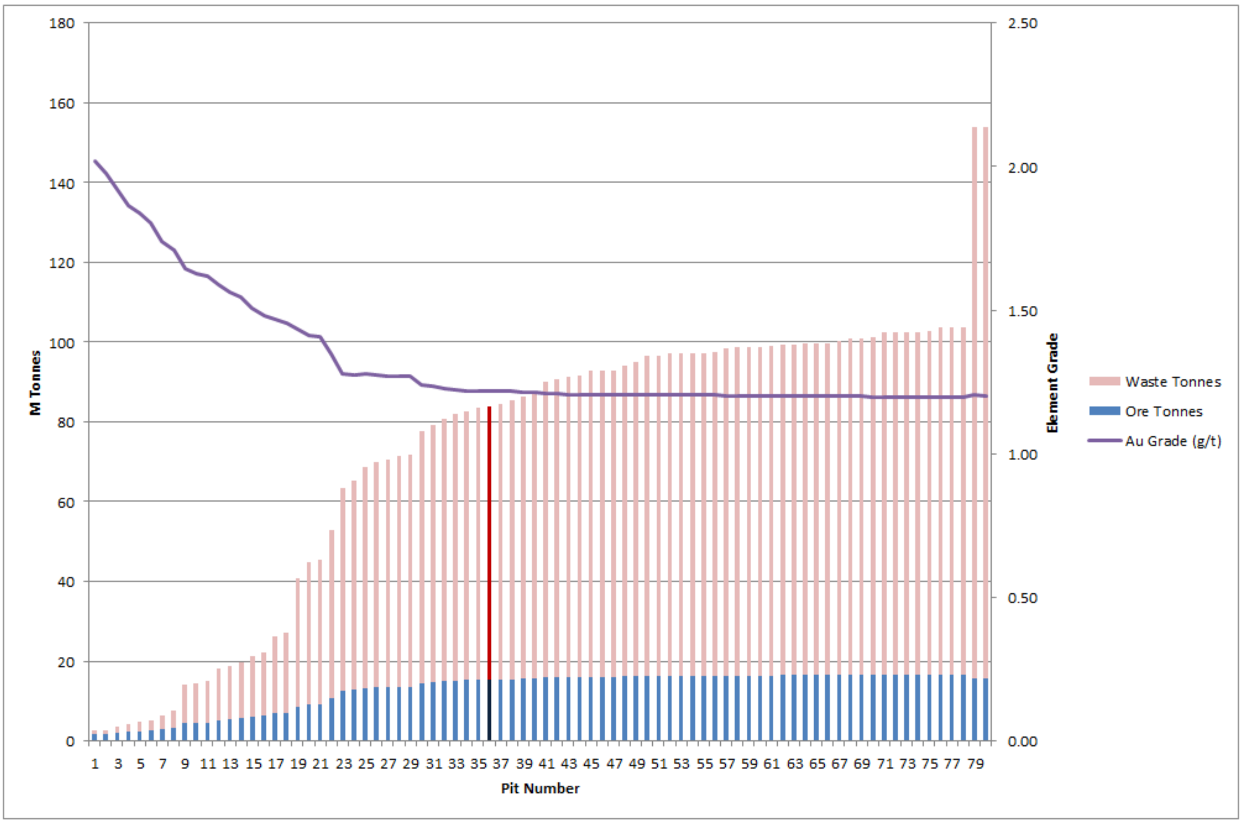

Figure 16-36: Porcupine Optimisation Results – Au Grade

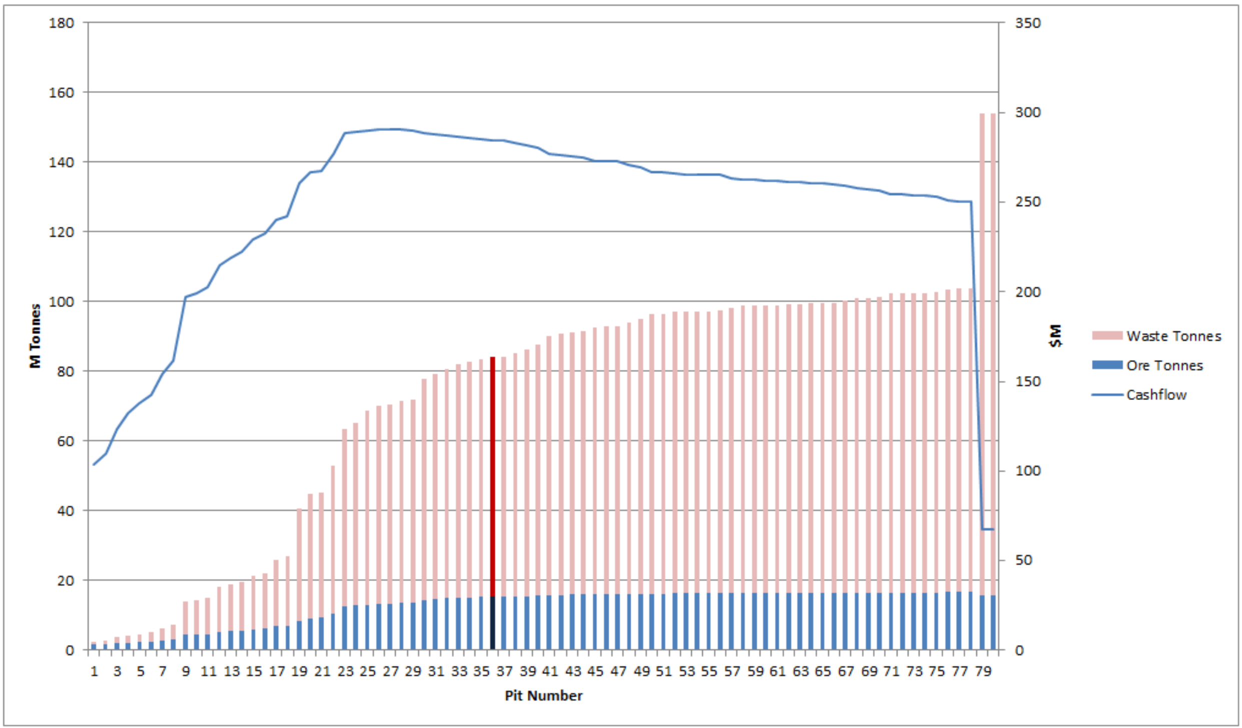

Figure 16-37: Porcupine Optimisation Results – Cashflow

16.3.2 Mine Design

Design Parameters

Table 16-22 summarises the parameters used for the pit designs. Due to the lack of geotechnical data, a series of assumptions have been made.

Further, it has been assumed there are no structural influences on the design parameters (e.g. bedding, dip etc).

Table 16-22: Pit Design Parameters (Upside Potential Case)

Parameter	Units	Kenge	Mbenge	Porcupine
Material Type		Fresh	Fresh	Fresh
Maximum Ramp Width	(m)	12.5	12.5	25
Bench Height (Kenge)	(m)	12.5	40	20
Batter Angle	(^{o})	85	76	78
Berm Width	(m)	15	5	5.5
Number of ramps per wall		2	4	3

To achieve the overall wall angles, there are significant sections within all three pits where single lane access has been designed. The designs where possible have incorporated passing bays, however SRK notes the single lane access could affect production.

Kenge

Figure 16-38 graphically shows the design conformance with the optimised shell.

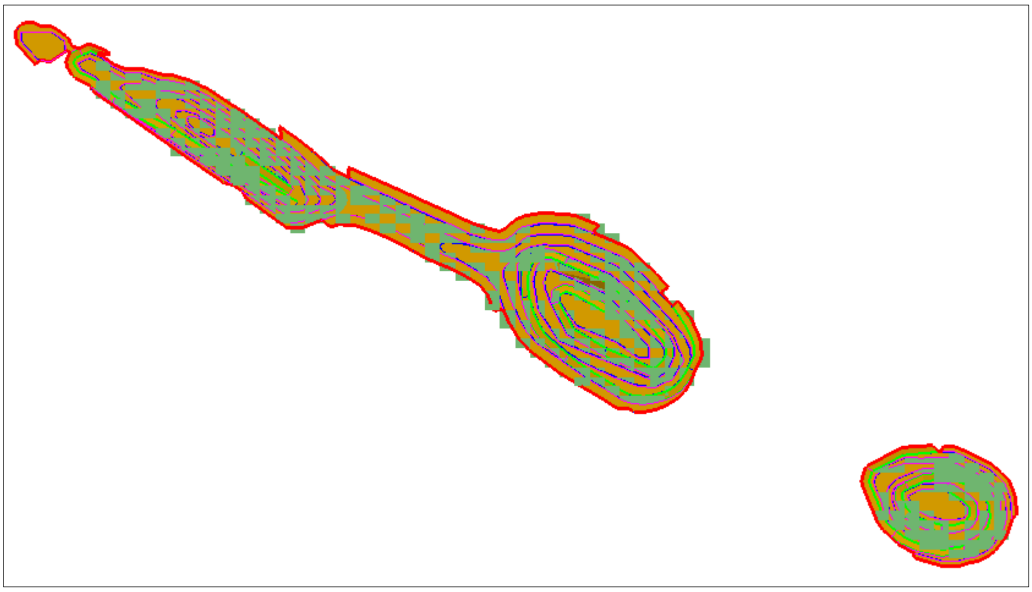

Figure 16-38: Kenge Mine Design (tan) conformance with Optimised Shell (teal)

Table 16-23 tabulates the designed pit conformance with the optimised shell.

Table 16-23: Kenge Designed Pit Conformance with Optimised Shell

	Whittle Tonnes	Designed Pit Tonnes	Conformance
Waste	20,939,141	22,284,430	106%
Inferred	2,081,758	2,031,239	98%
Indicated	1,829,593	1,627,089	89%
Measured	0	0	
Total	24,850,492	25,942,758	104%
Ounces	177,370	165,700	93%

Figure 16-39 and Figure 16-40 shows the Kenge pit design.

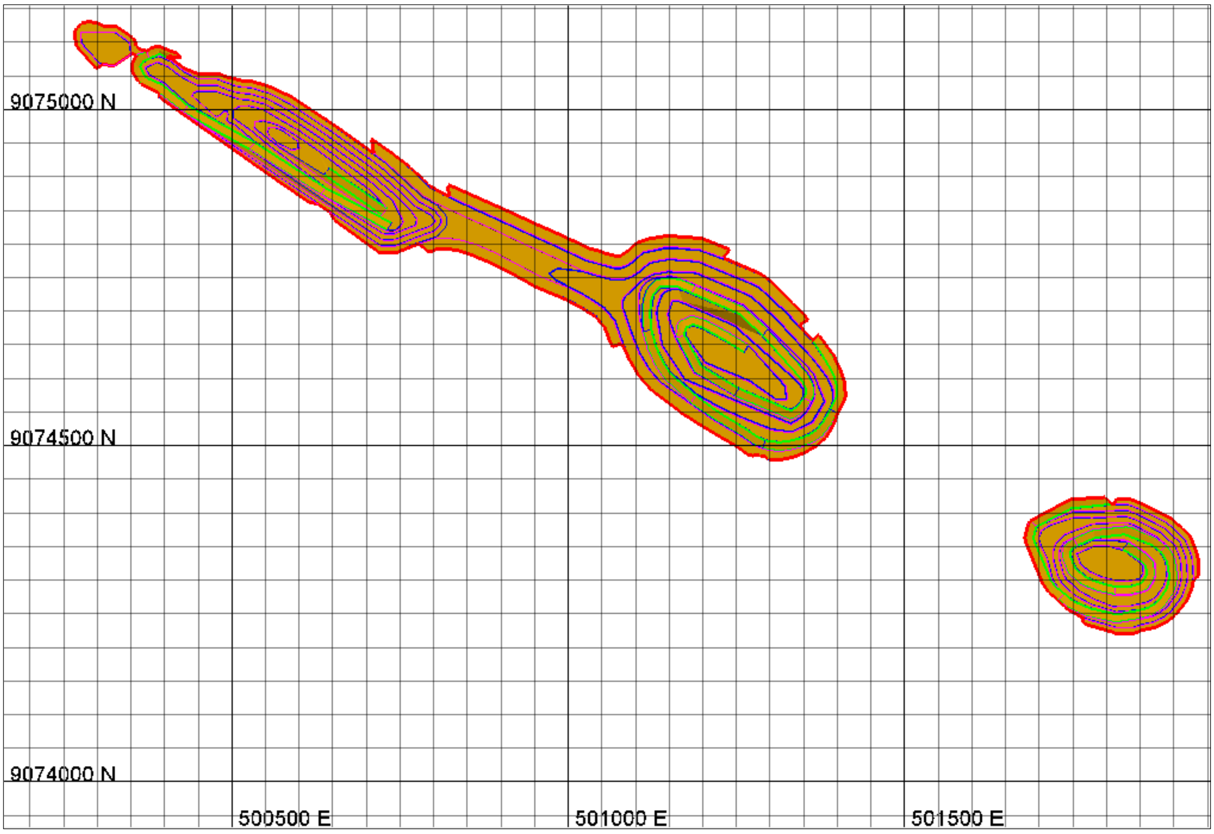

Figure 16-39: Kenge Mine Design – Plan View

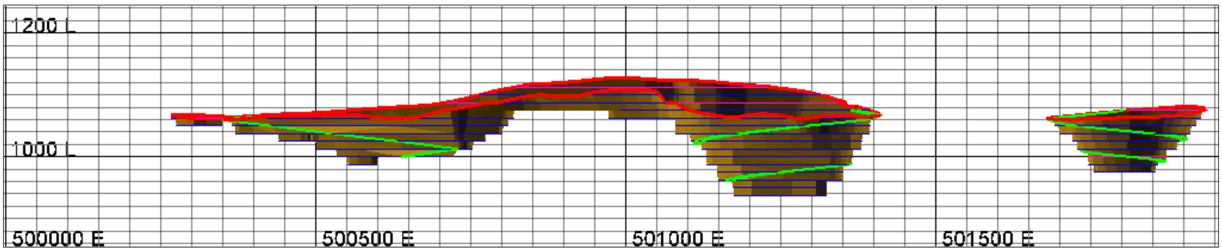

Figure 16-40: Kenge Mine Design – Looking North

Mbenge

Figure 16-41 graphically shows the design conformance with the optimised shell.

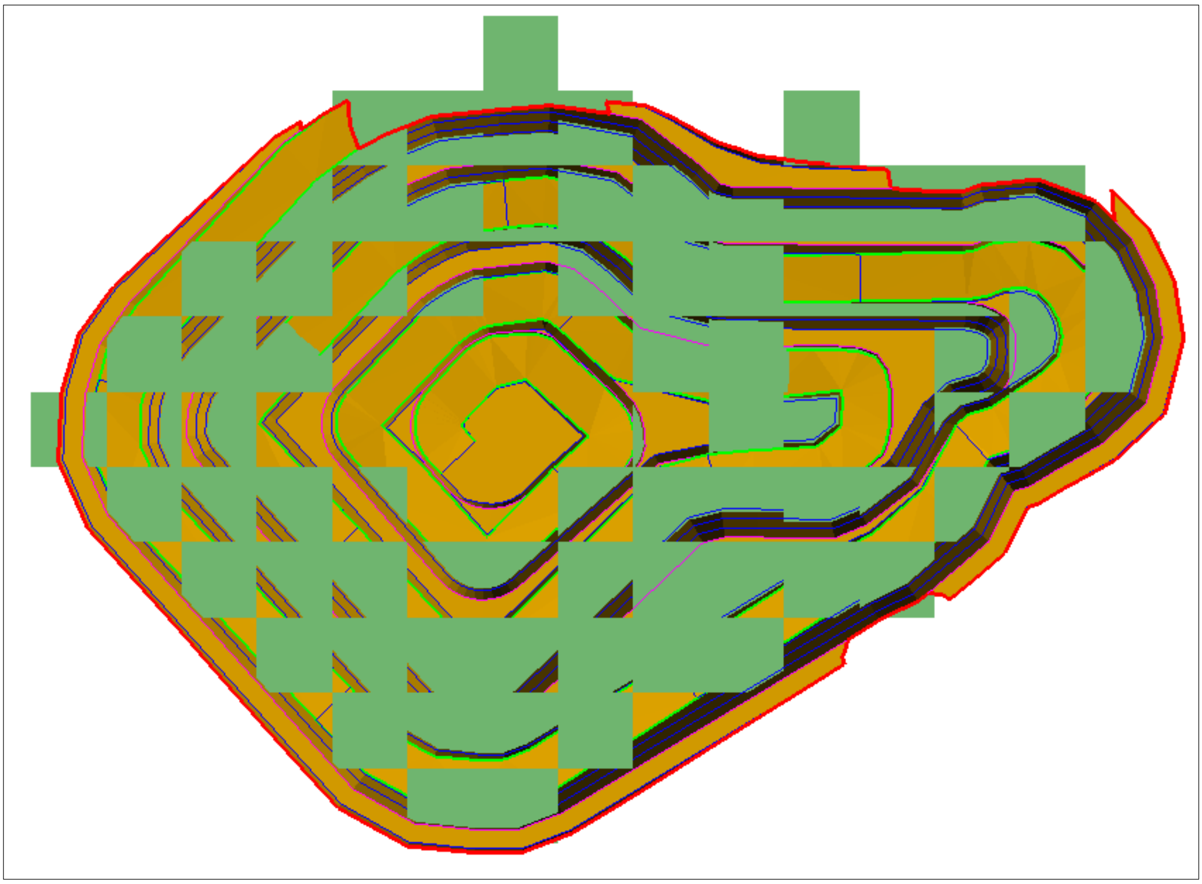

Figure 16-41: Mbenge Mine Design (tan) conformance with Optimised Shell (teal)

Table 16-24 tabulates the designed pit conformance with the optimised shell.

Table 16-24: Mbenge Designed Pit Conformance with Optimised Shell

	Optimised Tonnes	Designed Pit Tonnes	Conformance
Waste	4,005,041	4,194,911	105%
Inferred	0	0	-
Indicated	1,661,061	1,480,676	89%
Measured	0	0	-
Total	5,666,102	5,675,587	100%
Ounces	84,358	76,623	91%

Figure 16-42 and Figure 16-43 shows the Mbenge pit design.

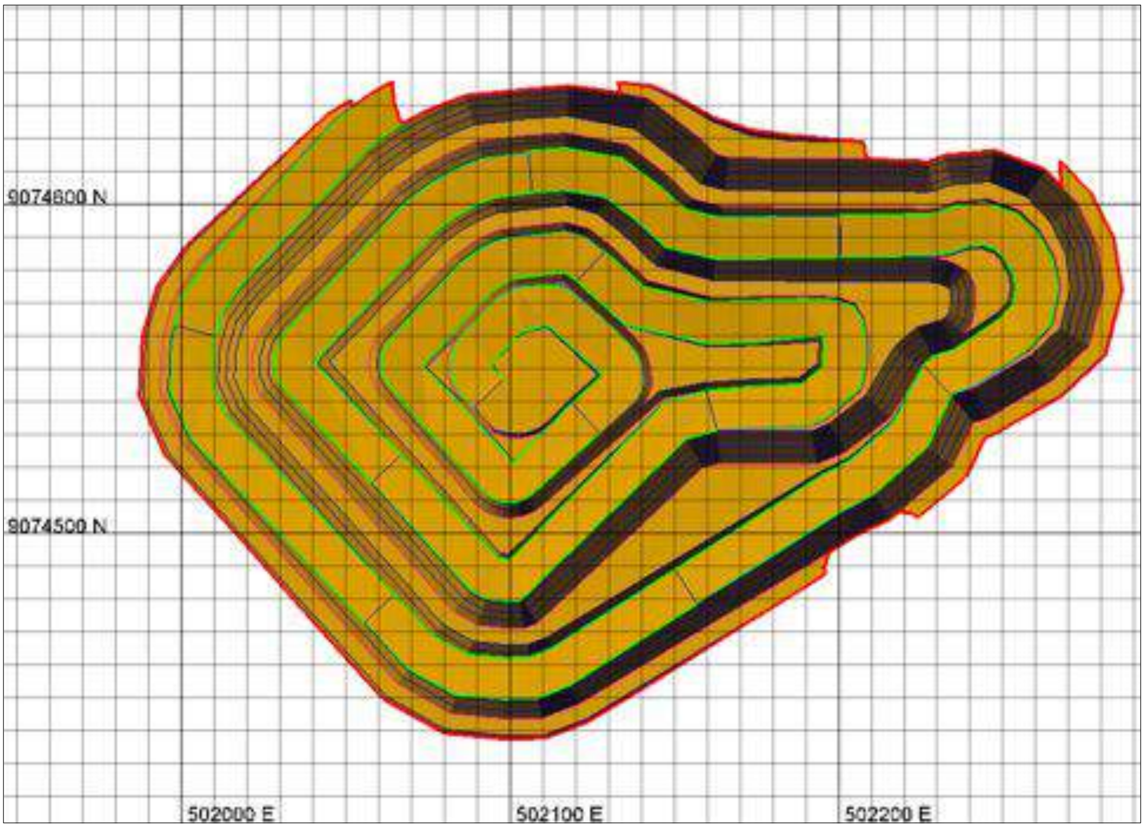

Figure 16-42: Mbenge Mine Design – Plan View

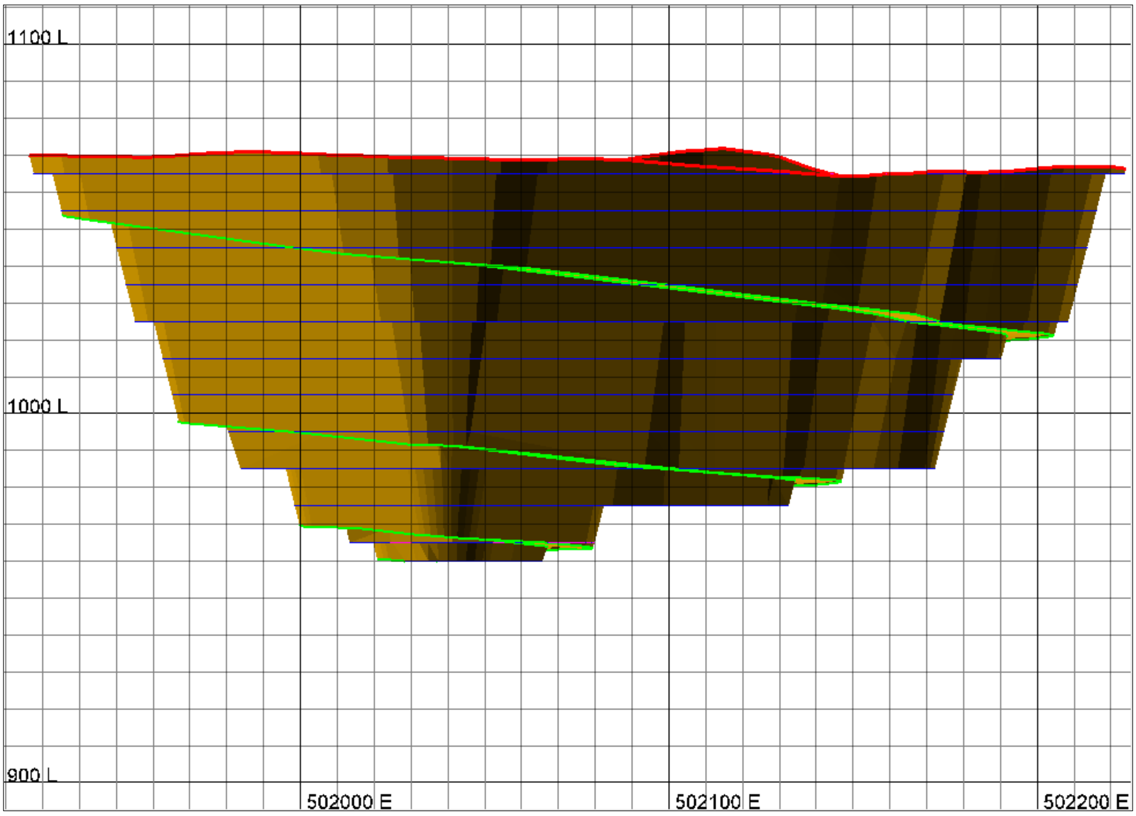

Figure 16-43: Mbenge Mine Design – Looking North

Porcupine

Figure 16-44 graphically shows the design conformance with the optimised shell.

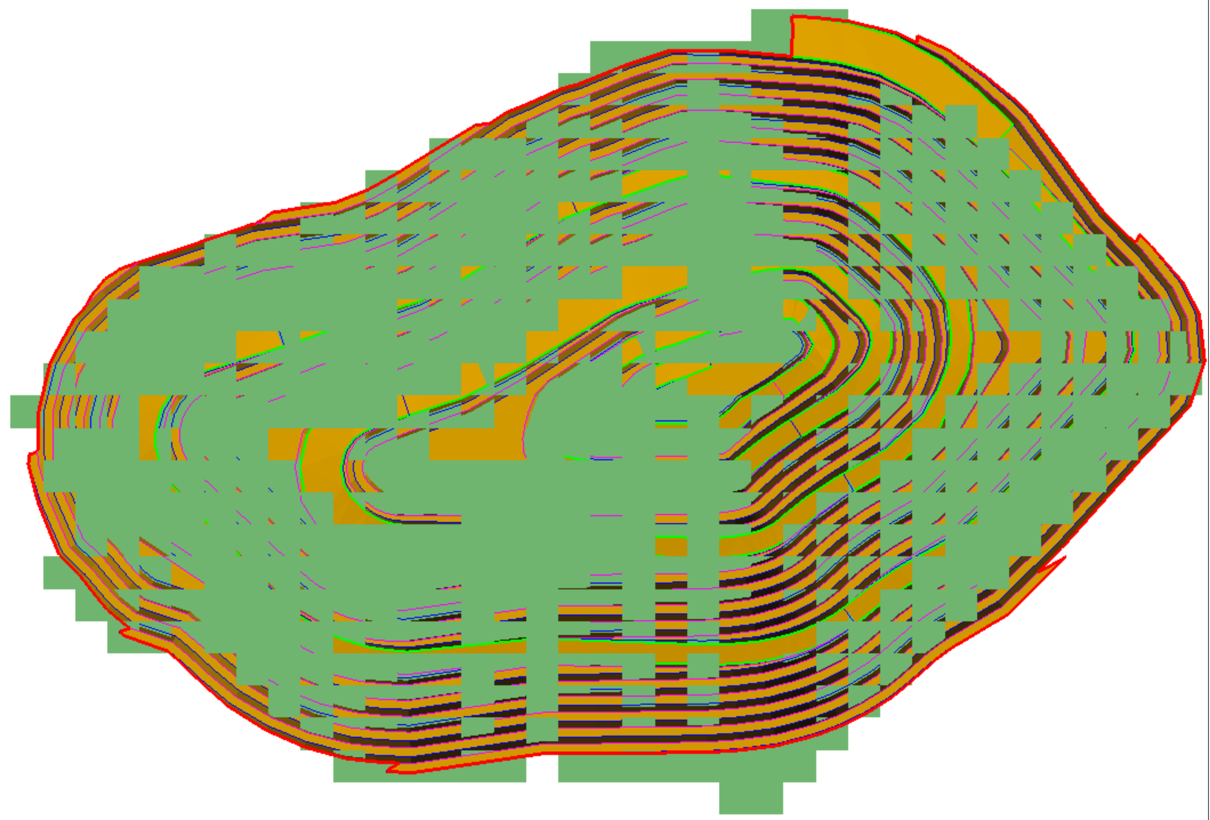

Figure 16-44: Porcupine Mine Design (tan) conformance with Optimised Shell (teal)

Table 16-25 tabulates the designed pit conformance with the optimised shell.

Table 16-25: Porcupine Designed Pit Conformance with Optimised Shell

	Optimised Tonnes	Designed Pit Tonnes	Conformance
Waste	69,998,756	77,469,769	111%
Inferred	69,026	176,528	256%
Indicated	2,120,766	2,034,963	96%
Measured	11,814,946	11,916,859	101%
Total	84,003,494	91,598,119	109%
Ounces	603,744	601,324	100%

Figure 16-45 and Figure 16-46 shows the Porcupine pit design.

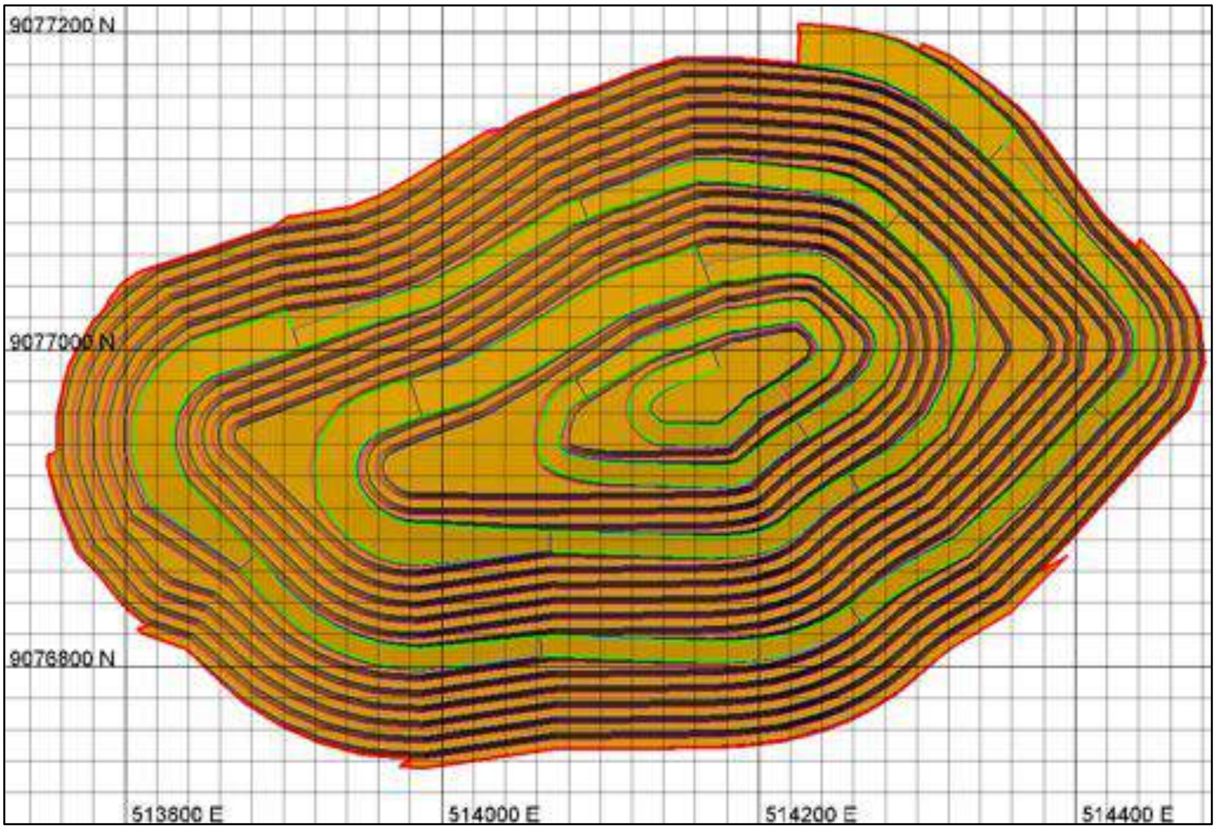

Figure 16-45: Porcupine Mine Design – Plan View

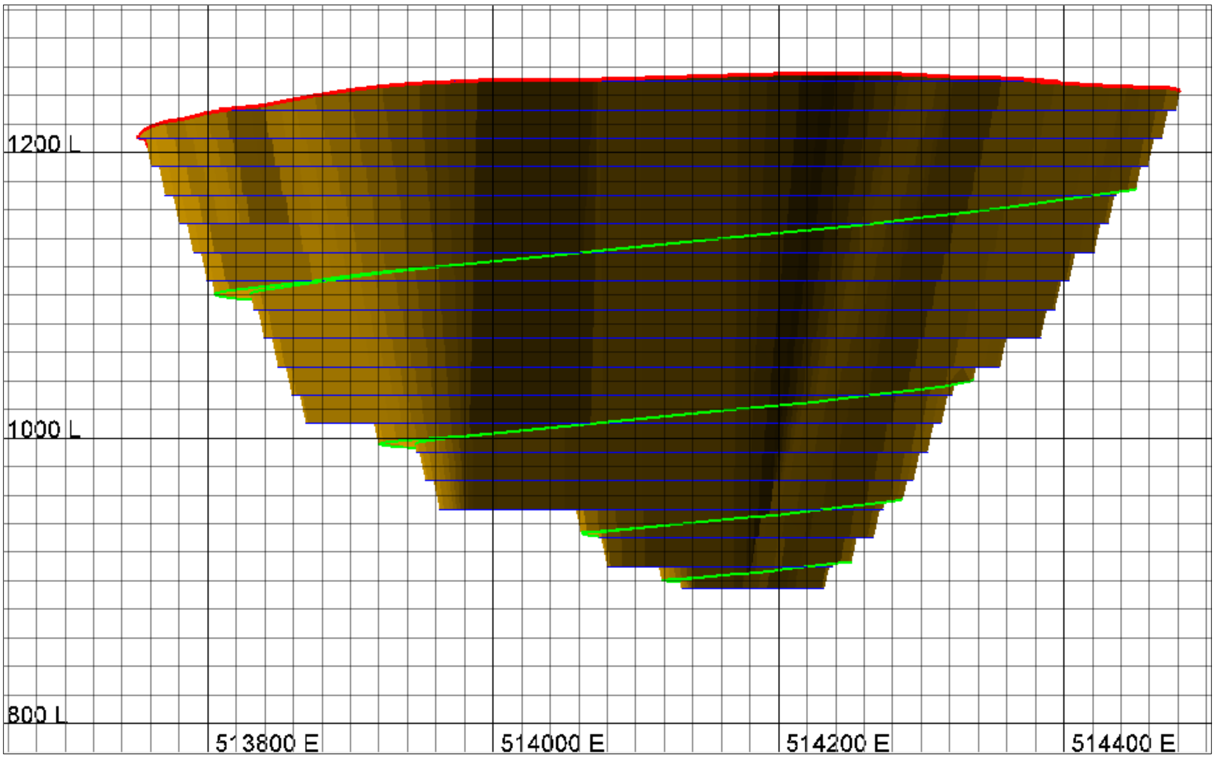

Figure 16-46: Porcupine Mine Design – Looking North

16.3.3 Waste Rock Dump Design

The following criteria have been used as part of the design process for both dump designs:

- 250 m standoff from pit design crest;
- Ramp width of 25 m;
- Batter angle of 37 degrees;
- Berm width of 45.8 m; and
- Target buffer capacity between 5-15%

SRK notes both waste dump designs have changed shape to allow for the larger volumes required.

Kenge Waste Dump

Table 16-26 shows the inputs used for calculating the volume requirements for the Kenge waste dump.

Table 16-26: Kenge Waste Dump (Upside Potential Case)

Pit	Units	Value
Kenge	Mm^3	8.1
Mbenge	Mm^3	6.8
Total	**Mm^3**	**14.9**
Swell Factor		1.3
Required Dump Capacity	Mm^3	19.4
Designed Dump Capacity	Mm^3	20.7
Buffer Capacity		6%
Approximate Dump Height	m	100

Figure 16-47 shows the waste dump with the Kenge and Mbenge pit designs.

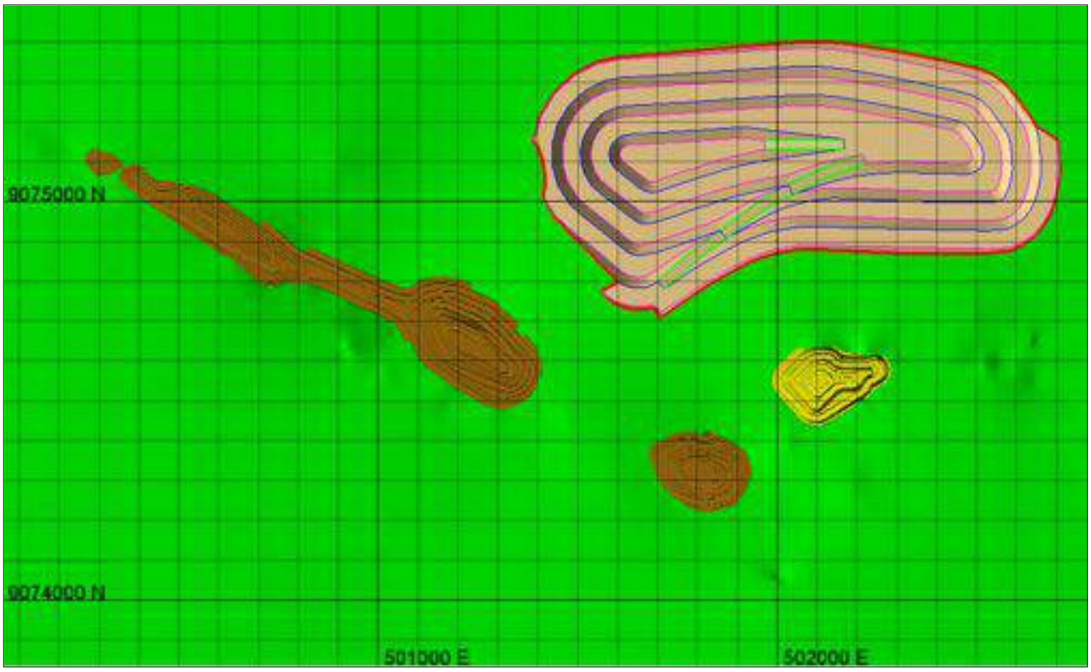

Figure 16-47: Kenge Waste Dump

Porcupine Waste Dump

A single waste dump has been designed capable of storing all waste material from the Porcupine deposit.

Table 16-27 shows the inputs used for calculating the volume requirements for the Porcupine waste dump.

Table 16-27: Porcupine Waste Dump (Upside Potential Case)

Pit	Units	Value
Porcupine	Mm^3	29.5
Swell Factor		1.3
Required Dump Capacity	Mm^3	38.3
Design Dump Capacity	Mm^3	43.2
Buffer Capacity		13%
Approximate Dump Height	m	100

Figure 16-48 shows the waste dump with the Porcupine pit design.

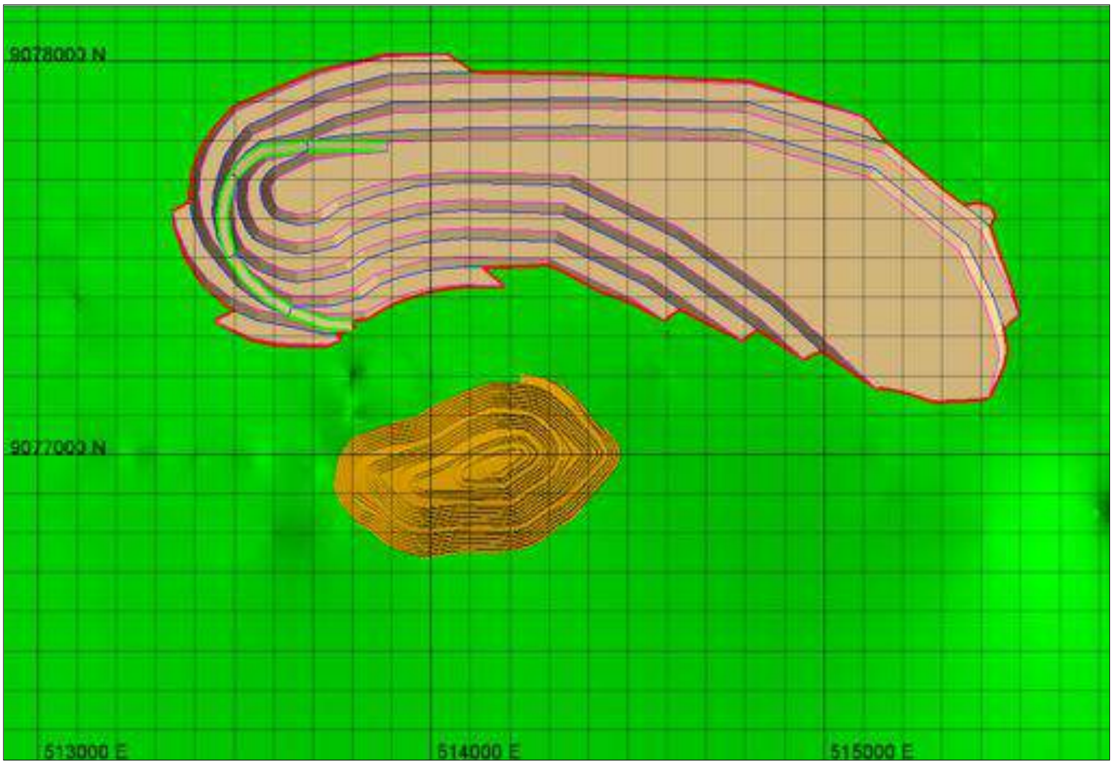

Figure 16-48: Porcupine Waste Dump

16.3.4 Mine Production

A target of approximately 100,000 oz per annum of gold was targeted for production at the request of Helio. No hard maximum material movement limit was placed on the schedule.

The operation is planned to be mined using a contract mining fleet, hence the fleet size was variable to minimise costs, whilst ensuring that a practical mining fleet configuration could be maintained.

Key features of the schedule include:

- Pits are to be mined in a general sequence of Porcupine, then Kenge followed by Mbenge;
- Maximum of two pits open at a single point in time to ensure supervisory commitments are manageable;
- First year scheduled in quarters, followed by annual increments;
- No ore is processed during the first year of the mining operation. This allows for the construction of the processing facility to occur whilst the mine is being developed. Any ore mined during this period will be stockpiled for feed into the plant once the plant is commissioned and operating. There is not expected to be any deterioration in ore quality during this stockpiling period;
- 1Mt ore is planned to be fed during the first year of the process facility's operation, 2.4 Mt ore to be fed during the second year of operation which is the planned maximum feed rate for the process facility; and
- A Run of Mine ("ROM") stockpile is to be utilised to 'smooth' the production profile and feed rate to the process facility. The maximum size of the ROM stockpile is approximately 260 kt.

Table 16-28, Figure 16-49, and Figure 16-50 detail the mine production schedule.

Table 16-28: SMP combined Mine Production (Upside Potential Case)

Period	Q1 2014	Q2 2014	Q3 2014	Q4 2014	2015	2016	2017	2018
Ore Tonnes (Mt)	0.0	0.0	0.1	0.2	1.2	2.0	2.5	2.6
Waste Tonnes (Mt)	4.5	4.5	4.5	4.4	13.4	14.5	13.9	14.8
Total Tonnes (Mt)	4.5	4.6	4.6	4.6	14.6	16.5	16.4	17.3
Au Grade (g/t)	1.09	0.86	1.13	1.07	1.23	1.51	1.17	1.25

Period	2019	2020	2021	2022	2023
Ore Tonnes (Mt)	2.3	2.5	2.4	2.5	1.0
Waste Tonnes (Mt)	12.5	5.8	6.2	3.0	0.6
Total Tonnes (Mt)	12.8	8.2	8.6	5.5	1.6
Au Grade (g/t)	1.34	1.23	1.28	1.43	1.18

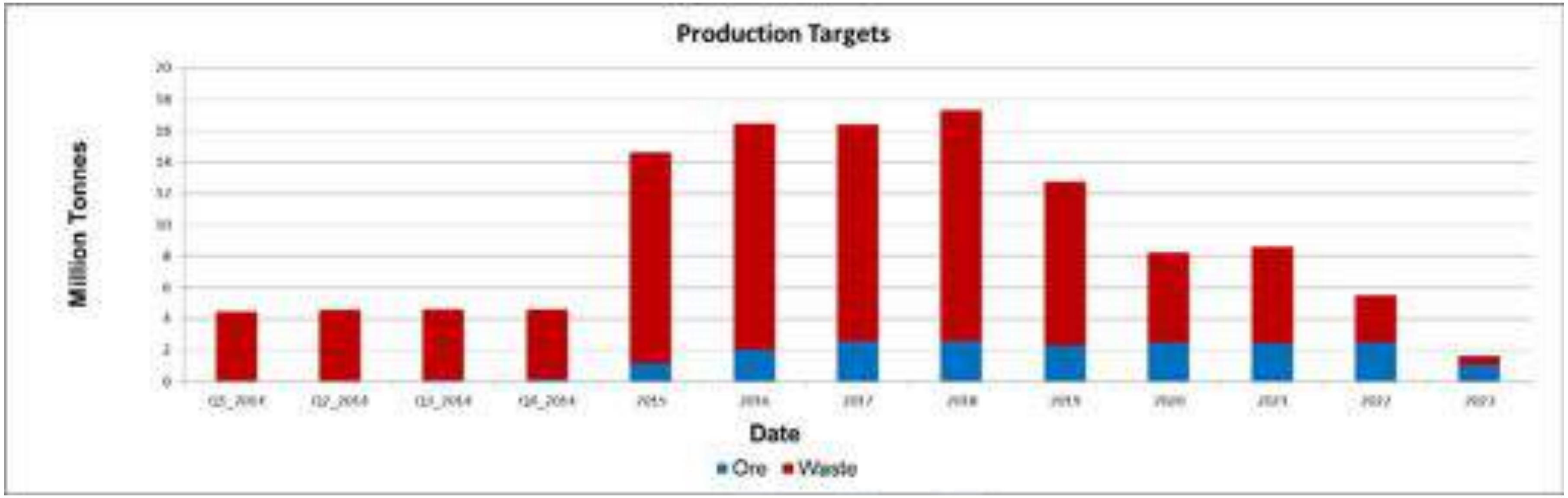

Figure 16-49: SMP Mine Production Schedule (Upside Potential Case)

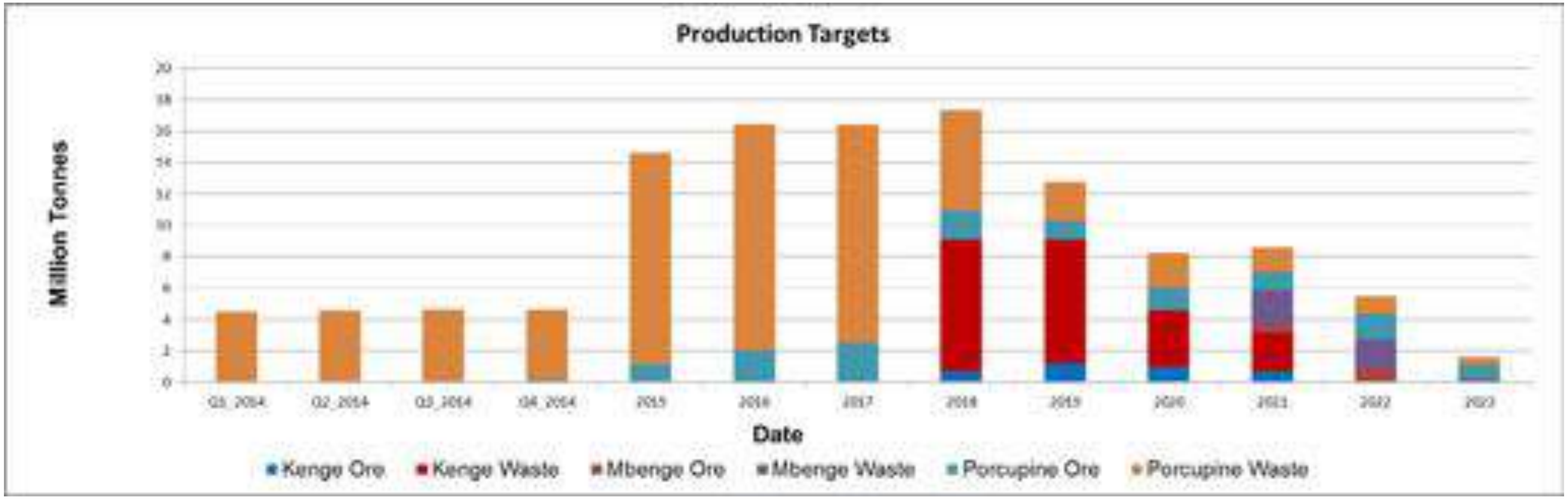

Figure 16-50: SMP Mine Production by deposit (Upside Potential Case)

16.3.5 Fleet Requirements

A first principles approach was utilised to estimate the equipment requirements. The process involved measuring the required haul distances and calculating the dynamic productivities for each piece of equipment. To estimate the requirements for auxiliary equipment, experience from current operating mines of similar size and scale has been used.

Table 16-29 summarises the estimated equipment requirements.

Table 16-29: Equipment requirements (Upside Potential Case)

Mining Equipment	Type	Maximum Number
Primary Equipment		
Haul Truck	Cat 777 (100t)	17
Loaders	Ex 1900 (190t machine)	3
Drills	6-9 inch diameter	2
Secondary Equipment	**(Auxiliary)**	
FEL – ROM ore pad	Cat 980 (260kW engine)	1
Track Dozer	Cat D10 (634kW engine)	2
Track Dozer	Cat D6 (433kW engine)	1
Wheel Dozer	Cat 834 (674kW engine)	1
Fuel Truck	~10,000L capacity	1
Water Truck	Cat 785 (1082kW engine)	1
Grader	Cat 16M (221kW engine)	1
Tertiary Equipment	**(Auxiliary)**	
Pick-up trucks	4 x 4 troop carriers	12
Lighting plants		12
Service vehicles	(hi lift, forklift, IT24)	5

The equipment type nominated is indicative only. The specific equipment will be determined as part of further studies.

Figure 16-51 details the loading unit requirements by period.

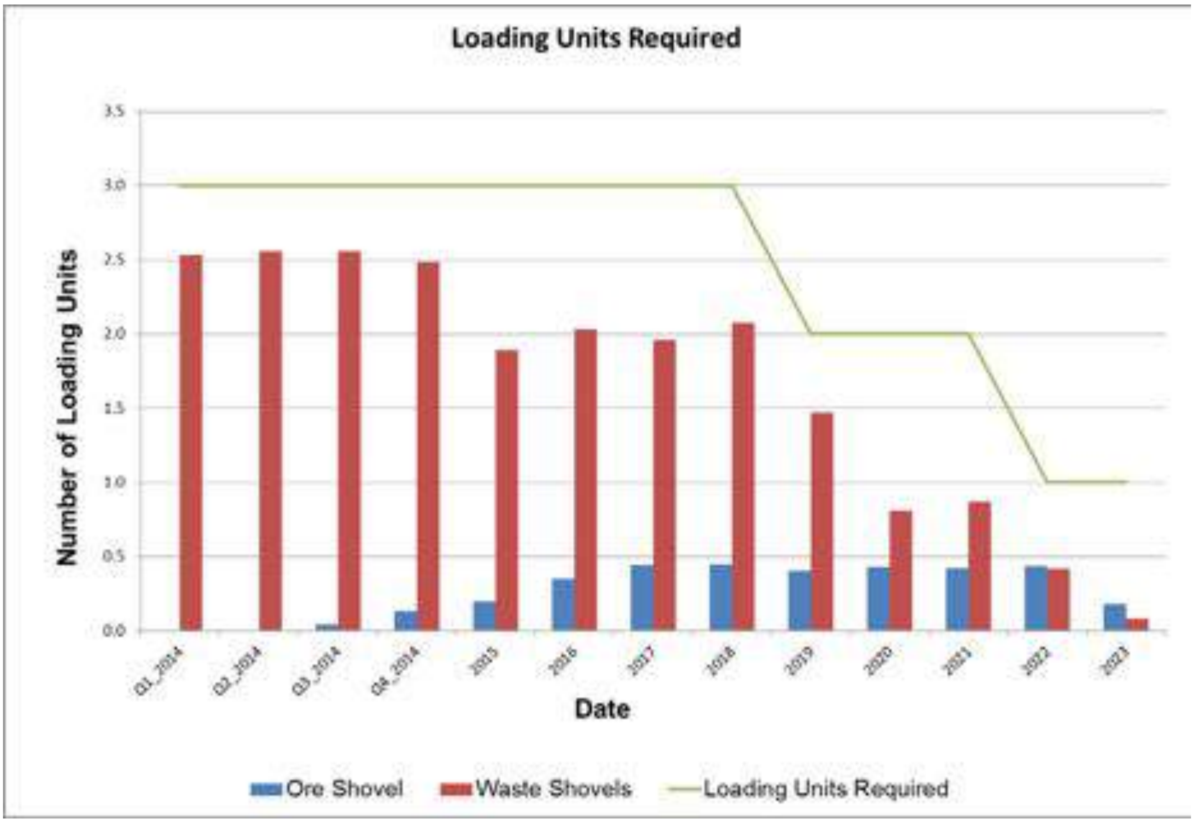

Figure 16-51: Loading Unit Requirements (Upside Potential Case)

Figure 16-52 details the trucking requirements by period.

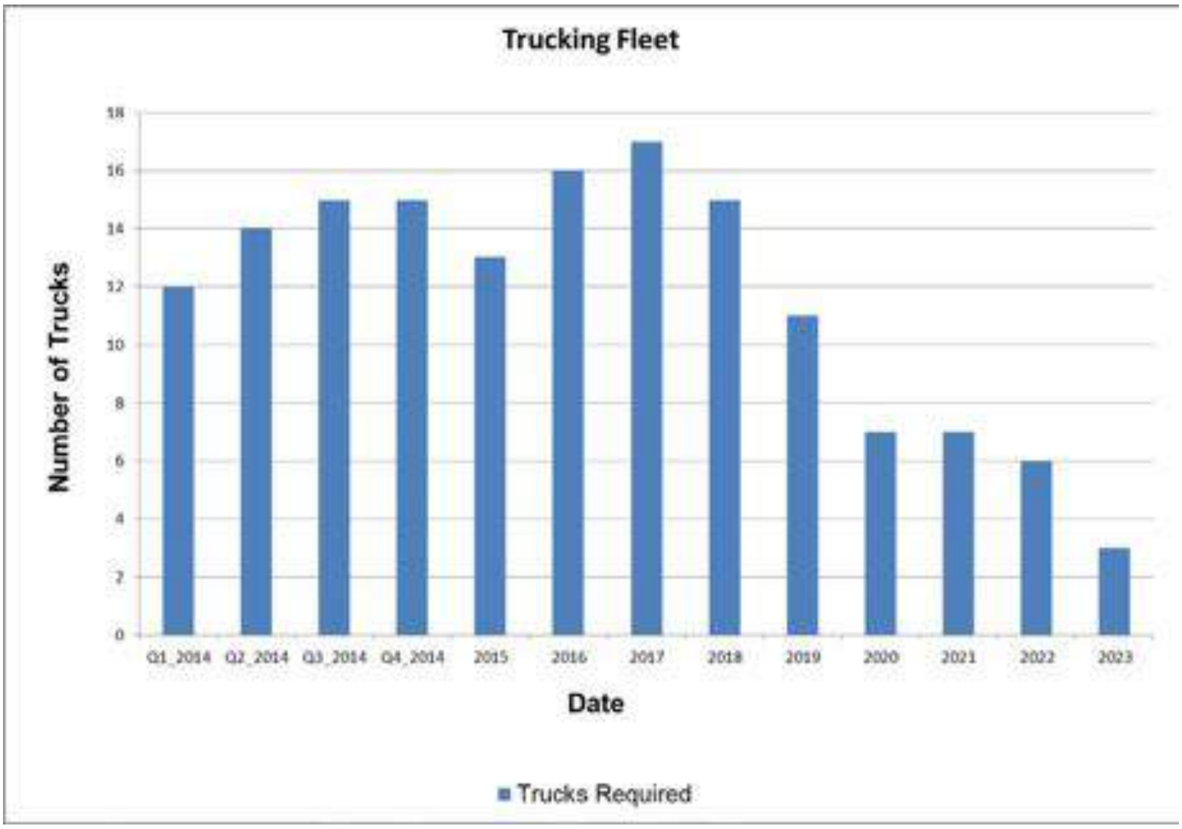

Figure 16-52: Truck Fleet Requirements (Upside Potential Case)

16.3.6 Labour Requirements

The site will be planned to operate 24 hours a day, 365 days a year.

The labour requirements have been developed using first principles and experience from other similarly sized operations. It is expected that the contractor will supply the workforce for the earthmoving activities. Nominally, it is expected the workforce will work a shift rotation of 2 weeks on 1 week off. This requires three operating shifts for the mine labour.

Figure 16-53 shows the estimated labour requirements for the mining operation throughout the life of the project.

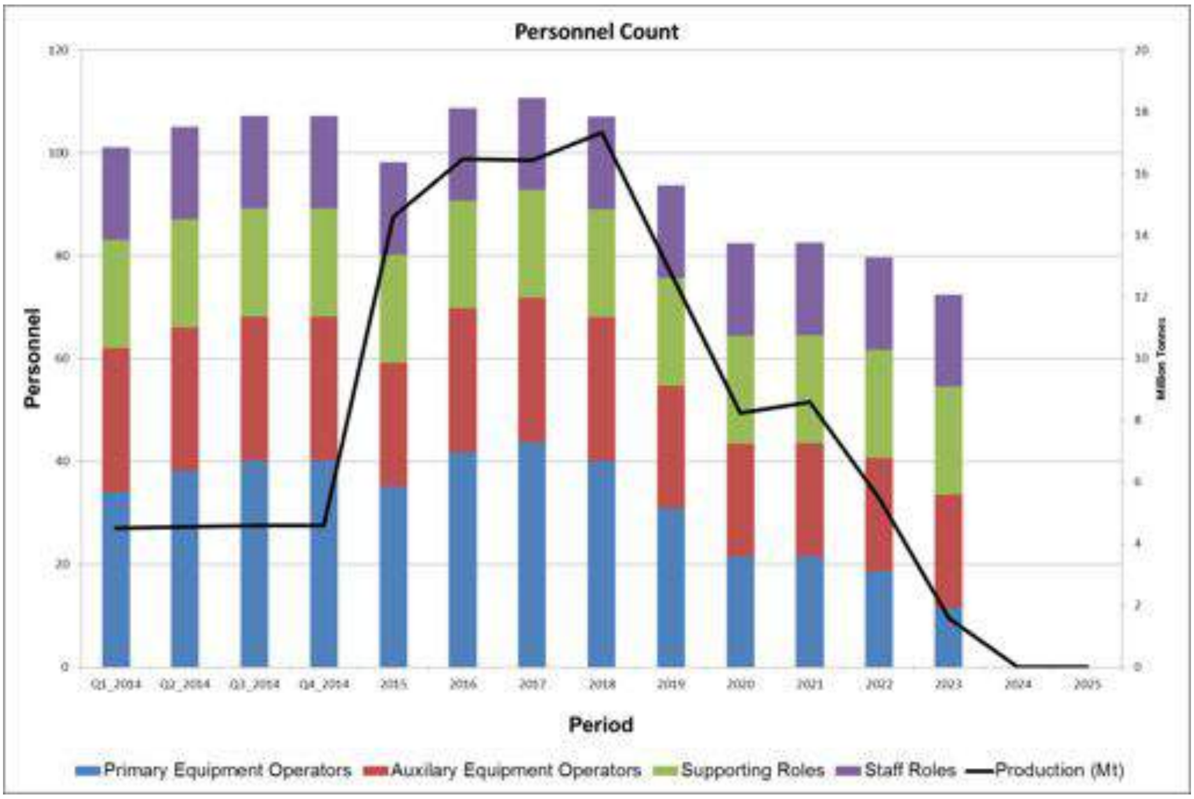

Figure 16-53: Mining Labour Requirements (Upside Potential Case)

16.3.7 Mining Cost Estimates

The estimated mining operating cost has been derived using first principles. The average mining cost for the life of the project is $2.24 / t total material mined, summarised in Table 16-30.

Table 16-30: Operating Cost Estimate (Upside Potential Case)

Parameter	Unit	Estimated Unit Cost
Equipment	USD / t	0.59
Fuel	USD / t	0.74
Over haulage	USD / t	0.06
Dewatering	USD / t	0.02
Labour	USD / t	0.26
Blasting	USD / t	0.20
Subtotal	**USD / t**	**1.87**
Contractor Premium (@20%)	USD / t	0.37
Total	**USD / t**	**2.24**

17 Recovery Methods

All ore is planned to be processed through a centralised processing facility. SRK used expected processing costs and metallurgical recoveries which were supplied by the client. These values were based on metallurgical testwork completed by SGS Lakefield Research Limited ("SGS") in Ontario, Canada.

Due to the preliminary nature of the metallurgical testwork completed, a detailed flowsheet has yet to be developed, however it is expected that due to the nature of the ore material and the indicative testwork results, a conventional gravity and cyanidation gold recovery technique will be investigated further.

18 Project Infrastructure

18.1 Site Selection

Initially the site for the proposed infrastructure was selected on the basis of proximity to the Porcupine deposit which is planned to be mined first. Following the site visit, a visual inspection of the proposed area indicated the area to the West of Porcupine deposit would be suitable for the required infrastructure.

Further work is required to ensure geotechnical suitability of the site and confirmation of sterilisation drilling.

A general site layout is shown in Figure 18-1.

18.2 Plant Personnel Transport

A bus service is proposed to be provided from the processing facility / administration area to the towns of Mkwajuni and Saza for personnel transport. It is planned that senior management will receive a 4 wheel drive vehicle. A bus parking area (including sufficient turning space will be required on site.

18.3 Bulk Services

Bulk services are limited to the supply of diesel and explosive products. Both products are planned to be stored in bulk containers, in secure bunded areas away from normal operations. Access to these areas will be restricted to authorised personnel only, with a log system in place to monitor access.

18.4 Laboratory Facility

An on-site laboratory is planned to be constructed and fitted out with appropriate analytical equipment. The role of the laboratory is to provide an analytical service to the gold plant and the mine by the assaying of samples received from these areas.

18.5 Power Supply and Distribution

A 33 kV line currently runs through the mine lease area. It is expected the company will have access to this power for use on site. The specific power requirements on site are yet to be determined.

Brown outs are experienced with the current power supply and it is expected that diesel generators will be required to act as backup for essential services on site.

18.6 Control and Instrumentation

The planned control system for the processing plant will use a programmable logic controller (PLC), with supervisory control and data acquisition (“SCADA”) systems to provide an operator interface. Ongoing review of operations will ensure as much “expert knowledge” is to be incorporated into the PLC programme as possible.

The modular system will be fault tolerant, easy to commission, fault find and maintain. The plant will be continually manned and be able to run automatically once started by the operators. All plant processes will be automatic with manual override facilities. On mains power failure, the plant will revert to a safe state.

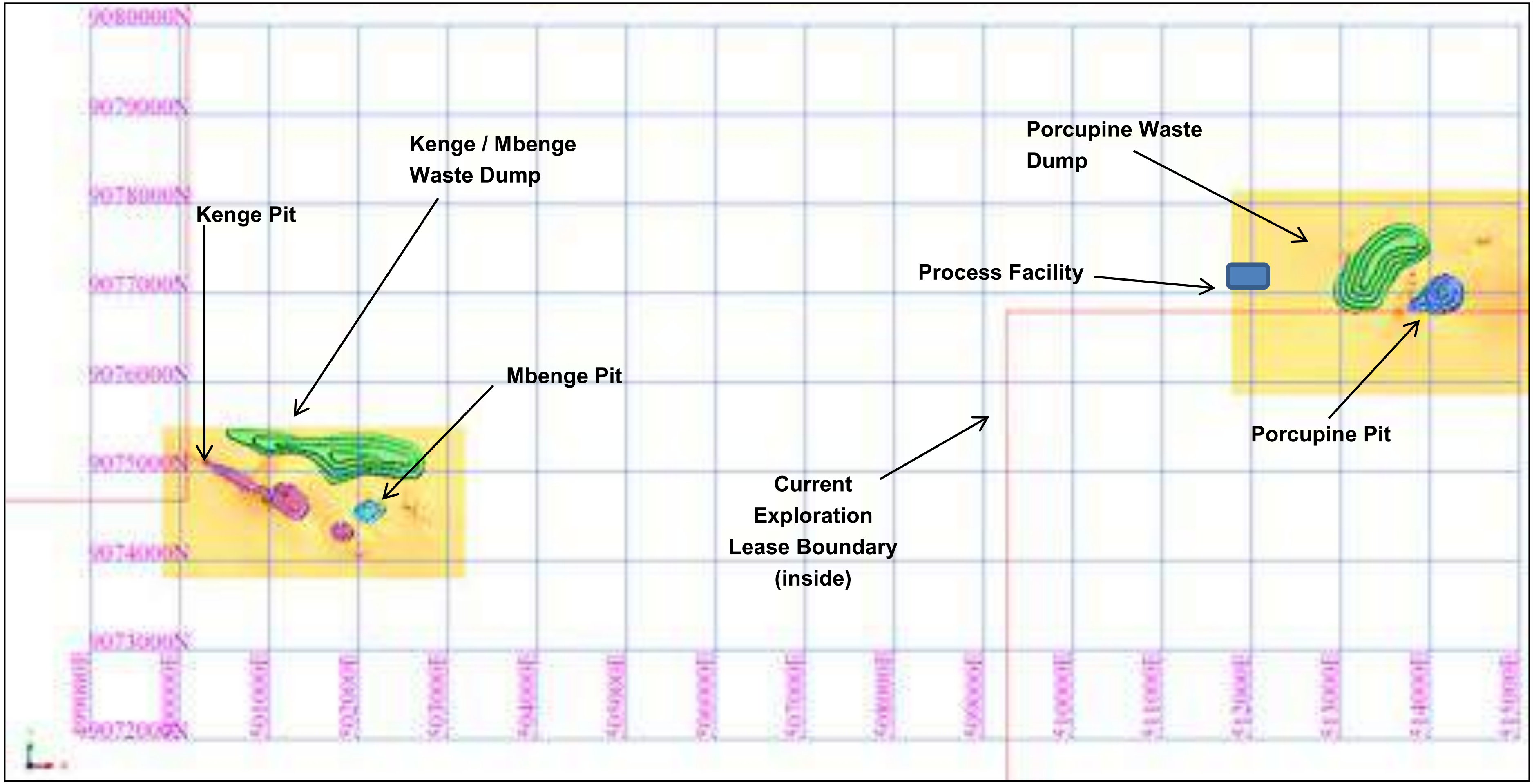

Figure 18-1: SMP – General Site Layout (Base Case)

18.6.1 Process Surveillance Equipment

Closed-circuit television ("CCTV") cameras will be used to give feedback to the control room on the specifically targeted areas within the processing facility.

Separate security camera systems will cover all security sensitive areas.

18.7 Water Infrastructure, Treatment and Distribution

18.7.1 Raw Water Storage and Distribution

Raw water is planned to be pumped from the production and the pit dewatering boreholes to a receiving tank at the water treatment facility (WTF) and the flotation process water tank. Raw water treatment includes coagulation, settling and filtration. Once treated, raw water will be distributed to the following areas:

- Crushing and milling for dust suppression; and
- Potable water treatment plant.

18.7.2 Potable Water Distribution

A portion of treated raw water is planned to undergo a chlorinating disinfection stage at the WTF to produce potable water. Potable water will be pumped to a potable water storage tank with a bottom section dedicated to fire water storage. Potable water will be fed via a gravity-pressurised pipe network throughout the processing facility, including the various safety showers throughout the plant.

The treatment plant is planned to be of a modular design installed on an elevated concrete slab. This will allow for any increase in size at a later stage of mine development. The plant will be sized to accommodate reagent make-up water requirements as well as the consumption requirements of all personnel on site. It is estimated that 150 litres of potable water per person per day must be provided for human consumption; both plant and mining facilities will be fed from this facility.

An elevated storage tank will be required to accommodate the single biggest water usage event during reagent make-up or the change house requirement for a shift change.

18.7.3 Fire Water Distribution

A pressurised fire water reticulation system is proposed to be installed throughout the plant, the mine and the office complex. A pump driven by an electrical motor connected to the main power supply will serve as the main pump, with a diesel driven pump on standby. A jockey pump will maintain the system pressure.

Fire water and potable water are planned to be stored in the same tank. This will prevent algae growth within the fire water tank. Fire water will be drawn from the bottom of the tank while potable water will be withdrawn from a higher level in the tank. This ensures that the minimum required amount of fire water will be available regardless of the potable water consumption.

A mobile fire truck will be used for areas not covered by hose reels or where booster facilities are required.

18.7.4 Process Water

The process flowsheet is still to be developed; therefore the specific flow of process water is also still to be developed. It is expected that significant focus will be made within the processing flowsheet to recover as much process water as practical.

18.7.5 Tailings Lines

The process flowsheet is still to be developed; therefore the specific flow of tailings lines is also still to be developed.

It is expected the tailings from the plant will be pumped to the Tailings Dam Facility (TDF) through High Density Polyethylene (HDPE) pipe lines. The processed tailings line will be placed within a lined trench, with collection sumps to return the water to the process water circuit.

18.8 Sewerage Collection and Treatment

Sewerage is planned to be collected and use gravity reticulation via buried sewerage pipes to be transported to the treatment facility. Sewage will be treated in a modular sewerage treatment plant (STP). The default capacity of the plant will be to treat the sewerage generated from 300 persons on site per day. The water output from the STP is expected to be suitable for use in dust suppression, vehicle washing, irrigation, fire water and process water.

The STP will also produce a small quantity of sludge, which will be dried in a sludge-drying bed located at a point lower than the plant. Dried sludge can be used as fertiliser in the rehabilitation of vegetation on the TDF slopes.

18.9 Storm Water Management

Rainfall runoff within the process plant and spillages from the process are classified as “dirty” water and is planned to be collected in a lined storm water pond. The storm water pond will be designed for a one in a hundred year rainfall event. As there is potential that excess storm water could be contaminated with cyanide, this water will be returned to a high sulphide return water dam for use in the CIL and intense cyanidation processes.

Any rainfall that occurs outside the disturbed plant area will be classified as clean run-off and will be diverted back to the natural watercourse.

18.10 Pollution Control Dam

All water collected onsite will be tested before being released into the natural watercourse. A lined pollution control dam will be required to collect and prevent any uncontrolled release of mine and process water into the natural watercourse.

Any water tested and deemed unsuitable for release into the natural watercourse will be returned to the process facility as process water.

18.11 Compressed Air Services

Compressed air will be required throughout the processing facility and mobile equipment workshop. This service will be supplied by semi-permanent air compressors located at each facility. These compressors will be either connected to the main electrical supply or diesel powered.

18.12 Refuse and Waste Disposal

All non-hazardous material will be collected and deposited onsite at a designated refuse disposal site.

Any hazardous material (e.g. oil, batteries) will be collected and disposed of by qualified and authorised third parties off site.

18.13 Buildings

It is expected the following buildings will be required on site to assist the mining operation:

- Administration Building (including engineering offices);
- Plant Engineering Building;
- Laboratory Building;
- Processing Facility (including engineering offices and plant control room);
- Explosive Storage Facilities;
- Mobile Mine Equipment Building (including engineering offices);
- Security Office;
- Change house;
- Gatehouse and First Aid facility; and
- Fencing.

18.14 General Facility Lighting

General lighting on site will be affixed to permanent structures. Roads will be illuminated by fixed lighting stands, with particular focus on intersections.

Mobile lighting towers will be utilised within the mining area. The mobile towers will be fuelled by diesel. A strict monitoring procedure will be required for the refuelling process.

18.15 Roads

Currently the area surrounding Mkwajuni, including access to the mine lease is serviced by a series of dirt roads. Short sections of the roads have been sealed. As part of the development of the mine, it is expected that additional sections of the road will be sealed to maintain access throughout the year.

Additional dirt roads on site are expected to be established as required to link all activities to the central processing facility and mining areas on site.

18.16 Security

The planned electronic security system will provide an integrated real time viewing security application utilising CCTV and intruder detection. All equipment will be non-proprietary, commercially available products. The integrated alarm and access control system will provide live-event recording of all movements in controlled areas and immediate indication of programmed alarm events. The surveillance facility will be located in a secure location with access control.

The security system will incorporate an access control system that will allow predetermined permission rights to card holders and interface to a database system for logging and reporting.

Within the area, there is activity of itinerant artisanal mining. This activity will likely require monitoring to ensure that activities do not continue once the mining operation commences.

18.17 Cranage

Within the processing facility, in-plant maintenance will be serviced by a centrally located tower crane. In the reagent mixing areas dedicated electric hoists are to be installed.

A fixed crane will also be installed within the mobile equipment workshop for use on mobile mine equipment.

For other lifting requirements mobile cranes will be utilised.

18.18 Mobile Equipment

A single telescopic handler (mobile crane) will be used for general plant and mobile equipment maintenance. This telescopic handler will be shared across the site and will be able to reach general areas not served by fixed means of cranage. The telescopic handler will be able to handle the typical weight of items to be handled such as pumps, motors and gearboxes.

Two long wheel base ("LWB") light delivery vehicles ("LDV") will be used for general plant maintenance. LDVs will have a minimum payload of 1 000 kg.

Maintenance and repairs of the mobile equipment will be carried out in the utility workshop located in the mobile equipment workshop facility.

18.19 Fire Fighting Services

Water cannons are planned to be installed throughout the plant and will be fed from a pressurised fire water distribution network. Additionally, fire hydrant points complete with fire hoses will be located throughout the plant. Hydrant points will be located next to access routes for easy access. Fire hydrant points will also be installed along the conveyor line and throughout the office complex buildings. These points will be fed from the pressurised fire water distribution network.

Foam spray fire extinguishers will be placed in accessible locations at each conveyor drive. Dry powder extinguishers will be placed at the entry and emergency exits of electrical rooms or where there is the potential for electrical fires occurring. Foam spray fire extinguishers will also be located throughout the workshops, stores, offices, change houses and other infrastructure facilities.

18.20 Product Transport

Dore bullion is planned to be packed within the gold vault and transported off-site to a selected refinery.

18.21 Information Technology and Communication

There is currently mobile phone coverage to the proposed mine sites and surrounding region, including the nearby township of Mkwajuni.

A two-way radio system is planned to be established on site and utilised between operators within the mine area.

19 Market Studies and Contracts

No market studies have been conducted as part of this Technical Report.

20 Environmental Studies, Permitting and Social or Community Impact

20.1 Introduction

This section of this report provides an overview of the Tanzanian legislation applicable to the Project (including environment, mining, health, safety and labour legislation). The permitting process is outlined and responsibilities of the parties are provided. Environmental monitoring, baseline studies, closure and reclamation are discussed. Plans to address health, safety and environmental concerns will be required in the Environmental Impact Statement (EIS) as part of a future study.

20.2 Tanzanian Legislation and Guidelines

Key legislation for the Project includes environmental, mining, health, safety and labour legislation.

20.3 Environmental and Social

The key environmental legislation in Tanzania is the *Environmental Management Act, 2004*. Environmental management in Tanzania is the responsibility of the Vice-President’s Office and the Division of Environment (DoE).

The Division of Environment (DoE) duties include:

- Policy formation on environment;
- Monitoring and co-ordinating environmental issues;
- Environmental planning; and
- Policy focused research.

An agency of the DoE is the National Environmental Management Council (NEMC). The role of the NEMC is to advise the Vice-Presidents Office on matters of environmental management and conservation. A key responsibility of the NEMC is to review Environmental Impact Statements (EIS), conduct monitoring and auditing.

The *Environmental Management Act, 2004* details the following:

- Environmental planning;
- Environmental management;
- Environmental impact assessments;
- Pollution management;
- Waste management; and
- Administration functions.

Part VI of the Act provides guidance on the EIS. It details the obligations to undertake an EIS, provides regulations and guidelines and outlines the responsibilities of the parties. The NEMC will determine the scope of the EIS by agreeing with the proponent on the persons and institutions to be consulted, as well as the study approach.

Following the submission of an EIS, the following is stipulated by the Act:

- Within 60 days of submission, the NEMC shall carry out a review, within which time the proponent must comply with any additional requests for information;
- Within 30 days of the submission, the Council may decide whether or not to convene a public hearing for the gathering of public comments;
- The NEMC shall circulate the EIS to institutions, government agencies and the public for comment;
- At the completion of the EIS review, the NEMC shall submit recommendations to the Minister;
- Within 30 days of receipt of the Council's recommendations, the minister may approve, disprove, or conditionally approve the EIS; and
- Successful applications are to receive an Environmental Impact Assessment Certificate.

Other notable environmental legislation includes:

- The *Forest Act, 2002*; and
- The *Water Resources Management Act, 2009*.

20.4 Minerals and Mining

The primary legislation governing mining in Tanzania is the *Mining Act, 2010*. The Ministry of Natural Resources and Tourism is the principle regulatory body that administers this law. The *Mining Act, 2010,* provides regulation relating to 'prospecting for minerals, mining, processing and dealing in minerals, to granting, renewal and termination of mineral rights, payment of royalties, fees and other charges and any other relevant matters.

The *Mining Act, 2010* requires that an application for a mineral right (e.g. mining lease) shall be made to the Minister and shall include:

- Identification of the relevant prospecting license;
- A description of the area, a sketch and detail of the maximum area of the lease;
- Describe the mineral deposit;
- Provide a feasibility study detailing the programme of mining operations, estimated recovery rate and an estimate of the mineral quantity to be produced for sale annually; and
- A statement detailing the duration, not exceeding ten years, for which the mining license is sought.

The *Mining Act, 2010 (Art 50)* provides circumstances where a mining license may not be granted and includes:

- Licence areas that are excessive to mining the deposit identified;
- Unsatisfactory employment and training programme for citizens of Tanzania;
- Applications over existing mining licences; and
- Financial and technical resources available to the applicant are insufficient to conduct mining operations.

The Zonal and Resident Mines Offices for the South Western Zone are Mbeya and Chunya.

20.5 Health, Safety and Labour

The principal health and safety and labour laws applicable in the mining industry include:

- The Public Health Act, 2009;
- The HIV and AIDS (Prevention and Control) Act, 2008;
- The Occupational Health and Safety Act, 2003;
- The Workers Compensation Act, 2008;
- The Employment and Labour Relations Act, 2004; and
- Trade Unions Act, 1998.

20.6 Project Permitting Process

The primary certificates and permits required for the development of the Project include an Environmental Impact Assessment Certificate and a Mining License. The Environmental Impact Assessment Certificate is granted by the Minster subsequent to the EIS review. The Minster shall notify the proponent of the outcome within 30 days of the EIS review. The EIS review shall be conducted by the Council within 60 days of the EIS submission.

To be granted a mining license, the proponent must submit a mining license application to the Minister. The application shall identify the area of application, include sketches, describe the mineral deposit and shall provide a feasibility study. The *Mining Act, 2010 (Art 50)* details the granting of a mining license and provides the responsibilities of the Minister in this regard (through Article 71), however, no distinct time periods for the processing of new applications is provided.

20.6.1 Baseline Studies

Detailed baseline studies are required for the development of an EIS. The required baseline studies include:

- Socioeconomic,
- Health; and
- Environment (climate, air quality, noise, hydrology, soil, fauna and flora).

Data collection is to be performed by approved experts under the *Mining Act, 2010* and the methods of data collection are to be negotiated with the Minister.

20.6.2 Surface Water

The dominant surface water feature of Mbeya is Lake Rukwa (west of the Project). Lake Rukwa is an endorheic basin having no external flows. Inflow occurs from the Lukwate, Rungwa, Wuku, Luika, Kikambo, Chambua, Luiche, Songwe and Kavuu rivers. The lake is approximately 6,000 km^2 with an average depth of 3 to 5 meters. The lake is considered cyclical in size, in previous years contracting to 50 km in length and also expanding to 135 km in length. The lake provides a surrounding drainage basin of approximately 81,000 km^2.

Several small creeks run seasonally throughout the wet season across the current exploration licenses.

20.6.3 Flora

The Project site consists primarily of brushland and shrub thicket, woodlands, wooded and brush grassland. A botanical survey has not been conducted to date. SRK does not consider this an issue for this level of study. SRK would consider it normal practice to include a botanical survey in an EIS document as part of future studies (i.e. commencement of the botanical survey as part of a pre-feasibility study, completion to be included in a feasibility study document).

20.6.4 Fauna

A fauna survey has not been completed on the project to date. SRK does not consider this an issue for this level of study. SRK would consider it normal practice to include a fauna survey in an EIS document as part of future studies (i.e. commencement of the fauna survey as part of a pre-feasibility study, completion to be included in a feasibility study document).

20.7 Monitoring

The client will have an obligation to carry out environmental monitoring in consultation with the NEMC. Monitoring will assess the possible change and potential impact to the environment. A Monitoring Plan (MP) will be developed and submitted with the EIS.

20.8 Rehabilitation and Closure

The client will be required to deposit an environmental performance bond with the Director of Environment as security for good environmental practice. Site rehabilitation to its natural state is the responsibility of the operator and is to be conducted at their own expense. The Director of Environment shall not discharge the environmental performance bond until the site satisfactory meets the conditions of performance as set by the Minister.

Requirements and costs of closure and rehabilitation will be presented in the EIS.

20.9 Current Status and SRK Comment

SRK is not aware of any completed or ongoing environmental compliance studies for the project. SRK considers this acceptable for a project of this level of early stage development. SRK expects an EIS will be completed as part of future studies (Prefeasibility and Feasibility).

21 Capital and Operating Costs

21.1 Base Case

21.1.1 Base Case Capital Costs

Table 21-1 summarises the estimated capital requirements for the project. The capital costs are expressed in US dollars and have the following provisions:

- Capital estimates are given in real June 2012 USD terms;
- The capital expenditure is phased according to project requirements and timing; and
- A flat contingency of 15% has been applied to all capital requirements.

Note: SRK recommends caution with this level of contingency; however, notes that this figure is broadly in line with other industry studies of this development.

Table 21-1: Summary of Estimated Capital Costs (Base Case)

Item	Project Capital (USDM)
Processing Facility	57.3
Tailings Disposal Facility	9.4
Laboratory	0.5
Mobile Equipment Workshop and Stores	3.5
Power Supply	3.5
Roads and Access	1.0
General Buildings	2.2
Fuel Storage and Distribution	0.5
Communications	0.5
Dewatering Infrastructure	0.5
General Site facilities upgrade	3.0
Contractor Mobilisation	5.0
Contractor Demobilisation	5.0
Sustaining Capital	32.8
Subtotal	**124.7**
Contingency (15%)	18.7
Total	**143.4**

21.1.2 Base Case Operating Costs

Table 21-2 and Table 21-3 summarise the estimated operating requirements for the project. The operating costs are expressed in US dollars and have the following provisions:

- Operating estimates are given in real June 2012 USD terms; and
- Contingencies of 5% have been applied to the mining, processing, tailings and environmental and social costs.

Note: SRK recommends caution with this level of contingency, however notes that this figure is broadly in line with other industry studies prepared at this stage of a project's development.

Table 21-2: Summary of Operating Costs (Base Case)

Description	Total (USDM)	Unit Cost (USD / t $_{processed}$)
Mining	161.6	14.44
Processing	115.3	10.30
Tailings Disposal	3.9	0.35
Admin (includes environmental)	56.0	5.00
Total Cash Operating Cost (before Contingency)	336.8	30.09
Contingency (@ 5%)	16.8	1.50
Royalty	34.9	3.12
Total Cash Operating Cost	**388.5**	**34.71**

Table 21-3: Summary of Operating Costs (per ounce) (Base Case)

Description	$ / oz
Mining	335.5
Processing	239.4
Tailings	8.1
Admin (including environmental)	116.2
Total Cash Operating Cost (before contingency)	**699.2**
Contingency	35.0
Royalty	72.5
Total Operating Cost	**806.7**

21.2 Upside Potential Case

21.2.1 Upside Potential Capital Costs

Table 21-4 summarises the estimated capital requirements for the project. The capital costs are expressed in US dollars and have the following provisions:

- Capital estimates are given in real June 2012 USD terms;
- The capital expenditure is phased according to project requirements and timing; and
- A flat contingency of 15% has been applied to all capital requirements.

Table 21-4: Summary of Estimated Capital Costs (Upside Potential Case)

Item	Project Capital (USDM)
Processing Facility	73.1
Tailings Disposal Facility	13.0
Laboratory	1.0
Mobile Equipment Workshop and Stores	3.75
Power Supply	3.5
Roads and Access	2.0
General Buildings	4.0
Fuel Storage and Distribution	1.25
Communications	1.0
Dewatering Infrastructure	0.5
General site facilities upgrade	3.0
Contractor Mobilisation	5.0
Contractor Demobilisation	5.0
Sustaining Capital	52.8
Subtotal	**168.9**
Contingency (15%)	25.3
Total	**194.2**

21.2.2 Upside Potential Operating Costs

Table 21-5 and

Table 21-6 summarise the estimated operating requirements for the project. The capital costs are expressed in US dollars and have the following provisions:

- Operating estimates are given in real June 2012 USD terms; and
- Contingencies of 5% have been applied to the mining, processing, tailings and environmental and social costs.

Note: SRK recommends caution with this level of contingency, however notes that this figure is broadly in line with other industry studies prepared at this stage of a project's development.

Table 21-5: Summary of Operating Costs (Upside Potential Case)

Description	Total (USDM)	Unit Cost (USD / t processed)
Mining	224.1	11.63
Processing	198.0	10.28
Tailings Disposal	6.7	0.35
Admin (includes environmental)	96.4	5.00
Total Cash Operating Cost (before Contingency)	525.2	27.25
Contingency (@ 5%)	26.3	1.36
Royalty	55.0	2.86
Total Cash Operating Cost	**606.5**	**31.47**

Table 21-6: Summary of Operating Costs (per ounce) (Upside Potential Case)

Description	$ / oz
Mining	295.2
Processing	260.9
Tailings	8.9
Admin (including environmental)	127.0
Total Cash Operating Cost (before contingency)	**692.0**
Contingency	34.6
Royalty	72.5
Total Operating Cost	**799.1**

22 Economic Analysis

The PEA and production schedule evaluate Measured and Indicated Mineral Resources only. Inferred Mineral Resources have not been included. SRK notes this assessment is preliminary in nature and there is no certainty that the preliminary assessment will be realised. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

SRK notes that the technical work was focussed on recovering Measured and Indicated ounces.

22.1 Base Case Fiscal and Economic Parameters

22.1.1 Royalties

A mineral royalty of 5% has been applied. This includes a 4% Royalty payable to the Government, and 1% to be paid as a Royalty to the previous owner.

22.1.2 Taxes

All economic analysis has been conducted on a pre-tax basis.

22.1.3 Currency

All costs have been estimated in US Dollars.

22.1.4 Inflation

All costs have been estimated in real terms, i.e., inflation has been ignored for all economic analysis.

22.1.5 Project Timing

Key assumptions for project timing used in developing the financial model include:

- Process facility commissioned at the end of Year 1;
- First ore mined in Year 2; and
- Project life is 9 years including the initial year of mining prestrip and infrastructure construction.

22.1.6 Financial Model

Table 22-1 presents the pre-tax pre-finance cashflow model for the project.

The following assumptions were made in the development of the model:

- Pre-tax financial model;
- 100% equity financed, therefore no financing costs have been incorporated;
- No working capital requirements have been included;
- Financial year runs 1 January to 31 December; and
- Sustaining capital requirements have been estimated at 5% per year of the initial capital costs (excluding the final year);

Table 22-1: Base Case Financial Model

	Units	Total	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
Gold	USD / oz		1,450	1,450	1,450	1,450	1,450	1,450	1,450	1,450	1,450
Production											
Mining											
Ore Tonnes	t	11,193,026	120,980	863,298	1,519,154	1,465,715	1,807,184	1,741,963	1,626,295	1,325,148	723,289
Waste Tonnes	t	73,420,723	8,551,520	9,174,202	6,715,846	13,134,285	10,967,816	9,208,037	5,693,705	6,274,706	3,700,606
Stockpile Overhaul	t	559,720	0	0	80,846	103,432	0	0	0	274,852	100,590
Mining Ore Grade	g/t		1.06	1.15	1.66	1.50	1.30	1.37	1.51	1.41	1.32
Processing											
Processing Capacity	t	12,000,000	0	800,000	1,600,000	1,600,000	1,600,000	1,600,000	1,600,000	1,600,000	1,600,000
Processed Tonnes	t	11,193,026	0	800,000	1,600,000	1,569,147	1,600,000	1,600,000	1,600,000	1,600,000	823,879
Processed Ore Grade	g/t		0.00	1.15	1.63	1.47	1.30	1.37	1.51	1.40	1.32
Gold metal produced	oz	509,688	0	29,556	83,897	74,353	66,849	70,443	77,765	71,908	34,916
Gold recovery to Dore	%	94.5%	0	94.5%	94.5%	94.5%	94.5%	94.5%	94.5%	94.5%	94.5%
Gold in Dore	oz	481,655	0	27,930	79,282	70,264	63,173	66,569	73,488	67,954	32,995
Subtotal Paid Metal Value	USD M	698	0.0	40.5	115.0	101.9	91.6	96.5	106.6	98.5	47.8
Capital Costs											
Processing Plant	USD M	57.29	57.3	0	0	0	0	0	0	0	0
Site Infrastructure	USD M	24.63	24.6	0	0	0	0	0	0	0	0
Contractor Mobilisation	USD M	5	5.0	0	0	0	0	0	0	0	0
Contractor Demobilisation	USD M	5	0.0	0	0	0	0	0	0	0	5
Sustaining Capital	USD M	32.8	0.0	4.1	4.1	4.1	4.1	4.1	4.1	4.1	4.1
Contingency	USD M	18.7	13.0	0.6	0.6	0.6	0.6	0.6	0.6	0.6	1.4
Subtotal Capital Costs	USD M	143.4	100.0	4.7	4.7	4.7	4.7	4.7	4.7	4.7	10.5

Operating Costs											
Mining Cost	USD M	161.6	15.9	18.1	16.4	23.2	22.2	19.9	15.7	17.0	13.1
Processing Cost	USD M	115.3	0	8.2	16.6	16.2	16.4	16.5	16.5	16.5	8.5
Tailings Disposal	USD M	3.9	0	0.3	0.6	0.5	0.6	0.6	0.6	0.6	0.3
Administration	USD M	56.0	0	4.0	8.0	7.8	8.0	8.0	8.0	8.0	4.1
Subtotal pre-contingency	USD M	336.8	15.9	30.5	41.5	47.8	47.2	45.0	40.8	42.0	26.0
Contingency	USD M	16.8	0.8	1.5	2.1	2.4	2.4	2.2	2.0	2.1	1.3
Subtotal Operating Costs	USD M	354	17	32	44	50	50	47	43	44	27
Subtotal Operating Costs	($/t)		1.93	3.19	5.29	3.44	3.88	4.31	5.86	5.80	6.18
Subtotal Operating Costs	($/t ore)		138.29	37.13	28.69	34.24	27.44	27.10	26.36	33.28	37.78

Operating Profit (before Depreciation)	USD M	344.8	-16.7	8.4	71.4	51.7	42.0	49.3	63.7	54.4	20.5

Royalty	0	5%	5%	5%	5%	5%	5%	5%	5%	5%	5%
Royalty Paid	USD M	34.9	0.0	2.0	5.7	5.1	4.6	4.8	5.3	4.9	2.4

Pretax Cashflow	USD M	166.5	-116.7	1.7	60.9	41.9	32.7	39.8	53.7	44.8	7.7
Discount Rate	8%										

Cumulative Cashflow	USD M	0	-116.7	-115.0	-54.1	-12.2	20.6	60.3	114.0	158.8	166.5

Net Present Value	85.6	USD M
Internal Rate of Return	24%	

Figure 22-1 and Table 22-2 show the sensitivity analysis performed on the project with respect to Net Present Value (NPV) and Internal Rate of Return (IRR).

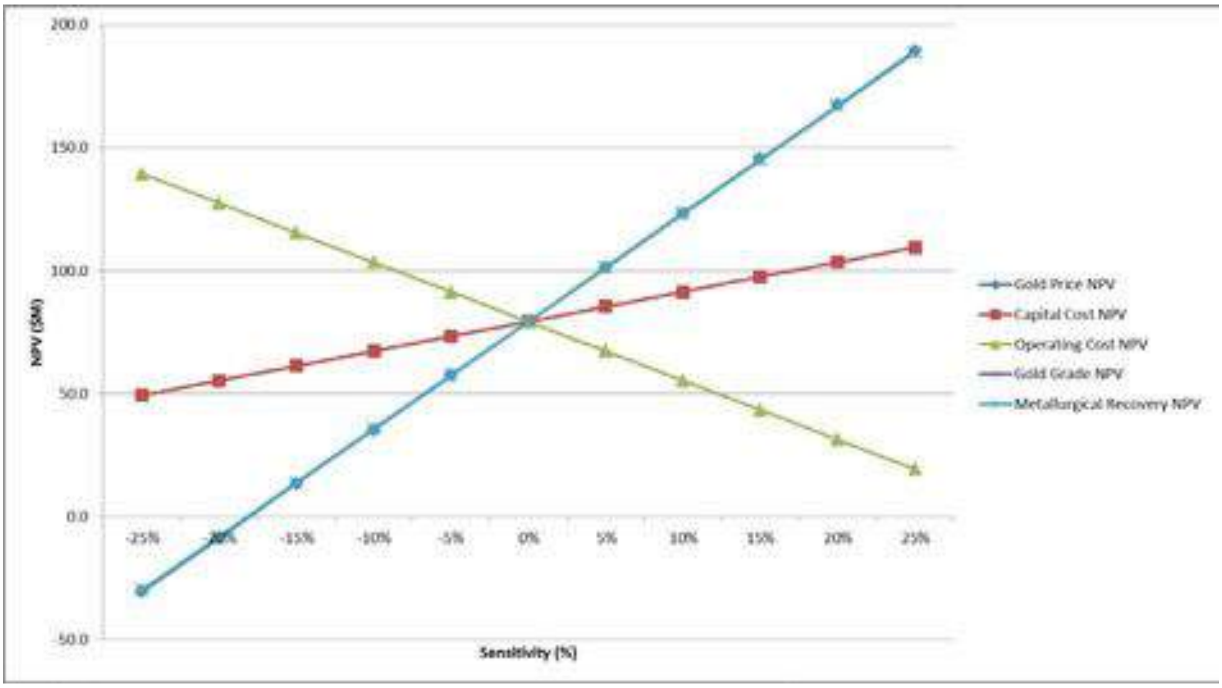

Figure 22-1: Base Case Financial Sensitivity Analysis

Table 22-2: Base Case Financial Sensitivity Analysis

	-25%	-20%	-15%	-10%	-5%	0%	5%	10%	15%	20%	25%
Gold Price NPV (USDM)	-33.3	-9.5	14.3	38.1	61.8	85.6	109.4	133.1	156.9	180.7	204.5
Gold Price IRR (%)	0%	6%	11%	16%	20%	24%	28%	31%	35%	38%	42%
Capital Cost NPV (USDM)	53.1	59.6	66.1	72.6	79.1	85.6	92.1	98.6	105.1	111.6	118.1
Capital Cost IRR (%)	17%	18%	19%	21%	22%	24%	26%	27%	29%	31%	34%
Operating Cost NPV (USDM)	150.5	137.5	124.6	111.6	98.6	85.6	72.6	59.6	46.7	33.7	20.7
Operating Cost IRR (%)	35%	33%	31%	28%	26%	24%	22%	19%	17%	15%	12%
Gold Grade NPV (USDM)	-32.6	-9.0	14.7	38.3	62.0	85.6	109.2	132.9	156.5	180.2	203.8
Gold Grade IRR (%)	0%	6%	11%	16%	20%	24%	28%	31%	35%	38%	42%
Metallurgical Recovery NPV (USDM)	-32.6	-9.0	14.7	38.3	62.0	85.6	109.2	132.9	156.5	180.2	203.8
Metallurgical Recovery IRR (%)	42%	38%	35%	31%	28%	24%	20%	16%	11%	6%	0%

Table 22-3 details the sensitivity analysis of the project with respect to the discount rate.

Table 22-3: Base Case Discount Rate Analysis

	3%	4%	5%	6%	7%	8%	9%	10%	11%	12%	13%
Discount Rate NPV (USDM)	131.5	121.2	111.5	102.4	93.8	85.6	77.9	70.6	63.7	57.1	50.9

22.2 Upside Potential Fiscal and Economic Parameters

22.2.1 Royalties

A mineral royalty of 5% has been applied. This includes a 4% Royalty payable to the Government, and 1% to be paid as a Royalty to the previous owner.

22.2.2 Taxes

All economic analysis has been conducted on a pre-tax basis.

22.2.3 Currency

All costs have been estimated in US Dollars.

22.2.4 Inflation

All costs have been estimated in real terms, i.e., inflation has been ignored for all economic analysis.

22.2.5 Project Timing

Key assumptions for project timing used in developing the financial model include:

- Process facility commissioned at the end of Year 1;
- First ore mined in Year 2; and
- Project life is 10 years including the initial year of mining prestrip and infrastructure construction.

22.2.6 Financial Model

Table 22-4 presents the pre-tax pre-finance cashflow model for the project.

The following assumptions were made in the development of the model:

- Pre-tax financial model;
- 100% equity financed, therefore no financing costs have been incorporated;
- No working capital requirements have been included; and
- Financial year runs 1 January to 31 December;

Sustaining capital requirements have been estimated at 5% per year of the initial capital costs (excluding the final year);

Table 22-4: Upside Potential Financial Model

	Units	Total	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
Gold	USD / oz		1,450	1,450	1,450	1,450	1,450	1,450	1,450	1,450	1,450	1,450
Production												
Mining												
Ore Tonnes	t	19,272,620	259,712	1,154,887	2,007,679	2,529,076	2,566,436	2,314,958	2,468,442	2,425,098	2,506,538	1,039,794
Waste Tonnes	t	100,484,904	17,990,288	13,445,113	14,462,321	13,895,924	14,771,064	10,460,042	5,766,558	6,167,459	2,968,462	557,672
Stockpile Overhaul	t	1,117,868	0	0	0	0	0	85,042	0	0	0	1,032,826
Mining Ore Grade	g/t		1.09	1.23	1.51	1.17	1.25	1.34	1.23	1.28	1.43	1.18
Processing												
Processing Capacity	t	19,600,000	0	1,000,000	1,800,000	2,400,000	2,400,000	2,400,000	2,400,000	2,400,000	2,400,000	2,400,000
Processed Tonnes	t	19,272,620	0	1,000,000	1,800,000	2,400,000	2,400,000	2,400,000	2,400,000	2,400,000	2,400,000	2,072,620
Processed Ore Grade	g/t		0.00	1.23	1.51	1.17	1.25	1.34	1.23	1.28	1.43	1.22
Gold metal produced	oz	803,146	0	39,598	87,430	90,470	96,479	103,264	95,232	98,882	110,209	81,582
Gold recovery to Dore	%	94.5%	0.0%	94.5%	94.5%	94.5%	94.5%	94.5%	94.5%	94.5%	94.5%	94.5%
Gold in Dore	oz	758,973	0	37,420	82,621	85,494	91,173	97,584	89,994	93,444	104,147	77,095
Subtotal Paid Metal Value	USD M	1,101	0.0	54.3	119.8	124.0	132.2	141.5	130.5	135.5	151.0	111.8

Capital Costs												
Processing Plant	USD M	73.07	73.1	0	0	0	0	0	0	0	0	0
Site Infrastructure	USD M	32.45	32.5	0	0	0	0	0	0	0	0	0
Contractor Mobilisation	USD M	5	5.0	0	0	0	0	0	0	0	0	0
Contractor Demobilisation	USD M	5	0.0	0	0	0	0	0	0	0	0	5
Sustaining Capital	USD M	47.5	0.0	5.3	5.3	5.3	5.3	5.3	5.3	5.3	5.3	5.3
Contingency	USD M	24.5	16.6	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	1.5
Subtotal Capital Costs	USD M	187.5	127.1	6.1	6.1	6.1	6.1	6.1	6.1	6.1	6.1	11.8

Operating Costs												
Mining Cost	USD M	224.1	27.6	24.1	28.3	29.1	29.1	23.5	17.8	18.3	15.5	10.7
Processing Cost	USD M	198.0	0	10.3	18.6	24.6	24.6	24.7	24.6	24.7	24.7	21.3
Tailings Disposal	USD M	6.7	0	0.4	0.6	0.8	0.8	0.8	0.8	0.8	0.8	0.7
Administration	USD M	96.4	0	5.0	9.0	12.0	12.0	12.0	12.0	12.0	12.0	10.4
Subtotal pre-contingency	USD M	525.2	27.6	39.8	56.5	66.5	66.6	61.1	55.3	55.8	53.0	43.1
Contingency	USD M	26.3	1.4	2.0	2.8	3.3	3.3	3.1	2.8	2.8	2.7	2.2
Subtotal Operating Costs	USD M	551	29	42	59	70	70	64	58	59	56	45
Subtotal Operating Costs	($/t)		1.59	2.86	3.60	4.25	4.03	5.02	7.05	6.82	10.17	28.30
Subtotal Operating Costs	($/t ore)		111.74	36.14	29.56	27.62	27.24	27.69	23.51	24.16	22.22	43.47

Operating Profit (before Depreciation	USD M	549.0	-29.0	12.5	60.5	54.1	62.3	77.4	72.5	76.9	95.3	66.6

Royalty	0	5%	5%	5%	5%	5%	5%	5%	5%	5%	5%	5%
Royalty Paid	USD M	55.0	0.0	2.7	6.0	6.2	6.6	7.1	6.5	6.8	7.6	5.6

Pretax Cashflow	USD M	306.6	-156.1	3.7	48.4	41.8	49.6	64.3	59.9	64.1	81.7	49.2
Discount Rate	8%											

Cumulative Cashflow	USD M	0	-156.1	-152.4	-104.0	-62.1	-12.5	51.7	111.6	175.7	257.4	306.6

Net Present Value	146.1	USD M
Internal Rate of Return	23%	

Figure 22-2 and Table 22-5 show the sensitivity analysis performed on the project with respect to Net Present Value (NPV) and Internal Rate of Return (IRR).

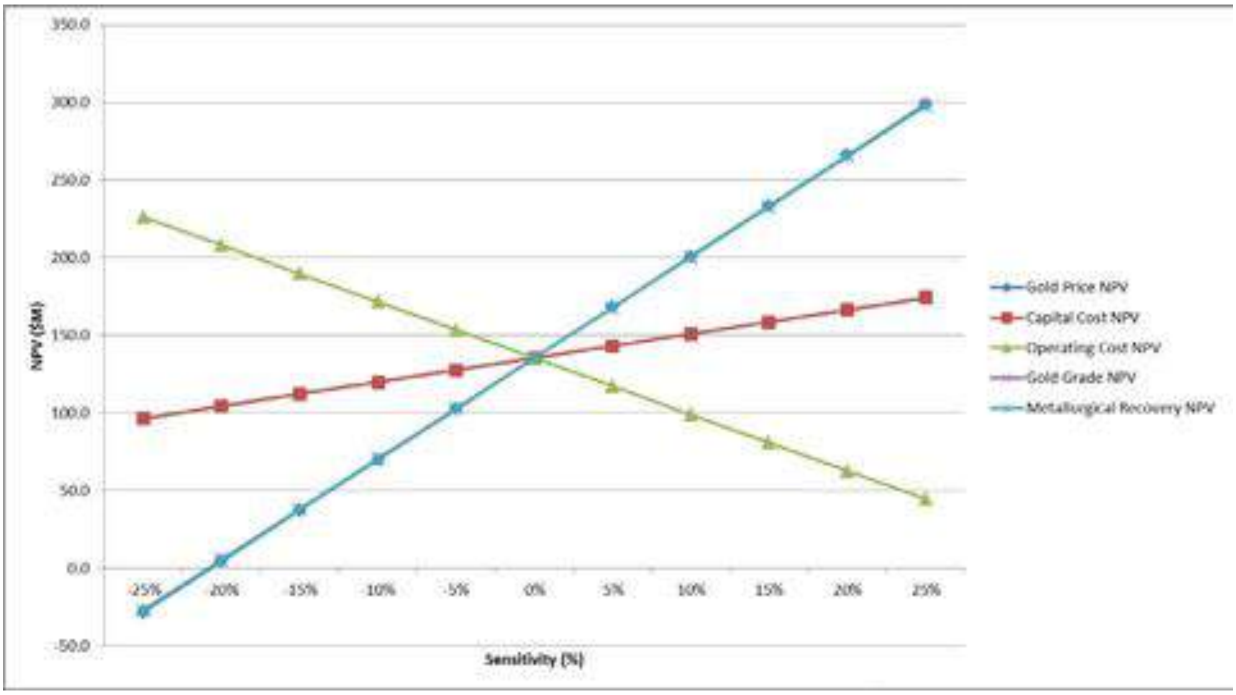

Figure 22-2: Base Case Financial Sensitivity Analysis

Table 22-5: Base Case Financial Sensitivity Analysis

	-25%	-20%	-15%	-10%	-5%	0%	5%	10%	15%	20%	25%
Gold Price NPV (USDM)	-30.5	4.8	40.1	75.5	110.8	146.1	181.4	216.8	252.1	287.4	322.7
Gold Price IRR (%)	4%	9%	13%	16%	20%	23%	26%	30%	33%	36%	38%
Capital Cost NPV (USDM)	104.1	112.5	120.9	129.3	137.7	146.1	154.5	162.9	171.3	179.7	188.1
Capital Cost IRR (%)	18%	19%	20%	21%	22%	23%	25%	26%	27%	29%	31%
Operating Cost NPV (USDM)	244.2	224.6	205.0	185.4	165.7	146.1	126.5	106.9	87.2	67.6	48.0
Operating Cost IRR (%)	33%	31%	29%	27%	25%	23%	21%	19%	17%	15%	13%
Gold Grade NPV (USDM)	-29.6	5.6	40.7	75.8	111.0	146.1	181.2	216.4	251.5	286.6	321.8
Gold Grade IRR (%)	4%	9%	13%	16%	20%	23%	26%	30%	33%	35%	38%
Metallurgical Recovery NPV (USDM)	-29.6	5.6	40.7	75.8	111.0	146.1	181.2	216.4	251.5	286.6	321.8
Metallurgical Recovery IRR (%)	4%	9%	13%	16%	20%	23%	26%	30%	33%	35%	38%

Table 22-6 details the sensitivity analysis of the Upside Potential Scenario with respect to the discount rate.

Table 22-6: Base Case Discount Rate Analysis

	3%	4%	5%	6%	7%	8%	9%	10%	11%	12%	13%
Discount Rate NPV (USDM)	235.4	214.9	195.9	178.1	161.6	146.1	131.7	118.1	105.5	93.6	82.5

23 Adjacent Properties

The SMP is in the north western portion of the Lupa Goldfield, it surrounded on all sides by prospecting licenses which are held by Tanzanian registered companies and individuals.

The Luika gold mine which is located to the West of the Kenge deposit is currently being developed by Shanta Mining Corporation Limited, an AIM listed company.

The first gold has been poured at this site 31 August, 2012.

24 Other Relevant Data and Information

There is no other relevant data available about the SMP deposit Project.

25 Interpretation and Conclusions

25.1 Geology

Helio acquired the first of the Prospecting Licenses for the Saza-Makongolosi Project (SMP) in 2006 and since then have carried out a series of exploration activities including twelve drilling programmes. The drillhole database supplied to SRK for the purpose of estimating a Mineral Resource contains 881 drillholes with a total length of 111,682 metres and 58,598 primary assay results. This database is entirely based on information collected by Helio. Data collected by previous operators within the SMP area are used by Helio for defining targets, but the historic exploration data are not verifiable, reliable or complete enough to be suitable for estimating Mineral Resources.

In September 2011 SRK visited the SMP site and the primary assay laboratory for the project. During these visits SRK reviewed the drilling, logging, sampling and assaying procedures. The qualified person considers these methods appropriate for collecting information to be used in a Mineral Resource estimate.

SRK carried out verification checks on the database, including comparing the records in the database files given to SRK against primary sources such as logging sheets and assay certificates. SRK also reviewed the results from QA/QC sampling programmes implemented by Helio, which include analyses of Certified Reference Material, blanks and duplicates. SRK recommends that Helio should also implement check assays by an umpire laboratory as part of the QA/QC programme. About 5 % of the pulverized samples prepared by the primary laboratory (AAL) should be sent to a second laboratory for analysis.

A second concern from the QA/QC review is that there appears to be a problem with mislabelling of standards and also of some blanks. For about 2% of the standards, the reported assay plots as an outlier that happens to neatly coincide with the value of a different standard. It is possible that the problem is confined to the QA/QC samples and is not representative of high rate of sample swaps also occurring among the primary assays, but identifying swaps among the primary samples is much more difficult than finding likely swaps among the standards.

Helio has recognised the problem of sample swaps among the standards and switched to a system where the logging geologist takes responsibility for the insertion of standards, instead of a field technician. From plotting of the analytical results in a time series, SRK identified that there is a noticeable decrease in the apparent frequency of standard swaps for the most recent two field seasons.

The concerns about the QA/QC data discussed above are not considered to be material and the qualified person's conclusion from the verification checks and the review of QA/QC data is that the database is of sufficient quality to support estimation of Mineral Resources.

The main objective of SRK's work was to update the Mineral Resource estimate for SMP. The total Mineral Resource estimate for all SMP targets, at a 0.5 g/t cut-off and with an effective date of February 10, 2012, is:

- Combined Measured and Indicated Mineral Resources of 24.1 Mt @ 1.32 g/t for 1,020,000 ounces Au; and
- Inferred Mineral Resources of 7.3 Mt @ 1.05 g/t for 250,000 ounces Au.

The results for the previous (effective data 30th November 2010) estimate, also at a 0.5 g/t cut-off were:

- Combined Measured and Indicated Mineral Resources of 10.8 Mt @ 1.43 g/t for 500,000 ounces Au; and
- Inferred Mineral Resources of 7.1 Mt @ 1.19 g/t for 270,000 ounces Au.

The previous Mineral Resource was prepared by Golder Associates (UK) Ltd. This previous Mineral Resource is now superseded by the February 10, 2012 Mineral Resource.

The main reason for the increase in Measured and Indicated tonnes and metal since the previous Mineral Resource estimate is the additional drilling Helio completed in 2011, particularly at the Porcupine target. A secondary reason for the increases is that SRK's interpretation of the mineralised zone at Kenge was a larger volume than the previous interpretation.

Helio has identified over 30 exploration targets within the area covered by the SMP licenses. In SRK's judgement, only five of these targets currently have sufficient information available to support the estimation of Mineral Resources: Porcupine, Kenge, Mbenge, Konokono and Tumbili. Considerable exploration potential remains at the targets where Mineral Resources have already been defined and also at many of the targets where as yet there is insufficient information for estimating Mineral Resources.

The Porcupine Main zone is open at depth. Closer-spaced drilling of some of the secondary mineralised structures around the Main zone (in particular the Quill zone) may add to the Mineral Resources defined for Porcupine.

The Kenge and Mbenge domains are also open at depth.

The Konokono target is currently covered by lines of RC drilling spaced 300 m apart. Around one of these lines several diamond holes have been drilled, about 25 m apart, which have made it possible to define an Inferred Mineral Resource. Tumbili is similar to Konokono: covered by lines of widely-spaced RC drilling (200 m) apart and around one of these lines four diamond holes have been drilled. An Inferred Mineral Resource was also defined for Tumbili. Further infill drilling at Konokono and Tumbili may lead to additional Mineral Resources being defined.

The Gap target, about 2 km northeast of Porcupine, has about 800 m of strike length that is covered by lines of RC and diamond drilling spaced 100 m apart. Further drilling at Gap may make it possible to estimate an initial Mineral Resource for this target.

25.2 Project

Exploration, block model interpretation and study work completed to date, including optimisation, mine design and mine scheduling indicate that the Project is economically viable.

A year has been allocated for process facility and other associated infrastructure development, including prestrip. Nine (9) years of ore production at approximately 1.6 Mt per annum follows, based on the current Resource Statement. The Project is anticipated to recover approximately 480 k oz. of gold over the life of the Project.

The PEA and production schedule evaluate Measured and Indicated Mineral Resources only. Inferred Mineral Resources have not been included. SRK notes this assessment is preliminary in nature and there is no certainty that the preliminary assessment will be realised. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

A preliminary high level cashflow financial model has been developed. Using a gold price of $1,450 / oz and a discount rate of 8%, the project reports the following:

- NPV of USD85.6M; and
- IRR of 24%.

Engineering and estimating has been undertaken to define the project scope and develop cost estimates, sufficient to support a Preliminary Economic Model (+/- 50%).

26 Recommendations

26.1 Geology

Helio's goal for the next twelve months or so is to substantially increase the inventory of Mineral Resources for the SMP, with particularly focus on defining higher-grade zones. SRK recommends the exploration program in Table 26-1 below. The recommended items are all single-phase.

Table 26-1: Estimated cost of the exploration program proposed for the SMP

Item	Comments	Drill Method	All-in cost per metre (USD)	Metres	Budget (USD)
Porcupine	Drilling of high grade zones in and around Porcupine Main domain	DD	300	2,000	600,000
Kenge	Drilling of high grade zones below current depth extent of Kenge domains	DD	300	1,500	450,000
Saza Mine	Drilling below old workings	DD	300	1,500	450,000
Konokono	Infill drilling to define additional Mineral Resources	RC	160	3,000	480,000
Tumbili	Infill drilling to define additional Mineral Resources	RC	160	3,000	480,000
Gap	Infill drilling to define initial Mineral Resources for this target	RC	160	3,000	480,000
Consulting Services	Update Mineral Resource estimate, prepare a Preliminary Economic Assessment, compile Technical Report				160,000
Total				**14,000**	**3,100,000**

26.2 Further technical studies

Based on the technical work completed to date, SRK recommends the following:

- Development of a metallurgical flowsheet to increase the understanding level of the processing requirements, including costs and recoveries;
- Development of a tailings disposal system, including costs and infrastructure requirements;
- Condemnation drilling of process facility and key infrastructure sites;
- Inclusion of a geotechnical drilling program to understand the geotechnical constraints for each of the deposits;
- Consider a geotechnical investigation for key infrastructure, including process facility, mobile equipment workshop and tailings storage facility;
- Development of an environmental management program, including baseline studies to understand the impact mining operations will have on the local environment. This will also assist in early identification of any environmental restrictions;
- Engage Tanzanian (or nearby) earthmoving contractors to better understand local cost of mining and productivities;
- Update the optimisation process and mine design process based on the revised values from the further investigation (processing, mining costs etc);
- Continued engagement with local landholders and resident of the local towns and villages;

- Continued engagement with government authorities relating to securing connection to the high voltage national power grid; and
- Assessment of ground and surface water volumes and qualities on site in conjunction with process and mine requirements, to understand the water balance of the site.

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Appendices

Appendix A: List of SMP Drillholes Grid system used for collar coordinates is UTM Zone 36 South, WGS84 datum

Hole Name	Drill Type	Collar X	Collar Y	Collar Z	Maximum Depth (m)	Dip	Azimuth	Intersects a Mineralised Domain?
GPD001	DD	514071	9077020	1224	121	-49	340	Yes
GPD002	DD	514071	9077019	1224	158	-70	342	Yes
GPD003	DD	514023	9077003	1225	126	-48	337	Yes
GPD004	DD	514062	9076966	1224	245	-48	341	Yes
GPD005	DD	514112	9077033	1216	40	-47	338	Yes
GPD005A	DD	514112	9077033	1216	198	-46	337	Yes
GPD006	DD	514173	9077064	1207	145	-50	343	
GPD007	DD	514219	9077132	1203	151	-50	342	
GPD008	DD	514164	9077129	1206	258	-48	342	Yes
GPD009	DD	513997	9076969	1218	208	-48	341	Yes
GPD010	DD	514112	9076989	1217	142	-48	342	Yes
GPD011	DD	513596	9076795	1197	243	-48	2	
GPD012	DD	513613	9076990	1194	218	-49	2	Yes
GPD013	DD	513596	9077007	1193	202	-45	182	
GPD014	DD	513606	9077204	1181	200	-48	181	
GPD015	DD	514112	9076904	1232	218	-49	339	Yes
GPD016	DD	514114	9076901	1231	222	-60	337	Yes
GPD017	DD	514113	9076902	1230	278	-75	340	Yes
GPD018	DD	514149	9076944	1224	132	-49	339	
GPD018A	DD	514149	9076944	1224	204	-49	341	Yes
GPD019	DD	514149	9076942	1224	213	-60	340	Yes
GPD020	DD	514152	9077005	1212	137	-49	336	Yes
GPD021	DD	514152	9077006	1212	175	-60	334	Yes
GPD022	DD	514204	9077018	1218	163	-51	340	Yes
GPD023	DD	514200	9076968	1221	217	-48	342	Yes
GPD024	DD	514252	9077037	1212	140	-49	342	Yes
GPD025	DD	514295	9077052	1214	187	-51	340	
GPD026A	DD	514200	9076967	1222	227	-60	340	Yes
GPD027	DD	514254	9076987	1225	221	-50	341	
GPD028	DD	514254	9076987	1225	240	-60	341	Yes
GPD029	DD	514026	9076944	1221	241	-48	341	Yes
GPD030	DD	514060	9076912	1228	273	-51	339	Yes
GPD031	DD	514060	9076912	1227	225	-64	340	Yes
GPD032	DD	514010	9076892	1223	306	-47	339	Yes
GPD033	DD	514010	9076892	1223	239	-61	342	Yes
GPD034	DD	514144	9076891	1234	258	-47	336	Yes
GPD035	DD	514095	9076871	1238	245	-47	342	Yes
GPD036	DD	514067	9076850	1237	260	-48	340	Yes
GPD037	DD	514067	9076849	1238	301	-76	340	Yes
GPD038	DD	513631	9076971	1199	117	-48	341	Yes
GPD039	DD	514005	9076836	1237	230	-48	341	Yes
GPD040	DD	513964	9076868	1217	219	-49	341	Yes
GPD041	DD	513951	9076826	1237	300	-47	337	Yes

Hole Name	Drill Type	Collar X	Collar Y	Collar Z	Maximum Depth (m)	Dip	Azimuth	Intersects a Mineralised Domain?
GPD042	DD	514006	9076836	1236	288	-65	338	Yes
GPD043	DD	514071	9076850	1237	275	-60	337	Yes
GPD044	DD	514188	9076901	1239	315	-65	338	Yes
GPD045	DD	513803	9077080	1204	239	-49	338	Yes
GPD046	DD	513719	9077043	1193	231	-48	339	
GPD047	DD	513905	9077112	1207	202	-48	339	
GPD048	DD	514188	9076905	1239	483	-79	340	Yes
GPD049	DD	514200	9076967	1223	335	-89	109	Yes
GPD050	DD	514245	9076986	1220	304	-76	341	Yes
GPD051	DD	514245	9076986	1220	407	-89	335	Yes
GPD052	DD	514317	9077008	1226	517	-88	29	Yes
GPD053	DD	513965	9076868	1217	260	-70	342	Yes
GPD054	DD	513951	9076819	1242	344	-69	347	Yes
GPD055	DD	513806	9076897	1210	451	-49	2	Yes
GPD056	DD	515410	9076798	1386	413	-49	360	
GPD057	DD	514410	9077043	1228	317	-60	341	
GPD058	DD	515697	9076854	1141	275	-47	3	
GPD059	DD	515698	9077111	1305	131	-49	181	
GPD060	DD	515152	9076901	1347	110	-45	357	
GPD061	DD	515143	9077012	1330	81	-48	182	
GPD063	DD	514110	9077029	1213	218	-50	269	Yes
GPD064	DD	514202	9077018	1216	260	-50	276	Yes
GPD065	DD	514317	9077007	1222	361	-51	275	Yes
GPD066	DD	514112	9077026	1217	209	-71	224	Yes
GPD067	DD	514072	9077101	1216	251	-48	223	
GPD068	DD	517050	9077843	1249	190	-49	166	
GPD069	DD	517087	9077651	1248	183	-47	346	
GPD070	DD	513797	9077191	1194	221	-50	180	Yes
GPD071	DD	513960	9077171	1200	221	-47	221	
GPD072	DD	517321	9077403	1239	148	-47	357	
GPD073	DD	514101	9077144	1205	112	-49	222	
GPD074	DD	517317	9077537	1243	141	-49	176	
GPD075	DD	514553	9077552	1213	172	-50	181	
GPD076	DD	514651	9077538	1212	150	-49	182	
GPD077	DD	517603	9077610	1239	201	-47	183	
GPD078	DD	514293	9077051	1215	266	-49	271	Yes
GPD079	DD	517601	9077374	1235	201	-49	1	
GPD080	DD	514288	9077102	1209	220	-49	271	Yes
GPD081	DD	517606	9077378	1236	157	-47	185	
GPD082	DD	517605	9077229	1232	131	-46	3	
GPD083	DD	517649	9077155	1232	306	-47	216	
GPD084	DD	517302	9077232	1234	300	-47	244	
GPD085	DD	516952	9077619	1252	161	-48	231	

Hole Name	Drill Type	Collar X	Collar Y	Collar Z	Maximum Depth (m)	Dip	Azimuth	Intersects a Mineralised Domain?
GPD086	DD	516896	9077833	1252	170	-47	232	
GPD087	DD	517423	9076852	1238	306	-48	35	
GPD088	DD	516955	9077100	1252	306	-48	65	
GPD089	DD	516733	9077703	1249	191	-47	48	
GPD090	DD	516808	9077511	1238	191	-48	51	
GPD091	DD	514024	9076943	1221	523	-69	219	
GPD092	DD	514023	9077000	1221	353	-70	220	Yes
GPD093	DD	513963	9076960	1215	360	-70	218	Yes
GPD094	DD	513843	9076844	1231	252	-50	358	Yes
GPD095	DD	513849	9076842	1231	149	-61	356	Yes
GPD096	DD	513838	9077214	1202	101	-51	181	Yes
GPD097	DD	513839	9077215	1201	104	-64	183	Yes
GPD098	DD	513900	9077241	1199	154	-48	176	Yes
GPD099	DD	513901	9077243	1199	135	-55	177	Yes
GPD100	DD	513949	9077239	1192	139	-48	179	
GPD101	DD	513950	9077239	1192	149	-60	179	
GPD102	DD	514001	9077251	1191	131	-48	177	
GPD103	DD	514001	9077252	1191	153	-73	175	
GPD104	DD	514199	9077202	1198	100	-48	178	
GPD105	DD	514200	9077200	1198	100	-48	1	
GPD106	DD	514201	9077303	1202	100	-50	177	
GPD107	DD	514198	9077300	1201	100	-50	355	
GPD108	DD	514201	9077402	1207	101	-49	180	
GPD109	DD	514200	9077400	1207	101	-47	358	
GPD110	DD	514201	9077508	1208	101	-48	181	
GPD111	DD	513942	9076945	1214	212	-48	338	Yes
GPD112	DD	513892	9076916	1216	206	-46	337	Yes
GPD113	DD	513846	9076847	1221	200	-88	113	
GPD114	DD	513881	9076834	1229	141	-48	3	Yes
GPD115	DD	513754	9076836	1211	124	-48	2	
GPD116	DD	515899	9077791	1259	120	-46	3	
GPD117	DD	515899	9077791	1259	152	-74	6	
GPD118	DD	515798	9077771	1252	119	-49	359	
GPD119	DD	515699	9077748	1252	106	-49	1	
GPD120	DD	515601	9077749	1250	146	-44	354	
GPD121	DD	511953	9076896	1175	250	-48	360	
GPD122	DD	515501	9077742	1252	135	-49	356	
GPD123	DD	516004	9077781	1257	203	-49	356	
GPD124	DD	511960	9077148	1220	250	-48	183	
GPD125	DD	516100	9077782	1257	201	-49	357	
GPD126	DD	513753	9077179	1193	154	-47	177	Yes
GPD127	DD	513649	9077143	1185	118	-50	183	
GPD128	DD	516550	9077999	1258	100	-48	180	

Hole Name	Drill Type	Collar X	Collar Y	Collar Z	Maximum Depth (m)	Dip	Azimuth	Intersects a Mineralised Domain?
GPD129	DD	513556	9077129	1180	130	-50	180	
GPD130	DD	511655	9077998	1258	101	-50	355	
GPD131	DD	513457	9077090	1178	109	-46	181	
GPD132	DD	516550	9077903	1259	102	-48	356	
GPD133	DD	516555	9077803	1259	100	-49	358	
GPD134	DD	516549	9077901	1258	100	-49	177	
GPD135	DD	516553	9078101	1258	100	-49	178	
GPR001	RC	516135	9078102	1297	76	-51	4	
GPR002	RC	516142	9078154	1298	50	-50	182	
GPR003	RC	515895	9077798	1287	124	-51	2	
GPR004	RC	514068	9077026	1237	80	-51	336	Yes
GPR005	RC	518264	9078452	1273	40	-50	180	
GPR006	RC	518023	9078301	1270	40	-50	179	
GPR007	RC	513397	9076805	1192	87	-50	360	
GPR008	RC	513399	9076900	1190	89	-50	360	
GPR009	RC	513400	9076898	1190	88	-50	180	
GPR010	RC	513398	9076997	1190	98	-50	360	
GPR011	RC	513400	9076994	1193	85	-50	180	
GPR012	RC	513402	9077105	1187	92	-50	180	
GPR013	RC	513790	9076798	1232	96	-50	360	
GPR014	RC	513805	9076900	1223	96	-50	360	
GPR015	RC	513801	9076897	1222	80	-50	180	Yes
GPR016	RC	513796	9077014	1219	94	-50	360	
GPR016A	RC	513797	9077016	1215	12	-50	360	
GPR017	RC	513794	9077013	1214	90	-50	180	Yes
GPR018	RC	513792	9077115	1206	83	-50	360	Yes
GPR019	RC	513798	9077114	1191	90	-50	180	Yes
GPR020	RC	513796	9077193	1181	70	-50	180	Yes
GPR021	RC	514387	9077205	1220	54	-50	360	
GPR022	RC	514387	9077210	1219	83	-50	180	
GPR023	RC	514384	9077251	1217	42	-50	180	
GPR024	RC	514385	9077243	1218	52	-50	360	
GPR025	RC	514385	9077302	1217	50	-50	180	
GPR026	RC	514389	9077297	1217	90	-51	357	
GPR027	RC	513102	9077100	1179	90	-55	183	
GPR028	RC	513100	9077001	1173	90	-51	177	
GPR029	RC	513097	9076997	1174	90	-51	360	
GPR030	RC	513102	9076901	1178	90	-51	177	
GPR031	RC	513102	9076898	1178	100	-50	3	
GPR032	RC	513100	9076802	1174	90	-51	5	
GPR033	RC	514602	9077600	1234	90	-51	179	
GPR034	RC	514602	9077512	1227	90	-51	2	
GPR035	RC	514604	9077514	1227	105	-51	182	

Hole Name	Drill Type	Collar X	Collar Y	Collar Z	Maximum Depth (m)	Dip	Azimuth	Intersects a Mineralised Domain?
GPR036	RC	514601	9077402	1225	100	-51	8	
GPR037	RC	514603	9077406	1226	85	-51	183	
GPR038	RC	514603	9077301	1232	100	-50	5	
GPR039	RC	514604	9077304	1235	70	-51	182	
GPR040	RC	514598	9077214	1246	90	-50	3	
GPR041	RC	514603	9077217	1242	75	-51	179	
GPR042	RC	514999	9077902	1248	100	-50	181	
GPR043	RC	515000	9077801	1242	90	-48	8	
GPR044	RC	515002	9077804	1242	90	-49	181	
GPR045	RC	514998	9077698	1240	90	-50	3	
GPR046	RC	515001	9077700	1240	90	-50	179	
GPR047	RC	515000	9077592	1237	100	-50	2	
GPR048	RC	515002	9077594	1237	90	-51	180	
GPR049	RC	514998	9077498	1242	90	-47	360	
GPR050	RC	515001	9077501	1242	76	-51	176	
GPR051	RC	515007	9077423	1253	80	-52	178	
GPR052	RC	515006	9077420	1254	100	-57	359	
GPR053	RC	515400	9077996	1262	90	-51	178	
GPR054	RC	515398	9077918	1262	85	-50	360	
GPR055	RC	515401	9077920	1262	95	-51	177	
GPR056	RC	515399	9077799	1262	90	-51	2	
GPR057	RC	515401	9077803	1258	90	-53	175	
GPR058	RC	515400	9077702	1258	90	-52	359	
GPR059	RC	515401	9077705	1254	100	-51	177	
GPR060	RC	515400	9077599	1247	85	-52	3	
GPR061	RC	515403	9077488	1258	110	-53	360	
GPR062	RC	515400	9077490	1258	75	-52	178	
GPR063	RC	515398	9077402	1262	90	-51	359	
GPR064	RC	515401	9077403	1261	80	-52	179	
GPR065	RC	515396	9077311	1267	90	-51	1	
GPR066	RC	515399	9077314	1267	85	-53	179	
GPR067	RC	515400	9077221	1273	85	-50	1	
GPR068	RC	515402	9077224	1273	74	-52	175	
GPR069	RC	515397	9077604	1246	90	-50	173	
GPR070	RC	515697	9078097	1275	90	-52	177	
GPR071	RC	515698	9078000	1270	90	-51	359	
GPR072	RC	515701	9078002	1271	95	-53	176	
GPR073	RC	515699	9077900	1264	90	-51	358	
GPR074	RC	515702	9077900	1266	90	-50	181	
GPR075	RC	515697	9077822	1262	90	-51	180	
GPR076	RC	515698	9077698	1263	90	-50	1	
GPR077	RC	515702	9077700	1260	90	-51	178	
GPR078	RC	515694	9077821	1262	90	-52	0	

Hole Name	Drill Type	Collar X	Collar Y	Collar Z	Maximum Depth (m)	Dip	Azimuth	Intersects a Mineralised Domain?
GPR079	RC	515697	9077602	1259	90	-51	1	
GPR080	RC	516199	9078500	1293	90	-52	182	
GPR081	RC	516200	9078398	1298	90	-51	354	
GPR082	RC	516204	9078400	1298	90	-50	175	
GPR083	RC	516200	9078298	1290	90	-50	0	
GPR084	RC	516198	9078298	1284	90	-50	178	
GPR085	RC	516196	9078200	1280	90	-49	178	
GPR086	RC	516194	9078198	1280	90	-50	359	
GPR087	RC	516201	9078105	1279	90	-51	179	
GPR088	RC	516198	9078102	1278	90	-51	359	
GPR089	RC	516196	9078001	1278	90	-51	358	
GPR090	RC	516198	9078003	1278	90	-50	181	
GPR091	RC	516198	9077903	1272	90	-51	178	
GPR092	RC	516200	9077900	1272	90	-50	3	
GPR093	RC	516200	9077798	1273	90	-52	358	
GPR094	RC	516197	9077801	1274	90	-52	184	
GPR095	RC	516202	9077702	1272	90	-52	2	
GPR096	RC	515696	9078098	1275	90	-50	4	
GPR097	RC	515699	9078197	1279	90	-50	181	
GPR098	RC	515399	9077993	1264	90	-51	2	
GPR099	RC	515401	9078097	1275	100	-51	180	
GPR100	RC	514998	9077890	1248	90	-51	3	
GPR101	RC	514999	9077998	1254	90	-51	179	
GPR102	RC	513102	9077099	1179	90	-50	3	
GPR103	RC	513097	9077202	1187	100	-52	178	
GPR104	RC	513099	9077199	1188	90	-50	1	
GPR105	RC	513098	9077299	1183	90	-50	3	
GPR106	RC	513101	9077302	1183	90	-51	178	
GPR107	RC	513101	9077400	1199	100	-52	180	
GPR108	RC	513098	9077397	1194	90	-50	359	
GPR109	RC	513100	9077497	1214	100	-51	177	
GPR110	RC	513098	9077495	1215	90	-57	360	
GPR111	RC	513100	9077601	1210	94	-51	177	
GPR112	RC	513098	9077697	1221	100	-51	180	
GPR113	RC	513098	9077599	1209	90	-50	359	
GPR114	RC	514395	9077602	1232	90	-50	180	
GPR115	RC	514401	9077507	1226	90	-50	2	
GPR116	RC	514404	9077400	1214	90	-50	360	
GPR117	RC	514406	9077403	1214	90	-52	176	
GPR118	RC	514397	9077508	1226	100	-52	178	
GPR119	RC	514392	9077101	1222	150	-49	179	
GPR120	RC	514389	9077101	1225	100	-50	3	
GPR121	RC	515279	9077241	1273	80	-49	178	

Hole Name	Drill Type	Collar X	Collar Y	Collar Z	Maximum Depth (m)	Dip	Azimuth	Intersects a Mineralised Domain?
GPR122	RC	515279	9077145	1287	99	-49	358	
GPR123	RC	516346	9077102	1292	80	-49	176	
GPR124	RC	516346	9077004	1302	95	-49	3	
GPR125	RC	516349	9077007	1302	75	-49	178	
GPR126	RC	516353	9076916	1317	95	-51	0	
GPR127	RC	516844	9078523	1275	75	-50	356	
GPR128	RC	516850	9078603	1273	75	-51	178	
GPR129	RC	516847	9078601	1273	85	-50	356	
GPR130	RC	516850	9078694	1269	80	-51	181	
GPR131	RC	516849	9078692	1269	105	-52	360	
GPR132	RC	516849	9078800	1264	95	-51	184	
GPR133	RC	518106	9078455	1253	83	-48	179	
GPR134	RC	518103	9078367	1251	80	-50	1	
GPR135	RC	518107	9078370	1251	100	-51	181	
GPR136	RC	518101	9078260	1249	100	-50	360	
GPR137	RC	518101	9078452	1254	66	-50	1	
GPR138	RC	516848	9078797	1264	90	-51	359	
GPR139	RC	516855	9078897	1264	90	-50	179	
GPR140	RC	516853	9078894	1264	95	-49	358	
GPR141	RC	516845	9079000	1270	100	-50	177	
GPR142	RC	518102	9078522	1254	70	-50	183	
GPR143	RC	513841	9077212	1201	100	-49	183	Yes
GPR144	RC	513756	9077159	1196	100	-50	180	Yes
GPR145	RC	513850	9076921	1221	100	-50	2	
GPR146	RC	513846	9076853	1234	100	-49	1	Yes
GPR147	RC	513748	9076821	1231	100	-49	2	
GPR148	RC	513696	9076802	1224	100	-49	359	
GPR149	RC	513749	9076955	1204	100	-50	359	Yes
GPR150	RC	513699	9076955	1212	100	-49	2	Yes
GPR151	RC	513498	9076992	1198	114	-50	1	
GPR152	RC	513300	9077004	1191	120	-51	2	Yes
GPR153	RC	513199	9077011	1186	108	-48	1	
GPR154	RC	513546	9076989	1204	114	-51	2	Yes
GPR155	RC	513449	9076998	1194	100	-51	357	
GPR156	RC	513347	9077000	1189	110	-51	3	Yes
GPR157	RC	513248	9077000	1182	100	-50	1	Yes
ILD001	DD	509400	9077094	1179	102	-49	359	
ILD002	DD	509399	9077208	1185	100	-47	176	
ILD003	DD	509401	9077196	1185	100	-50	358	
ILD004	DD	509399	9077300	1192	206	-50	177	
ILD005	DD	508901	9077402	1188	83	-47	2	
ILD006	DD	508903	9077487	1199	110	-53	178	
ILD007	DD	509699	9076801	1165	101	-52	5	

Hole Name	Drill Type	Collar X	Collar Y	Collar Z	Maximum Depth (m)	Dip	Azimuth	Intersects a Mineralised Domain?
ILD008	DD	509700	9076900	1173	101	-53	181	
ILD009	DD	509701	9076895	1173	100	-50	2	
ILD010	DD	509701	9076999	1176	55	-51	177	
ILD011	DD	509705	9077075	1178	100	-53	177	
ILD012	DD	508901	9077550	1232	121	-47	179	
ILD013	DD	508898	9077401	1195	91	-52	175	
ILD014	DD	508902	9077298	1195	91	-45	358	
ILD015	DD	508899	9077303	1178	91	-52	181	
ILD016	DD	508900	9077199	1177	91	-52	357	
ILD017	DD	508898	9077202	1176	91	-53	179	
ILD018	DD	508901	9077093	1172	91	-52	360	
ILD019	DD	508902	9077100	1169	91	-50	182	
ILD020	DD	508899	9076999	1171	88	-50	359	
ILD021	DD	508901	9077007	1168	91	-51	180	
ILD022	DD	508900	9076902	1170	91	-50	355	
ILD023	DD	508991	9077446	1205	147	-48	182	
ILD024	DD	508854	9077572	1209	128	-50	177	
KWR001	RC	521698	9078448	1270	90	-51	269	
KWR002	RC	521594	9078447	1262	90	-50	91	
KWR003	RC	520217	9079659	1270	77	-51	55	
KWR004	RC	521502	9076828	1276	100	-50	354	
KWR005	RC	521450	9076904	1274	80	-51	179	
KWR006	RC	521454	9076901	1274	96	-51	360	
KWR007	RC	521451	9077001	1273	84	-49	180	
KWR008	RC	521449	9077000	1274	90	-49	1	
KWR009	RC	521450	9077098	1270	86	-51	181	
KWR010	RC	521447	9077096	1270	92	-52	360	
KWR011	RC	521457	9077200	1265	86	-51	180	
KWR012	RC	521455	9077198	1265	90	-50	0	
KWR013	RC	521451	9077300	1263	86	-50	181	
KWR014	RC	521449	9077298	1262	90	-50	358	
KWR015	RC	521450	9077402	1257	90	-51	179	
KWR016	RC	521053	9077301	1272	90	-50	181	
KWR017	RC	521051	9077184	1289	100	-50	360	
KWR018	RC	521053	9077188	1288	60	-49	179	
KWR019	RC	521046	9077103	1298	90	-51	1	
KWR020	RC	521049	9077107	1297	74	-51	176	
KWR021	RC	521050	9077011	1317	110	-49	359	
KWR022	RC	521054	9077013	1318	150	-51	182	
KWR023	RC	521450	9077403	1257	90	-51	2	
KWR024	RC	521450	9077505	1258	90	-50	181	
KWR025	RC	521450	9077502	1258	94	-51	1	
KWR026	RC	521450	9077601	1256	90	-50	179	

Hole Name	Drill Type	Collar X	Collar Y	Collar Z	Maximum Depth (m)	Dip	Azimuth	Intersects a Mineralised Domain?
KWR027	RC	520302	9079438	1261	66	-51	1	
KWR028	RC	520301	9079506	1259	65	-51	180	
KWR029	RC	520298	9079506	1259	90	-49	357	
KWR030	RC	520300	9079601	1255	90	-49	178	
KWR031	RC	520297	9079600	1255	90	-49	360	
KWR032	RC	520300	9079700	1253	95	-50	182	
KWR033	RC	520298	9079698	1253	91	-54	4	
KWR034	RC	520304	9079800	1247	99	-50	181	
KWR035	RC	520302	9079799	1247	90	-50	5	
KWR036	RC	522199	9077105	1246	72	-50	2	
KWR037	RC	522200	9077156	1246	54	-51	179	
KWR038	RC	522200	9077155	1245	140	-49	2	
KWR039	RC	522218	9077303	1242	93	-49	179	
KWR040	RC	522201	9077240	1244	80	-49	181	
KWR041	RC	522200	9077238	1244	65	-50	357	
KWR042	RC	520301	9079903	1241	90	-49	177	
KWR043	RC	520299	9079901	1241	95	-50	359	
KWR044	RC	520300	9079998	1237	90	-49	177	
KWR045	RC	518581	9079502	1263	70	-50	357	
KWR046	RC	518576	9079571	1259	72	-49	180	
KWR047	RC	518576	9079570	1259	140	-50	357	
KWR048	RC	518566	9079727	1259	145	-50	180	
KWR049	RC	518568	9079725	1260	70	-51	357	
KWR050	RC	518581	9079799	1258	70	-49	178	
KWR051	RC	518869	9079489	1293	102	-49	1	
KWR052	RC	518868	9079513	1297	75	-49	0	
MND001	DD	510910	9080725	1247	500	-50	159	
MND002	DD	511350	9080592	1245	547	-50	160	
MND003	DD	511601	9080601	1241	502	-50	161	
MND004	DD	511967	9080626	1254	502	-50	163	
MND005	DD	514403	9081053	1269	562	-51	180	
MNR001	RC	513899	9081200	1260	90	-51	177	
MNR002	RC	513899	9081101	1270	95	-48	356	
MNR003	RC	513897	9081104	1270	90	-49	178	
MNR004	RC	513899	9081007	1273	95	-48	1	
MNR005	RC	513901	9081009	1272	90	-50	175	
MNR006	RC	513899	9080905	1284	95	-49	360	
MNR007	RC	513900	9080905	1283	45	-49	179	
MNR008	RC	513900	9080850	1291	48	-50	1	
MNR009	RC	513900	9080851	1291	90	-50	179	
MNR010	RC	512649	9081299	1225	90	-51	182	
MNR011	RC	512651	9081207	1231	90	-50	1	
MNR012	RC	512652	9081208	1231	90	-51	177	

Hole Name	Drill Type	Collar X	Collar Y	Collar Z	Maximum Depth (m)	Dip	Azimuth	Intersects a Mineralised Domain?
MNR013	RC	512656	9081102	1232	90	-48	1	
MNR014	RC	512656	9081103	1232	90	-48	178	
MNR015	RC	512651	9081003	1238	90	-47	1	
MNR016	RC	511600	9081401	1192	90	-48	182	
MNR017	RC	511601	9081305	1198	90	-50	1	
MNR018	RC	511599	9081305	1198	84	-49	178	
MNR019	RC	511600	9081206	1201	84	-48	0	
MNR020	RC	511602	9081207	1201	74	-43	181	
MNR021	RC	511604	9081128	1208	74	-50	1	
MNR022	RC	511607	9081129	1208	102	-48	177	
MNR023	RC	511604	9081006	1216	104	-48	2	
MNR024	RC	511600	9081008	1215	92	-49	181	
MNR025	RC	511600	9080904	1232	100	-48	3	
MNR026	RC	511602	9080905	1230	90	-49	180	
MNR027	RC	511596	9080804	1238	90	-50	1	
MNR028	RC	511595	9080805	1242	90	-49	181	
MNR029	RC	511596	9080705	1237	90	-50	2	
MNR030	RC	511597	9080704	1231	60	-50	180	
MNR031	RC	511599	9080636	1227	60	-49	358	
MNR032	RC	511597	9080641	1228	55	-49	182	
MNR033	RC	511599	9080603	1240	38	-49	2	
SER001	RC	505281	9076003	1125	90	-49	178	
SER002	RC	505270	9075904	1125	85	-48	1	
SER003	RC	505506	9076125	1129	90	-48	180	
SER004	RC	505500	9076025	1131	80	-48	1	
SER005	RC	505501	9076027	1131	90	-48	180	
SER006	RC	505500	9075924	1120	81	-48	2	
SER007	RC	505498	9075926	1120	90	-48	181	
SER008	RC	505500	9075833	1111	80	-49	4	
SER009	RC	505702	9075057	1109	90	-50	1	
SER010	RC	505702	9075057	1108	100	-50	180	
SER011	RC	505703	9075169	1123	110	-50	184	
SER012	RC	505703	9074947	1096	80	-48	2	
SER013	RC	505701	9074948	1096	90	-49	182	
SER014	RC	504794	9075185	1086	90	-49	182	
SER015	RC	504790	9075095	1083	90	-49	2	
SER016	RC	504787	9075096	1083	100	-49	181	
SER017	RC	504805	9074982	1086	90	-48	2	
SER018	RC	504806	9074982	1086	90	-48	184	
SER019	RC	504806	9074906	1078	70	-48	1	
SER020	RC	504805	9074907	1077	90	-49	181	
SER021	RC	504814	9074803	1076	85	-48	4	
SER022	RC	504502	9074799	1091	90	-49	4	

Hole Name	Drill Type	Collar X	Collar Y	Collar Z	Maximum Depth (m)	Dip	Azimuth	Intersects a Mineralised Domain?
SER023	RC	504506	9074901	1088	90	-48	181	
SER024	RC	504507	9074899	1088	90	-47	0	
SER025	RC	504500	9075002	1088	90	-48	180	
SER026	RC	504499	9074999	1088	80	-50	2	
SER027	RC	504510	9075082	1089	80	-49	182	
SER028	RC	504508	9075080	1089	90	-48	3	
SER029	RC	504506	9075180	1092	90	-48	183	
SER030	RC	505102	9075055	1089	85	-48	-1	
SER031	RC	505104	9075056	1089	80	-49	180	
SER032	RC	505099	9075147	1090	90	-49	184	
SER033	RC	505103	9074967	1085	70	-49	2	
SER034	RC	505102	9074969	1085	95	-51	183	
SER035	RC	505102	9074847	1091	108	-47	-1	
SER036	RC	505303	9075106	1091	100	-48	175	Yes
SER037	RC	505511	9075164	1117	168	-48	181	Yes
SSD001	DD	506795	9072822	1076	202	-54	358	Yes
SSD002	DD	506795	9072822	1076	200	-75	354	Yes
SSD003	DD	506754	9072895	1067	112	-49	359	
SSD004	DD	506843	9072894	1076	116	-49	3	Yes
SSR001	RC	507797	9073104	1084	90	-49	183	
SSR002	RC	507795	9073001	1080	90	-51	2	
SSR003	RC	507797	9073003	1080	110	-50	181	
SSR004	RC	507805	9072889	1085	110	-50	358	
SSR005	RC	507803	9072892	1087	90	-51	180	
SSR006	RC	507803	9072807	1084	90	-50	7	
SSR007	RC	507807	9072811	1084	90	-50	181	
SSR008	RC	507800	9072711	1087	100	-51	358	
SSR009	RC	506800	9073010	1073	90	-51	359	
SSR010	RC	506799	9073013	1073	110	-50	181	
SSR011	RC	506795	9072891	1073	108	-51	359	Yes
SSR012	RC	506792	9072893	1072	65	-49	180	
SSR013	RC	506794	9072821	1074	68	-52	357	
SSR014	RC	506793	9072824	1074	120	-51	183	
SSR015	RC	506796	9072698	1069	110	-52	1	
SSR016	RC	506801	9073101	1076	90	-50	180	
SSR016a	RC	506801	9073099	1076	48	-51	183	
SSR017	RC	507550	9072853	1060	90	-51	353	
SSR018	RC	507516	9072943	1051	80	-48	179	
SSR019	RC	507524	9073049	1060	100	-50	179	
SSR020	RC	507527	9073049	1061	90	-48	0	
SSR021	RC	507556	9073140	1060	90	-49	178	
SSR022	RC	507511	9072942	1050	90	-47	2	
SSR023	RC	507302	9072919	1055	70	-48	1	

Hole Name	Drill Type	Collar X	Collar Y	Collar Z	Maximum Depth (m)	Dip	Azimuth	Intersects a Mineralised Domain?
SSR024	RC	507302	9072987	1053	70	-53	180	
SSR025	RC	507304	9072987	1052	100	-50	0	
SSR026	RC	507290	9073108	1064	110	-49	179	
SSR027	RC	506990	9073193	1070	80	-48	181	
SSR028	RC	507002	9073113	1070	80	-49	1	
SSR029	RC	507004	9073113	1070	100	-49	180	
SSR030	RC	507001	9073005	1070	100	-48	360	
SSR031	RC	507003	9073004	1070	80	-49	186	
SSR032	RC	507004	9072921	1067	80	-48	355	
SSR033	RC	507002	9072920	1067	110	-49	178	
SSR034	RC	507027	9072806	1061	100	-49	2	
SSR035	RC	507024	9072809	1061	100	-48	179	
SSR036	RC	506990	9072707	1051	90	-47	358	
SSR037	RC	506606	9073181	1056	90	-47	180	
SSR038	RC	506605	9073081	1058	90	-49	1	
SSR039	RC	506601	9073083	1058	80	-47	178	
SSR040	RC	506606	9072997	1057	80	-49	1	
SSR041	RC	506404	9072958	1052	100	-47	178	
SSR042	RC	506394	9072848	1045	100	-49	1	
SSR043	RC	506395	9072849	1045	80	-48	178	
SSR044	RC	506393	9072753	1047	80	-47	2	
SSR045	RC	506392	9072753	1048	90	-46	181	
SSR046	RC	506397	9072651	1046	80	-46	179	
SSR047	RC	506402	9072956	1052	90	-48	2	
SSR048	RC	506405	9073045	1056	90	-49	183	
SSR049	RC	506403	9073043	1056	90	-48	0	
SSR050	RC	506402	9073149	1056	90	-48	178	
SSR051	RC	506603	9073000	1055	90	-49	181	
SSR052	RC	506587	9072905	1055	81	-47	1	
SWR001	RC	499198	9073719	1027	100	-49	2	
SWR002	RC	499195	9073717	1026	100	-49	179	
SWR003	RC	499201	9073801	1027	100	-51	181	
SWR004	RC	499198	9073800	1027	100	-50	0	
SWR005	RC	499198	9073890	1023	100	-50	360	
SWR006	RC	499200	9073891	1023	100	-51	180	
SWR007	RC	499198	9073998	1032	100	-49	177	
SWR008	RC	499196	9073997	1032	100	-51	358	
SWR009	RC	499198	9074100	1035	100	-51	178	
SWR010	RC	498498	9072702	1005	106	-51	359	
SWR011	RC	498491	9072801	997	100	-50	358	
SWR012	RC	498489	9072803	997	100	-50	178	
SWR013	RC	498492	9072892	990	90	-48	175	
SWR014	RC	498494	9072891	990	100	-51	359	

Hole Name	Drill Type	Collar X	Collar Y	Collar Z	Maximum Depth (m)	Dip	Azimuth	Intersects a Mineralised Domain?
SWR015	RC	498501	9073006	995	100	-50	180	
SWR016	RC	498498	9073005	995	100	-50	1	
SWR017	RC	498502	9073101	1000	100	-50	179	
SWR018	RC	498499	9073100	1000	100	-50	360	
SWR019	RC	498497	9073194	1002	90	-49	180	
SWR020	RC	498495	9073193	1002	90	-49	360	
SWR021	RC	498497	9073303	1004	100	-50	180	
SWR022	RC	498499	9073301	1004	100	-49	1	
SWR023	RC	498501	9073405	1008	94	-50	179	
SWR024	RC	498499	9073403	1007	92	-50	358	
SWR025	RC	498492	9073500	1010	90	-50	178	
SWR026	RC	495248	9073999	1047	90	-50	181	
SWR027	RC	495247	9073898	1055	100	-51	357	
SWR028	RC	495247	9073899	1055	84	-51	181	
SWR029	RC	495239	9073813	1054	85	-50	358	
SWR030	RC	495240	9073813	1052	120	-50	179	
SWR031	RC	495244	9073688	1045	100	-51	360	
SWR032	RC	495245	9073691	1035	80	-50	178	
SWR033	RC	495244	9073612	1027	70	-50	358	
SWR034	RC	495244	9073615	1028	110	-51	182	
SWR035	RC	495255	9073501	1018	105	-49	356	
SZD001	DD	501837	9074133	1058	173	-60	160	
SZD002	DD	501775	9074291	1065	184	-60	31	Yes
SZD003	DD	500595	9074883	1065	152	-60	30	Yes
SZD004	DD	501176	9074757	1113	161	-60	210	
SZD005	DD	502140	9074369	1062	188	-60	160	
SZD006	DD	502187	9074279	1054	170	-60	160	Yes
SZD007	DD	501801	9074329	1071	80	-60	30	Yes
SZD008	DD	501746	9074242	1063	230	-59	34	Yes
SZD009	DD	501717	9074288	1060	146	-60	33	Yes
SZD010	DD	501806	9074242	1064	203	-59	30	
SZD011	DD	501831	9074282	1068	125	-56	22	Yes
SZD012	DD	501744	9074330	1065	111	-59	31	Yes
SZD013	DD	500690	9074843	1070	95	-50	43	Yes
SZD014	DD	500716	9074797	1077	141	-60	32	Yes
SZD015	DD	500635	9074752	1059	302	-59	30	
SZD016	DD	500662	9074796	1064	265	-58	26	
SZD017	DD	500635	9074847	1061	176	-59	31	Yes
SZD018	DD	500608	9074799	1058	259	-60	27	
SZD019	DD	501193	9074583	1081	170	-51	38	Yes
SZD020	DD	501147	9074597	1077	203	-60	29	Yes
SZD021	DD	501169	9074629	1096	155	-60	32	Yes
SZD022	DD	501202	9074630	1094	115	-59	28	Yes

Hole Name	Drill Type	Collar X	Collar Y	Collar Z	Maximum Depth (m)	Dip	Azimuth	Intersects a Mineralised Domain?
SZD023	DD	501252	9074578	1081	140	-60	28	Yes
SZD024	DD	501228	9074538	1074	206	-60	29	Yes
SZD025	DD	501935	9074173	1062	180	-47	158	Yes
SZD026	DD	501983	9074187	1063	113	-48	158	Yes
SZD027	DD	502234	9074309	1060	228	-47	160	Yes
SZD028	DD	502116	9074253	1061	232	-46	160	
SZD029	DD	502021	9074233	1067	240	-48	162	Yes
SZD030	DD	501973	9074210	1064	223	-46	166	Yes
SZD031	DD	502187	9074282	1054	100	-47	340	
SZD032	DD	501862	9074241	1066	209	-47	27	
SZD033	DD	501686	9074338	1061	157	-48	30	Yes
SZD034	DD	501635	9074341	1057	166	-47	30	Yes
SZD035	DD	501575	9074344	1054	206	-47	29	Yes
SZD036	DD	501503	9074420	1055	149	-47	29	Yes
SZD037	DD	501413	9074470	1062	104	-47	28	
SZD038	DD	501317	9074493	1068	152	-48	27	Yes
SZD039	DD	501288	9074542	1074	131	-47	28	Yes
SZD040	DD	500993	9074589	1072	283	-47	32	Yes
SZD041	DD	500923	9074629	1066	254	-47	26	Yes
SZD042	DD	500491	9074908	1062	192	-46	30	Yes
SZD043	DD	501239	9074554	1077	164	-58	30	Yes
SZD044	DD	501276	9074581	1079	85	-49	29	
SZD045	DD	501276	9074580	1078	152	-70	23	Yes
SZD046	DD	501262	9074590	1083	78	-48	29	Yes
SZD047	DD	501292	9074573	1080	68	-46	28	Yes
SZD048	DD	501321	9074553	1075	69	-47	33	
SZD049	DD	501320	9074552	1074	152	-71	33	Yes
SZD050	DD	501342	9074536	1068	84	-48	28	Yes
SZD051	DD	501331	9074518	1069	130	-47	29	Yes
SZD052	DD	501255	9074606	1069	91	-48	33	Yes
SZD053	DD	501255	9074606	1069	49	-70	31	Yes
SZD054	DD	501195	9074627	1094	64	-48	29	Yes
SZD055	DD	500230	9075061	1064	107	-48	29	Yes
SZD056	DD	501820	9074262	1066	128	-47	28	Yes
SZD057	DD	501840	9074307	1073	71	-48	30	Yes
SZD058	DD	501856	9074326	1080	47	-48	29	Yes
SZD059	DD	501786	9074310	1068	105	-48	29	Yes
SZD060	DD	501762	9074267	1061	200	-60	32	Yes
SZD061	DD	501731	9074308	1062	139	-60	29	Yes
SZD062	DD	500982	9074800	1127	74	-45	207	Yes
SZD063	DD	501045	9074730	1113	200	-48	31	Yes
SZD064	DD	501044	9074725	1114	174	-69	30	Yes
SZD065	DD	501083	9074705	1113	66	-48	30	Yes

Hole Name	Drill Type	Collar X	Collar Y	Collar Z	Maximum Depth (m)	Dip	Azimuth	Intersects a Mineralised Domain?
SZD066	DD	501082	9074704	1113	100	-69	27	Yes
SZD067	DD	500854	9074801	1104	63	-48	28	Yes
SZD068	DD	500853	9074799	1104	55	-71	25	Yes
SZD069	DD	501133	9074665	1111	54	-49	28	Yes
SZD070	DD	501133	9074665	1112	83	-70	24	Yes
SZD071	DD	500559	9074937	1074	76	-50	30	Yes
SZD072	DD	501883	9074316	1079	49	-43	23	Yes
SZD073	DD	501883	9074315	1079	71	-69	25	
SZD074	DD	502189	9074533	1069	251	-48	357	Yes
SZD075	DD	502242	9074531	1066	64	-48	357	Yes
SZD076	DD	502135	9074532	1071	134	-47	360	Yes
SZD077	DD	502160	9074488	1067	196	-47	360	Yes
SZD078	DD	502212	9074494	1064	160	-48	356	Yes
SZD079	DD	501100	9074619	1078	182	-49	31	Yes
SZD080	DD	501099	9074619	1078	200	-60	30	Yes
SZD081	DD	501092	9074658	1090	175	-47	30	Yes
SZD082	DD	501092	9074657	1090	257	-70	29	Yes
SZD083	DD	501041	9074670	1102	193	-48	29	Yes
SZD084	DD	501040	9074670	1100	260	-70	28	Yes
SZD085	DD	501002	9074700	1104	109	-47	24	Yes
SZD086	DD	501001	9074698	1103	257	-71	23	Yes
SZD087	DD	500951	9074704	1097	187	-49	28	
SZD088	DD	500951	9074704	1097	208	-71	28	Yes
SZD089	DD	500918	9074740	1093	181	-47	30	Yes
SZD089A	DD	500918	9074740	1093	80	-47	27	Yes
SZD090	DD	500917	9074735	1095	221	-70	32	Yes
SZD091	DD	500865	9074775	1098	83	-48	30	Yes
SZD092	DD	500867	9074775	1098	191	-70	32	Yes
SZD092A	DD	500868	9074776	1098	93	-70	29	Yes
SZD093	DD	500823	9074783	1091	91	-48	28	Yes
SZD094	DD	500822	9074782	1091	212	-70	28	Yes
SZD095	DD	500776	9074800	1087	84	-48	27	Yes
SZD096	DD	500775	9074799	1087	218	-70	26	Yes
SZD097	DD	501803	9074243	1064	161	-49	30	Yes
SZD098	DD	501803	9074242	1064	236	-71	31	
SZD099	DD	501690	9074249	1058	182	-51	28	Yes
SZD100	DD	501691	9074248	1058	212	-61	28	Yes
SZD101	DD	502214	9074543	1068	69	-50	358	Yes
SZD102	DD	502214	9074527	1067	99	-48	359	Yes
SZD103	DD	502239	9074507	1066	139	-47	359	
SZD104	DD	502211	9074469	1065	199	-48	360	Yes
SZD105	DD	502186	9074464	1066	311	-48	357	Yes
SZD106	DD	502236	9074549	1067	87	-48	2	Yes

Hole Name	Drill Type	Collar X	Collar Y	Collar Z	Maximum Depth (m)	Dip	Azimuth	Intersects a Mineralised Domain?
SZD107	DD	502262	9074548	1068	244	-49	356	Yes
SZD108	DD	502262	9074521	1064	251	-48	359	Yes
SZD109	DD	502239	9074484	1064	277	-49	357	
SZD110	DD	502109	9074506	1071	136	-48	354	Yes
SZD111	DD	502080	9074473	1073	205	-49	354	Yes
SZD112	DD	502060	9074499	1073	112	-47	358	
SZD113	DD	502023	9074499	1070	144	-48	358	
SZD114	DD	501174	9074539	1063	203	-48	29	Yes
SZD115	DD	501174	9074539	1063	240	-60	27	Yes
SZD116	DD	501174	9074539	1062	262	-70	27	Yes
SZD117	DD	501285	9074483	1061	175	-48	25	Yes
SZD118	DD	501285	9074484	1062	191	-60	27	Yes
SZD119	DD	501284	9074483	1062	202	-70	25	Yes
SZD120	DD	501225	9074488	1065	211	-49	27	Yes
SZD121	DD	501225	9074487	1065	260	-61	28	Yes
SZD122	DD	500720	9074818	1080	84	-49	27	Yes
SZD123	DD	500706	9074837	1080	84	-48	29	Yes
SZD124	DD	500670	9074871	1071	117	-50	30	Yes
SZD125	DD	500667	9074870	1071	127	-69	28	Yes
SZD126	DD	500647	9074886	1069	115	-47	29	Yes
SZD127	DD	500644	9074890	1069	141	-69	26	Yes
SZD128	DD	501175	9074492	1066	226	-49	30	Yes
SZD129	DD	501175	9074491	1066	251	-61	29	Yes
SZD130	DD	500625	9074896	1073	123	-47	30	Yes
SZD131	DD	500626	9074896	1072	122	-69	28	Yes
SZD132	DD	500608	9074908	1073	112	-49	34	Yes
SZD133	DD	501112	9074532	1063	250	-49	31	Yes
SZD134	DD	501112	9074532	1063	268	-61	32	Yes
SZD135	DD	500606	9074906	1074	108	-66	37	Yes
SZD136	DD	501066	9074557	1066	245	-48	31	Yes
SZD137	DD	501066	9074557	1066	280	-61	29	Yes
SZD138	DD	500583	9074923	1075	80	-49	30	Yes
SZD139	DD	500583	9074923	1075	107	-70	31	Yes
SZD140	DD	501016	9074577	1066	252	-48	28	Yes
SZD141	DD	501016	9074577	1066	278	-60	29	Yes
SZD142	DD	500548	9074910	1069	102	-48	29	Yes
SZD143	DD	500548	9074909	1069	140	-70	28	Yes
SZD144	DD	501176	9074656	1107	73	-48	26	Yes
SZD145	DD	501176	9074656	1106	37	-63	25	Yes
SZD146	DD	500528	9074925	1068	102	-48	28	Yes
SZD147	DD	500528	9074925	1068	143	-69	26	Yes
SZD148A	DD	501220	9074640	1102	121	-69	25	Yes
SZD149	DD	501292	9074592	1071	37	-48	29	Yes

Hole Name	Drill Type	Collar X	Collar Y	Collar Z	Maximum Depth (m)	Dip	Azimuth	Intersects a Mineralised Domain?
SZD150	DD	501292	9074591	1072	110	-70	26	Yes
SZD151	DD	500489	9074945	1066	99	-47	32	Yes
SZD152	DD	500489	9074944	1066	131	-71	29	Yes
SZD153	DD	501266	9074606	1089	130	-70	25	Yes
SZD154	DD	502140	9074461	1069	329	-48	360	Yes
SZD155	DD	500252	9075059	1064	90	-49	31	Yes
SZD156	DD	500252	9075059	1064	131	-70	30	Yes
SZD157	DD	501266	9074606	1089	23	-50	27	Yes
SZD158	DD	501314	9074579	1068	111	-60	33	Yes
SZD159	DD	501331	9074555	1073	110	-70	29	Yes
SZD160	DD	502097	9074480	1073	187	-48	1	Yes
SZD161	DD	500293	9075032	1061	93	-48	33	Yes
SZD162	DD	500292	9075031	1061	127	-71	31	Yes
SZD163	DD	501330	9074556	1075	39	-49	31	
SZD164	DD	500358	9075005	1059	92	-48	31	Yes
SZD165	DD	500355	9075005	1059	127	-70	33	Yes
SZD166	DD	501347	9074547	1069	72	-48	31	
SZD167	DD	500431	9074947	1063	114	-48	31	Yes
SZD168	DD	500430	9074947	1063	169	-71	30	Yes
SZD169	DD	500383	9074974	1061	131	-49	32	Yes
SZD170	DD	500383	9074974	1061	143	-69	34	Yes
SZD171	DD	502096	9074405	1066	271	-48	2	Yes
SZD172	DD	505401	9075097	1122	130	-49	181	Yes
SZD173	DD	502117	9074528	1071	91	-48	1	Yes
SZD174	DD	502027	9074400	1065	266	-49	359	Yes
SZD175	DD	505346	9075099	1112	151	-47	182	Yes
SZD176	DD	502305	9074615	1068	243	-48	2	Yes
SZD177A	DD	505378	9075156	1117	262	-48	181	Yes
SZD178	DD	505452	9075103	1120	130	-49	180	
SZD179A	DD	505424	9075163	1124	248	-48	180	Yes
SZD180	DD	502318	9074485	1064	174	-48	1	
SZD181	DD	502063	9074400	1067	269	-47	0	Yes
SZD182	DD	502324	9074625	1069	124	-48	2	Yes
SZD183	DD	502273	9074628	1067	120	-47	359	Yes
SZD184	DD	501976	9074402	1073	57	-49	0	
SZD185	DD	501972	9074304	1072	206	-49	359	
SZD186	DD	501700	9074298	1060	152	-48	359	Yes
SZD187	DD	501605	9074293	1053	246	-50	3	Yes
SZD188	DD	502000	9074051	1046	120	-54	355	
SZD189	DD	502000	9074051	1046	144	-60	357	
SZD190	DD	502110	9074627	1060	121	-47	360	
SZD191	DD	502222	9074631	1060	118	-47	360	Yes
SZD192	DD	502222	9074631	1060	151	-59	359	Yes

Hole Name	Drill Type	Collar X	Collar Y	Collar Z	Maximum Depth (m)	Dip	Azimuth	Intersects a Mineralised Domain?
SZD193	DD	502522	9074585	1065	151	-48	359	
SZD194	DD	502602	9074580	1074	165	-49	358	
SZD195	DD	502050	9074061	1043	136	-50	2	Yes
SZD196	DD	502050	9074061	1043	146	-61	359	
SZD197	DD	501960	9074054	1046	125	-48	359	
SZD198	DD	501960	9074053	1046	139	-60	359	
SZR001	RC	500305	9073549	1023	57	-50	180	
SZR002	RC	500303	9073555	1023	60	-50	360	
SZR003	RC	500299	9073601	1023	60	-50	180	
SZR004	RC	500302	9073603	1025	103	-50	360	
SZR005	RC	500301	9073707	1028	103	-50	180	
SZR006	RC	500302	9073710	1028	67	-50	360	
SZR007	RC	500301	9073707	1028	100	-70	360	
SZR008	RC	500298	9073810	1033	70	-50	180	
SZR009	RC	500919	9074625	1074	150	-50	40	
SZR010	RC	501192	9074583	1076	146	-48	43	Yes
SZR011	RC	500689	9074844	1073	130	-51	43	Yes
SZR012	RC	501008	9074747	1118	80	-50	40	Yes
SZR013	RC	501898	9074250	1068	90	-50	180	
SZR014	RC	501899	9074259	1069	100	-50	360	
SZR015	RC	502001	9074055	1053	112	-55	360	Yes
SZR016	RC	502000	9074182	1061	122	-50	180	Yes
SZR017	RC	502000	9074189	1062	80	-50	360	
SZR018	RC	501900	9074363	1083	123	-50	180	
SZR019	RC	501900	9074369	1082	86	-50	360	
SZR020	RC	501898	9074452	1075	90	-50	180	
SZR021	RC	505393	9074920	1103	68	-50	180	
SZR022	RC	505393	9074924	1104	94	-50	360	
SZR023	RC	505396	9074994	1107	80	-50	180	
SZR024	RC	505396	9074996	1110	90	-50	360	
SZR025	RC	505397	9075097	1117	100	-50	180	Yes
SZR026	RC	505398	9075100	1119	85	-50	360	
SZR027	RC	505394	9075190	1116	75	-50	180	
SZR028	RC	505394	9075199	1115	90	-50	360	
SZR029	RC	505695	9074657	1108	100	-50	360	
SZR030	RC	505695	9074757	1108	116	-50	180	
SZR031	RC	505698	9074754	1111	100	-50	360	
SZR032	RC	505707	9074855	1111	110	-50	180	
SZR033	RC	505707	9074860	1111	102	-50	360	
SZR034	RC	500701	9073803	1035	100	-50	360	
SZR035	RC	500700	9073801	1036	100	-50	180	
SZR036	RC	500702	9073803	1036	100	-54	354	
SZR037	RC	500689	9073896	1036	100	-53	183	

Hole Name	Drill Type	Collar X	Collar Y	Collar Z	Maximum Depth (m)	Dip	Azimuth	Intersects a Mineralised Domain?
SZR038	RC	500688	9073897	1036	97	-52	358	
SZR039	RC	500691	9073989	1035	100	-52	359	
SZR040	RC	500694	9073987	1035	100	-51	179	
SZR041	RC	500698	9074098	1037	100	-52	179	
SZR042	RC	501020	9073803	1038	100	-50	358	
SZR043	RC	501011	9073894	1044	100	-50	178	
SZR044	RC	501014	9073901	1044	125	-52	357	
SZR045	RC	501015	9073984	1048	107	-51	181	
SZR046	RC	501018	9073998	1048	100	-51	357	
SZR047	RC	501024	9074100	1049	100	-51	178	
SZR048	RC	501030	9074104	1049	100	-52	357	
SZR049	RC	501030	9074200	1045	100	-50	183	
SZR050	RC	501767	9074299	1068	60	-90	0	
SZR051	RC	501631	9074344	1057	54	-90	0	
SZR052	RC	502191	9074534	1069	107	-51	358	Yes
SZR053	RC	502187	9074632	1073	67	-50	182	
SZR054	RC	502187	9074634	1072	140	-50	180	Yes
SZR055	RC	502186	9074628	1072	110	-51	3	
SZR056	RC	502187	9074706	1074	98	-50	181	
SZR057	RC	502185	9074706	1074	100	-51	5	
SZR058	RC	502301	9074615	1066	100	-52	360	Yes
SZR059	RC	502308	9074623	1070	100	-51	180	
SZR060	RC	502303	9074726	1071	100	-50	357	
SZR061	RC	502303	9074724	1071	100	-50	179	
SZR062	RC	502295	9074838	1072	77	-50	177	
SZR063	RC	502299	9074318	1060	107	-50	357	
SZR064	RC	502296	9074415	1060	100	-51	0	
SZR065	RC	502296	9074425	1059	100	-51	182	
SZR066	RC	502326	9074519	1065	107	-51	359	
SZR067	RC	502325	9074521	1065	100	-51	184	
SZR068	RC	502753	9074324	1073	89	-49	357	
SZR069	RC	502761	9074392	1071	95	-50	179	
SZR070	RC	502762	9074398	1071	100	-51	356	
SZR071	RC	502745	9074502	1083	100	-50	179	
SZR072	RC	502742	9074498	1083	112	-49	356	
SZR073	RC	502754	9074624	1088	119	-49	180	
SZR074	RC	502751	9074621	1088	90	-49	357	
SZR075	RC	502757	9074697	1087	90	-50	179	
SZR076	RC	502758	9074701	1087	100	-51	360	
SZR077	RC	502752	9074797	1083	97	-50	179	
SZR078	RC	503994	9074911	1078	110	-50	180	
SZR079	RC	503994	9074904	1078	120	-50	0	
SZR080	RC	503971	9075028	1077	126	-51	179	

Hole Name	Drill Type	Collar X	Collar Y	Collar Z	Maximum Depth (m)	Dip	Azimuth	Intersects a Mineralised Domain?
SZR081	RC	503973	9075024	1077	100	-51	356	
SZR082	RC	503977	9075097	1079	90	-51	180	
SZR083	RC	503974	9075091	1080	60	-50	358	
SZR084	RC	504003	9074790	1077	118	-51	358	
SZR085	RC	503996	9074789	1077	80	-50	179	
SZR086	RC	504003	9074720	1072	60	-51	1	
SZR087	RC	503999	9074730	1074	110	-51	180	
SZR088	RC	503995	9074602	1078	84	-50	358	
SZR089	RC	503993	9074604	1077	110	-50	182	
SZR090	RC	504007	9074496	1083	100	-50	360	
SZR091	RC	504002	9074498	1083	100	-49	182	
SZR092	RC	503998	9074405	1083	96	-50	358	
SZR093	RC	501026	9073806	1038	108	-51	182	
SZR094	RC	501027	9073700	1037	94	-50	359	
SZR095	RC	500480	9075004	1067	46	-49	29	Yes
SZR096	RC	500789	9074880	1106	50	-49	210	Yes
SZR097	RC	500850	9074873	1118	108	-49	210	Yes
SZR098	RC	500887	9074844	1119	80	-48	207	Yes
SZR099	RC	500931	9074809	1120	46	-47	208	Yes
SZR100	RC	500427	9075026	1068	40	-47	28	Yes
SZR101	RC	500390	9075037	1062	41	-47	30	Yes
SZR102	RC	500345	9075072	1061	42	-45	29	Yes
SZR103	RC	500297	9075080	1064	47	-49	30	Yes
SZR104	RC	500263	9075113	1067	40	-48	31	Yes
SZR105	RC	500222	9075137	1074	34	-45	31	Yes
SZR106	RC	500183	9075151	1075	42	-47	29	Yes
SZR107	RC	500111	9075161	1072	64	-46	27	Yes
SZR108	RC	500089	9075203	1082	46	-47	33	Yes
SZR109	RC	500147	9075121	1067	92	-48	27	Yes
SZR110	RC	500414	9074974	1063	88	-47	32	Yes
SZR111	RC	500557	9074938	1073	82	-50	29	Yes
SZR112	RC	500511	9074940	1067	92	-46	30	Yes
SZR113	RC	501387	9074513	1064	78	-49	29	
SZR114	RC	501471	9074468	1056	74	-51	32	
SZR115	RC	501571	9074415	1056	52	-50	31	Yes
SZR116	RC	505966	9075198	1127	110	-50	6	
SZR117	RC	505959	9075306	1133	86	-51	179	
SZR118	RC	505963	9075304	1133	100	-50	2	
SZR119	RC	505973	9075403	1137	90	-51	181	
SZR120	RC	505979	9075397	1136	90	-50	1	
SZR121	RC	505966	9075495	1143	90	-51	177	
SZR122	RC	501698	9074298	1061	22	-50	360	
SZR123	RC	501497	9074331	1055	150	-50	360	Yes

Hole Name	Drill Type	Collar X	Collar Y	Collar Z	Maximum Depth (m)	Dip	Azimuth	Intersects a Mineralised Domain?
SZR124	RC	501397	9074343	1061	150	-51	1	
SZR125	RC	501300	9074300	1059	105	-50	359	
SZR126	RC	501596	9074185	1052	100	-50	2	
SZR127	RC	502646	9074552	1085	100	-50	1	
SZR128	RC	502496	9074600	1073	100	-50	3	
SZR129	RC	502410	9074599	1067	100	-49	359	
SZR130	RC	502646	9074651	1088	100	-49	182	
SZR131	RC	502643	9074651	1082	100	-50	4	
SZR132	RC	502399	9074701	1071	100	-50	181	
SZR133	RC	502495	9074714	1070	100	-47	180	
SZR134	RC	500899	9074902	1115	90	-52	359	
SZR135	RC	500899	9075003	1104	90	-50	179	
SZR136	RC	500901	9075002	1104	92	-50	360	
SZR137	RC	500899	9075101	1080	90	-49	179	
SZR138	RC	502357	9074629	1070	134	-50	359	
SZR139	RC	502453	9074630	1069	120	-51	360	
SZR140	RC	502559	9074607	1082	120	-51	0	
SZR141	RC	502589	9074607	1082	120	-49	358	
SZR142	RC	502690	9074605	1085	130	-50	359	
SZR143	RC	503253	9074503	1078	100	-51	177	
SZR144	RC	503254	9074392	1073	100	-51	0	
SZR145	RC	503256	9074394	1073	85	-49	177	
SZR146	RC	503254	9074304	1069	85	-48	358	
SZR147	RC	503255	9074307	1070	95	-49	175	
SZR148	RC	503244	9074201	1064	95	-50	359	
SZR149	RC	503242	9074202	1065	72	-50	176	
SZR150	RC	503239	9074120	1065	76	-51	356	

Appendix B: List of Intersections used for Mineral Resource Estimation

Hole Name	Drill Type	Depth From	Depth To	Mineralised Domain	Mean Au grade (g/t)	Intersection Length (m)
GPD001	DD	13.4	64.9	Porcupine Main	3.27	51.5
GPD002	DD	22.8	79.1	Porcupine Main	1.83	56.3
GPD003	DD	4.2	23.4	Porcupine Main	0.90	19.2
GPD003	DD	27.9	59.6	Porcupine Main	0.65	31.7
GPD004	DD	50.2	103.5	Porcupine Main	3.08	53.3
GPD005	DD	16.7	39.6	Porcupine Main	1.37	22.9
GPD005A	DD	16.6	71.3	Porcupine Main	3.14	54.7
GPD008	DD	3.9	15.4	Porcupine Main	1.22	11.5
GPD009	DD	22.4	26.7	Porcupine Main	0.60	4.3
GPD009	DD	35.9	42.5	Porcupine Main	0.43	6.6
GPD009	DD	62.9	79.0	Porcupine Main	0.91	16.1
GPD010	DD	58.9	112.7	Porcupine Main	1.69	53.8
GPD012	DD	0.0	45.2	Porcupine Northwest 322	0.69	45.2
GPD015	DD	140.3	194.1	Porcupine Main	2.00	53.7
GPD016	DD	155.4	201.5	Porcupine Main	1.80	46.2
GPD017	DD	195.3	262.2	Porcupine Main	0.90	66.9
GPD017	DD	274.1	278.2	Porcupine Main	0.42	4.1
GPD018A	DD	126.5	172.1	Porcupine Main	2.02	45.6
GPD019	DD	137.2	187.3	Porcupine Main	1.58	50.1
GPD020	DD	64.0	111.6	Porcupine Main	1.59	47.5
GPD021	DD	74.8	123.9	Porcupine Main	1.85	49.1
GPD022	DD	105.0	120.5	Porcupine Main	0.69	15.5
GPD023	DD	136.7	174.9	Porcupine Main	0.83	38.2
GPD024	DD	106.9	115.0	Porcupine Main	0.43	8.2
GPD026A	DD	142.2	189.6	Porcupine Main	0.88	47.3
GPD028	DD	166.7	187.3	Porcupine Main	0.72	20.6
GPD029	DD	51.0	70.6	Porcupine Main	0.87	19.7
GPD029	DD	77.7	86.0	Porcupine Main	0.67	8.4
GPD029	DD	98.9	104.9	Porcupine Main	0.44	6.0
GPD030	DD	100.2	144.8	Porcupine Main	1.37	44.6
GPD030	DD	149.0	156.5	Porcupine Main	0.77	7.6
GPD031	DD	110.5	165.2	Porcupine Main	1.76	54.7
GPD032	DD	81.6	134.7	Porcupine Main	0.80	53.1
GPD033	DD	94.5	152.1	Porcupine Main	0.72	57.6
GPD033	DD	193.1	202.7	Porcupine Main	0.54	9.5
GPD034	DD	166.4	212.2	Porcupine Main	1.01	45.8
GPD035	DD	150.4	214.8	Porcupine Main	1.31	64.4
GPD036	DD	150.4	200.5	Porcupine Main	1.22	50.0
GPD037	DD	224.1	280.3	Porcupine Main	0.80	56.2
GPD038	DD	20.0	68.0	Porcupine Northwest 322	0.27	48.0
GPD039	DD	125.6	192.4	Porcupine Main	0.80	66.8
GPD040	DD	74.0	75.9	Porcupine Main	0.30	1.8
GPD040	DD	91.0	105.4	Porcupine Main	0.82	14.3
GPD040	DD	109.4	136.4	Porcupine Main	0.95	27.0

Hole Name	Drill Type	Depth From	Depth To	Mineralised Domain	Mean Au grade (g/t)	Intersection Length (m)
GPD040	DD	148.5	167.4	Porcupine Main	0.56	18.9
GPD041	DD	114.7	126.3	Porcupine Main	0.67	11.6
GPD041	DD	129.5	146.5	Porcupine Main	1.18	17.0
GPD041	DD	158.6	167.6	Porcupine Main	1.29	9.0
GPD041	DD	179.4	192.9	Porcupine Main	0.58	13.5
GPD041	DD	207.9	226.8	Porcupine Main	0.90	18.9
GPD041	DD	234.0	242.8	Porcupine Main	0.73	8.8
GPD042	DD	174.8	181.6	Porcupine Main	0.68	6.8
GPD042	DD	189.9	216.2	Porcupine Main	0.83	26.4
GPD042	DD	227.0	275.2	Porcupine Main	2.00	48.3
GPD043	DD	181.5	207.4	Porcupine Main	1.08	25.9
GPD044	DD	201.6	254.3	Porcupine Main	1.78	52.8
GPD045	DD	38.7	59.0	Quill	1.67	20.3
GPD045	DD	84.9	98.5	Quill	0.75	13.6
GPD048	DD	258.3	288.1	Porcupine Main	0.93	29.8
GPD048	DD	300.2	357.0	Porcupine Main	0.91	56.7
GPD049	DD	244.0	292.1	Porcupine Main	3.20	48.1
GPD050	DD	184.2	232.8	Porcupine Main	1.23	48.5
GPD050	DD	279.6	289.1	Porcupine Main	0.41	9.5
GPD050	DD	295.1	303.8	Porcupine Main	0.88	8.7
GPD051	DD	227.7	326.6	Porcupine Main	1.80	98.9
GPD052	DD	270.5	344.7	Porcupine Main	0.75	74.2
GPD053	DD	107.5	120.1	Porcupine Main	0.99	12.6
GPD053	DD	146.5	148.0	Porcupine Main	0.75	1.5
GPD053	DD	203.4	240.2	Porcupine Main	0.72	36.8
GPD054	DD	167.4	174.4	Porcupine Main	0.77	7.0
GPD054	DD	233.3	247.7	Porcupine Main	0.97	14.3
GPD054	DD	260.7	278.8	Porcupine Main	0.98	18.2
GPD055	DD	98.6	111.2	Porcupine Main	0.71	12.6
GPD063	DD	26.9	100.0	Porcupine Main	1.33	73.1
GPD064	DD	116.0	180.8	Porcupine Main	1.66	64.7
GPD065	DD	194.4	258.5	Porcupine Main	3.09	64.1
GPD065	DD	298.3	316.7	Porcupine Main	3.20	18.4
GPD066	DD	88.2	172.6	Porcupine Main	2.88	84.4
GPD066	DD	195.3	200.0	Porcupine Main	0.26	4.6
GPD070	DD	23.8	32.5	Quill	0.39	8.7
GPD070	DD	47.6	59.8	Quill	0.42	12.2
GPD078	DD	136.3	203.7	Porcupine Main	1.27	67.5
GPD080	DD	98.2	105.6	Porcupine Main	0.85	7.4
GPD092	DD	18.1	112.9	Porcupine Main	0.76	94.8
GPD092	DD	132.7	157.5	Porcupine Main	0.78	24.8
GPD092	DD	195.2	259.7	Porcupine Main	1.46	64.5
GPD092	DD	260.1	262.2	Porcupine Main	0.44	2.1
GPD093	DD	16.8	80.9	Porcupine Main	1.03	64.1

Hole Name	Drill Type	Depth From	Depth To	Mineralised Domain	Mean Au grade (g/t)	Intersection Length (m)
GPD093	DD	85.0	133.4	Porcupine Main	0.84	48.4
GPD093	DD	180.1	187.1	Porcupine Main	0.62	7.1
GPD093	DD	213.7	246.9	Porcupine Main	1.55	33.2
GPD094	DD	62.0	82.1	Porcupine Main	1.49	20.2
GPD094	DD	159.4	185.3	Porcupine Main	0.82	25.9
GPD095	DD	63.3	102.7	Porcupine Main	1.14	39.5
GPD096	DD	17.9	34.2	Quill	0.33	16.2
GPD096	DD	51.4	70.5	Quill	1.72	19.1
GPD096	DD	71.2	78.4	Quill	0.55	7.3
GPD097	DD	19.6	70.2	Quill	0.53	50.6
GPD098	DD	16.6	22.2	Quill	0.60	5.6
GPD099	DD	17.6	24.9	Quill	0.60	7.3
GPD111	DD	6.6	24.4	Porcupine Main	0.91	17.8
GPD112	DD	19.3	23.6	Porcupine Main	0.49	4.3
GPD112	DD	98.8	100.3	Porcupine Main	0.30	1.5
GPD112	DD	109.2	131.4	Porcupine Main	0.72	22.3
GPD114	DD	102.3	107.7	Porcupine Main	0.42	5.4
GPD126	DD	96.4	101.8	Quill	0.54	5.4
GPD126	DD	105.7	111.7	Quill	0.48	6.0
GPR004	RC	2.1	62.6	Porcupine Main	1.50	60.5
GPR015	RC	33.6	62.4	Porcupine Main	0.73	28.8
GPR017	RC	56.5	77.1	Porcupine Main	0.52	20.6
GPR017	RC	79.5	90.0	Porcupine Main	0.47	10.6
GPR018	RC	0.0	19.4	Quill	1.12	19.4
GPR018	RC	38.7	73.1	Quill	0.46	34.3
GPR019	RC	0.0	12.9	Quill	0.36	12.9
GPR020	RC	19.8	34.2	Quill	0.49	14.4
GPR020	RC	42.0	60.2	Quill	0.87	18.2
GPR143	RC	13.2	32.7	Quill	3.22	19.6
GPR143	RC	45.5	62.8	Quill	0.29	17.3
GPR144	RC	36.0	53.9	Quill	2.39	17.9
GPR144	RC	81.5	90.2	Quill	0.78	8.7
GPR146	RC	58.6	80.0	Porcupine Main	1.66	21.4
GPR149	RC	46.0	50.0	Porcupine Northwest 322	0.76	4.0
GPR150	RC	52.0	66.0	Porcupine Northwest 322	0.86	14.0
GPR152	RC	42.0	62.0	Porcupine Northwest 321	0.52	20.0
GPR154	RC	40.0	56.0	Porcupine Northwest 322	0.83	16.0
GPR156	RC	14.0	40.0	Porcupine Northwest 321	0.45	26.0
GPR157	RC	40.0	54.0	Porcupine Northwest 321	0.60	14.0
SER036	RC	28.4	49.2	Konokono	1.29	20.8
SER036	RC	50.3	55.2	Konokono	0.40	4.9
SER037	RC	26.7	58.0	Konokono	0.77	31.3
SER037	RC	66.6	77.0	Konokono	0.62	10.4
SSD001	DD	99.5	118.9	Tumbili	0.74	19.4

Hole Name	Drill Type	Depth From	Depth To	Mineralised Domain	Mean Au grade (g/t)	Intersection Length (m)
SSD002	DD	129.0	134.2	Tumbili	0.54	5.2
SSD004	DD	22.5	35.8	Tumbili	0.59	13.3
SSR011	RC	34.1	54.5	Tumbili	1.47	20.5
SZD002	DD	83.9	104.1	Kenge Southeast	3.01	20.3
SZD003	DD	81.7	91.3	Kenge Hanging Wall	0.34	9.6
SZD006	DD	66.0	76.0	Mbenge South 216	3.01	10.0
SZD007	DD	11.0	50.0	Kenge Southeast	0.54	39.0
SZD008	DD	181.5	198.3	Kenge Southeast	0.81	16.8
SZD009	DD	101.0	122.7	Kenge Southeast	1.45	21.7
SZD011	DD	63.0	92.1	Kenge Southeast	2.20	29.1
SZD012	DD	41.1	44.5	Kenge Southeast	0.11	3.4
SZD013	DD	42.2	70.2	Kenge Hanging Wall	1.61	28.0
SZD014	DD	71.5	80.5	Kenge Hanging Wall	0.01	9.0
SZD017	DD	89.4	101.5	Kenge Hanging Wall	0.74	12.2
SZD019	DD	125.1	143.2	Kenge Footwall	1.94	18.1
SZD020	DD	151.8	175.7	Kenge Footwall	2.81	23.8
SZD021	DD	120.1	135.5	Kenge Footwall	2.02	15.4
SZD022	DD	23.9	47.3	Kenge Hanging Wall	0.02	23.3
SZD023	DD	96.2	122.0	Kenge Footwall	5.81	25.8
SZD024	DD	144.3	167.3	Kenge Footwall	0.91	23.0
SZD025	DD	29.7	34.5	Mbenge South 213	0.97	4.9
SZD026	DD	24.1	32.9	Mbenge South 213	1.98	8.8
SZD027	DD	76.3	81.9	Mbenge South 216	0.91	5.6
SZD029	DD	62.2	72.8	Mbenge South 213	2.64	10.6
SZD029	DD	156.8	163.5	Mbenge South 215	0.82	6.8
SZD030	DD	46.6	56.5	Mbenge South 213	2.43	10.0
SZD033	DD	50.4	58.2	Kenge Southeast	0.45	7.9
SZD034	DD	56.8	101.5	Kenge Southeast	0.28	44.7
SZD035	DD	89.1	122.0	Kenge Southeast	0.20	32.9
SZD036	DD	62.8	78.2	Kenge Southeast	0.36	15.4
SZD038	DD	123.0	131.2	Kenge Footwall	0.61	8.2
SZD039	DD	90.1	110.1	Kenge Footwall	1.17	19.9
SZD040	DD	219.8	262.4	Kenge Footwall	1.23	42.5
SZD041	DD	213.1	217.2	Kenge Footwall	1.72	4.1
SZD042	DD	103.0	112.5	Kenge Hanging Wall	1.23	9.5
SZD043	DD	118.7	142.5	Kenge Footwall	1.90	23.8
SZD045	DD	84.7	120.2	Kenge Footwall	1.66	35.5
SZD046	DD	32.6	33.9	Kenge Hanging Wall	0.01	1.2
SZD047	DD	32.7	38.7	Kenge Hanging Wall	0.02	6.0
SZD049	DD	93.1	112.1	Kenge Footwall	2.27	19.0
SZD050	DD	56.8	65.3	Kenge Footwall	0.09	8.5
SZD051	DD	93.7	97.0	Kenge Footwall	3.10	3.3
SZD052	DD	9.2	20.7	Kenge Hanging Wall	4.28	11.5
SZD053	DD	19.6	31.9	Kenge Hanging Wall	1.31	12.3

Hole Name	Drill Type	Depth From	Depth To	Mineralised Domain	Mean Au grade (g/t)	Intersection Length (m)
SZD054	DD	27.1	48.3	Kenge Hanging Wall	1.98	21.2
SZD055	DD	73.8	79.3	Kenge Hanging Wall	1.12	5.5
SZD056	DD	81.8	103.3	Kenge Southeast	3.62	21.5
SZD057	DD	28.4	58.9	Kenge Southeast	0.74	30.5
SZD058	DD	3.4	34.8	Kenge Southeast	0.87	31.4
SZD059	DD	41.7	69.0	Kenge Southeast	1.06	27.3
SZD060	DD	133.5	173.4	Kenge Southeast	1.24	39.9
SZD061	DD	78.4	96.0	Kenge Southeast	0.22	17.5
SZD062	DD	27.8	61.6	Kenge Hanging Wall	0.54	33.7
SZD063	DD	25.5	36.6	Kenge Hanging Wall	0.45	11.1
SZD064	DD	30.0	49.2	Kenge Hanging Wall	0.71	19.2
SZD065	DD	30.5	35.8	Kenge Hanging Wall	2.43	5.3
SZD066	DD	39.4	60.4	Kenge Hanging Wall	0.49	21.0
SZD067	DD	22.2	33.6	Kenge Hanging Wall	0.59	11.4
SZD068	DD	29.0	45.0	Kenge Hanging Wall	0.33	16.0
SZD069	DD	26.7	48.2	Kenge Hanging Wall	2.81	21.4
SZD070	DD	38.0	71.4	Kenge Hanging Wall	2.15	33.4
SZD071	DD	50.0	70.6	Kenge Hanging Wall	1.33	20.6
SZD072	DD	27.1	35.5	Kenge Southeast	0.28	8.4
SZD074	DD	23.0	49.0	Mbenge 201	1.54	26.0
SZD075	DD	13.8	19.2	Mbenge 201	0.98	5.4
SZD076	DD	19.0	62.5	Mbenge 201	2.32	43.5
SZD077	DD	65.0	95.7	Mbenge 201	1.82	30.7
SZD077	DD	99.1	102.2	Mbenge 201	0.68	3.0
SZD077	DD	108.3	112.0	Mbenge 201	0.83	3.8
SZD078	DD	91.2	109.4	Mbenge 201	2.32	18.2
SZD079	DD	147.9	163.2	Kenge Footwall	1.76	15.3
SZD080	DD	169.9	185.6	Kenge Footwall	2.15	15.8
SZD081	DD	35.4	76.0	Kenge Hanging Wall	1.77	40.6
SZD082	DD	167.3	184.5	Kenge Footwall	0.96	17.2
SZD083	DD	148.3	151.4	Kenge Footwall	0.72	3.1
SZD084	DD	175.3	184.6	Kenge Footwall	1.10	9.3
SZD085	DD	59.1	71.0	Kenge Hanging Wall	0.77	11.9
SZD086	DD	170.8	180.8	Kenge Footwall	1.84	10.0
SZD088	DD	201.0	201.9	Kenge Footwall	0.01	0.9
SZD089	DD	43.4	59.8	Kenge Hanging Wall	0.51	16.4
SZD089A	DD	45.1	59.3	Kenge Hanging Wall	0.27	14.2
SZD090	DD	193.9	196.7	Kenge Footwall	0.44	2.8
SZD091	DD	34.0	44.3	Kenge Hanging Wall	0.36	10.3
SZD092	DD	43.2	66.8	Kenge Hanging Wall	1.15	23.6
SZD092A	DD	42.2	63.6	Kenge Hanging Wall	0.13	21.3
SZD093	DD	47.3	52.3	Kenge Hanging Wall	0.42	5.0
SZD094	DD	185.9	195.3	Kenge Footwall	0.35	9.4
SZD095	DD	55.6	61.0	Kenge Hanging Wall	0.42	5.4

Hole Name	Drill Type	Depth From	Depth To	Mineralised Domain	Mean Au grade (g/t)	Intersection Length (m)
SZD096	DD	170.1	180.2	Kenge Footwall	0.45	10.1
SZD097	DD	101.8	125.3	Kenge Southeast	0.95	23.5
SZD099	DD	143.5	158.2	Kenge Southeast	1.14	14.7
SZD100	DD	152.9	176.6	Kenge Southeast	0.50	23.7
SZD101	DD	20.2	34.8	Mbenge 201	1.82	14.6
SZD101	DD	51.5	57.6	Mbenge 201	0.46	6.1
SZD102	DD	29.8	31.5	Mbenge 201	0.58	1.8
SZD102	DD	37.8	53.9	Mbenge 201	2.61	16.1
SZD102	DD	56.2	68.8	Mbenge 201	1.38	12.6
SZD102	DD	76.8	79.9	Mbenge 201	0.78	3.1
SZD104	DD	131.0	137.0	Mbenge 201	0.67	6.1
SZD105	DD	127.1	134.0	Mbenge 201	0.69	6.8
SZD105	DD	262.3	279.9	Mbenge 203	1.62	17.6
SZD106	DD	6.0	24.9	Mbenge 201	1.42	18.9
SZD106	DD	29.7	35.7	Mbenge 201	0.76	6.1
SZD106	DD	35.8	59.2	Mbenge 201	2.96	23.5
SZD107	DD	172.2	184.7	Mbenge 203	0.81	12.5
SZD108	DD	198.1	207.7	Mbenge 203	1.55	9.6
SZD110	DD	55.5	69.5	Mbenge 201	0.98	14.0
SZD110	DD	79.1	82.8	Mbenge 201	0.51	3.6
SZD110	DD	89.0	100.4	Mbenge 201	2.10	11.3
SZD111	DD	97.7	106.8	Mbenge 201	0.47	9.0
SZD111	DD	117.4	146.4	Mbenge 201	1.49	29.0
SZD114	DD	162.9	173.4	Kenge Footwall	1.18	10.5
SZD115	DD	190.4	200.0	Kenge Footwall	1.41	9.5
SZD116	DD	216.9	234.7	Kenge Footwall	1.01	17.8
SZD117	DD	141.6	154.8	Kenge Footwall	1.87	13.2
SZD118	DD	163.7	167.9	Kenge Footwall	1.25	4.2
SZD119	DD	163.9	177.9	Kenge Footwall	1.63	13.9
SZD120	DD	172.7	184.4	Kenge Footwall	1.13	11.7
SZD121	DD	189.7	191.1	Kenge Footwall	2.15	1.5
SZD122	DD	51.7	68.8	Kenge Hanging Wall	1.96	17.0
SZD123	DD	43.8	64.1	Kenge Hanging Wall	1.56	20.4
SZD124	DD	43.9	56.9	Kenge Hanging Wall	1.42	13.0
SZD125	DD	51.6	62.6	Kenge Hanging Wall	1.02	11.0
SZD126	DD	43.5	57.7	Kenge Hanging Wall	3.20	14.2
SZD127	DD	50.6	59.7	Kenge Hanging Wall	0.37	9.2
SZD128	DD	203.0	210.8	Kenge Footwall	1.48	7.8
SZD129	DD	221.5	231.7	Kenge Footwall	1.05	10.2
SZD130	DD	41.6	56.0	Kenge Hanging Wall	1.56	14.3
SZD131	DD	63.4	79.2	Kenge Hanging Wall	0.35	15.9
SZD132	DD	46.4	59.7	Kenge Hanging Wall	1.31	13.2
SZD133	DD	210.4	224.5	Kenge Footwall	1.05	14.1
SZD134	DD	232.8	249.6	Kenge Footwall	0.74	16.8

Hole Name	Drill Type	Depth From	Depth To	Mineralised Domain	Mean Au grade (g/t)	Intersection Length (m)
SZD135	DD	59.1	74.3	Kenge Hanging Wall	0.37	15.2
SZD136	DD	223.5	235.3	Kenge Footwall	0.66	11.8
SZD137	DD	247.3	253.2	Kenge Footwall	1.11	5.9
SZD138	DD	39.9	63.2	Kenge Hanging Wall	0.93	23.3
SZD139	DD	66.9	87.1	Kenge Hanging Wall	1.46	20.2
SZD140	DD	227.1	238.4	Kenge Footwall	2.10	11.3
SZD141	DD	254.1	267.4	Kenge Footwall	0.27	13.3
SZD142	DD	83.2	94.4	Kenge Hanging Wall	0.92	11.2
SZD143	DD	104.5	127.0	Kenge Hanging Wall	0.33	22.4
SZD144	DD	12.7	37.4	Kenge Hanging Wall	2.44	24.7
SZD145	DD	11.5	37.5	Kenge Hanging Wall	1.19	25.9
SZD146	DD	81.7	90.5	Kenge Hanging Wall	3.06	8.8
SZD147	DD	108.2	119.7	Kenge Hanging Wall	0.63	11.5
SZD148A	DD	11.3	31.8	Kenge Hanging Wall	1.26	20.5
SZD149	DD	10.6	19.2	Kenge Hanging Wall	1.10	8.6
SZD150	DD	72.0	95.3	Kenge Footwall	0.53	23.3
SZD151	DD	70.7	82.3	Kenge Hanging Wall	0.73	11.6
SZD152	DD	110.9	114.9	Kenge Hanging Wall	0.54	4.0
SZD153	DD	13.0	17.0	Kenge Hanging Wall	1.71	4.0
SZD153	DD	80.3	97.1	Kenge Footwall	1.08	16.8
SZD154	DD	134.6	134.7	Mbenge 201	0.80	0.1
SZD154	DD	148.2	153.4	Mbenge 201	0.52	5.2
SZD154	DD	285.3	289.3	Mbenge 203	0.89	4.1
SZD155	DD	63.8	66.9	Kenge Hanging Wall	0.96	3.1
SZD156	DD	86.1	89.2	Kenge Hanging Wall	0.07	3.1
SZD157	DD	3.0	15.5	Kenge Hanging Wall	2.19	12.5
SZD158	DD	6.3	19.7	Kenge Hanging Wall	4.85	13.4
SZD159	DD	82.6	91.7	Kenge Footwall	1.78	9.1
SZD160	DD	79.7	81.2	Mbenge 201	0.39	1.5
SZD160	DD	83.0	113.0	Mbenge 201	2.13	29.9
SZD160	DD	119.5	139.4	Mbenge 201	3.51	19.9
SZD161	DD	69.3	76.4	Kenge Hanging Wall	0.71	7.1
SZD162	DD	94.3	101.7	Kenge Hanging Wall	0.13	7.4
SZD164	DD	63.6	73.8	Kenge Hanging Wall	0.50	10.1
SZD165	DD	87.6	95.7	Kenge Hanging Wall	1.03	8.2
SZD167	DD	89.0	99.1	Kenge Hanging Wall	0.80	10.1
SZD168	DD	137.6	144.7	Kenge Hanging Wall	1.33	7.1
SZD169	DD	80.9	91.5	Kenge Hanging Wall	1.20	10.7
SZD170	DD	110.9	124.2	Kenge Hanging Wall	0.62	13.4
SZD171	DD	169.0	174.5	Mbenge 201	0.65	5.5
SZD171	DD	176.8	184.9	Mbenge 201	1.45	8.1
SZD171	DD	199.5	202.3	Mbenge 201	0.60	2.9
SZD171	DD	207.6	227.6	Mbenge 201	1.01	20.0
SZD172	DD	13.4	35.0	Konokono	1.02	21.6

Hole Name	Drill Type	Depth From	Depth To	Mineralised Domain	Mean Au grade (g/t)	Intersection Length (m)
SZD173	DD	30.0	64.1	Mbenge 201	1.46	34.2
SZD174	DD	196.7	199.4	Mbenge 201	0.43	2.8
SZD175	DD	13.4	35.8	Konokono	2.20	22.5
SZD176	DD	54.6	58.5	Mbenge 202	0.69	3.9
SZD176	DD	61.5	65.8	Mbenge 202	0.75	4.3
SZD177A	DD	76.0	101.7	Konokono	0.54	25.7
SZD179A	DD	63.5	75.0	Konokono	0.73	11.5
SZD181	DD	201.9	206.0	Mbenge 201	0.65	4.1
SZD181	DD	222.1	226.0	Mbenge 201	0.63	3.9
SZD181	DD	231.5	233.8	Mbenge 201	0.42	2.2
SZD182	DD	32.3	44.2	Mbenge 202	1.63	11.9
SZD183	DD	40.1	63.2	Mbenge 202	2.76	23.1
SZD186	DD	99.0	118.0	Kenge Southeast	0.26	19.0
SZD187	DD	132.5	142.3	Kenge Southeast	0.64	9.8
SZD191	DD	32.9	57.1	Mbenge 202	0.04	24.2
SZD192	DD	40.4	61.6	Mbenge 202	0.06	21.2
SZD195	DD	67.1	96.9	Mbenge South 215	0.14	29.9
SZR010	RC	116.0	136.0	Kenge Footwall	1.32	20.0
SZR011	RC	41.0	70.0	Kenge Hanging Wall	1.05	29.0
SZR012	RC	20.8	32.3	Kenge Hanging Wall	0.74	11.4
SZR015	RC	52.6	57.7	Mbenge South 215	0.58	5.1
SZR015	RC	87.7	100.7	Mbenge South 215	7.80	13.0
SZR016	RC	14.8	25.4	Mbenge South 213	2.02	10.6
SZR016	RC	109.8	113.3	Mbenge South 215	0.13	3.5
SZR016	RC	117.1	122.0	Mbenge South 215	0.55	4.9
SZR025	RC	9.7	31.6	Konokono	1.76	21.9
SZR052	RC	22.8	49.7	Mbenge 201	1.04	26.9
SZR054	RC	109.4	120.9	Mbenge 201	0.73	11.5
SZR054	RC	137.0	140.0	Mbenge 201	0.70	3.1
SZR058	RC	62.9	67.5	Mbenge 202	0.50	4.6
SZR095	RC	19.0	27.0	Kenge Hanging Wall	1.70	8.0
SZR096	RC	17.6	32.0	Kenge Hanging Wall	0.75	14.4
SZR097	RC	71.0	98.0	Kenge Hanging Wall	0.19	27.0
SZR098	RC	19.0	70.0	Kenge Hanging Wall	1.64	51.0
SZR099	RC	10.0	24.0	Kenge Hanging Wall	1.10	14.0
SZR100	RC	14.0	30.0	Kenge Hanging Wall	0.40	16.0
SZR101	RC	18.0	30.0	Kenge Hanging Wall	0.74	12.0
SZR102	RC	12.0	18.0	Kenge Hanging Wall	0.04	6.0
SZR103	RC	23.0	31.0	Kenge Hanging Wall	1.30	8.0
SZR104	RC	15.0	22.0	Kenge Hanging Wall	0.18	7.0
SZR105	RC	14.0	16.0	Kenge Hanging Wall	0.06	2.0
SZR106	RC	22.0	34.0	Kenge Hanging Wall	0.11	12.0
SZR107	RC	52.0	57.0	Kenge Hanging Wall	1.23	5.0
SZR108	RC	30.0	33.0	Kenge Hanging Wall	0.41	3.0

Hole Name	Drill Type	Depth From	Depth To	Mineralised Domain	Mean Au grade (g/t)	Intersection Length (m)
SZR109	RC	62.0	72.0	Kenge Hanging Wall	0.58	10.0
SZR110	RC	65.0	76.0	Kenge Hanging Wall	0.90	11.0
SZR111	RC	50.0	70.0	Kenge Hanging Wall	2.44	20.0
SZR112	RC	66.0	79.0	Kenge Hanging Wall	0.78	13.0
SZR115	RC	9.0	18.0	Kenge Southeast	0.18	9.0
SZR123	RC	146.0	150.0	Kenge Southeast	1.09	4.0

Appendix C: Charts of Analytical Quality Control Data

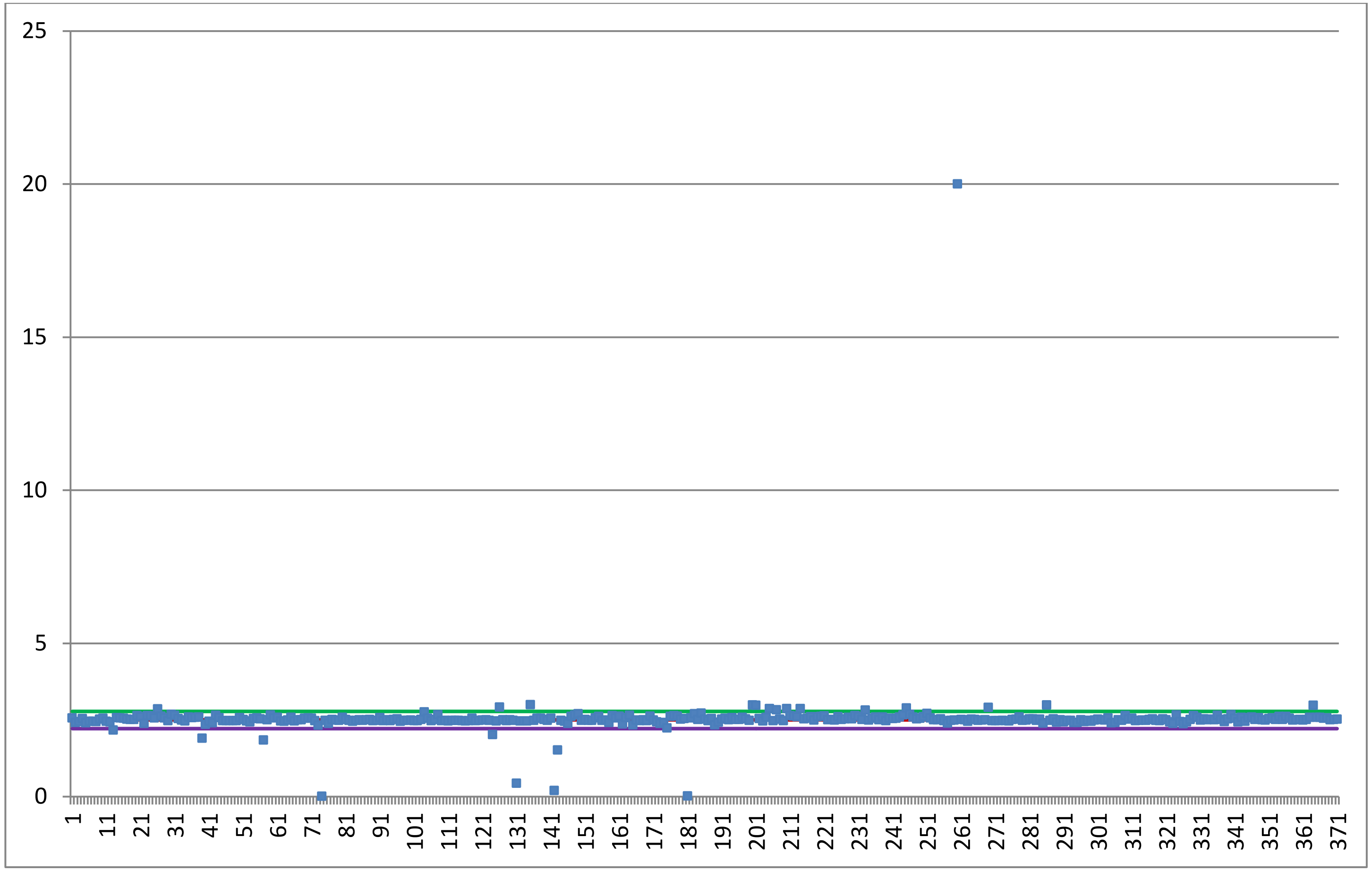

Figure C1: Analyses of standard G302-2

Certified mean of 2.50 g/t and standard deviation of 0.14. Control lines are drawn at ± 2 standard deviations.

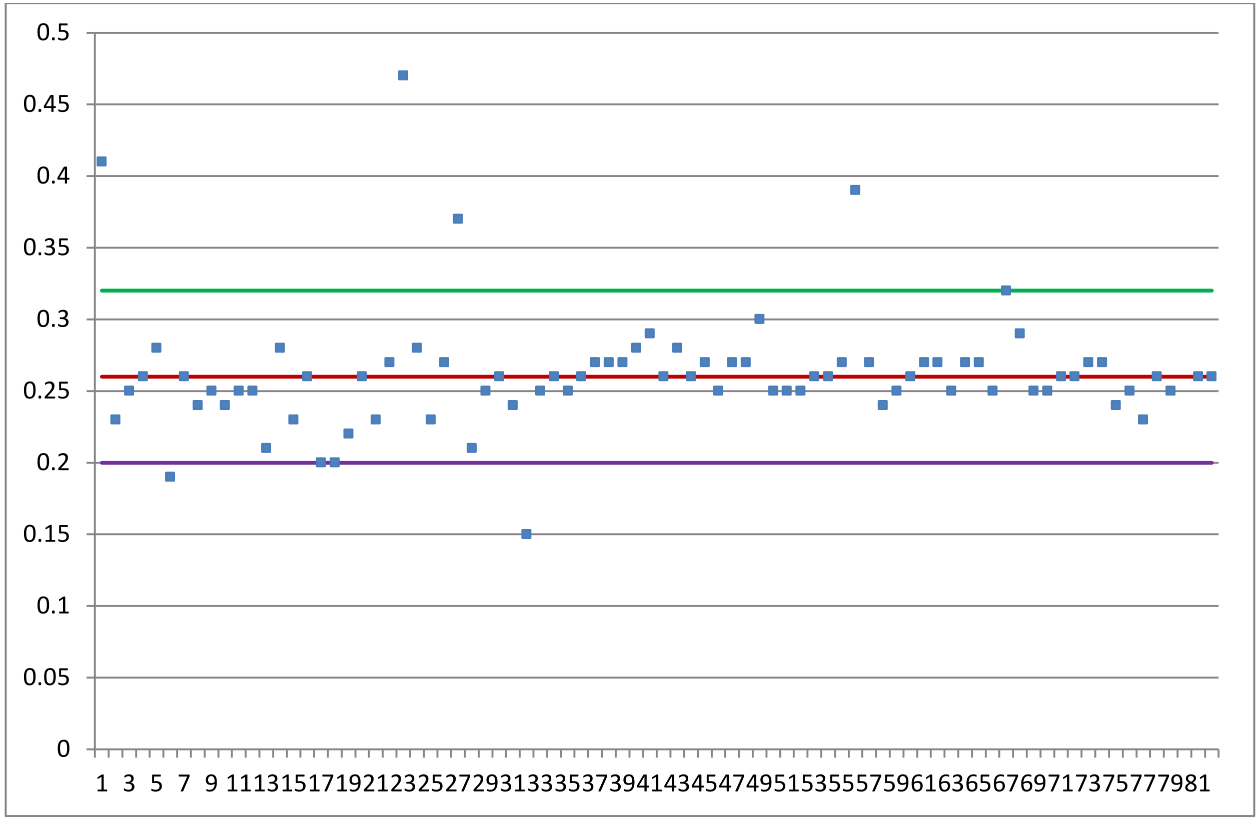

Figure C2: Analyses of standard G303-8

Certified mean of 0.26 g/t and standard deviation of 0.03. Control lines are drawn at ± 2 standard deviations.

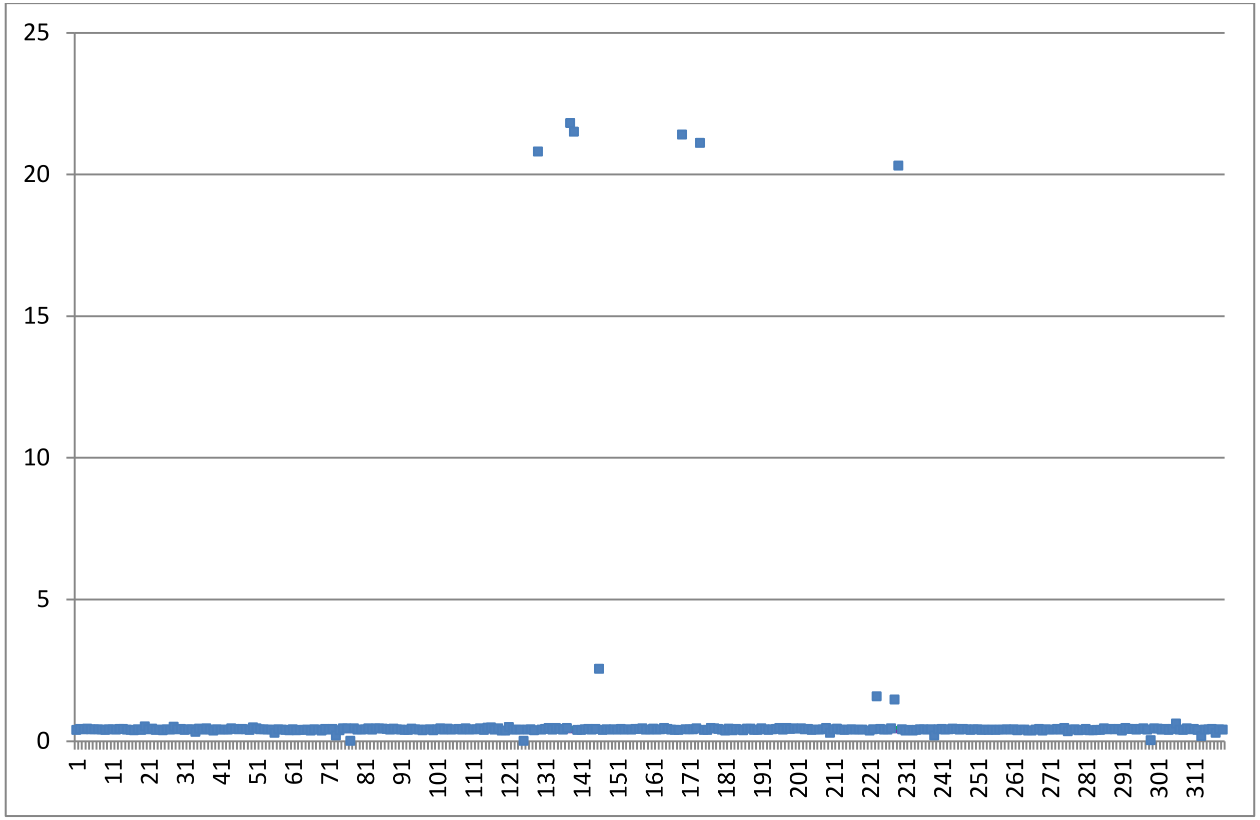

Figure C3: Analyses of standard G306-1

Certified mean of 0.41 g/t and standard deviation of 0.03. Control lines are drawn at ± 2 standard deviations.

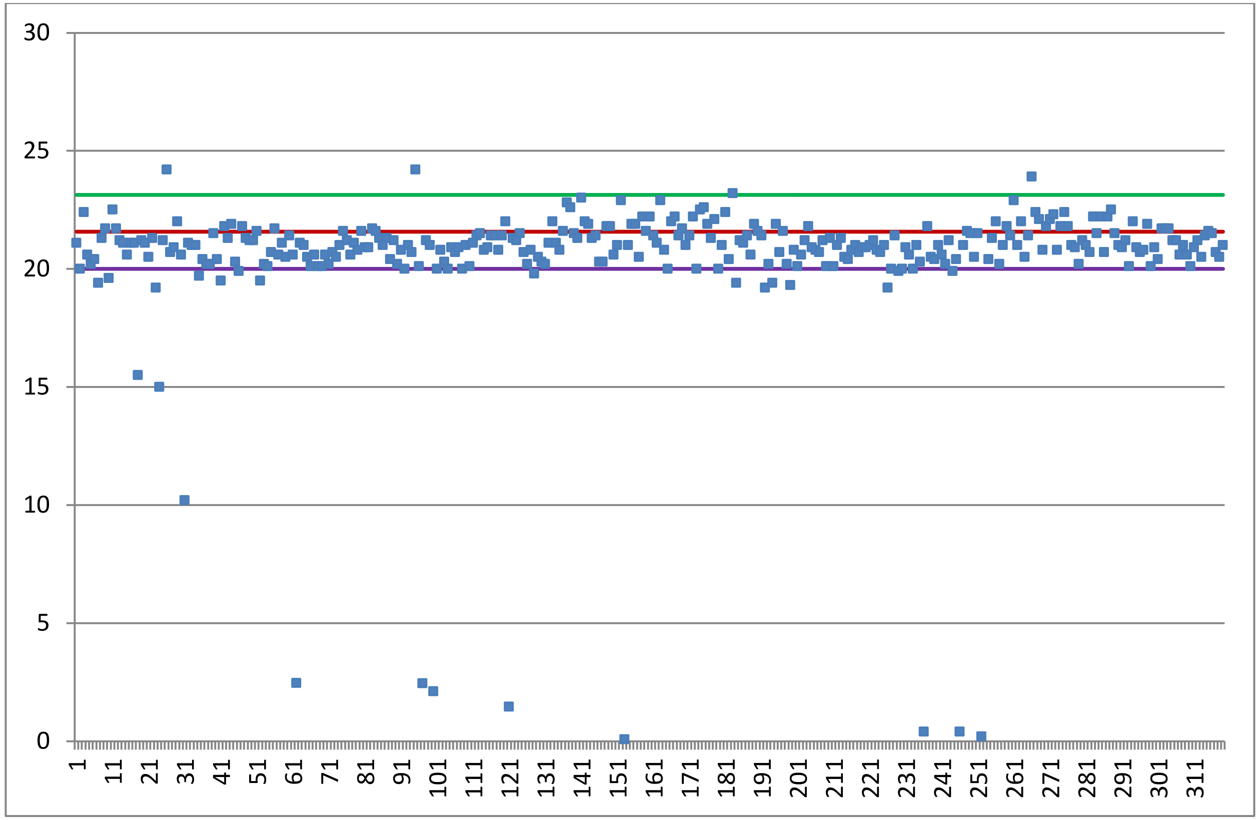

Figure C4: Analyses of standard G306-4

Certified mean of 21.57 g/t and standard deviation of 0.78. Control lines are drawn at ± 2 standard deviations.

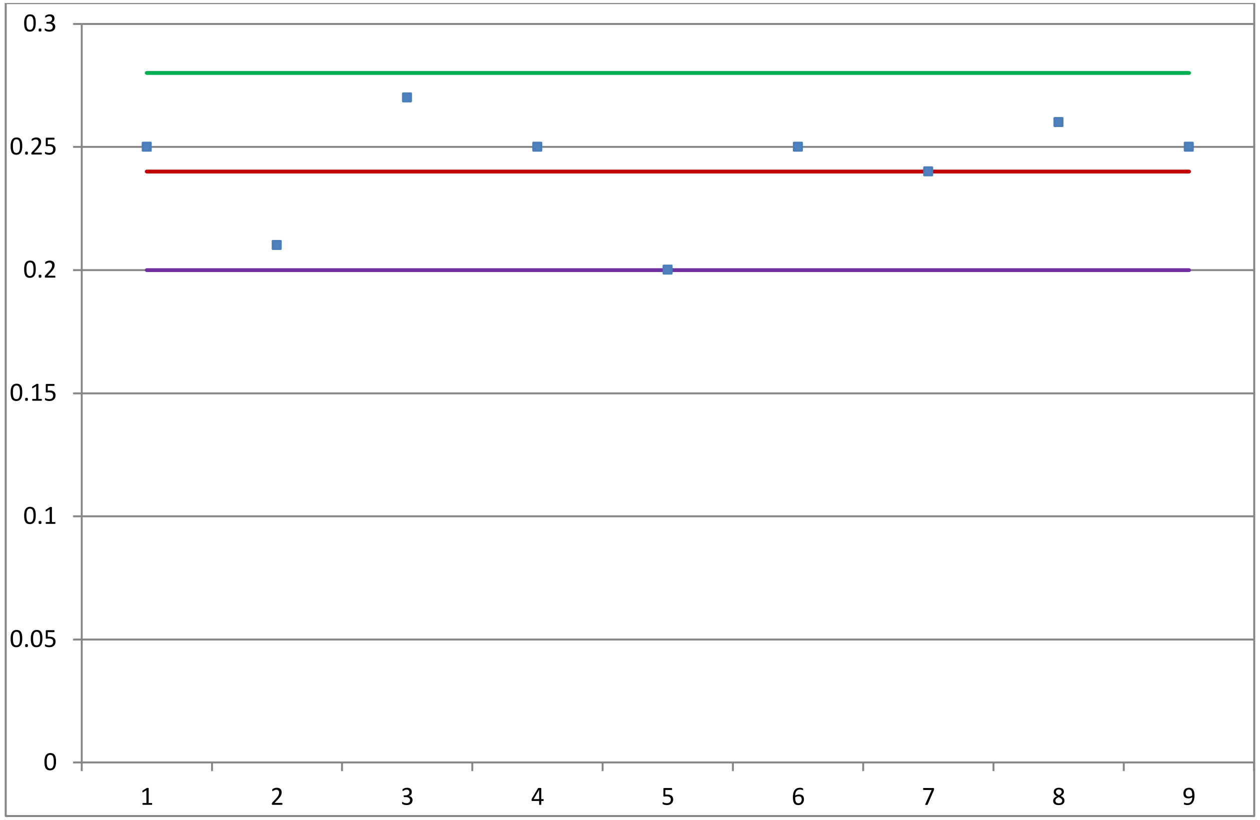

Figure C5: Analyses of standard G307-3

Certified mean of 0.24 g/t and standard deviation of 0.02. Control lines are drawn at ± 2 standard deviations.

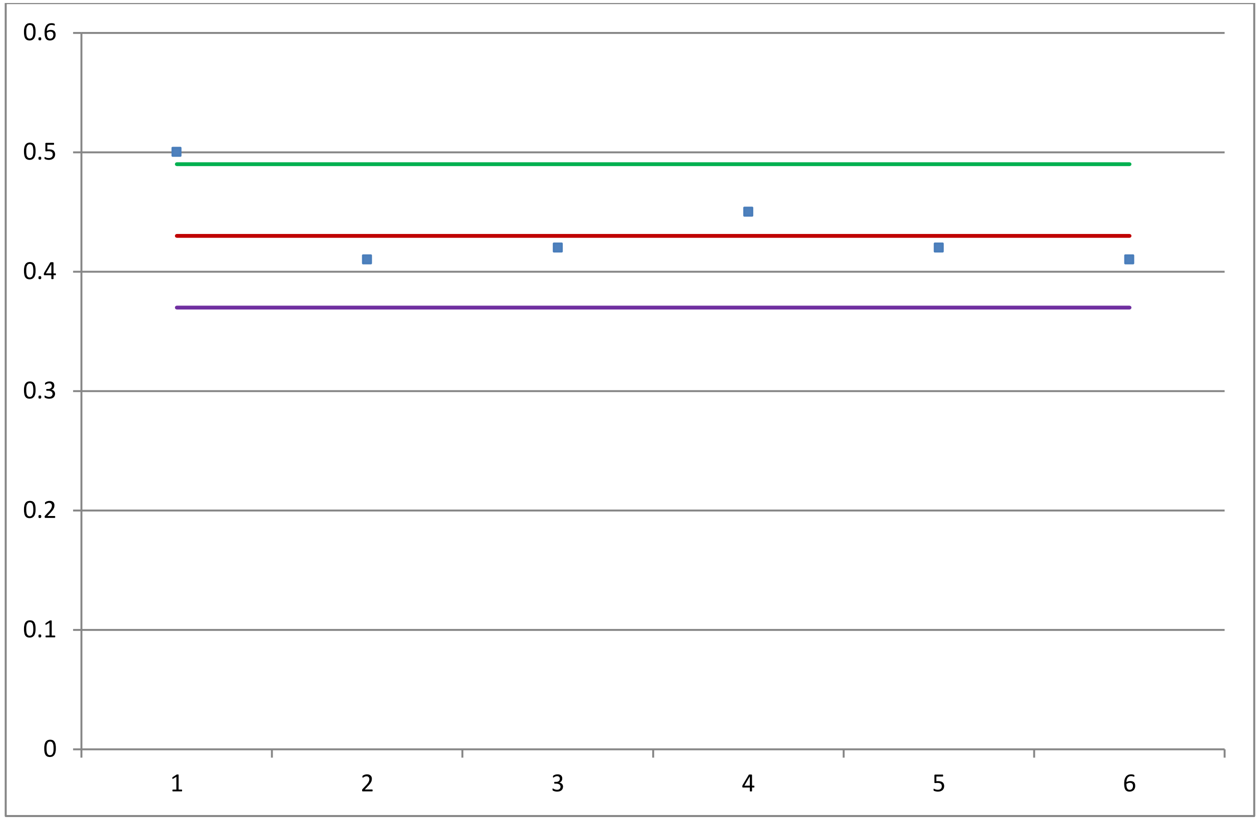

Figure C6: Analyses of standard G310-4

Certified mean of 0.43 g/t and standard deviation of 0.03. Control lines are drawn at ± 2 standard deviations.

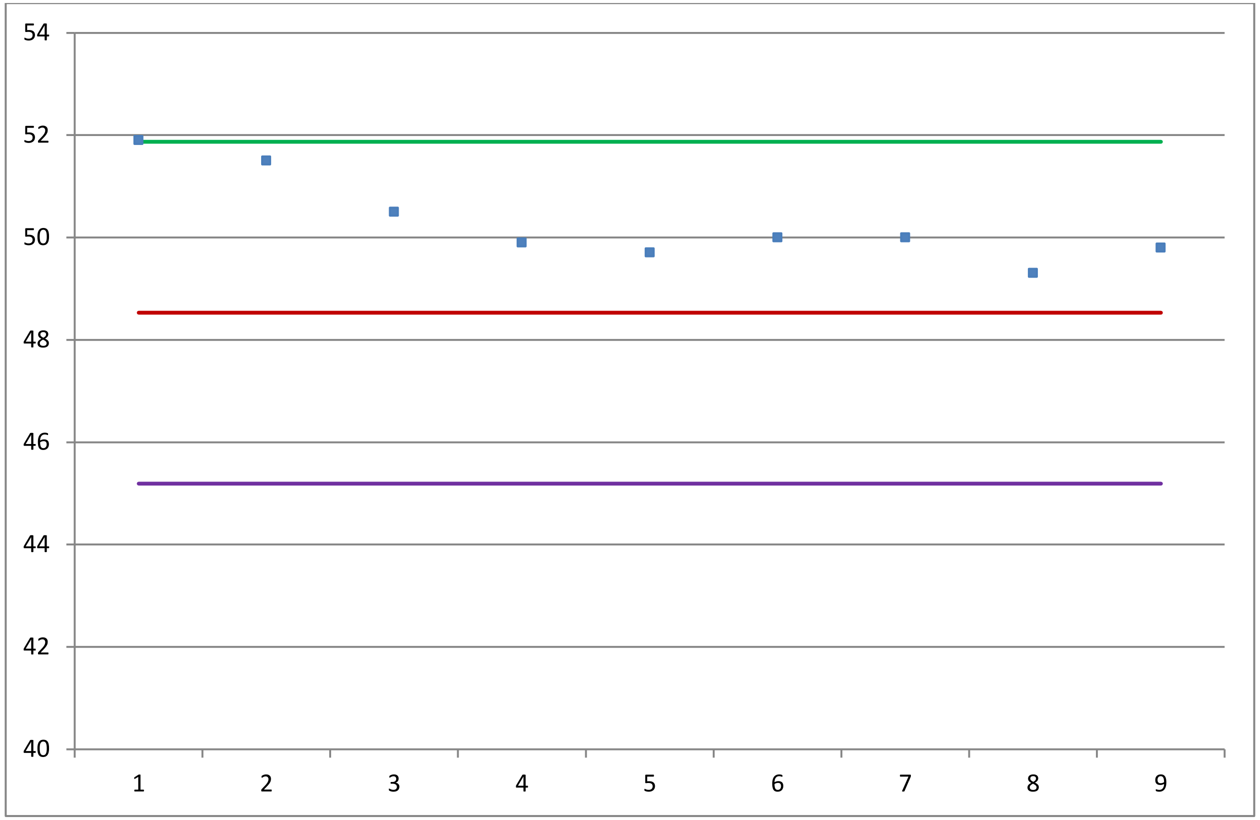

Figure C7: Analyses of standard G310-10

Certified mean of 48.53 g/t and standard deviation of 1.67. Control lines are drawn at ± 2 standard deviations.

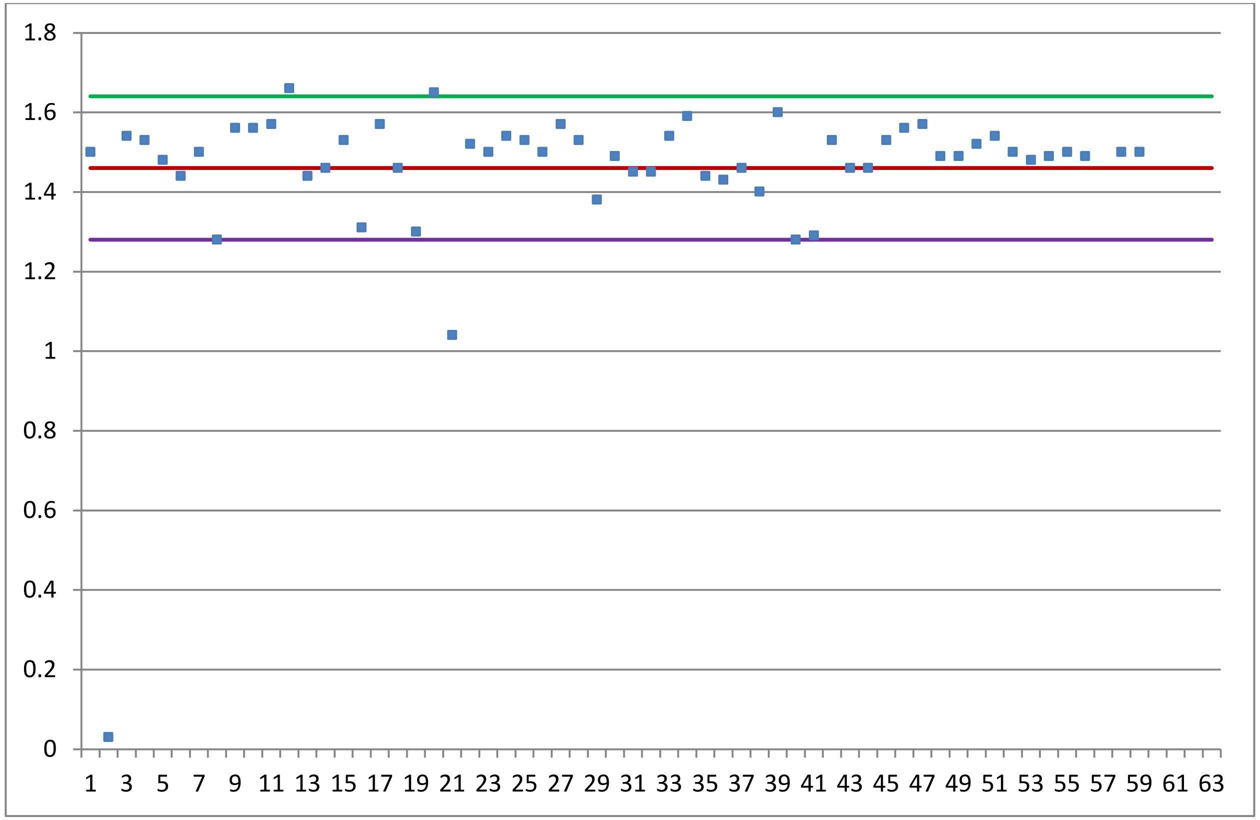

Figure C8: Analyses of standard G399-2

Certified mean of 1.46 g/t and standard deviation of 0.09. Control lines are drawn at ± 2 standard deviations.

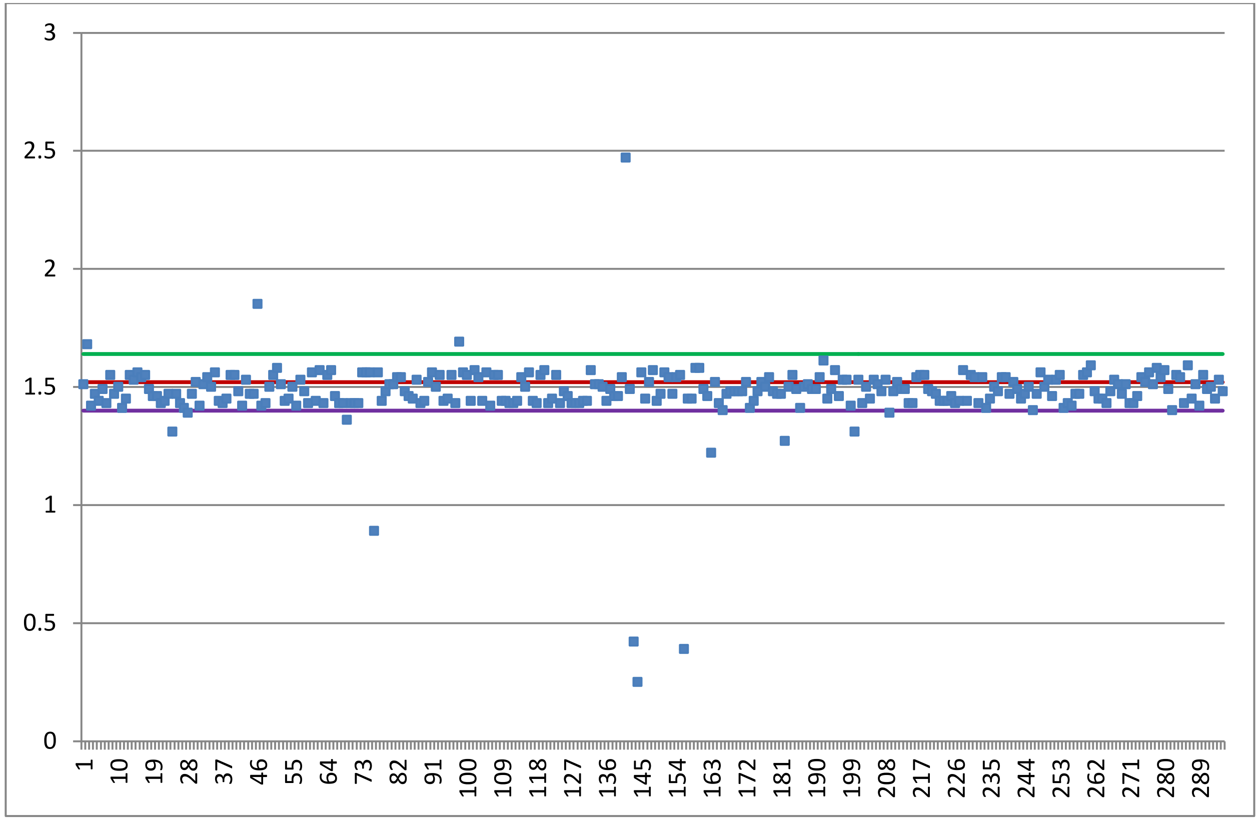

Figure C9: Analyses of standard G901-7

Certified mean of 1.52 g/t and standard deviation of 0.06. Control lines are drawn at ± 2 standard deviations.

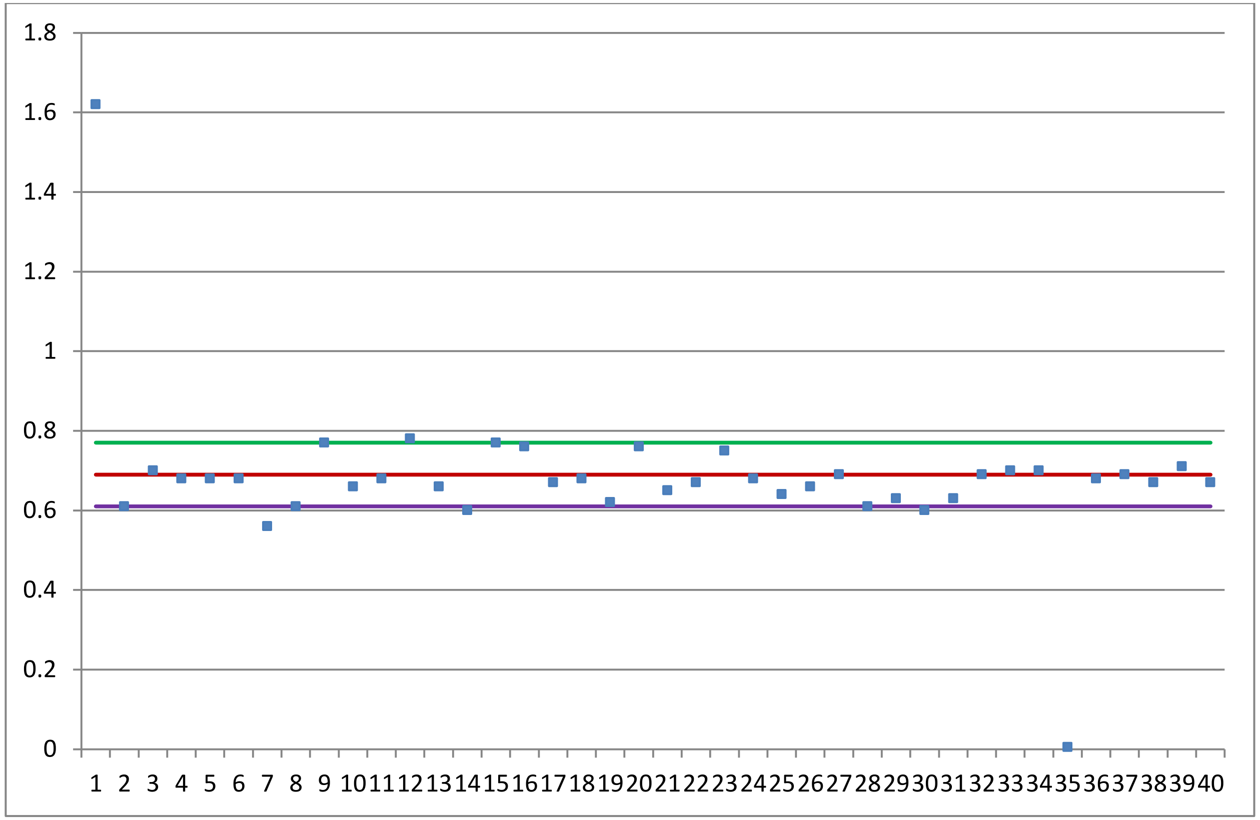

Figure C10: Analyses of standard G901-9

Certified mean of 0.69 g/t and standard deviation of 0.04. Control lines are drawn at ± 2 standard deviations.

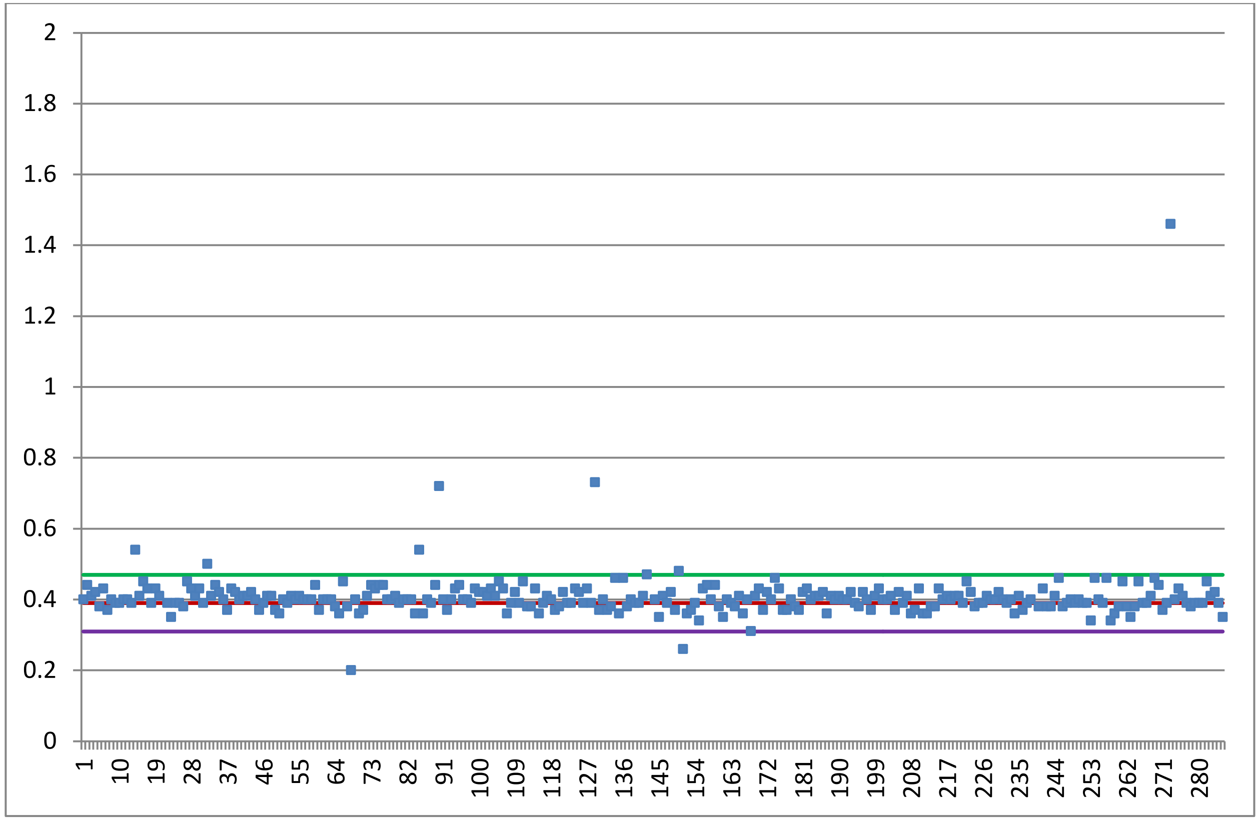

Figure C11: Analyses of standard G902-1

Certified mean of 0.39 g/t and standard deviation of 0.04. Control lines are drawn at ± 2 standard deviations.

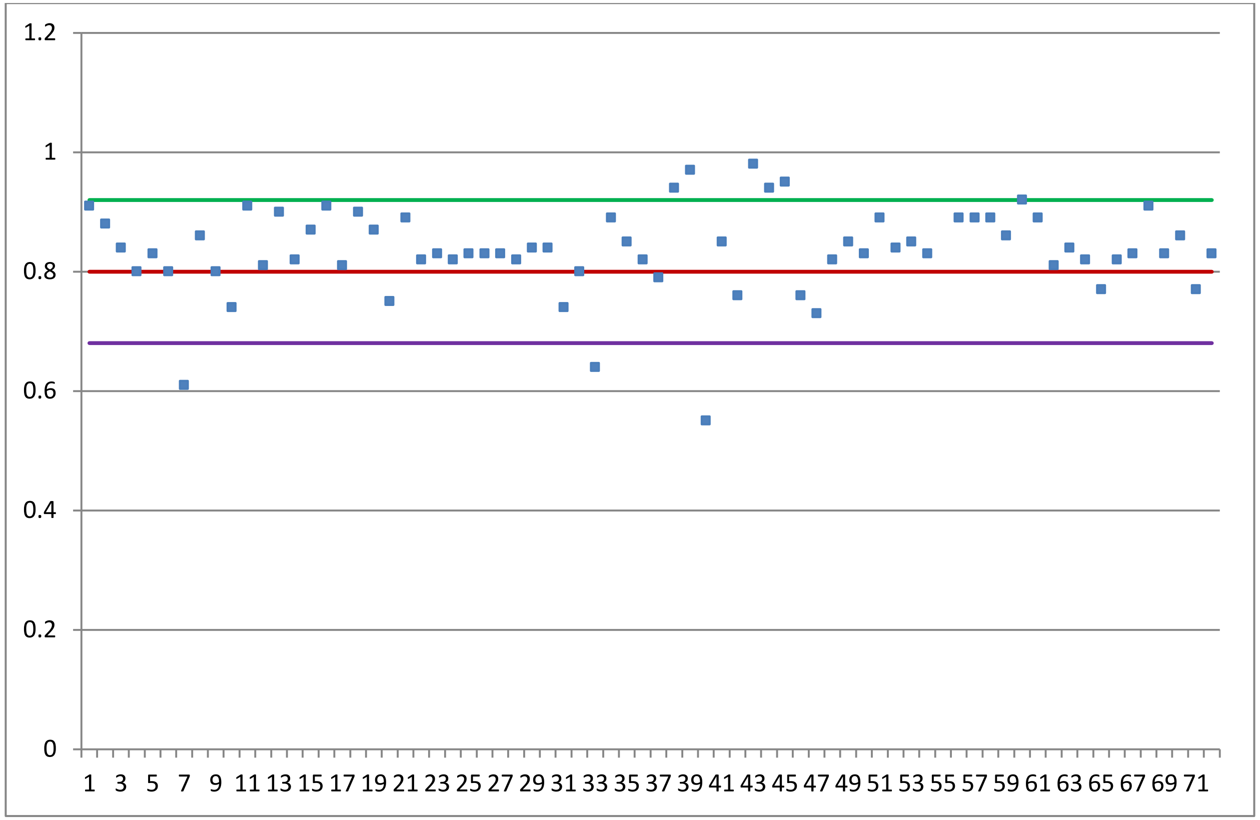

Figure C12: Analyses of standard G998-6

Certified mean of 0.80 g/t and standard deviation of 0.06. Control lines are drawn at ± 2 standard deviations.

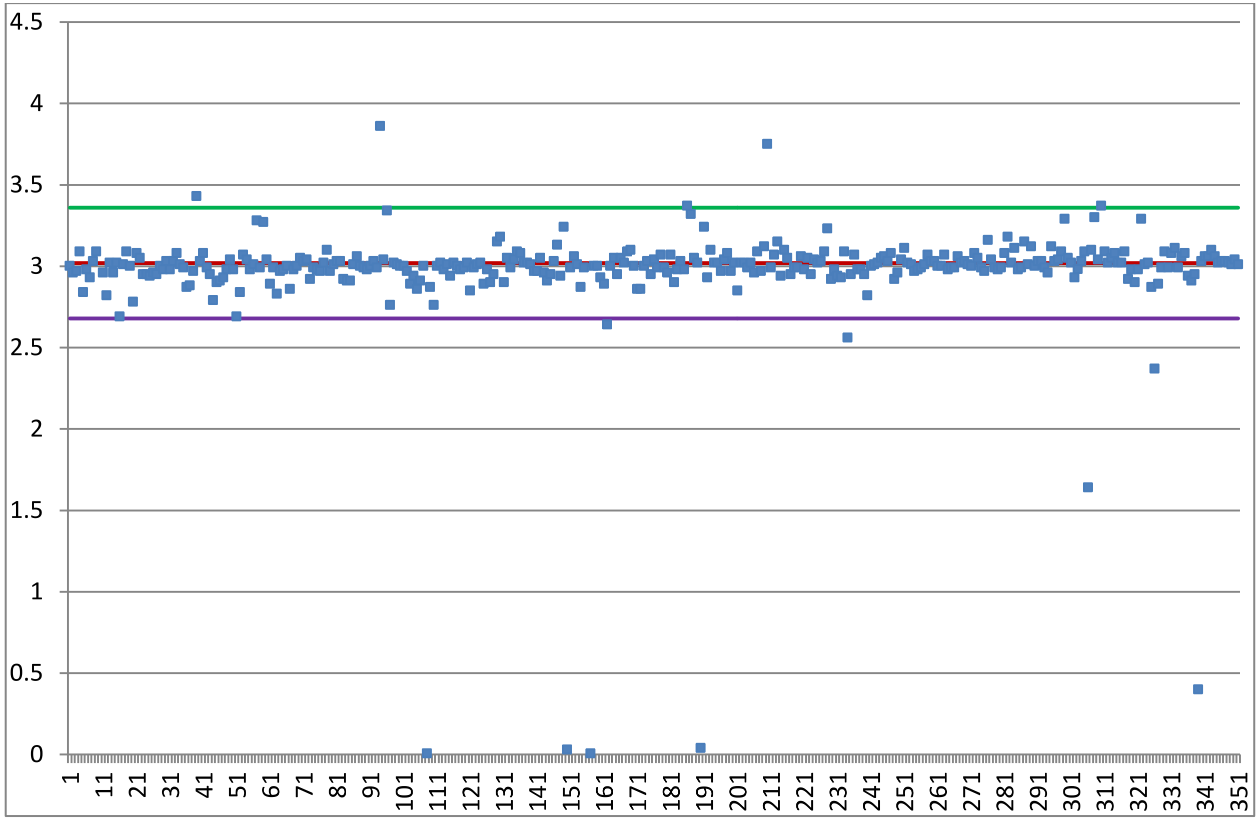

Figure C13: Analyses of standard G999-4

Certified mean of 3.02 g/t and standard deviation of 0.17. Control lines are drawn at ± 2 standard deviations.

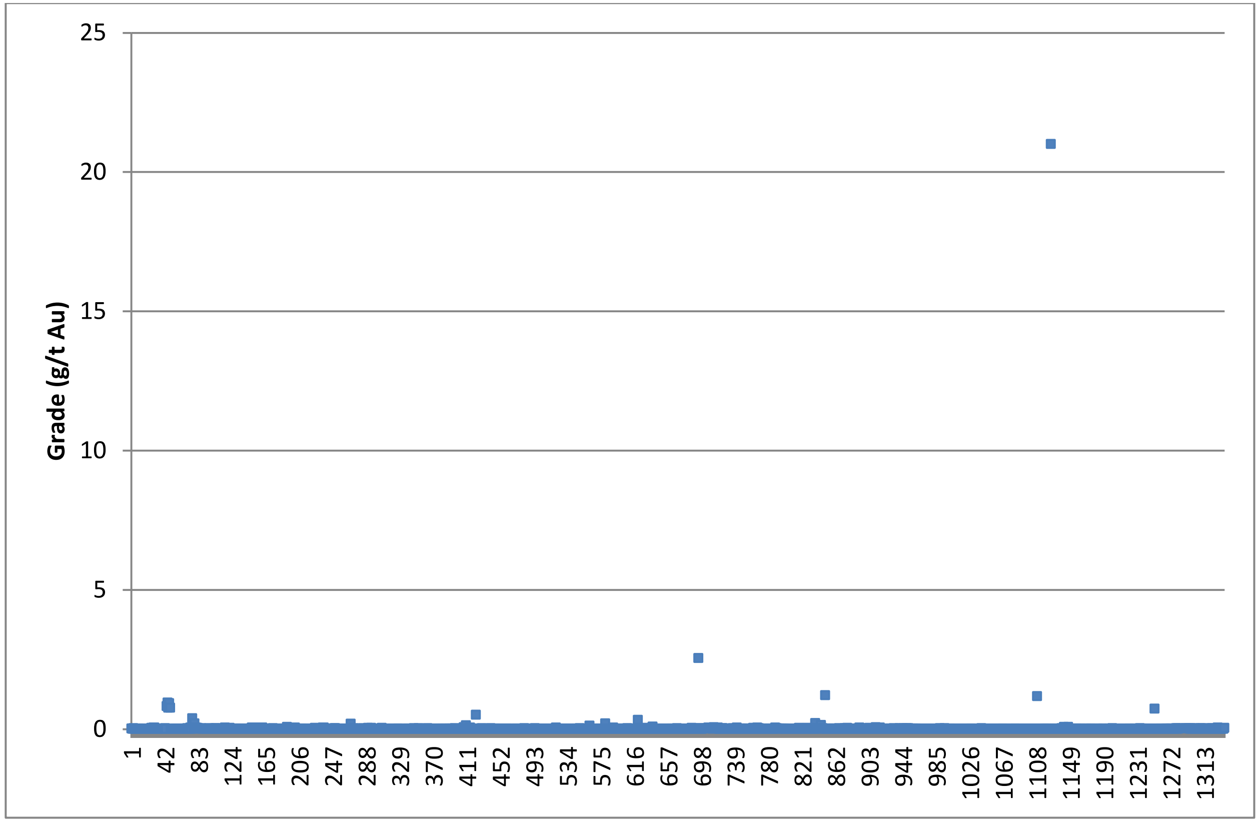

Figure C14: Analyses of blanks

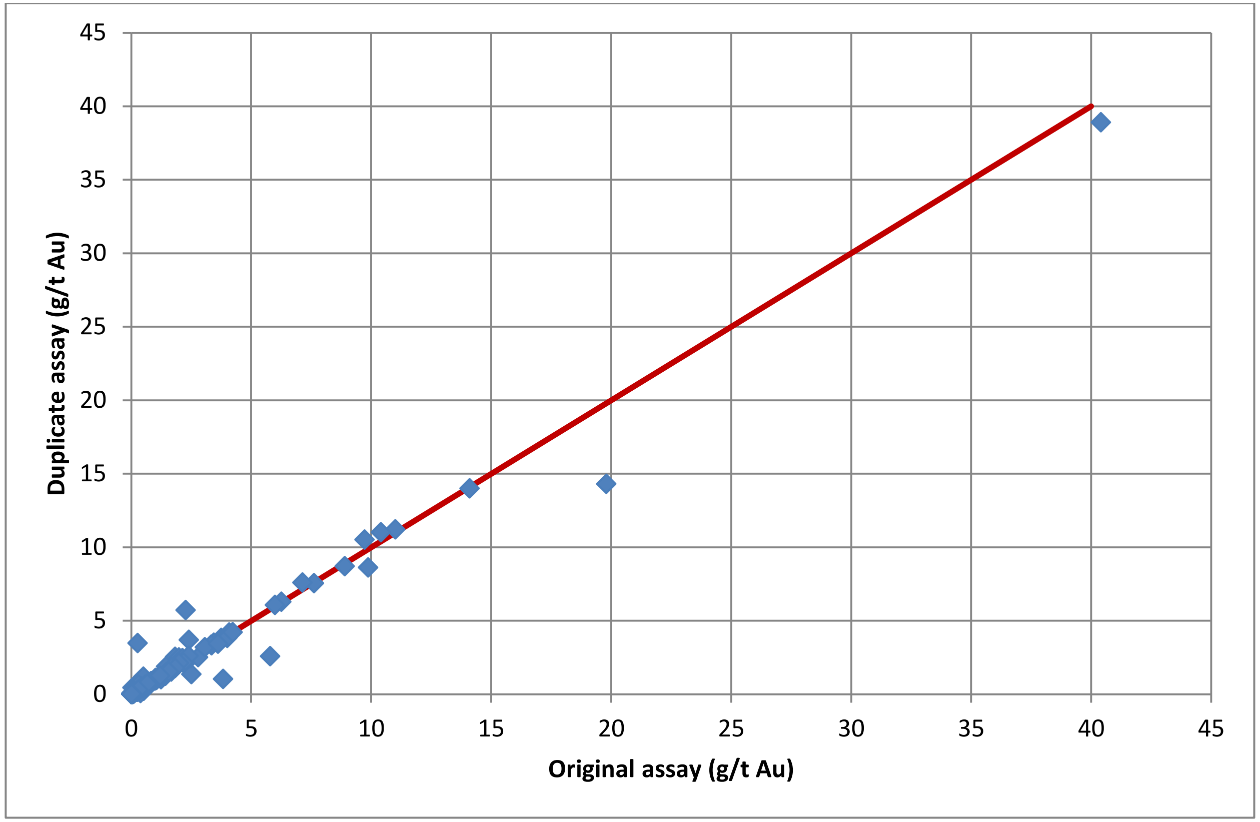

Figure C15: Scatterplot of DD duplicates from Kenge and Mbenge

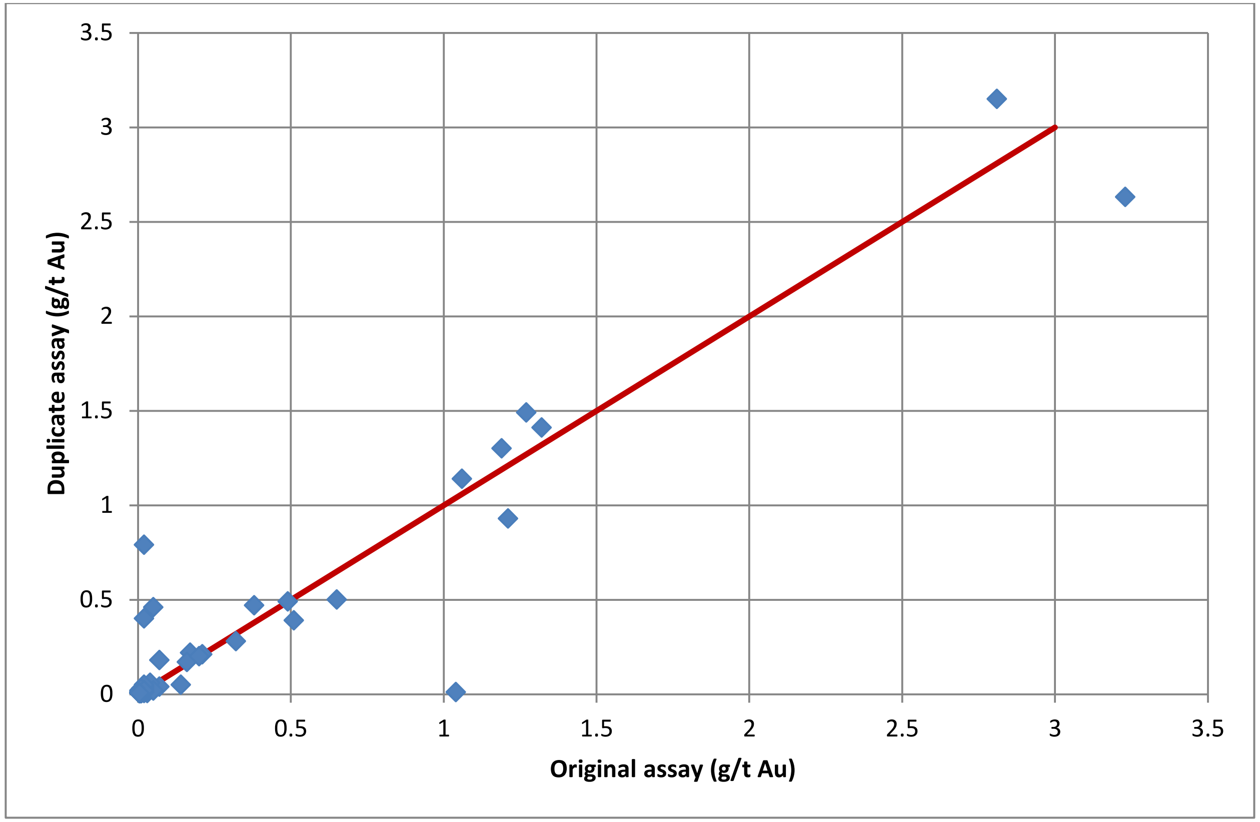

Figure C16: Scatterplot of RC duplicates from Kenge and Mbenge

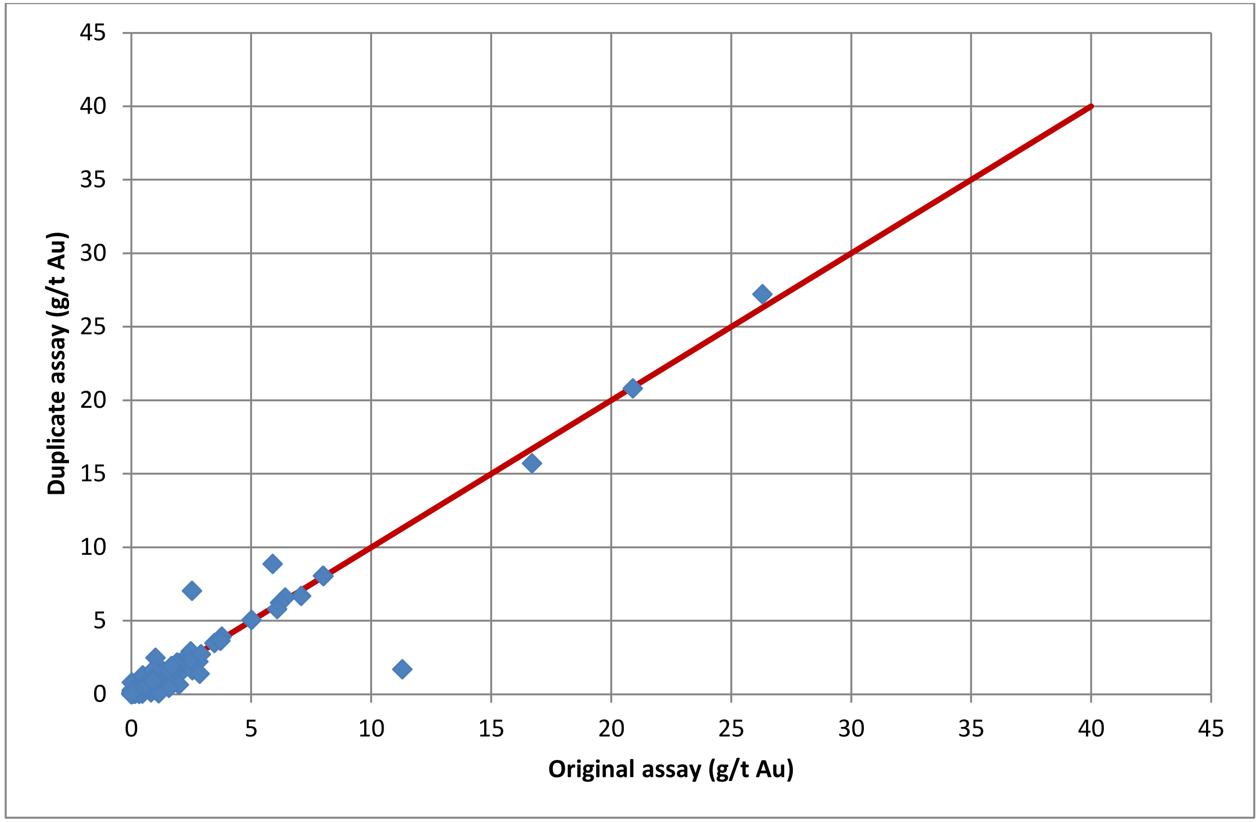

Figure C17: Scatterplot of DD duplicates from Porcupine, Konokono and Tumbili

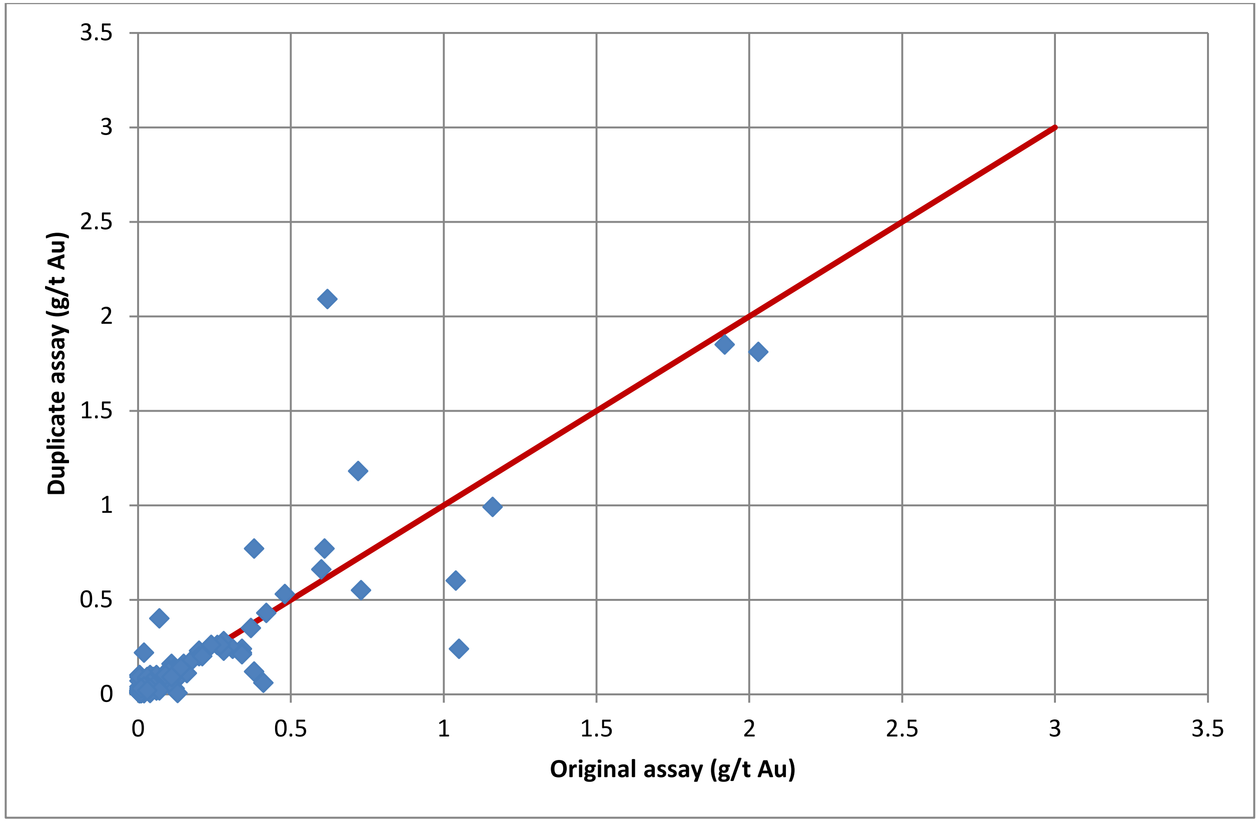

Figure C18: Scatterplot of RC duplicates from Porcupine, Konokono and Tumbili

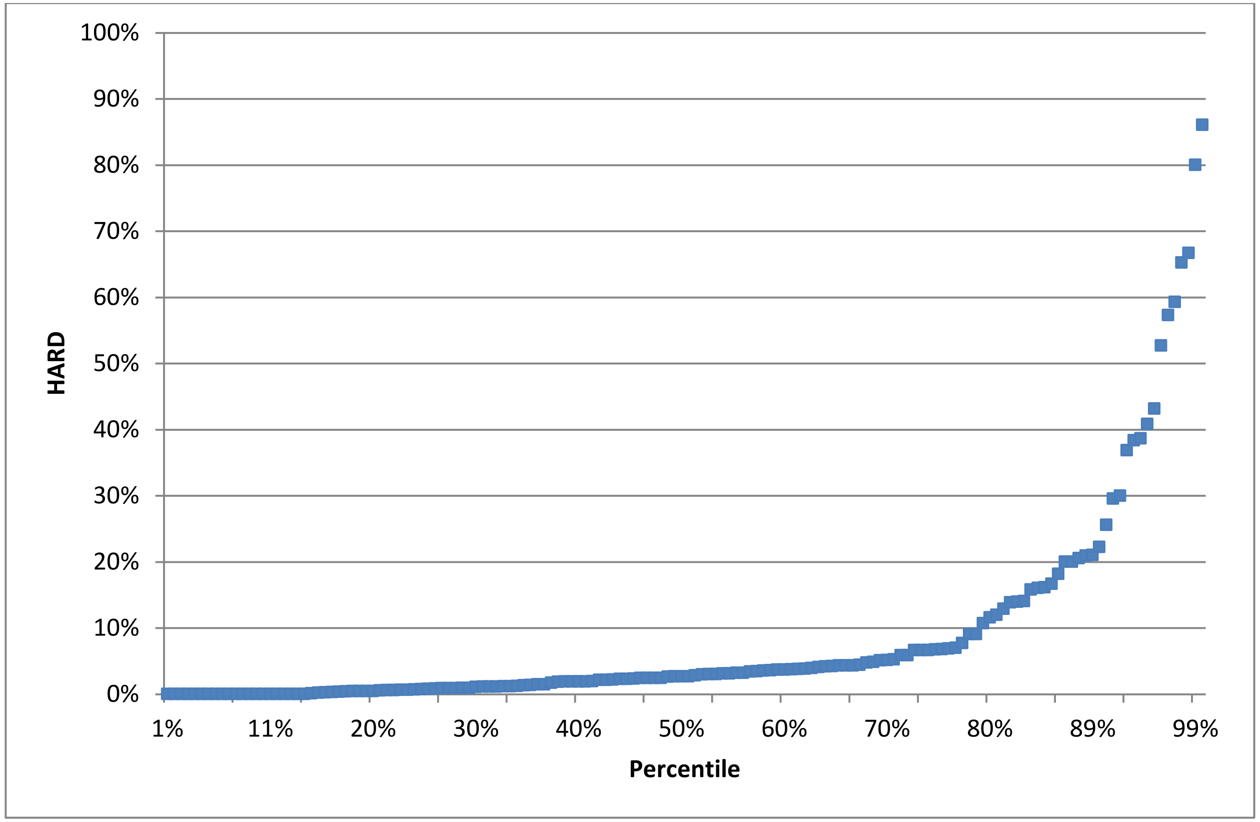

Figure C19: HARD (Half Absolute Relative Difference) plot for DD duplicates from Kenge and Mbenge

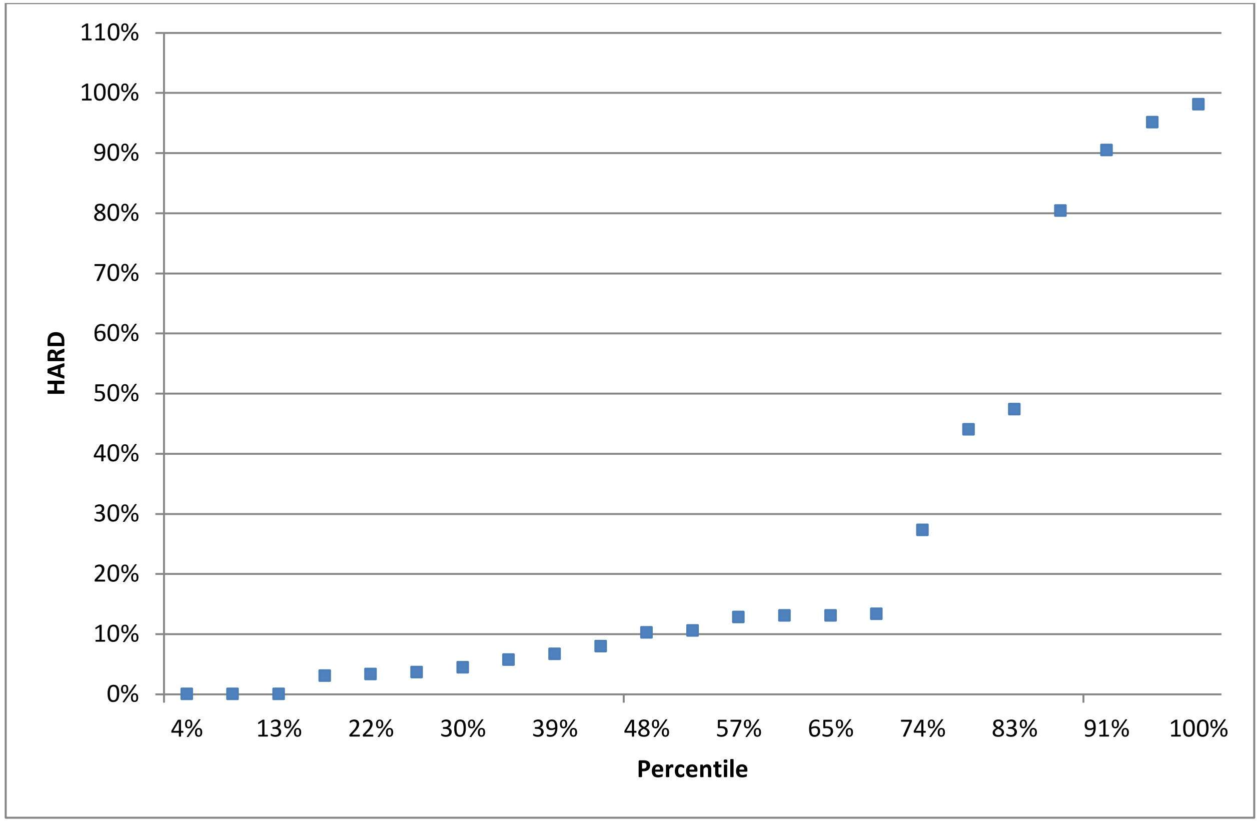

Figure C20: HARD (Half Absolute Relative Difference) plot for RC duplicates from Kenge and Mbenge

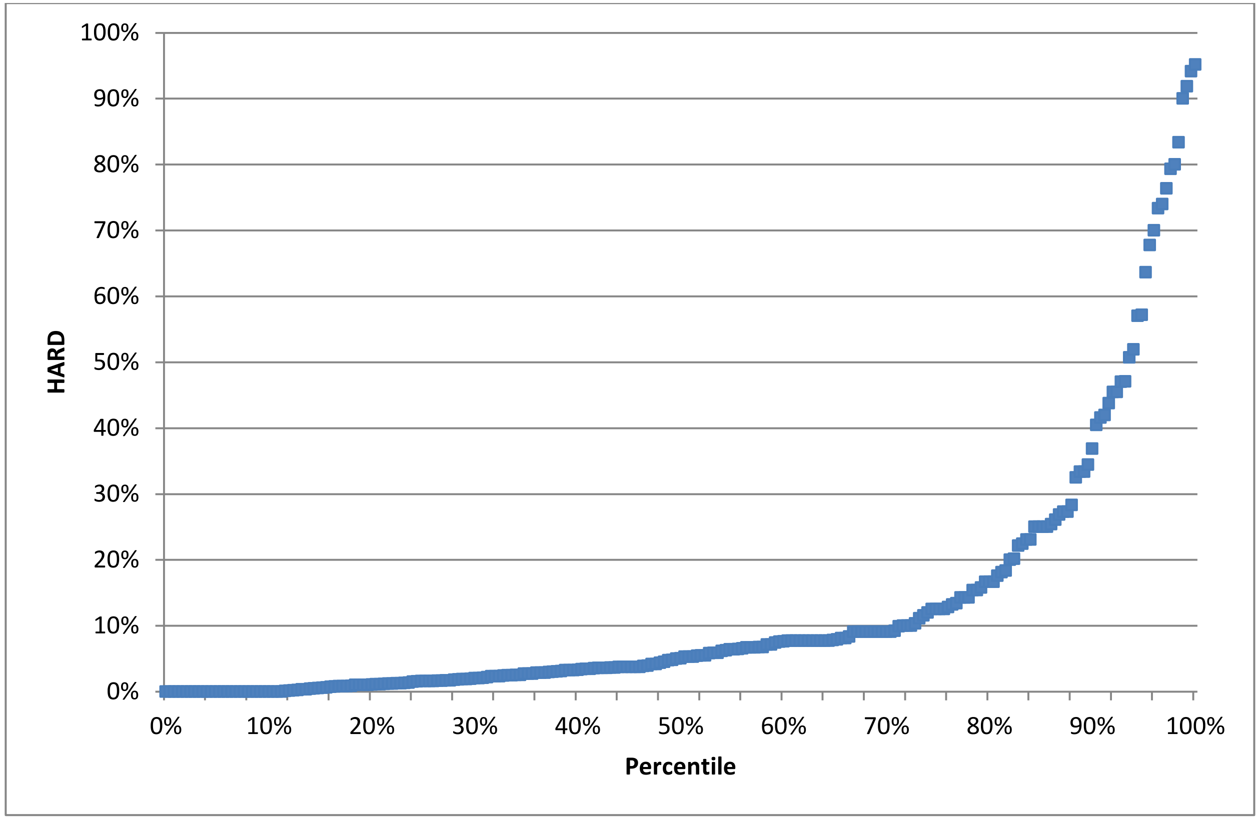

Figure C21: HARD (Half Absolute Relative Difference) plot for DD duplicates from Porcupine, Konokono and Tumbili

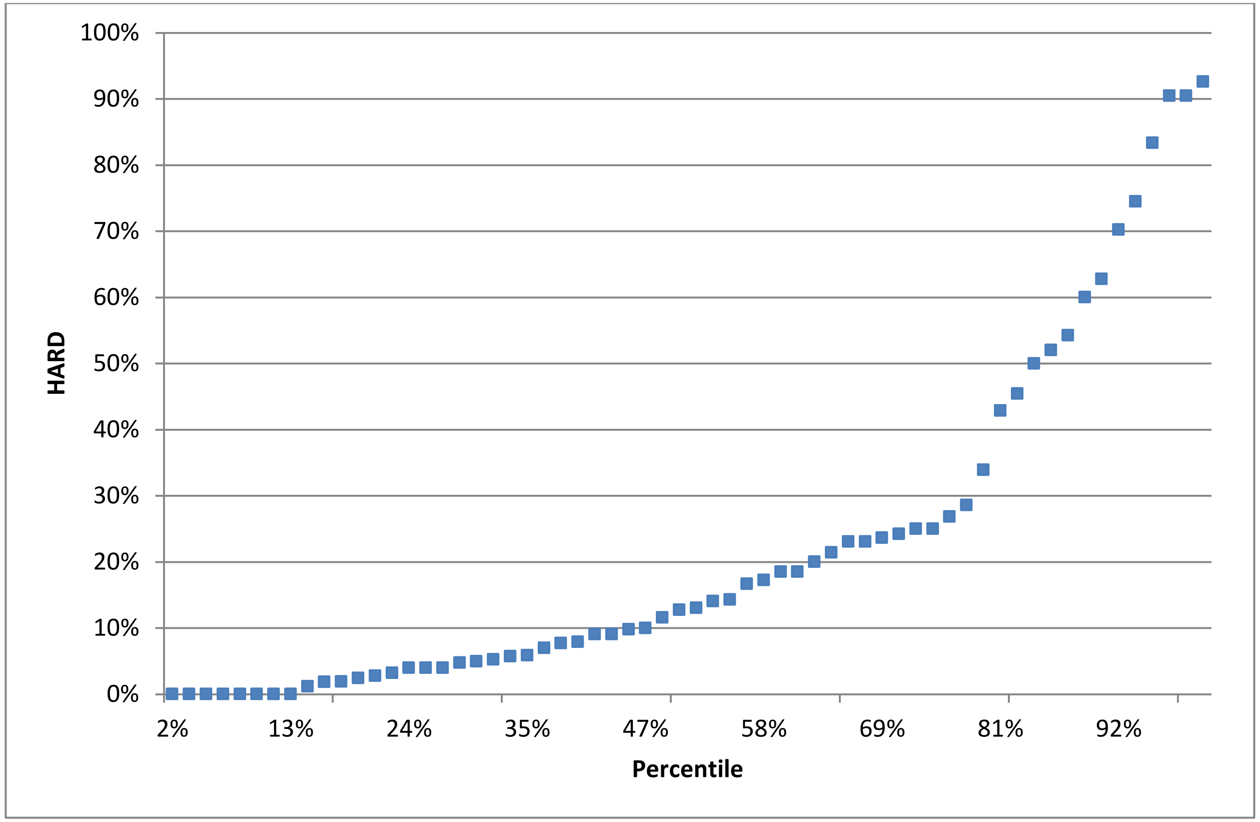

Figure C22: HARD (Half Absolute Relative Difference) plot for RC duplicates from Porcupine, Konokono and Tumbili

Appendix D: SGS Metallurgical

An Investigation of

THE RECOVERY OF GOLD FROM SAZA-MAKONGOLOSI PROJECT SAMPLES

prepared for

HELIO RESOURCE CORP/BAFEX TANZANIA LTD.

Project 11940-001 – Final Report
August 6, 2008

NOTE:
This report refers to the samples as received.

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Table of Contents

Introduction

This report presents the results from testwork on samples representing Helio Resource Corporation's Saza-Makongolosi Project located in Tanzania. The purpose of the program was to evaluate the processing characteristics of the ore at a scoping level, and to develop a preliminary process flowsheet. The program incorporated ore characterization tests (head analysis, mineralogy and comminution tests) as well as the evaluation of a number of processing options, including; gravity separation, flotation and cyanidation.

The test program was directed by Mr. Chris MacKenzie of Helio Resource Corp/BAFEX Tanzania Ltd. Test results were forwarded to Mr. MacKenzie as they became available.

James MacDonald
Project Manager

Inna Dymov, P.Eng
Gold Group Manager

Experimental Work by: *R. Huaraz, G. Thompson*
Report Prepared by: *R. Huaraz, J. MacDonald*

Testwork Summary

1. Sample Receipt, Preparation and Characterisation

1.1. Sample Receipt and Preparation

A single composite sample representing the Saza-Makongolosi Project was received in two boxes at SGS Minerals Services (Lakefield) on June 2, 2008. The boxes were assigned receipt number 0003-JUN08.

The composite sample was processed as follows:

1. The content of the two boxes were combined and labelled as SMP Comp 1.
2. The sample was crushed to nominally pass 6 mesh. One ~10-kg charge was riffled out for standard Bond ball mill work index (BWi) @ 100 mesh (150µm).
3. The balance of the sample was crushed to nominally pass 10 mesh.
4. The minus 10 mesh sample was rotary split into 2-kg and 1-kg test charges.
5. 2 x 1-kg samples were submitted for screened metallics analysis for gold at +/-150 mesh. The +150 mesh fraction was assayed to extinction and duplicate riffled cuts from the minus 150 mesh fraction were also assayed to extinction.
6. An additional 500-g representative sample was submitted for S, $S^{=}$ and ICP scan analysis.

The assay results are shown in Table 1.

1.2. Head Analysis

Screened metallics analyses for gold results are shown in Table 1. Two ~1,000-g tests were completed on the sample. The -150 mesh Au and Ag, g/t "a" and "b" designations refer to the duplicate riffled (~20 to 25-g) cuts from the -150 mesh fraction.

Table 1. Head Analysis, Screened Metallics for Gold

Calculated Head Grade, Au, g/t		+150 Mesh		-150 Mesh			% Au Distribution	
Avg.	Indiv.	% Mass	Au, g/t	% Mass	Au, g/t a	Au, g/t b	+150 Mesh	-150 Mesh
3.60	**3.47**	2.81	7.18	97.2	3.41	3.32	5.8	94.2
	3.73	3.18	11.6	96.8	3.41	3.54	9.9	90.1

Additional head analyses are contained in Table 2.

Table 2. Additional Head Analysis

Element	Assay	Element	Assay
S %	1.52	Mg g/t	10,000
$S^{=}$ %	1.04	Mn g/t	390
Semi-quantitative ICP Scan		Mo g/t	40
Ag g/t	<2	Na g/t	6,500
Al g/t	58,000	Ni g/t	< 20
As g/t	<30	P g/t	700
Ba g/t	710	Pb g/t	<30
Be g/t	0.84	Sb g/t	<10
Bi g/t	< 20	Se g/t	< 30
Ca g/t	32,000	Sn g/t	< 20
Cd g/t	<2	Sr g/t	90
Co g/t	22	Ti g/t	3,700
Cr g/t	56	Tl g/t	< 30
Cu g/t	220	U g/t	< 20
Fe g/t	38,000	V g/t	99
K g/t	26,000	Y g/t	6.1
Li g/t	< 5	Zn g/t	68

1.3. Comminution Testwork

Results from the standard Bond ball mill work index test completed on SMP Comp-1 are given in Table 3. The Saza-Makongolosi result is plotted against the SGS Grinding Specialists database in Figure 1.

Table 3. Bond Ball Mill Grindability Test Results

Feed (F_{80}), µm	Product (P_{80}), µm	Closing screen µm	BWi Imperial	BWi Metric
2,285	113	150	13.6	**15.0**

The detailed results from test represented in Table 3 are contained in the Details of Tests section at the end of this report.

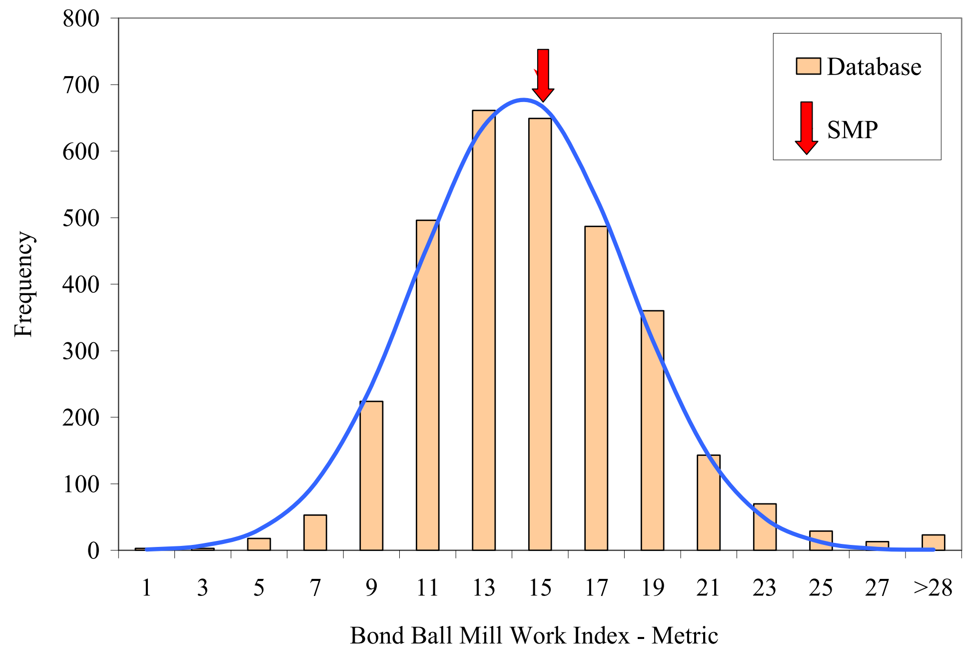

Figure 1. SMP Comp-1 Plotted with SGS Grinding Specialists Database

At 15, the SMP Comp-1 Bond work index falls at the 55th percentile, very near the database average. The ore is therefore considered to be of intermediate hardness in Bond work index terms.

1.4. Mineralogical Evaluation

A representative portion of SMP Comp-1 was submitted for mineralogical evaluation. The standard "rapid mineral scan" examination package was applied. The -10 mesh sample was submitted for polished section preparation and XRD (X-ray diffraction) analysis. Polished sections were examined using an optical microscope for mineral speciation, grain counting and grain size estimation. Based on the XRD results and optical microscopic data the abundance, size range, liberation and association of the major minerals were determined, with particular attention being paid to sulphide species. Photomicrographs were taken to illustrate the mineralogical composition, grain size and liberation data.

The investigation indicated that pyrite was the major sulphide present while minor amounts of chalcopyrite and galena were also noted. The detailed results from the RMS evaluation are contained in Appendix A.

2. Metallurgical Test Program

The metallurgical test program consisted of:

- Conventional (Lakefield type) gravity separation testing of the whole ore (SMP Comp-1) applying a Knelson MD-3 laboratory concentrator and Mozley C-800 Lab Separator,
- Flotation testing of both whole ore and gravity tailing,
- Conventional cyanidation of whole ore and gravity tailing, and
- Cyanide leaching of the flotation concentrate.

2.1. Gravity Separation Testwork

The potential for gold recovery by gravity separation was evaluated at a grind size of ~150 µm (P_{80}). The two gravity separation tests were completed using the standard scoping level program charge mass of 10-kg. A Knelson MD-3 concentrator was used as the primary gravity gold recovery unit. The Knelson concentrate was recovered and further upgraded by treatment on a Mozley mineral separator. Approximately 0.1% mass was targeted as the Mozley concentrate. The gravity concentrate was assayed to extinction for gold.

The Knelson and Mozley tailings were recombined, blended and divided into representative 1-kg (dry equivalent) charges for downstream flotation and cyanidation testwork. Gravity separation results are given in Table 4.

Table 4. Gravity Separation Test Results

Test No.	Feed Size P_{80}, µm	Tests Completed on Gravity Tailing	Product	Mass %	Assays, g/t Au	% Distribution Au
G-1	126	F-1, F-2, F-3, CN-1, CN-2, CN-3, CIL-1	Mozley Concentrate	0.130	972	35.9
			Combined Tailing	99.87	2.26	64.1
			Head (Calculated)	100.0	3.52	100.0
G-2	92	F-7	Mozley Concentrate	0.088	1,228	33.4
			Combined Tailing	99.91	2.15	66.6
			Head (Calculated)	100.0	3.23	100.0
			Head (Direct.)		**3.60**	

Note that Test G-2 was completed primarily for the purpose of generating flotation concentrate (in Test F-7) for subsequent cyanidation testwork. The finer feed size selected for that test (92 µm) was based on indications that flotation gold recovery was maximised at ~100 µm (P_{80}).

In both cases the combined gravity tailings were not assayed directly. The gold assays indicated for the tailings in Table 4 are the average calculated heads from the several tests completed on the combined gravity tailing products.

Gold recovery in both gravity separation tests was quite good ranging between 34 and 36%. It is very likely that the SMP ore process will include a gravity recovery circuit of some sort.

2.2. Flotation Testwork

Flotation testwork was conducted on the gravity separation tailing generated in Tests G-1 and G-2 and on the SMP Comp-1 whole ore.

Three rougher kinetics tests were conducted on the gravity separation tailing generated in Test G-1 in order to evaluate the effect of primary grind size on flotation response. A standard set of bulk sulphide collectors consisting of xanthate (PAX) and a dithiophosphate (Cytec 208) was applied. The conditions indicated in Table 5 were applied in all flotation tests within the scope of this program.

Table 5. Flotation Test Conditions

Stage	Reagents added, g/t			Time, minutes			pH
				Cond.	Froth		
	PAX	R208	MIBC		Ind.	Cum.	
Rougher 1	10	7.5	7.5	1	4	4	8.0
Rougher 2	10	7.5	5	1	4	8	
Rougher 3	10	5	5	1	4	12	
Rougher 4	10	5	5	1	4	16	
Rougher 5	10	5	5	1	4	20	
Rougher 6	10	5	5	1	4	24	8.0
Total	***60***	***35***	***32.5***	***6***	***24***		

Stage	Rougher
Flotation Cell	2000 g D-1
Speed: r.p.m.	1800

Three flotation tests were completed on whole ore at the same grind sizes targets as tested in the gravity tailing flotation testwork.

The gravity tailing flotation results are given in Table 6.

Table 6. Gravity Tailing Flotation Results

Feed =	Flot Test No.	Feed Size, P_{80}, µm	Product (cumulative)	Mass %	Assays, g/t, % Au	Assays, g/t, % $S^=$	% Distribution Au Flot	% Distribution Au Grav + Flot	% Distribution $S^=$
Gravity Tailing (Test G-1)	F-1	126	*Gravity Concentrate*	*0.130*				*35.9*	
			Rougher Conc. 4 min.	7.63	24.5	14.0	79.8	87.1	93.4
			Rougher Conc. 8 min.	10.9	18.7	10.1	87.2	91.8	95.7
			Rougher Conc. 12 min.	13.6	15.3	8.10	89.1	93.0	96.2
			Rougher Conc. 16 min.	17.3	12.2	6.37	90.3	93.8	96.4
			Rougher Conc. 20 min.	20.6	10.3	5.35	91.1	94.3	96.5
			Rougher Conc. 24 min.	22.9	9.35	4.82	91.7	**94.7**	96.6
			Rougher Tail.	77.1	0.25	0.05	8.25		3.37
			Head (calc.)	100.0	2.34	1.14	100.0		100.0
	F-2	75	*Gravity Concentrate*	*0.130*				*35.9*	
			Rougher Conc. 4 min.	7.86	24.6	13.8	85.4	90.6	92.3
			Rougher Conc. 8 min.	12.8	15.8	8.82	89.1	93.0	96.0
			Rougher Conc. 12 min.	17.9	11.5	6.35	91.0	94.2	96.5
			Rougher Conc. 16 min.	22.3	9.32	5.09	92.0	94.9	96.7
			Rougher Conc. 20 min.	25.8	8.12	4.42	92.6	95.3	96.8
			Rougher Conc. 24 min.	28.9	7.27	3.94	93.1	**95.6**	97.0
			Rougher Tail.	71.1	0.22	0.05	6.91		3.02
			Head (calc.)	100.0	2.26	1.18	100.0		100.0
	F-3	60	*Gravity Concentrate*	*0.130*				*35.9*	
			Rougher Conc. 4 min.	6.69	28.5	16.9	85.1	90.5	93.3
			Rougher Conc. 8 min.	11.5	17.3	10.1	88.9	92.9	96.1
			Rougher Conc. 12 min.	15.0	13.5	7.81	90.2	93.7	96.5
			Rougher Conc. 16 min.	18.0	11.4	6.50	91.3	94.4	96.6
			Rougher Conc. 20 min.	22.3	9.21	5.25	91.9	94.8	96.8
			Rougher Conc. 24 min.	25.4	8.16	4.62	92.5	**95.2**	96.9
			Rougher Tail.	74.6	0.23	0.05	7.5		3.08
			Head (calc.)	100.0	2.24	1.21	100.0		100.0
Gravity Tailing (Test G-2)	F-7	92	*Gravity Concentrate*	*0.088*				*33.4*	
			Rougher Conc. 36 min.	12.4	15.7	N/A	90.8	**93.9**	N/A
			Rougher Tail.	87.6	0.23		9.16		
			Head (calc.)	100.0	2.15		100.0		

Whole Ore test results are contained in Table 7. The flotation results from both sets of tests are graphically compared in Figure 2.

Table 7. Whole Ore Flotation Results

Flot Test No.	Feed Size, P_{80}, µm	Product (cumulative)		Mass %	Assays, g/t, %		% Distribution	
					Au	$S^=$	Au	$S^=$
F-5	171	Rougher Conc.	4 min.	5.12	61.7	22.9	85.2	93.2
		Rougher Conc.	8 min.	7.84	42.8	15.4	90.3	96.0
		Rougher Conc.	12 min.	10.5	32.7	11.6	92.1	96.4
		Rougher Conc.	16 min.	12.7	27.2	9.57	93.0	96.5
		Rougher Conc.	20 min.	14.6	23.8	8.32	93.5	96.6
		Rougher Conc.	24 min.	16.2	21.5	7.49	**93.9**	96.7
		Rougher Tail.		83.8	0.27	0.05	6.09	3.33
		Head (calc.)		100.0	3.71	1.26	100.0	100.0
F-6	125	Rougher Conc.	4 min.	5.61	60.4	21.6	88.0	94.2
		Rougher Conc.	8 min.	8.95	39.2	13.9	91.0	96.5
		Rougher Conc.	12 min.	12.5	28.4	10.0	92.0	96.6
		Rougher Conc.	16 min.	15.6	22.9	7.98	92.6	96.7
		Rougher Conc.	20 min.	18.2	19.7	6.83	93.0	96.8
		Rougher Conc.	24 min.	20.6	17.5	6.05	**93.4**	96.9
		Rougher Tail.		79.4	0.32	0.05	6.60	3.09
		Head (calc.)		100.0	3.85	1.29	100.0	100.0
F-4R	100	Rougher Conc.	4 min.	7.37	43.4	16.7	91.0	95.3
		Rougher Conc.	8 min.	10.6	31.0	11.8	93.4	96.5
		Rougher Conc.	12 min.	14.6	22.7	8.56	94.3	96.7
		Rougher Conc.	16 min.	17.9	18.6	6.97	94.8	96.8
		Rougher Conc.	20 min.	21.4	15.6	5.85	95.3	97.0
		Rougher Conc.	24 min.	24.1	14.0	5.21	**95.6**	97.1
		Rougher Tail.		75.9	0.21	0.05	4.43	2.94
		Head (calc.)		100.0	3.52	1.29	100.0	100.0

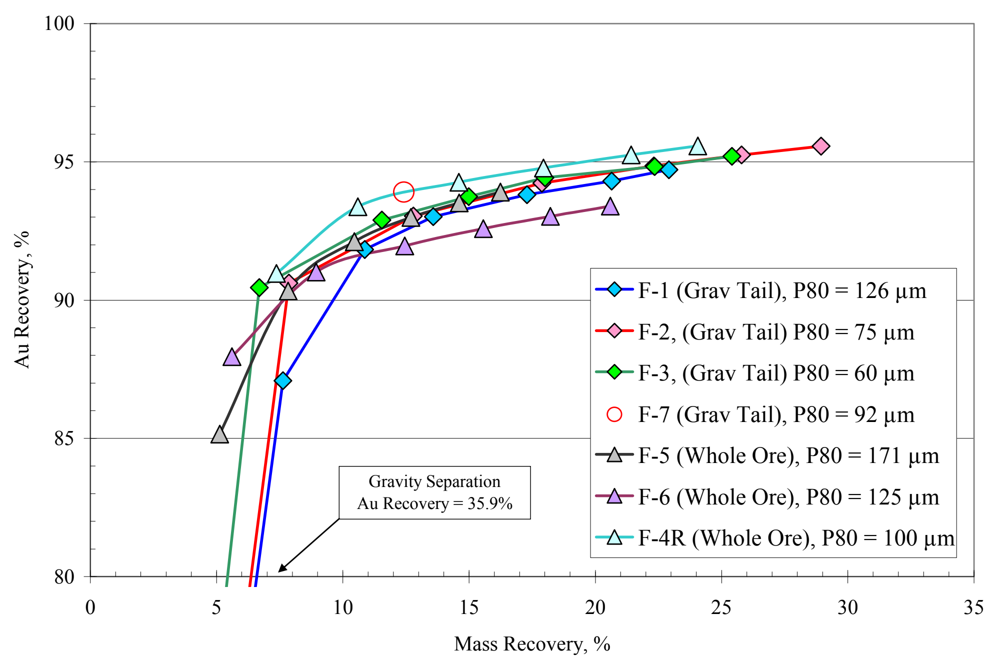

Figure 2. Flotation Results, Gravity Separation Tailing versus Whole Ore

The response to flotation in both test series was excellent, with overall gold recoveries ranging from ~93% to almost 96%. The impact of grind size appeared to be minimal across the size range tested here, although the whole ore test, F-5, completed at a rather coarse 171 µm, appeared to yield somewhat less satisfactory results than the others in the series. It is likely that the metallurgically optimum grind size is finer than 171 µm (P_{80}).

Sulphide recovery was very consistent in all tests. Gold recovery, while obviously tied very closely to sulphide recovery, did vary somewhat with mass pull. The data illustrated in Figure 2 appear to reveal a clear trend indicating that mass pull played a more significant role in gold recovery than grind size.

While the general similarity of the two sets of flotation data (Gravity Tailing versus Whole Ore) may seem to imply that whole ore flotation could be pursued, the high proportion of gravity recoverable gold already identified in the SMP Comp-1 ore clearly indicate that it would be prudent to include gravity separation in the flowsheet designed to process this material.

2.3. Cyanidation Testwork

2.3.1. Gravity Tailing and Whole Ore Testwork

Tests were completed on gravity tailing and whole ore samples to evaluate the effect of grind size. The grind size range evaluated was from ~150 µm to ~75 µm (P_{80}'s). The standard bottle roll test conditions applied were:

Pulp Density	=	40% solids (w/w)
pH	=	10.5 – 11 (maintained with lime)
Cyanide Concentration	=	0.5 g/L NaCN (maintained)
Retention Time	=	48 hours, *with pregnant solution sub-samples submitted for Au analysis at 8, 24 and 48 hours.*

Pulps were preconditioned for 1 hour with injected air (at leach pH) to ensure that dissolved oxygen levels were in the 6-8 mg/L O_2 range.

Applying the same conditions as indicated above, a single carbon-in-leach (CIL) test was completed on a gravity tailing sample. The test was completed at the grind size indicated as optimal in the initial grind series tests. All whole ore and gravity tailing cyanidation test results are presented in Table 8.

Table 8. Gravity Tailing and Whole Ore Cyanidation Results

Feed	Test No.	Feed Size P_{80}, µm	Reag. Consumption kg/t of CN Feed		% Au Extraction/Recovery				Residue Au, g/t	Head (calc) Au, g/t
			NaCN	CaO	6 h	24 h	48 h	O'all Grav + CN		
Test G-1 Gravity Tailing	CN-1	126	0.06	0.43	71	82	**84.3**	***89.9***	0.38	2.39
	CN-2	65	0.08	0.53	83	88	**89.3**	***93.1***	0.24	2.24
	CN-3	59	0.09	0.58	85	90	**91.4**	***94.5***	0.19	2.21
	CIL-1	58	0.45	0.68	--	--	**90.8**	***93.9***	0.20	2.13
Whole Ore	CN-4	96	0.11	0.37	79	84	***86.7***	--	0.36	2.68
	CN-7	63	0.05	0.65	69	88	***91.1***	--	0.34	3.81
	CN-6	58	0.04	0.47	78	89	***92.5***	--	0.26	3.42
	CN-5	52	0.07	0.48	81	90	***91.9***	--	0.24	2.98

Generally speaking, overall gold recoveries were higher in the tests completed on the gravity tailing than in the tests completed on the whole ore. There appeared to be a positive correlation between finer grinding and increased gold extraction (and improved extraction kinetics) in both test series.

There was no additional gold recovery/extraction realised in the single CIL test completed on the gravity tailing (compare Tests CN-3 to CIL-1).

Cyanide consumptions were quite low, ranging from ~0.04 to ~0.11-kg/t in the direct cyanidation tests and ~0.45-kg/t in the single CIL test. The reason for the much higher consumption in Test CIL-1 is not known.

2.3.2. Flotation Concentrate Cyanidation

Two tests were completed on the flotation concentrate generated in Test G-2/F-7 for the purpose of evaluating the impact of regrinding on gold extraction. One test was completed on the flotation concentrate "as-is" and the second, reground to ~13 µm (P_{80}).

Test conditions applied in both cases were as follows:

Pulp Density	=	20% solids (w/w)
pH	=	10.5 -11 (maintained with lime)
Cyanide Concentration	=	20 g/L NaCN (maintained)
Dissolved Oxygen	=	~20 mg/L (maintained with periodic additions of hydrogen peroxide)
Retention Time	=	24 hours, *with pregnant solution sub-samples submitted for Au analysis at 2, 6 and 24 hours.*

At the termination of the tests the pulps were filtered and washed well with fresh water. Filter cakes were submitted for duplicate gold assays and size analysis. Results from these tests are summarised in Table 9.

Table 9. Intensive Cyanidation Testwork, Test F-7 Rougher Concentrate

Test No.	Feed Size P_{80}, µm	Reag. Consumption kg/t of CN Feed		% Au Extraction/Recovery				Residue Au, g/t	Head (calc) Au, g/t
		NaCN	CaO	2 h	6 h	24 h	O'all Grav+ Flot Conc CN		
CN-8	29	4.77	0.50	95	--	**86.2**	***83.6***	2.24	16.2
CN-9	13	10.3	1.19	--	--	**96.4**	***92.0***	0.56	15.3

O’all Au Rec’ry,% = Grav Rec’ry (%) + (100 - Grav Rec’ry (%)) x Ro Flot. Rec’ry (%) x Flot Conc CN Extrac (%)

Based on the leach test unit extractions, there was a definite advantage to regrinding the flotation concentrates prior to cyanidation. Cyanide consumptions, while very high, are fairly typical of this sort of process. If future testwork is undertaken along the same lines (i.e., flotation concentrate cyanidation), we would recommend that significantly lower cyanide levels be tested and that the flowsheet in general, reflect a more conventional concentrate leach approach.

2.4. Overall Metallurgical Results

The metallurgical response of the Saza-Makongolosi (SMP Comp-1) material was quite positive on all fronts evaluated within the scope of this program. The overall (optimum) circuit responses of the ore to the various flowsheets evaluated in the program are compared in Figure 3.

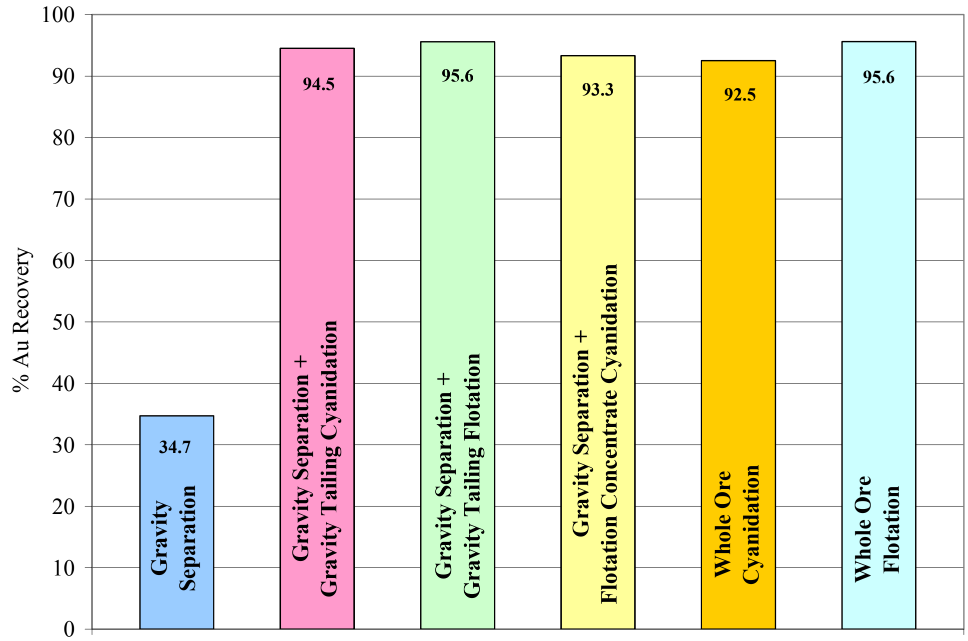

Figure 3. Comparison of Overall Flowsheet Gold Recoveries

Considering the quite successful round of tests completed on the SMP Comp-1 material, further metallurgical testwork is clearly warranted. We recommend that the next steps toward a robust metallurgical process flowsheet should focus on the gravity separation + gravity tailing flotation + flotation concentrate cyanidation. Specific flowsheet parameters that require further investigation are:

- Optimum (or maximum) flotation feed size. The testwork to date indicates that it is likely in the ~100 to ~170 µm range.
- Flotation flowsheet configuration. Given the rather high mass pulls observed in this program (generally >20%) it may be worthwhile investigating a simple flotation cleaner circuit. A brief evaluation of the requirement (or effects) of rougher concentrate regrinding prior to cleaning should be encompassed in the study.
- Conventional flotation concentrate cyanidation protocols should be investigated.

3. Preliminary Environmental Testwork

Samples of final tailing products were subjected to a preliminary environmental evaluation. A sample of Test CN-2 final tailing solids was submitted for acid-base accounting (ABA) and net acid generation (NAG) tests. Final leach solution from the same test was submitted for broad spectrum (ICP) scan analysis. The purpose of these tests was to expose potentially significant environmental issues at an early stage of the Saza-Makongolosi project. Tests results are presented in Tables 10 (ABA), 11 (NAG) and 12 (solution analysis).

Table 10. Acid-Base Accounting Test Results

Parameter		Test CN-2 Final Tailing Solids
Paste pH	units	8.33
Final pH	units	1.6
NP	t $CaCO_3$/1000t	84.8
AP	t $CaCO_3$/1000 t	30.9
Net NP	t $CaCO_3$/1000 t	54
NP/AP	**ratio**	**2.75**
S	%	1.34
$S^=$	%	0.99
SO_4	%	0.35
$C_{(T)}$	%	1.04
CO_3	%	4.49

Table 11. Net Acid Generation Test Results

Parameter		Test CN-2 Final Tailing Solids
Sample	weight (g)	1.48
H_2O_2	mL	150
Final pH	units	10.2
NaOH	Normality	0.1
NaOH to pH = 4.5	mL	0.0
NaOH to pH = 7.0	mL	0.0
NAG (kg H_2SO_4/tonne)	@ **pH = 4.5**	**0.0**
	@ **pH = 7.0**	**0.0**

Generally speaking, samples with NP/AP ratios >3 are considered to be non-acid generating. Samples with NP/AP ratios between 1 and 3 may be acid generating while samples with ratios of <1 are very likely to be acid generating.

Based on the data presented in Tables 10 and 11, it seems unlikely that SMP Comp-1 final tailing solids will generate acid.

Table 12. Final Tailing Solution Analysis

Parameter		Assays Test CN-2 Final Solution	Parameter		Assays Test CN-2 Final Solution
Ag	mg/L	< 0.05	Mo	mg/L	0.094
Al	mg/L	0.47	Na	mg/L	269
As	mg/L	< 0.008	Ni	mg/L	0.48
Ba	mg/L	0.0639	P	mg/L	0.02
Be	mg/L	< 0.0001	Pb	mg/L	0.006
B	mg/L	< 0.009	Sb	mg/L	< 0.01
Bi	mg/L	< 0.03	Se	mg/L	< 0.02
Ca	mg/L	56.9	Si	mg/L	5.92
Cd	mg/L	0.012	Sn	mg/L	< 0.03
Co	mg/L	0.041	Sr	mg/L	0.18
Cr	mg/L	0.001	Ti	mg/L	< 0.001
Cu	mg/L	10.6	Tl	mg/L	< 0.01
Fe	mg/L	0.69	U	mg/L	< 0.6
K	mg/L	17.2	V	mg/L	0.006
Li	mg/L	< 0.002	W	mg/L	< 0.01
Mg	mg/L	0.218	Y	mg/L	< 0.0004
Mn	mg/L	0.002	Zn	mg/L	0.85

Conclusions and Recommendations

The testwork completed on the SMP Comp-1 indicated the following:

Ore Characterisation

- The ore's head grade was 3.6 g/t Au with 1.04% $S^{=}$.
- At 15 (metric), the Bond ball mill work index is considered to be intermediate in terms of grindability.

Metallurgical Testing

- A simple, low mass yield, gravity circuit (Knelson) would likely yield gold recoveries in the 35% range. Full GRG testing would be required to gain an understanding of gold liberation relative to grind size.
- Flotation, at grind sizes ranging from ~170 µm to ~60 µm, gave good gold recovery in the seven tests conducted (gravity tail and whole ore). Gold recovery by gravity separation + rougher flotation ranged from ~93.4% to ~95.6%. Further development of the flotation option, including optimising primary grind size, an analysis of rougher concentrate cleaning and the impact of regrinding on cleaner circuit grade and recovery, is clearly warranted.
- The cyanidation of gravity separation tailing yielded an excellent response with approximately 94.5% of the gold being recovered in the gravity + cyanidation flowsheet at ~59 µm. Additional testwork will be required to elaborate on the effect of grind size on cyanidation gold extraction.
- A comparison of direct cyanidation and carbon-in-leach cyanidation indicated no preg robbing activity.
- The cyanidation of whole ore yielded a good response as well, with ~92.5% of the gold being recovered at P_{80} = 58 µm. Given the relatively high proportion of gravity recoverable gold in this material, we advise that gravity separation should be included in the flowsheet designed for treatment of the SMP Comp-1 ore.

- An intensive cyanidation test completed on flotation concentrate yielded a unit gold extraction of ~96% when the flotation concentrate was reground to 13 µm (P_{80}). Gravity + flotation concentrate cyanidation = ~92% gold extraction. We recommend further testwork to evaluate a more conventional concentrate cyanidation approach.

Environmental

- Testwork completed in this phase of the program indicates very low potential for acid mine drainage.

Details of Tests

Appendix A

Rapid Mineral Scan Report

Appendix E: SGS Metallurgical

May 12, 2009

Mr. Chris MacKenzie
Helio Resource Corp. (BAFEX Tanzania)
UK Branch: Belmayne House
99 Clarkehouse Road
Sheffield, S10 2LN
S. Yorkshire, UK

Re: Saza-Makongolosi Project (Kenge Ore) Heap Leach Amenability Test Results (SGS Project CALR-11940-002)

Mr. MacKenzie:

The following report presents the results from the two coarse ore bottle roll cyanidation tests completed on a BAFEX Tanzania, Saza-Makongolosi Project, Kenge target ore sample. The sample tested had been in storage at SGS Minerals Services (Lakefield) since the completion of a scoping level metallurgical test program conducted in 2008 (11940-001). The results from that program were reported in a document issued on August 7, 2008.

Background and Sample Description

Based on the very positive metallurgical response of the Kenge material in the 2008 scoping program, a request was received for the completion of additional testwork intended to assess the ore's amenability to coarse processing (i.e., heap leaching). Due to the limited availability of appropriately sized ore on hand at SGS (Lakefield), the program was limited to preliminary scoping level tests only. Typically, in programs evaluating heap leaching, even at a cursory level, ore as coarse as 1 inch or coarser would be evaluated. In this case, the testwork was completed on the coarsest ore available, specifically -6 mesh (3.35 mm) and -10 mesh (1.7 mm) material.

A single 1,000-g charge of the -6 mesh SMP Composite was prepared from the Bond ball mill test feed remaining from the previous test program. A 1,000-g charge of -10 mesh SMP Composite was retrieved from storage.

Testwork

Both 1-kg ore charges were subjected to heap leach amenability tests (coarse ore bottle roll cyanidation) applying the following test conditions:

Pulp Density	=	40% solids (w/w)
pH	=	10.5 – 11 (maintained with lime)
Cyanide Concentration	=	0.5 g/L NaCN (maintained)
Retention Time	=	14 days, subsampled for Au assay at the intervals indicated in Table 1.

In order to avoid excessive breakage and attrition of the ore, the leach vessels were not rolled continuously but rather intermittently (1 minute every hour) over the duration of the test. This method has been applied to numerous "greenfield" projects and is a very cost effective method of approximating metallurgy (including reagent requirements) and thereby potentially reducing the need to conduct more costly column scale leach tests.

Solution sub-samples were taken and assayed periodically over the test period. Free cyanide concentration and solution pH were monitored and maintained throughout the test. After 14 days, the pulps were filtered and the cakes washed, dried then crushed to pass 10 mesh (if required) and sampled in duplicate for gold analysis.

Considerable assay variation was noted in the initial paired residue gold assays. Both leach residues were therefore resampled and reassayed for gold in duplicate. The repeat assays did little to validate the residue assay. The assays are listed on the attached test sheets. The assay variation is likely a direct reflection of the relatively high proportion of coarse and/or liberated gold in the ore. Note that the average gravity separation gold recovery observed in the previous test program was ~35%.

Results from the two heap leach amenability tests are presented in Table 1.

Table 1. Heap Leach Amenability Test Results

Test No.	Feed Size P_{80}, mm	Reag. Consumption kg/t of CN Feed		% Au Extraction Days							*	Residue Au, g/t	Head (calc), Au, g,t
		NaCN	CaO	1	2	3	4	7	10	14	*Norm 14 day*		
HL-1	2.16	0.53	0.96	34	41	45	47	56	62	**67.4**	***62.8***	1.54	4.72
HL-2	1.32	0.57	0.96	50	57	60	61	64	66	**66.7**	***71.4***	1.19	3.56

* The normalised 14 day extractions are calculated by comparing the final residue assay to the average head grade (i.e., 4.14 g/t Au)

The kinetic gold extraction curves are given in Figure 1.

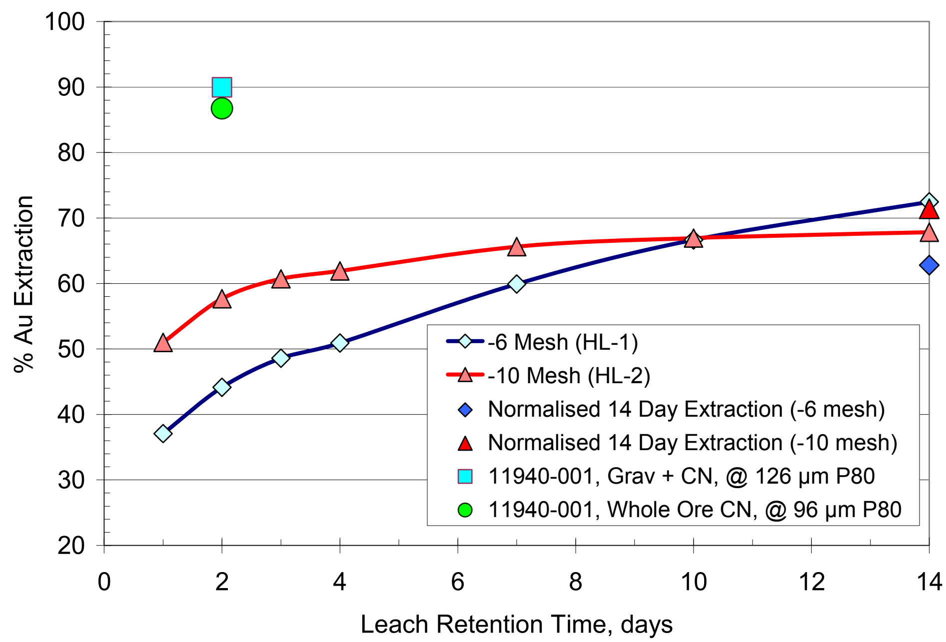

Figure 1. Heap Leach Amenability Gold Extraction Kinetics

The pronounced difference in calculated gold head grades between the two tests makes a direct comparison of gold extractions difficult. Comparing final residue grades however, indicates that as expected, gold recovery was somewhat higher from the finer crushed feed sample (Test HL-2 at -10 mesh). Comparing the final residue grades to the average calculated gold head grade (= 4.14 g/t Au) resulted in the "Normalised 14 Day Gold Extraction" values given in Table 1 and illustrated in Figure 1. Considering the normalised extraction values and the apparent trend toward lower extraction with coarser crushing, it does not appear likely that gold extraction from a heap leach operation, presumably operated at a much coarser crush size, would exceed ~70%. This should certainly be verified with comparative tests at much coarser crush sizes (-1 inch, -¾ inch, -½ inch and -¼ inch).

Conclusions and Recommendations

While gold recoveries approaching 70% by heap leaching may be considered as being quite reasonable in many cases, caution should be exercised in evaluating the results from the tests completed in this program. The ore tested in this case was considerably finer than is usual in current industrial practice. The grade of the Kenge ore (3.5 to 4.7 g/t Au) is certainly much higher than is typical in industrial heap leach operations. The excellent response of this ore to conventional fine grind + gravity separation + gravity tailing cyanidation processes or to whole ore cyanidation (refer to “11940-001, Grav + CN @ 126 µm P_{80}” and “11940-001, Whole Ore CN @ 96 µm P_{80}” in Figure 1) may be cause to contemplate processing this ore in a conventional circuit rather than a heap leach operation.

If the heap leach option will be further evaluated, we recommend that future testwork focus on evaluating coarser crush sizes.

All test details are contained in Appendix A.

Best Regards,

James MacDonald
Senior Metallurgist
SGS Minerals Services (Lakefield)

Appendix F: Porcupine Metallurgical

An Investigation of

THE RECOVERY OF GOLD FROM A SMP PORCUPINE TARGET SAMPLE

prepared for

HELIO RESOURCE CORP/BAFEX TANZANIA LTD.

Project 11940-003 – Final Report
August 28, 2009

NOTE:
This report refers to the samples as received.

Table of Contents

Introduction

This report presents the results from testwork on Helio Resource Corporation's (BAFEX Tanzania) Saza-Makongolosi Project, Porcupine target ore. The project is located in Tanzania. The purpose of the program was to evaluate the processing characteristics of the ore at a scoping level, and to develop a preliminary process flowsheet. The program, similar in scope to the previously completed test program on their Kenge ore, incorporated ore characterization tests (head analysis, mineralogy and comminution tests) as well as the evaluation of a number of processing options, including; gravity separation, flotation and cyanidation.

The test program was directed by Mr. Chris MacKenzie of Helio Resource Corp/BAFEX Tanzania Ltd. Test results were forwarded to Mr. MacKenzie as they became available.

James MacDonald
Project Manager

Inna Dymov, P.Eng
Gold Group Manager

Experimental Work by: *P. Mercer*
Report Prepared by: *R. Huaraz, J. MacDonald*

Testwork Summary

1. Sample Receipt, Preparation and Characterisation

1.1. Sample Receipt and Preparation

A single composite sample representing Porcupine target sample from the Saza-Makongolosi Project (SMP) was received in two plastic crates at SGS Minerals Services (Lakefield) on April 9, 2009. The crates were assigned receipt number 0098-APR09.

The composite sample was processed as follows:

- The contents of the two crates were combined and labelled as SMP Comp 2.
- The sample was crushed to nominally pass one inch. One ~50-kg charge was riffled out and the remainder was stored at -1 inch.
- The ~50-kg charge was crushed to nominally pass 6 mesh (3.35 mm). One ~10-kg charge was riffled out for standard Bond ball mill work index (BWi) @ 100 mesh (150µm).
- The balance of the sample was crushed to nominally pass 10 mesh (1.7 mm).
- The minus 10 mesh sample was rotary split into 2-kg and 1-kg test charges.
- Two representative 1-kg samples were submitted for screened metallics analysis for gold at +/- 150 mesh. The plus 150 mesh fraction was assayed to extinction and duplicate riffled cuts from the minus 150 mesh fraction were also assayed to extinction.
- An additional 500-g representative sample was submitted for S, S= and ICP scan analysis.

The assay results are shown in Table 1.

1.2. Head Analysis

Screened metallics analysis results are shown in Table 1. Two ~1,000-g tests were completed on the sample. The minus 150 mesh Au, g/t "a" and "b" designations refer to the duplicate riffled (~25 to 30-g) cuts from the minus 150 mesh fraction.

Table 1. Head Analysis, Screened Metallics for Gold

Calculated Head Grade, Au, g/t		+150 Mesh		-150 Mesh			% Au Distribution	
Avg.	Indiv.	% Mass	Au, g/t	% Mass	Au, g/t a	Au, g/t b	+150 Mesh	-150 Mesh
2.35	2.39	2.51	10.8	97.5	2.11	2.23	11.4	88.6
	2.32	2.72	6.16	97.3	2.18	2.25	7.2	92.8

Additional head analyses are presented in Table 2.

Table 2. Additional Head Analysis

Element	Assay	Element	Assay
S %	0.50	Mg g/t	3,600
$S^{=}$ %	0.43	Mn g/t	220
Semi-quantitative ICP Scan		Mo g/t	9
Ag g/t	5	Na g/t	18,000
Al g/t	61,000	Ni g/t	< 20
As g/t	<30	P g/t	170
Ba g/t	820	Pb g/t	<30
Be g/t	1.6	Sb g/t	<10
Bi g/t	< 20	Se g/t	< 30
Ca g/t	9,800	Sn g/t	< 20
Cd g/t	<2	Sr g/t	87
Co g/t	<6	Ti g/t	1,800
Cr g/t	47	Tl g/t	< 30
Cu g/t	100	U g/t	< 20
Fe g/t	21,000	V g/t	17
K g/t	34,000	Y g/t	56
Li g/t	<10	Zn g/t	31.1

1.3. Comminution Testwork

Results from the standard Bond ball mill work index test completed on SMP Comp-2 are given in Table 3. The Saza-Makongolosi (Porcupine target) result is plotted against the SGS Grinding Specialists database in Figure 1.

Table 3. Bond Ball Mill Grindability Test Results

Feed (F_{80}), µm	Product (P_{80}), µm	Closing screen µm	BWi	
			Imperial	Metric
2,240	123	150	14.3	**15.7**

With a Bond ball mill work index of 15.7 (metric), the SMP Comp-2 ore falls at the 62nd percentile compared to the database. In terms of ball mill grindability, the material is considered to be moderately hard. Detailed results from this test are presented in the Details of Tests section of this report.

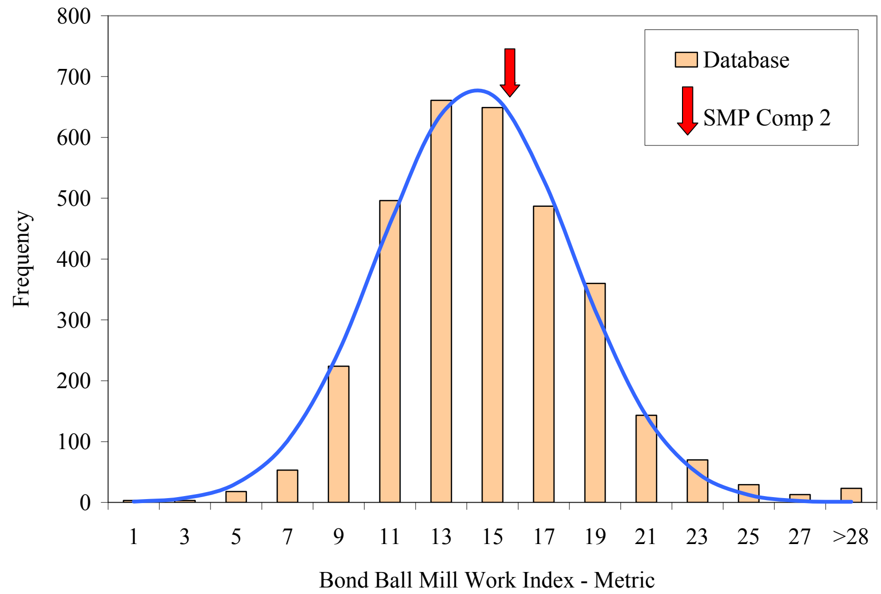

Figure 1. SMP Comp-1 Plotted with SGS Grinding Specialists Database

1.4. Mineralogical Evaluation

A representative portion of SMP Comp-2 was submitted for mineralogical evaluation. The standard "rapid mineral scan" examination package was applied. The -10 mesh sample was submitted for polished section preparation and XRD (X-ray diffraction) analysis. Polished sections were examined using an optical microscope for mineral speciation, grain counting and grain size estimation. Based on the XRD results and optical microscopic data, the abundance, size range, liberation and association of the major minerals were determined, with particular attention being paid to sulphide species. Photomicrographs were taken to illustrate the mineralogical composition, grain size and liberation data.

The investigation indicated that pyrite was the major sulphide present while minor amounts of chalcopyrite, covellite and chalcocite were also noted. The detailed results from the RMS evaluation are contained in Appendix A.

2. Metallurgical Test Program

- The metallurgical test program consisted of:
- Conventional (Lakefield type) gravity separation testing of the whole ore (SMP Comp-2) applying a Knelson MD-3 laboratory concentrator and Mozley C-800 Lab Separator,
- Flotation testing of both whole ore and gravity tailing,
- Conventional cyanidation of whole ore and gravity tailing, and
- Cyanide leaching of the flotation concentrate.

2.1. Gravity Separation Testwork

The potential for gold recovery by gravity separation was evaluated within a grind size range of 105 to 133 µm (P_{80}). The two gravity separation tests were completed using 10-kg of SMP Comp-2. A Knelson MD-3 concentrator was utilised as the primary gravity gold recovery unit. The Knelson concentrate was recovered and further upgraded by treatment on a Mozley mineral separator. Approximately 0.1% mass was targeted as the Mozley concentrate. The gravity concentrate was assayed to extinction for gold.

The Knelson and Mozley tailings were recombined, blended and divided into representative 1-kg (dry equivalent) charges for downstream flotation and cyanidation testwork. Gravity separation results are given in Table 4.

Table 4. Gravity Separation Test Results

Test No.	Feed Size P_{80}, µm	Tests on Grav. Tail.	Product	Mass %	Assays Au, g/t	% Distribution Au
G-1	133	F1-F3 CN4-CN6 and CIL1	Mozley Concentrate	0.090	426	16.8
			Combined Tailing	99.9	1.90	83.2
			Head (Calculated)	100.0	2.28	100.0
G-2	105	F8	Mozley Concentrate	0.159	312	22.0
			Combined Tailing	99.8	1.76	78.0
			Head (Calculated)	100.0	2.25	100.0
			Head (Direct)		**2.35**	

Note that Test G-2 was completed primarily for the purpose of generating flotation concentrate (in Test F-8) for subsequent cyanidation testwork. The finer feed size selected for that test (105 µm) was based on indications that flotation gold recovery was maximised at a slightly finer grind.

In both cases the combined gravity tailing was not assayed directly. The gold assays indicated for the tailings in Table 4 are the average calculated heads from the several tests subsequently completed on the combined gravity tailing product.

Gold recovery in both gravity separation tests was quite good ranging from ~17 to 22%. It is very likely that the SMP ore process will include a gravity recovery circuit of some sort.

2.2. Flotation Testwork

Flotation testwork was conducted on the gravity separation tailing generated in Tests G-1 and G-2 and on the SMP Comp-2 whole ore.

Three rougher kinetics tests were conducted on the gravity separation tailing generated in Test G-1 in order to evaluate the effect of primary grind size on flotation response. A standard set of bulk sulphide collectors, consisting of xanthate (PAX) and a dithiophosphate (Cytec 208) was applied. The conditions indicated in Table 5 were applied in all flotation tests completed within the scope of this program.

Table 5. Flotation Test Conditions

Stage	Reagents added, g/t			Time, minutes	Froth		pH
	PAX	R208	MIBC	Cond.	Ind.	Cum.	
Grind							
Rougher 1	15	10	12.5	2	3	3	8.0
Rougher 2	15	10	10	1	4	7	
Rougher 3	10	5	5	1	4	11	
Rougher 4	10	5	5	1	4	15	
Rougher 5	10	5	5	1	4	19	
Rougher 6	10	5	5	1	4	23	8.0
Total	**70**	**40**	**42.5**	**7**	**23**		

Stage	Rougher
Flotation Cell	2000 g D-1
Speed: r.p.m.	1800

The gravity tailing flotation results are given in Table 6.

Three flotation tests were completed on whole ore at similar grind sizes as tested in the gravity tailing flotation testwork. Whole Ore test results are contained in Table 7. The flotation results from both sets of tests are graphically compared in Figure 2.

The response to flotation in both test series was excellent. Overall gold recoveries (gravity + flotation) ranged from 91.6% at P_{80} = 133 μm (F-1) to ~93% at P_{80} = 111 μm. Finer grinding did not significantly improve gold recovery.

In the whole ore flotation tests, gold recovery ranged from ~92.3 % at P_{80} = 193 in Test F-5 to ~92.8% at P_{80} = 144 μm in Test F-6. Finer grinding, to P_{80} = 61 μm, appears to have resulted in a slight improvement in gold recovery (to 94.8% in Test F-7). In light of the very high calculated gold head grade in that test however, and the similarity in final tailing grades comparing Tests F-6 and F-7, there may in fact be no significant increase in gold recovery with finer grinding.

In the whole ore test, F-4, completed at a rather coarse 256 μm P_{80}, somewhat less satisfactory results were achieved. It is likely that the metallurgically optimum grind size is finer than 133 μm (P_{80}).

Table 6. Gravity Tailing Flotation Results

Feed =	Flot Test No.	Feed Size, P_{80}, µm	Product (cumulative)	Mass %	Assays, g/t, %		% Distribution		
					Au	$S^=$	Au Flot	Au Grav + Flot	$S^=$
Gravity Tailing (Test G-1)	F-1	133	*Gravity Concentrate*	*0.090*				*16.9*	
			Rougher Conc. 3 min.	2.21	61.8	23.0	74.6	78.9	90.6
			Rougher Conc. 7 min.	4.78	32.9	10.8	85.6	88.0	91.5
			Rougher Conc. 11 min.	6.24	25.7	8.26	87.5	89.6	91.7
			Rougher Conc. 15 min.	7.35	22.1	7.02	88.5	90.4	91.8
			Rougher Conc. 19 min.	8.32	19.7	6.21	89.2	91.0	91.8
			Rougher Conc. 23 min.	9.34	17.7	5.54	89.9	**91.6**	91.9
			Rougher Tail.	90.7	0.21	< 0.05	10.1		8.06
			Head (calc.)	100.0	1.84	0.56	100.0		100.0
	F-2	111	*Gravity Concentrate*	*0.090*				*16.9*	
			Rougher Conc. 3 min.	2.31	64.6	20.7	80.3	83.6	89.5
			Rougher Conc. 7 min.	4.91	33.2	9.93	87.8	89.9	91.1
			Rougher Conc. 11 min.	6.64	25.1	7.35	89.5	91.3	91.3
			Rougher Conc. 15 min.	7.93	21.2	6.16	90.4	92.0	91.4
			Rougher Conc. 19 min.	9.28	18.2	5.27	91.0	92.5	91.5
			Rougher Conc. 23 min.	10.3	16.4	4.74	91.3	**92.8**	91.6
			Rougher Tail.	89.7	0.18	< 0.05	8.69		8.39
			Head (calc.)	100.0	1.86	0.53	100.0		100.0
	F-3	89	*Gravity Concentrate*	*0.090*				*16.9*	
			Rougher Conc. 3 min.	2.99	52.6	15.6	83.4	86.2	89.3
			Rougher Conc. 7 min.	5.80	28.8	8.18	88.8	90.7	91.0
			Rougher Conc. 11 min.	8.09	21.0	5.88	90.2	91.9	91.2
			Rougher Conc. 15 min.	9.94	17.2	4.79	91.0	92.5	91.4
			Rougher Conc. 19 min.	11.5	15.0	4.16	91.5	92.9	91.5
			Rougher Conc. 23 min.	12.9	13.4	3.71	91.9	**93.3**	91.6
			Rougher Tail.	87.1	0.18	< 0.05	8.10		8.35
			Head (calc.)	100.0	1.88	0.52	100.0		100.0
Gravity Tailing (Test G-2)	F-8	105	*Gravity Concentrate*	*0.159*				*22.4*	
			Rougher Conc. 36 min.	11.3	14.0	4.05	91.5	**93.4**	91.2
			Rougher Tail.	88.7	0.17	< 0.05	8.49		8.85
			Head (calc.)	100.0	1.72	0.50	100.0		

Sulphide recovery was very consistent in all tests. Gold recovery, while obviously tied very closely to sulphide recovery, did vary somewhat with mass pull. The data illustrated in Figure 2 appear to reveal a clear trend indicating that mass pull played a more significant role in gold recovery than grind size.

Table 7. Whole Ore Flotation Results

Flot Test No.	Feed Size, P_{80}, µm	Product (cumulative)		Mass %	Assays, g/t, %		% Distribution	
					Au	$S^=$	Au	$S^=$
F-4	256	Rougher Conc.	3 min.	2.55	77.0	19.6	80.5	89.8
		Rougher Conc.	7 min.	5.11	41.7	10.0	87.2	91.4
		Rougher Conc.	11 min.	6.69	32.4	7.63	88.7	91.6
		Rougher Conc.	15 min.	8.02	27.2	6.38	89.5	91.7
		Rougher Conc.	19 min.	9.34	23.5	5.48	90.0	91.9
		Rougher Conc.	23 min.	10.4	21.2	4.91	**90.5**	92.0
		Rougher Tail.		89.6	0.26	< 0.05	9.54	8.04
		Head (calc.)		100.0	2.44	0.56	100.0	100.0
F-5	193	Rougher Conc.	3 min.	2.45	80.7	20.7	83.7	90.1
		Rougher Conc.	7 min.	5.42	38.8	9.49	89.2	91.5
		Rougher Conc.	11 min.	7.35	29.1	7.01	90.7	91.8
		Rougher Conc.	15 min.	8.85	24.3	5.83	91.4	91.9
		Rougher Conc.	19 min.	10.1	21.5	5.13	91.8	92.0
		Rougher Conc.	23 min.	11.6	18.8	4.47	**92.3**	92.1
		Rougher Tail.		88.4	0.21	< 0.05	7.69	7.87
		Head (calc.)		100.0	2.36	0.56	100.0	100.0
F-6	144	Rougher Conc.	3 min.	2.69	67.3	18.3	80.1	89.0
		Rougher Conc.	7 min.	5.32	37.6	9.50	88.4	91.3
		Rougher Conc.	11 min.	7.22	28.3	7.02	90.3	91.6
		Rougher Conc.	15 min.	8.92	23.2	5.69	91.5	91.8
		Rougher Conc.	19 min.	10.6	19.7	4.80	92.5	91.9
		Rougher Conc.	23 min.	12.1	17.4	4.22	**92.8**	92.1
		Rougher Tail.		87.9	0.19	< 0.05	7.20	7.95
		Head (calc.)		100.0	2.26	0.55	100.0	100.0
F-7	61	Rougher Conc.	3 min.	3.11	72.3	15.8	86.4	89.9
		Rougher Conc.	7 min.	7.64	31.4	6.56	91.9	91.6
		Rougher Conc.	11 min.	10.53	23.1	4.77	93.3	91.8
		Rougher Conc.	15 min.	12.76	19.2	3.95	93.9	92.0
		Rougher Conc.	19 min.	15.7	15.7	3.22	94.5	92.3
		Rougher Conc.	23 min.	17.2	14.3	2.94	**94.8**	92.4
		Rougher Tail.		82.8	0.17	< 0.05	5.24	7.56
		Head (calc.)		100.0	2.61	0.55	100.0	100.0

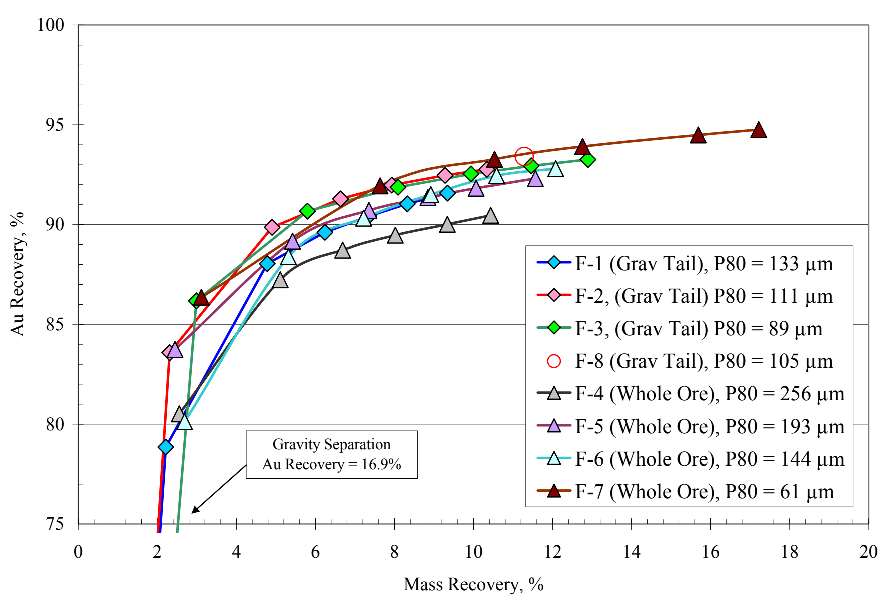

Figure 2. Flotation Results, Gravity Separation Tailing versus Whole Ore

While the general similarity of the two sets of flotation data (Gravity Tailing versus Whole Ore) may seem to imply that whole ore flotation could be pursued, the high proportion of gravity recoverable gold already identified in the SMP Comp-2 ore clearly indicate that it would be prudent to include gravity separation in the flowsheet designed to process this material.

2.3. Cyanidation Testwork

2.3.1. Gravity Tailing and Whole Ore Testwork

Tests were completed on gravity tailing and whole ore samples to evaluate the effect of grind size. The grind size range evaluated was from ~150 µm to ~75 µm (P_{80}'s). The standard bottle roll test conditions applied were:

Pulp Density	=	40% solids (w/w)
pH	=	10.5 – 11 (maintained with lime)
Cyanide Concentration	=	0.5 g/L NaCN (maintained)
Retention Time	=	48 hours, with pregnant solution sub-samples submitted for Au analysis at 8, 24 and 48 hours.

Pulps were preconditioned for 1 hour with injected air (at leach pH) to ensure that dissolved oxygen levels were in the 6-8 mg/L O_2 range.

Applying the same conditions as indicated above, a single carbon-in-leach (CIL) test was completed on a gravity tailing sample. The test was completed at the grind size indicated as optimal in the initial grind series tests. All whole ore and gravity tailing cyanidation test results are presented in Table 8.

Table 8. Gravity Tailing and Whole Ore Cyanidation Results

Feed	Test No.	Feed Size P_{80}, µm	Reag. Consumption kg/t of CN Feed		% Au Extraction/Recovery				Residue Au, g/t	Head (Calc) Au, g/t	
			NaCN	CaO	5 h	24 h	48 h	O'all Grav + CN		CN	O'all Grav + CN
Whole Ore	CN-1	406	0.11	0.28	52.0	67.7	***70.3***	--	0.69	2.31	
	CN-2	294	0.19	0.25	52.9	66.6	***73.6***	--	0.62	2.33	
	CN-3	129	0.61	0.25	42.0	81.1	***86.3***	--	0.40	2.91	
	CN-7	75	0.39	0.28	65.9	88.4	***88.9***	--	0.26	2.29	
Test G1 Gravity Tailing	CN-4	174	0.07	0.21	61.2	73.3	78.4	***82.0***	0.42	1.94	2.28
	CN-5	108	0.26	0.24	62.7	79.9	84.9	***87.5***	0.30	1.99	
	CN-6	79	0.62	0.20	55.2	80.3	86.9	***89.1***	0.26	1.99	
	CIL-1	71	0.19	0.12	--	--	87.0	***89.2***	0.23	1.77	

Generally speaking, overall gold recoveries were slightly higher in the tests completed on the gravity tailing than in the tests completed on the whole ore. There appeared to be a positive correlation between finer grinding and increased gold extraction (and improved extraction kinetics) in both test series.

There was no additional gold recovery/extraction realised in the single CIL test completed on the gravity tailing (compare Tests CN-6 to CIL-1). This indicates that there is no potential preg robbing.

Cyanide consumptions ranged from 0.11 to 0.62-kg/t in the direct cyanidation tests and 0.19-kg/t in the single CIL test. This sample (Porcupine target) appears to consume more cyanide compared with previous testwork completed on SMP Comp 1. Further testwork is required to clarify this relationship.

2.3.2. Flotation Concentrate Cyanidation

Two tests were completed on the flotation concentrate generated in Test G-2/F-8 for the purpose of evaluating the impact of regrinding on gold extraction. One test was completed on the flotation concentrate "as-is" and the second, reground to 12 µm (P_{80}).

Test conditions applied in both cases were as follows:

Pulp Density	=	20% solids (w/w)
pH	=	10.5 -11 (maintained with lime)
Cyanide Concentration	=	20 g/L NaCN (maintained)
Dissolved Oxygen	=	~20 mg/L (maintained with periodic additions of hydrogen peroxide)
Retention Time	=	24h, with pregnant solution sub-samples submitted for Au analysis at 2, 6 and 24 hours.

At the termination of the tests the pulps were filtered and washed well with fresh water. Filter cakes were submitted for duplicate gold assays and size analysis. Results from these tests are summarised in Table 9.

Table 9. Intensive Cyanidation Testwork, Test F-8 Rougher Concentrate

Test No.	Feed Size P_{80}, µm	Reag. Consumption kg/t of CN Feed		% Au Extraction/Recovery				Residue Au, g/t	Head (calc) Au, g/t
		NaCN	CaO	2 h	6 h	24 h	O'all Grav+ Flot		
CN-8	21	44.85	0.06	80	76	**86.2**	***83.6***	1.97	14.3
CN-9	12	25.7	0.67	--	99	**97.8**	***91.9***	0.32	14.3

O'all Au Rec'ry,% = Grav Rec'ry (%) + (100 - Grav Rec'ry (%)) x Ro Flot. Rec'ry (%) x Flot Conc CN Extrac (%)

Based on the leach test unit extractions, there was a definite advantage to regrinding the flotation concentrate prior to cyanidation. Cyanide consumption, while very high, are fairly typical of this sort of process. If future testwork is undertaken along the same lines (i.e., flotation concentrate cyanidation), we would recommend that significantly lower cyanide levels be tested and that the flowsheet in general, reflect a more conventional concentrate leach approach.

2.4. Overall Metallurgical Results

The metallurgical response of the SMP Comp-2 (Porcupine target) material was quite positive on all fronts evaluated within the scope of this program. The overall (optimum) circuit responses of the ore to the various flowsheets evaluated in the program are compared in Figure 3.

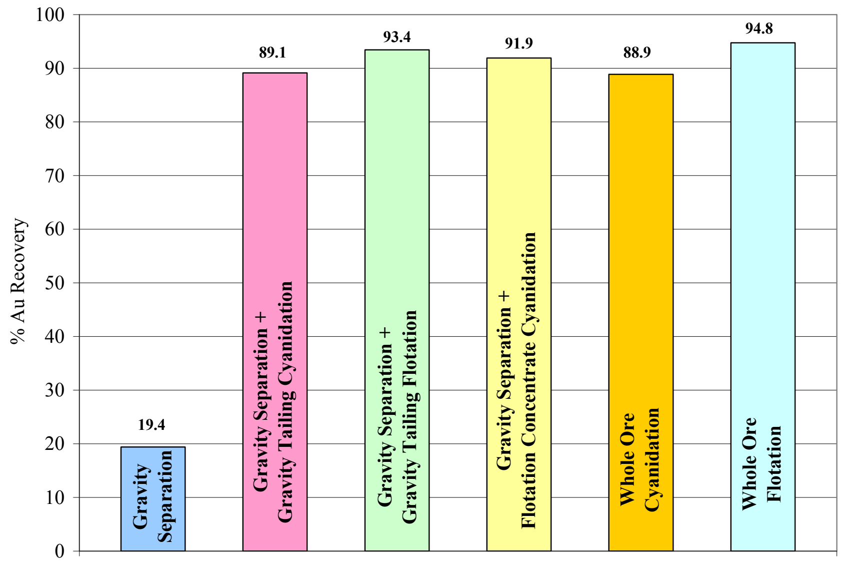

Figure 3. Comparison of Overall Flowsheet Gold Recoveries

Considering this quite successful round of tests completed on the SMP Comp-2 material, further metallurgical testwork is clearly warranted. We recommend that the next steps toward a robust metallurgical process flowsheet should focus on the gravity separation + gravity tailing flotation + flotation concentrate cyanidation flowsheet. Specific flowsheet parameters that require further investigation are:

- Optimum (or maximum) flotation feed size. The testwork to date indicates that it is likely in the ~75 to ~100 µm range.
- Flotation flowsheet configuration. Given the rather high mass pulls observed in this program (in the ~10 to ~ 17% range) it may be worthwhile investigating a simple flotation cleaner circuit. A brief evaluation of the requirement (or effects) of rougher concentrate regrinding prior to cleaning should be encompassed in the study.
- Conventional flotation concentrate cyanidation protocols should be investigated.

3. Preliminary Environmental Testwork

Samples of final tailing products were subjected to a preliminary environmental evaluation. A sample of Test CN-6 final tailing solids was submitted for acid-base accounting (ABA) and net acid generation

(NAG) tests. Final leach solution from the same test was submitted for broad spectrum (ICP) scan analysis. The purpose of these tests was to expose potentially significant environmental issues at an early stage of the Saza-Makongolosi project. Tests results are presented in Tables 10 (ABA), 11 (NAG) and 12 (solution analysis).

Table 10. Acid-Base Accounting Test Results

Parameter		Test CN-6 Final Tailing Solids
Paste pH	units	8.94
Final pH	units	1.62
NP	t $CaCO_3$/1000t	29.0
AP	t $CaCO_3$/1000 t	12.6
Net NP	t $CaCO_3$/1000 t	16.4
NP/AP	**ratio**	**2.3**
S	%	0.53
$S^=$	%	0.40
SO_4	%	0.13
$C_{(T)}$	%	0.45
CO_3	%	1.50

Table 11. Net Acid Generation Test Results

Parameter		Test CN-6 Final Tailing Solids
Sample	weight (g)	1.51
H_2O_2	mL	150
Final pH	units	7.4
NaOH	Normality	0.1
NaOH to pH = 4.5	mL	0.0
NaOH to pH = 7.0	mL	0.0
NAG (kg H_2SO_4/tonne)	@ **pH = 4.5**	**0.0**
	@ **pH = 7.0**	**0.0**

Generally speaking, samples with NP/AP ratios >3 are considered to be non-acid generating. Samples with NP/AP ratios between 1 and 3 may be acid generating while samples with ratios of <1 are very likely to be acid generating.

Based on the data presented in Tables 10 and 11, it seems unlikely that SMP Comp-1 final tailing solids will generate acid.

Table 12. Final Tailing Solution Analysis

Parameter		Assays Test CN-6 Final Solution
Ag	mg/L	2.4
Al	mg/L	0.6
As	mg/L	< 0.3
Ba	mg/L	0.1
Be	mg/L	< 0.002
Bi	mg/L	< 0.02
Ca	mg/L	12
Cd	mg/L	< 0.005
Co	mg/L	0.03
Cr	mg/L	< 0.1
Cu	mg/L	4.8
Fe	mg/L	85
K	mg/L	13
Li	mg/L	< 0.2
Mg	mg/L	0.07
Mn	mg/L	< 0.04

Parameter		Assays Test CN-6 Final Solution
Mo	mg/L	0.1
Na	mg/L	370
Ni	mg/L	0.6
P	mg/L	< 5
Pb	mg/L	< 0.01
Sb	mg/L	< 0.02
Se	mg/L	< 0.3
Sn	mg/L	< 0.05
Sr	mg/L	0.07
Ti	mg/L	< 0.02
Tl	mg/L	< 0.01
U	mg/L	< 0.01
V	mg/L	< 0.2
W	mg/L	< 0.01
Y	mg/L	< 0.005
Zn	mg/L	0.6

Conclusions and Recommendations

The testwork completed on the SMP Comp-2 (Porcupine target) ore indicated the following:

Ore Characterisation

- The ore's head grade was 2.35 g/t Au with 0.43% $S^{=}$.
- At 15.7 (metric), the Bond ball mill work index is considered to be moderately hard in terms of grindability.

Metallurgical Testing

- A simple, low mass yield, gravity circuit (Knelson) would likely yield gold recoveries in the 20% range. Full GRG testing would be required to gain an understanding of gold liberation relative to grind size.
- Flotation, at grind sizes ranging from ~193 µm to ~60 µm, gave good gold recovery in the seven tests conducted (on gravity tailing and whole ore). Gold recovery by gravity separation + rougher flotation ranged from ~92.3% to ~94.8%. Further development of the flotation option, including optimising primary grind size, an analysis of rougher concentrate cleaning and the impact of regrinding on cleaner circuit grade and recovery, is clearly warranted.
- The cyanidation of gravity separation tailing yielded a good response with approximately 89.1% of the gold being recovered in the gravity + cyanidation flowsheet at ~79 µm. Additional testwork will be required to elaborate on the effect of grind size on cyanidation gold extraction.
- A comparison of direct cyanidation and carbon-in-leach cyanidation indicated no preg robbing activity.
- The cyanidation of whole ore yielded a good response as well, with 88.9% of the gold being recovered (extracted) at P_{80} = 75 µm. Given the relatively high proportion of gravity recoverable gold in this material, we advise that gravity separation should be included in the flowsheet designed for treatment of the SMP Comp-1 ore.
- An intensive cyanidation test completed on flotation concentrate yielded a unit gold extraction of 97.8% when the flotation concentrate was reground to 12 µm (P_{80}). Gravity + flotation concentrate cyanidation = 91.9% gold extraction. We recommend further testwork to evaluate a more conventional concentrate cyanidation approach.

Environmental

- Testwork completed in this phase of the program indicates very low potential for acid mine drainage.

Details of Tests

Appendix A

Rapid Mineral Scan Report

Appendix G: Base Statistics and / or Variograms

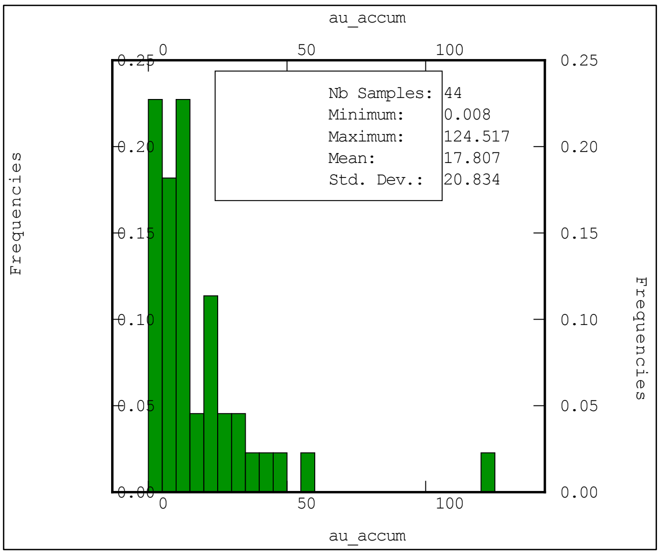

Figure G1: Histogram of accumulation values for composites from Kenge Footwall domain

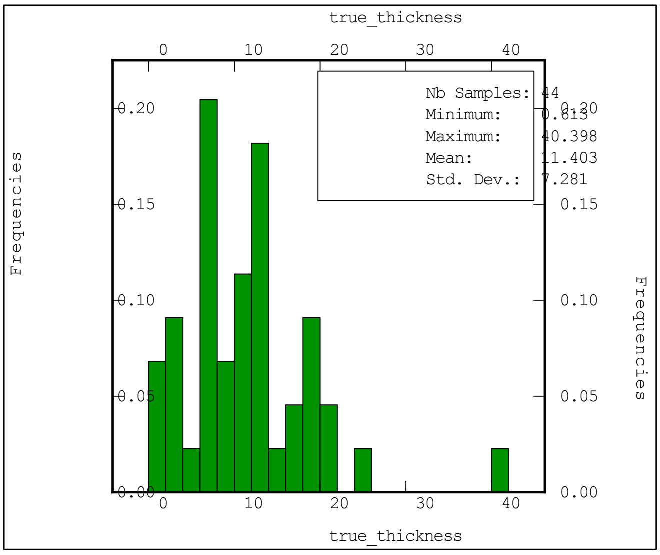

Figure G2: Histogram of true thicknesses for composites from Kenge Footwall domain

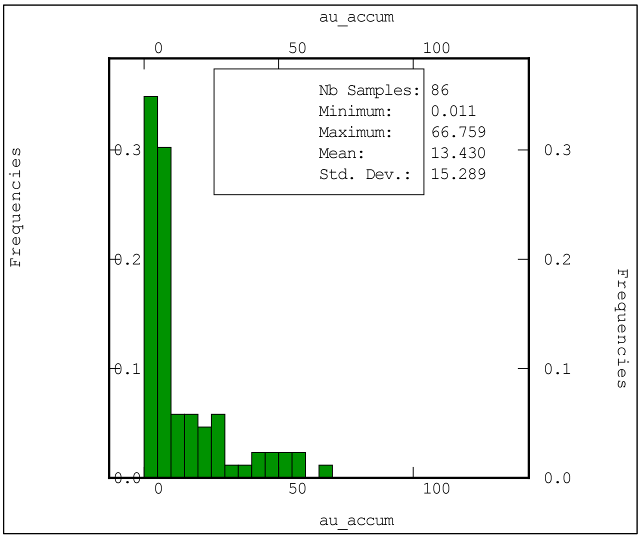

Figure G3: Histogram of accumulation values for composites from Kenge Hanging Wall domain

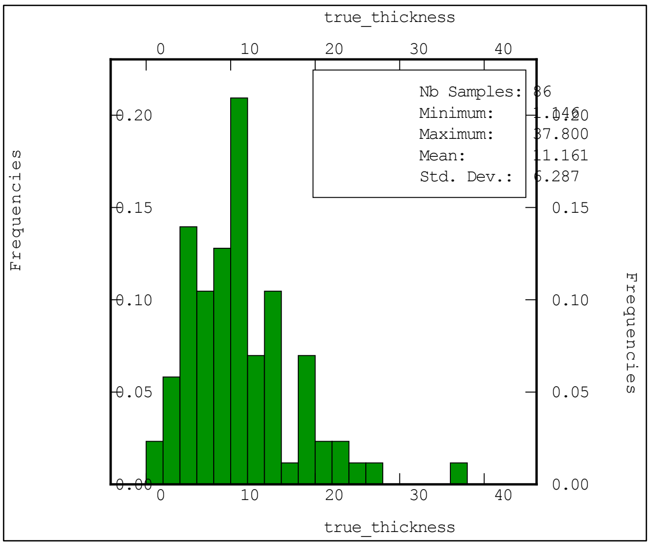

Figure G4: Histogram of true thicknesses for composites from Kenge Hanging Wall domain

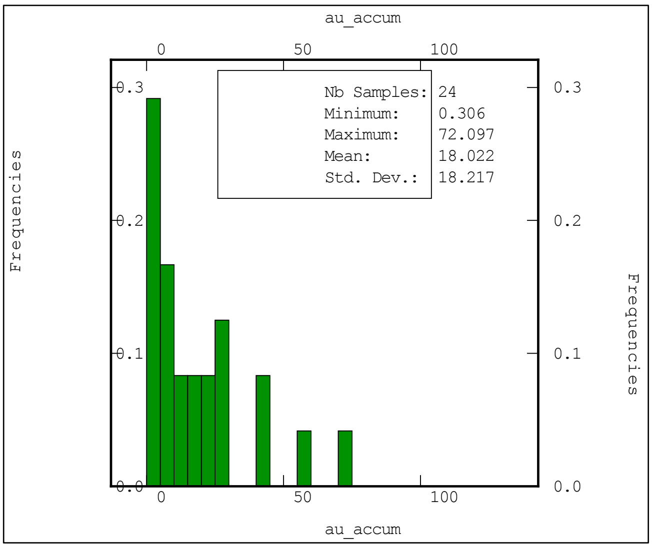

Figure G5: Histogram of accumulation values for composites from Kenge Southeast domain

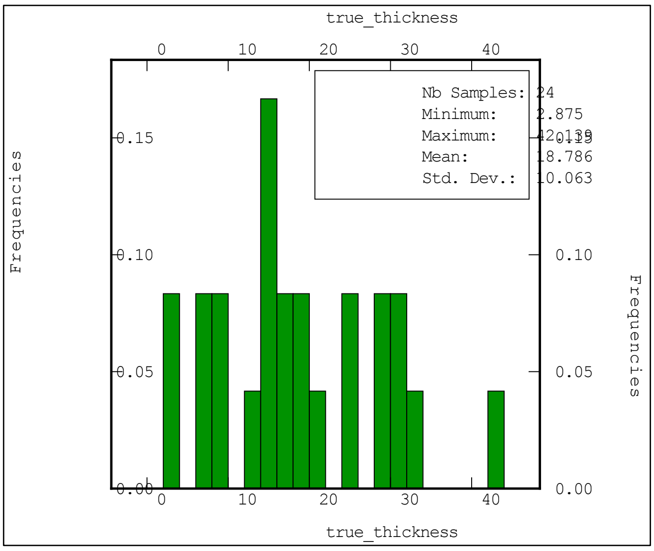

Figure G6: Histogram of true thicknesses for composites from Kenge Southeast domain

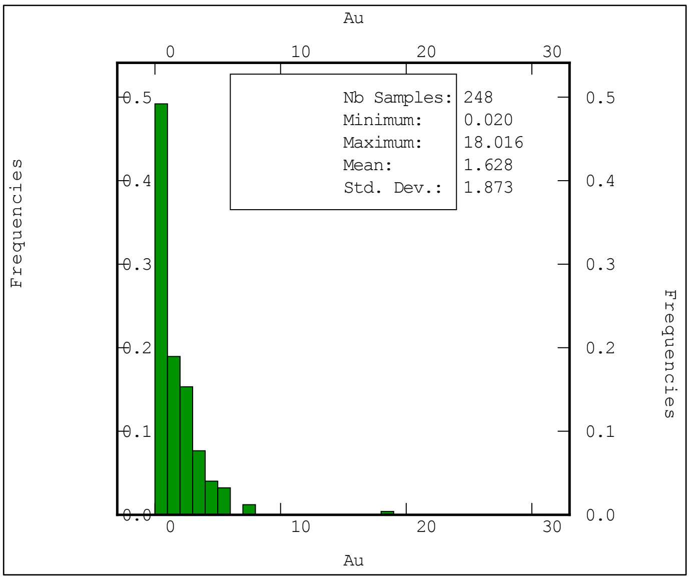

Figure G7: Histogram of composite grades from Mbenge domain 201

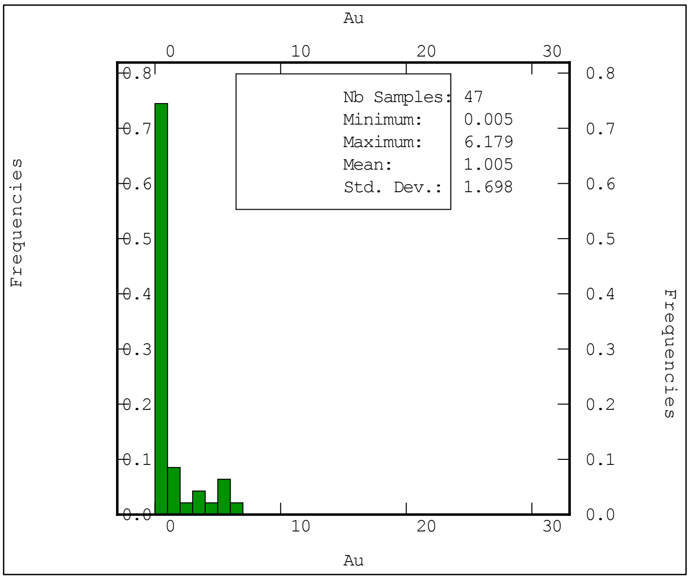

Figure G8: Histogram of composite grades from Mbenge domain 202

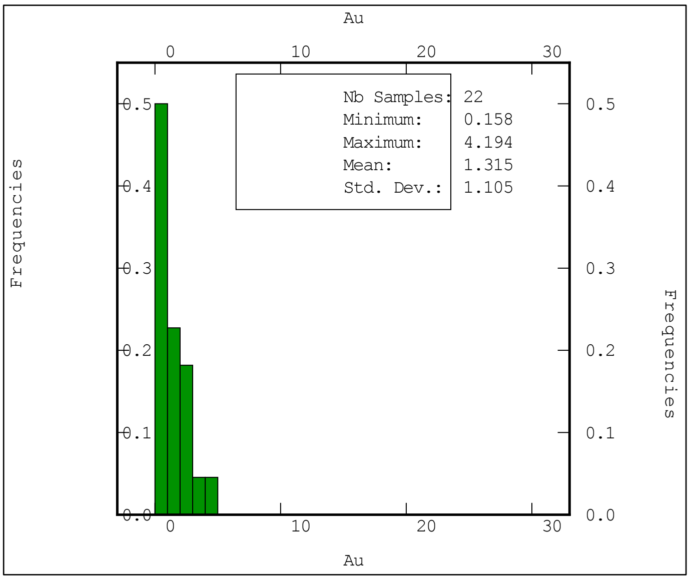

Figure G9: Histogram of composite grades from Mbenge domain 203

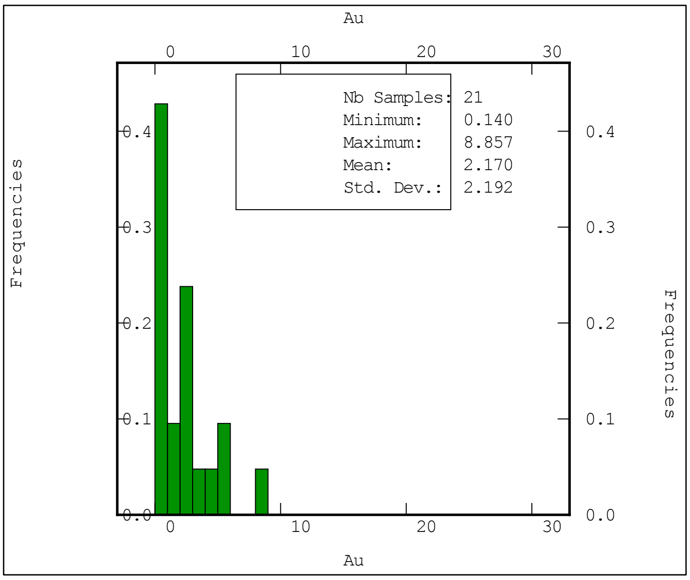

Figure G10: Histogram of composite grades from Mbenge South domain 213

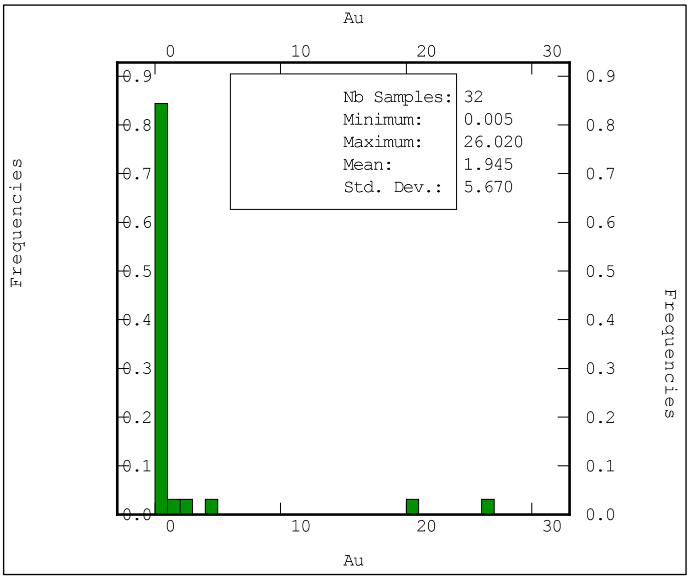

Figure G11: Histogram of composite grades from Mbenge South domain 215

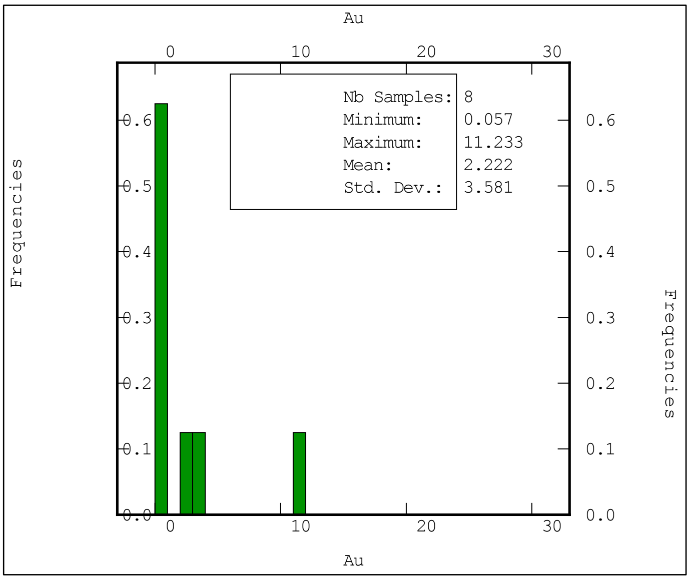

Figure G12: Histogram of composite grades from Mbenge South domain 216

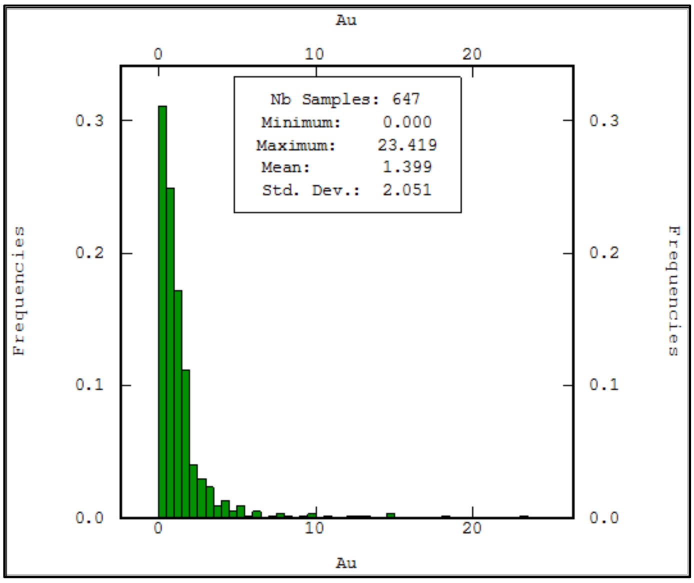

Figure G13: Histogram of composite grades from Porcupine Main domain

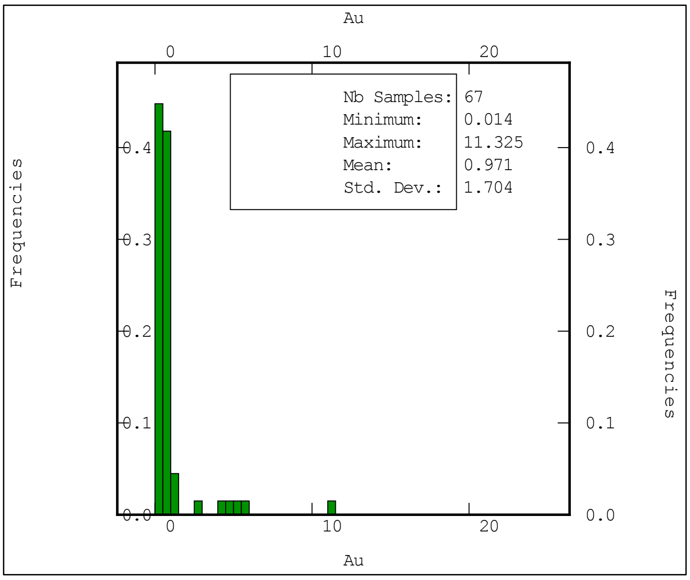

Figure G14: Histogram of composite grades from Porcupine Quill domain

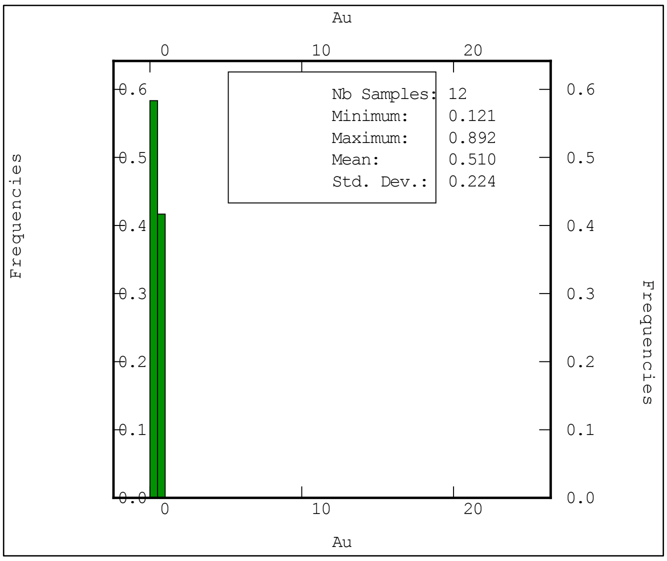

Figure G15: Histogram of composite grades from Porcupine Northwest domain 321

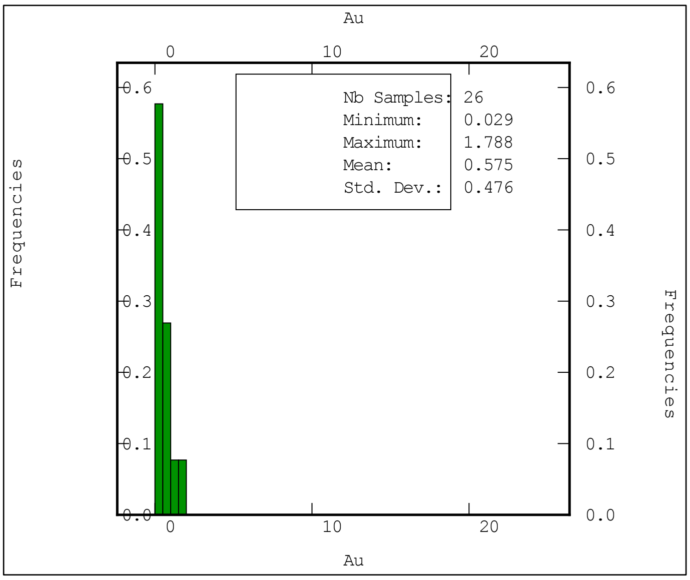

Figure G16: Histogram of composite grades from Porcupine Northwest domain 322

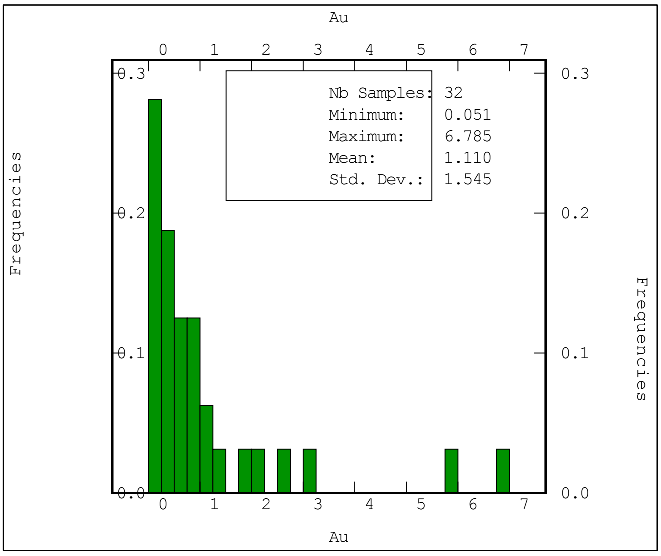

Figure G17: Histogram of composite grades from Konokono

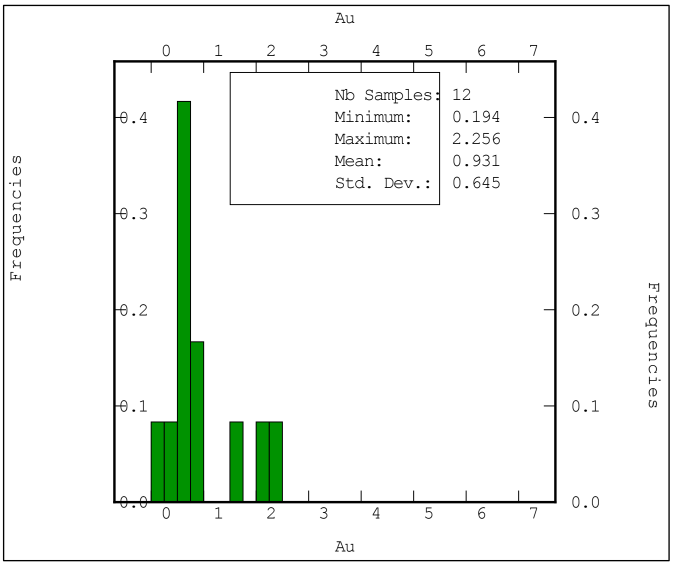

Figure G18: Histogram of composite grades from Tumbili

Appendix H: Analytical Results For SRK Verification Samples

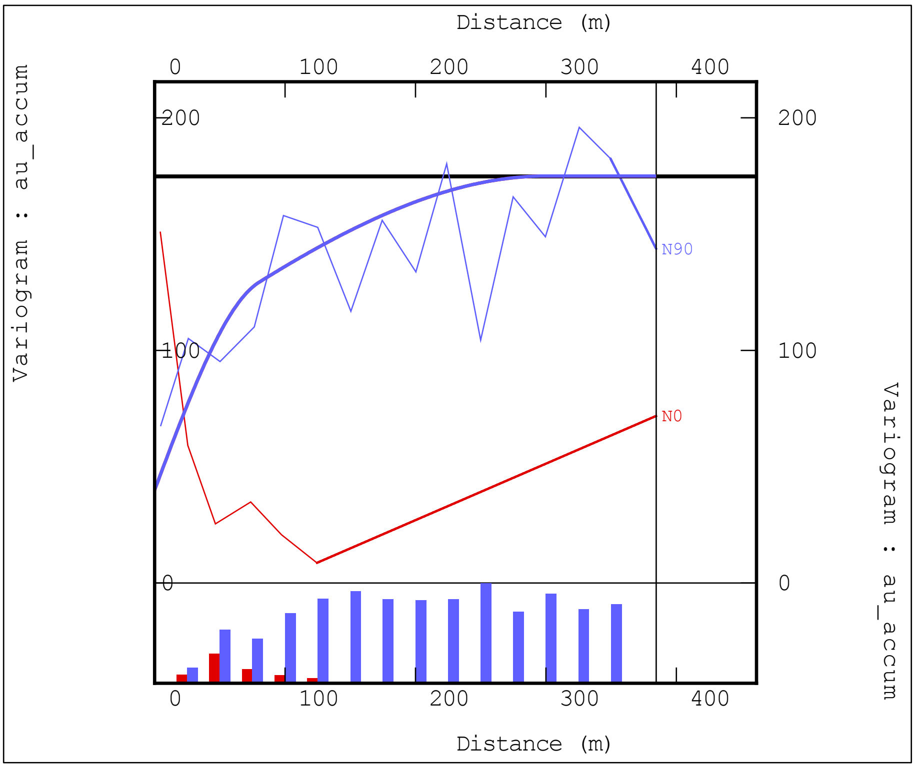

Figure H1: Experimental and modelled variograms for Kenge gold accumulation

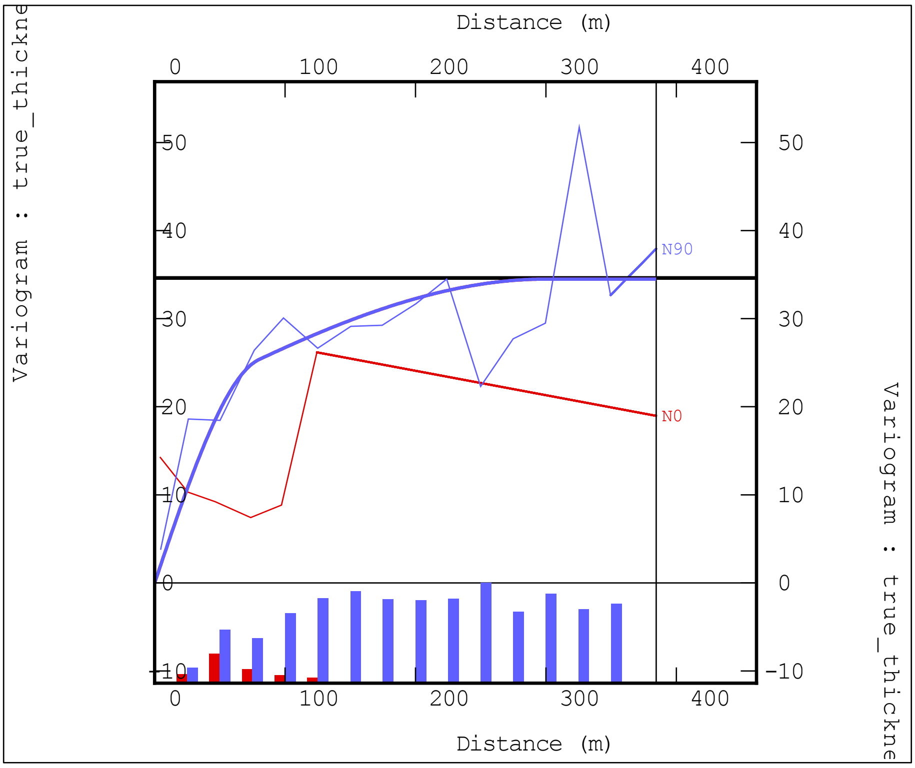

Figure H2: Experimental and modelled variograms for Kenge true thickness

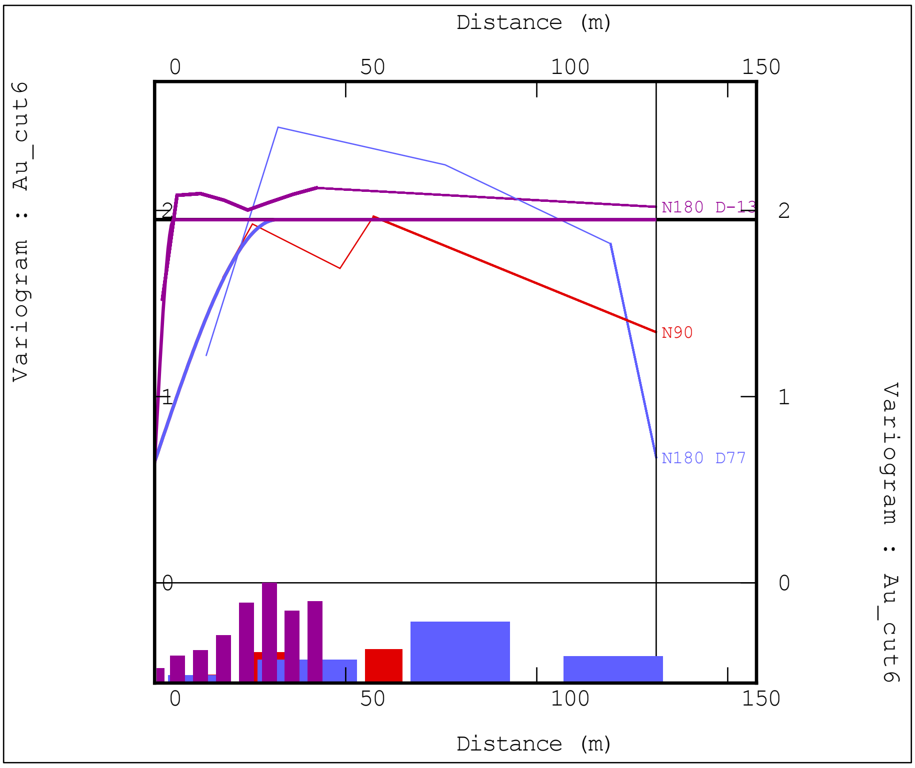

Figure H3: Experimental and modelled variograms for Mbenge Au grade

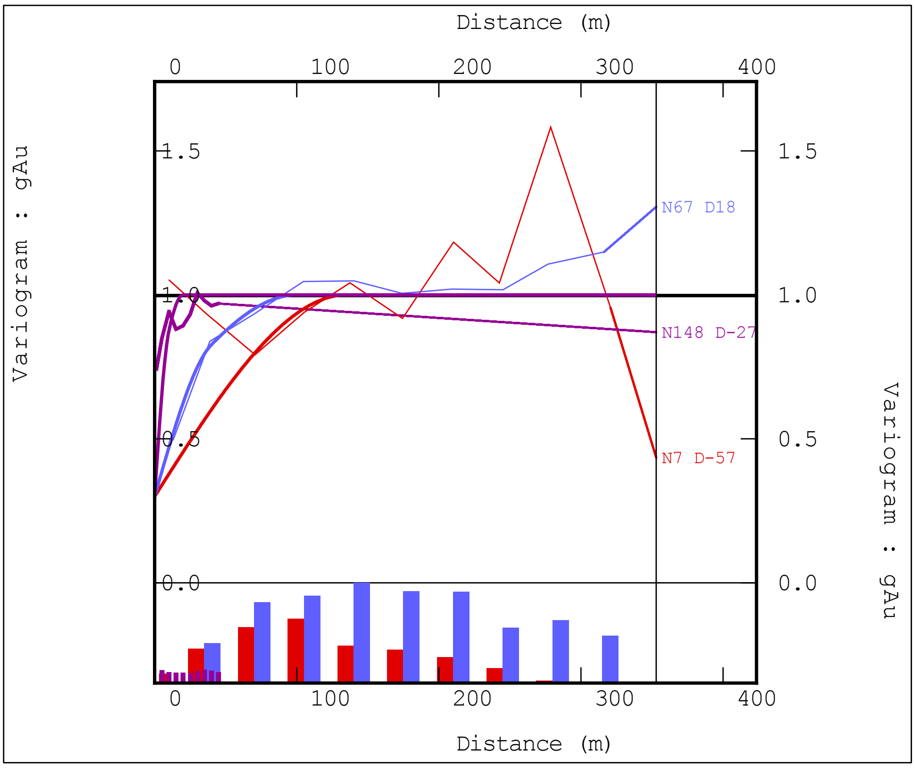

Figure H4: Experimental and modelled variograms for Porcupine Main Au grade (Gaussian space)

Appendix I: Block Model Validation

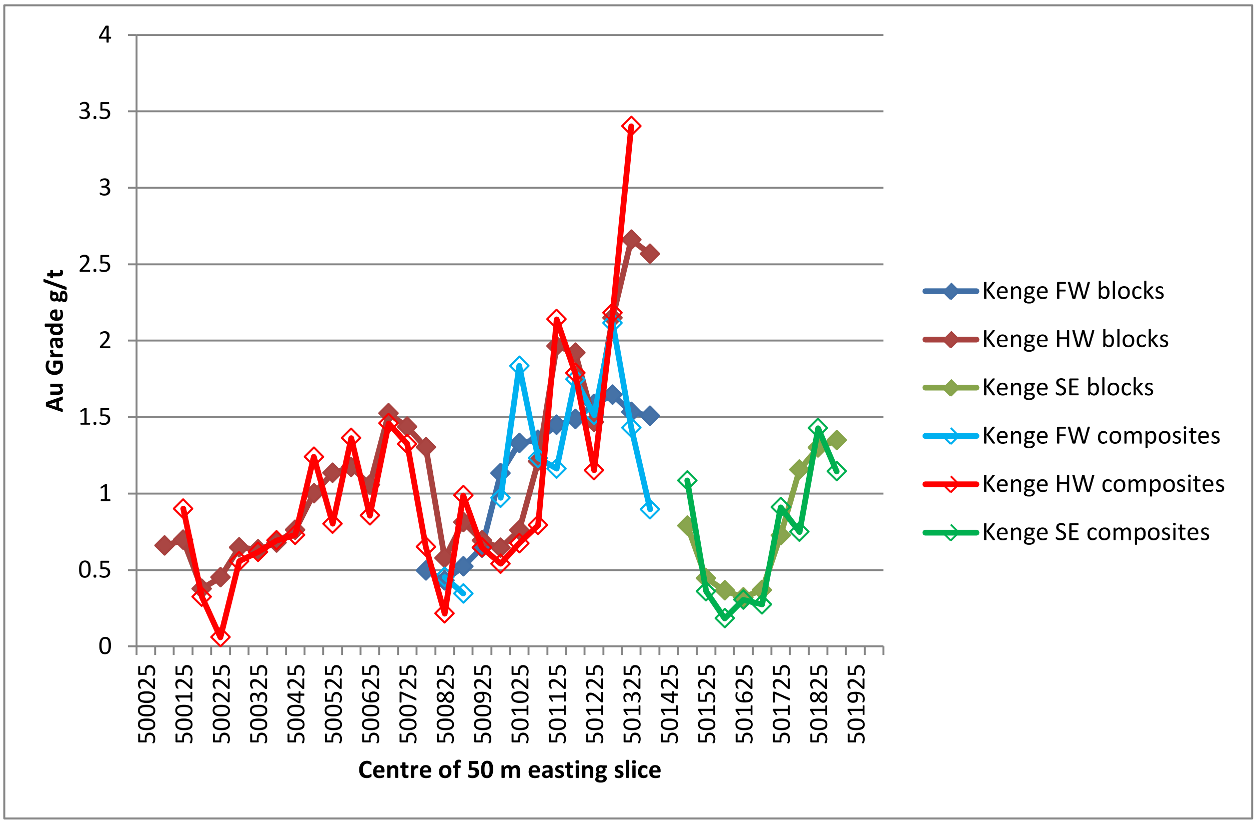

Figure I1: Swath plots of mean block and composite grades within 50 m thick easting slices through Kenge

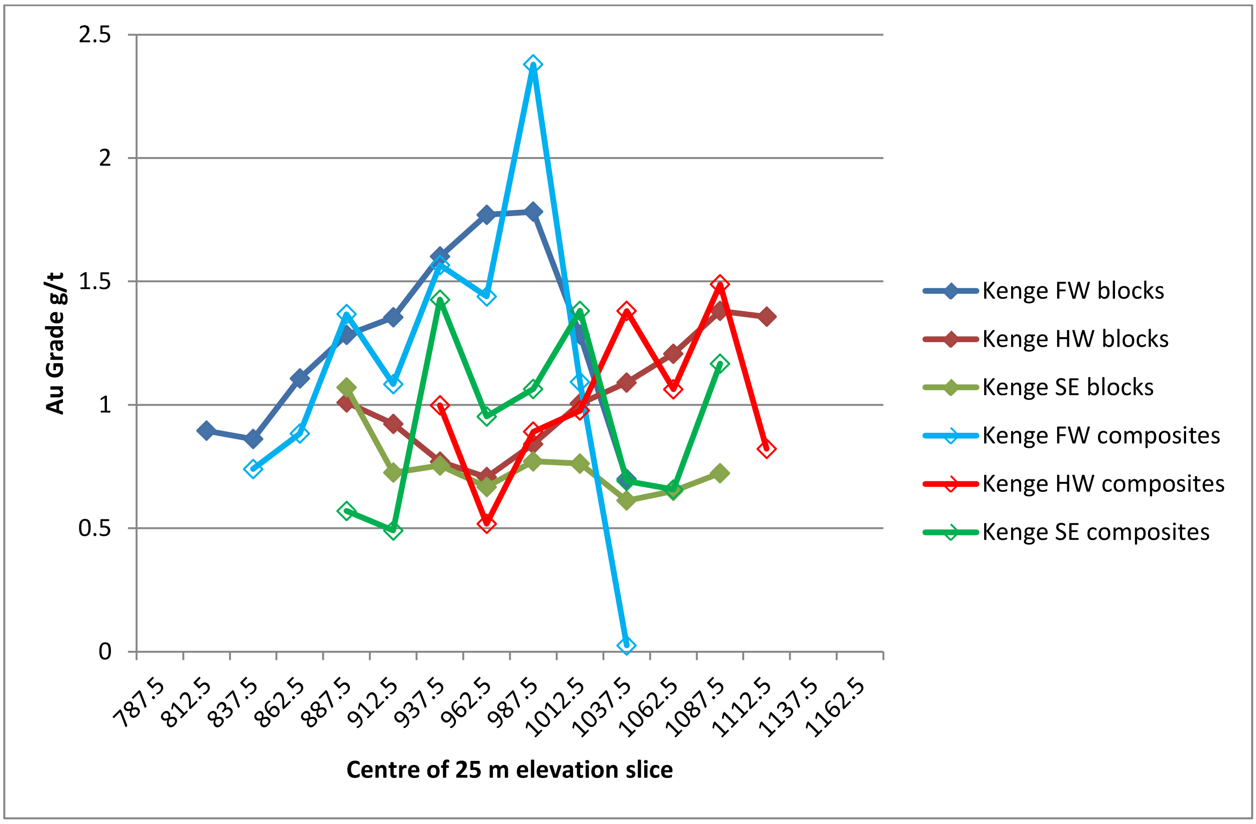

Figure I2: Swath plots of mean block and composite grades within 25 m thick elevation slices through Kenge

Figure I3: **Swath plots of mean block and composite grades within 50 m thick easting slices through Porcupine**

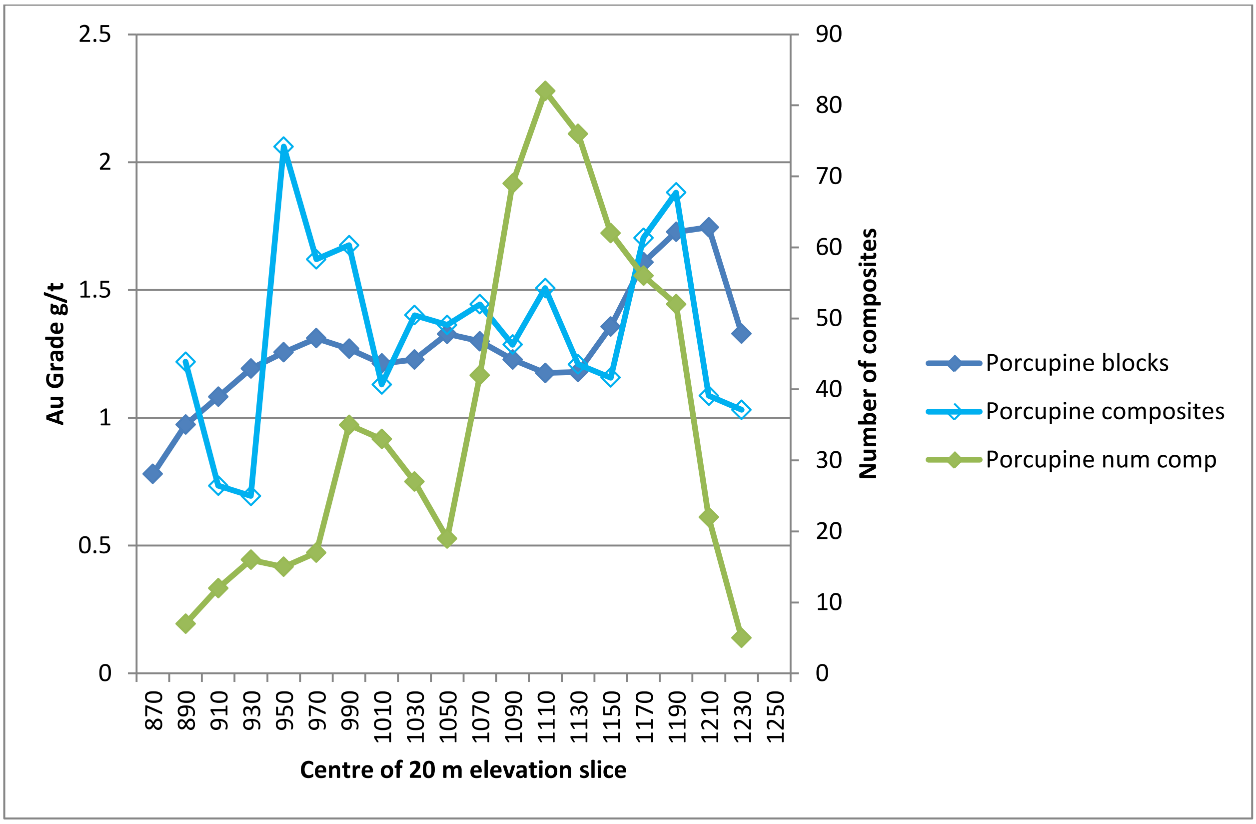

Figure I4: **Swath plots of mean block and composite grades within 20 m thick easting slices through Porcupine**

Appendix J: Results from the Previous Mineral Resource Estimate, Issued 30 November 2010 (Harrison, 2011)

Table J1: 30 November 2010 Mineral Resource Estimate for all SMP deposits combined

Cut-off	Class	Mean Au grade (g/t)[1]	Tonnage (Mt)[2]	Metal Au (koz)[3]
0.3	Measured	1.35	6.4	280
0.3	Indicated	1.32	5.4	230
0.3	Measured+Indicated	1.34	11.8	510
0.3	Inferred	0.98	9.7	310
0.5	Measured	1.45	5.9	270
0.5	Indicated	1.41	4.9	220
0.5	Measured+Indicated	1.43	10.8	500
0.5	Inferred	1.19	7.1	270
0.7	Measured	1.58	5.1	260
0.7	Indicated	1.52	4.2	210
0.7	Measured+Indicated	1.55	9.3	470
0.7	Inferred	1.40	5.2	240
0.9	Measured	1.73	4.2	240
0.9	Indicated	1.67	3.5	190
0.9	Measured+Indicated	1.70	7.8	430
0.9	Inferred	1.55	4.2	210

Table J2: 30 November 2010 Mineral Resource Estimate for Porcupine domains

Cut-off	Class	Mean Au grade (g/t)[1]	Tonnage (Mt)[2]	Metal Au (koz)[3]
0.3	Measured	1.35	6.4	280
0.3	Indicated	1.18	2.0	80
0.3	Measured+Indicated	1.31	8.5	360
0.3	Inferred	0.87	4.9	120
0.5	Measured	1.45	5.9	270
0.5	Indicated	1.27	1.8	70
0.5	Measured+Indicated	1.41	7.7	350
0.5	Inferred	1.17	3.0	110
0.7	Measured	1.58	5.1	260
0.7	Indicated	1.39	1.5	70
0.7	Measured+Indicated	1.54	6.6	260
0.7	Inferred	1.43	2.0	90
0.9	Measured	1.73	4.2	240
0.9	Indicated	1.58	1.2	60
0.9	Measured+Indicated	1.70	5.4	300
0.9	Inferred	1.57	1.7	90

Table J3: 30 November 2010 Mineral Resource Estimate for Kenge and Mbenge domains

Cut-off	Class	Mean Au grade (g/t)[1]	Tonnage (Mt)[2]	Metal Au (koz)[3]
0.3	Indicated	1.40	3.4	140
0.3	Inferred	1.10	4.8	150
0.5	Indicated	1.49	3.1	150
0.5	Inferred	1.21	4.1	160
0.7	Indicated	1.60	2.7	140
0.7	Inferred	1.39	3.1	140
0.9	Indicated	1.59	2.3	130
0.9	Inferred	1.54	2.5	110

1: Rounded to two decimal places

2: Rounded to nearest 0.1 Mt

3: Rounded to nearest 10 koz