NI 43-101 TECHNICAL REPORT
MINERAL RESOURCE AND RESERVE UPDATE
DECEMBER 31, 2014
TWANGIZA GOLD MINE
DEMOCRATIC REPUBLIC OF THE CONGO

Prepared For

TWANGIZA MINING SA,

A subsidiary of Banro Corporation

Effective Date of Report: July 29, 2015

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Report Prepared by

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UK6391

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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 date of this report, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report.

This report contains forward-looking statements which address activities, events or developments which are believed, expected or anticipated will or may occur in the future (including, without limitation, statements regarding estimates and/or assumptions in respect of net present values and future cash flows, Mineral Resource and Mineral Reserve estimates, future gold production, costs, mine life, gold recoveries, potential Mineral Resources and Mineral Reserves and production, development and exploration plans and objectives).

These forward-looking statements reflect current expectations based on information currently available and are subject to a number of risks and uncertainties that may cause the actual results to differ materially from those discussed in the forward-looking statements; these include, among other things: uncertainty of estimates of capital and operating costs, production estimates and estimated economic return; the possibility that actual circumstances will differ from the estimates and assumptions used in economic studies; failure to establish estimated Mineral Resources and Mineral Reserves; fluctuations in gold prices and currency exchange rates; inflation; gold recoveries being less than those indicated by the metallurgical testwork carried out to date; uncertainties relating to the availability and costs of financing needed in the future; changes in equity markets; political developments in the DRC; lack of infrastructure; failure to procure or maintain, or delays in procuring or maintaining, permits and approvals; lack of availability at a reasonable cost or at all, of plants, equipment or labour; inability to attract and retain key management and personnel; changes to regulations affecting mining activities; the uncertainties involved in interpreting drilling results and other geological data.

Although it is believed that the assumptions inherent in the forward-looking statements are reasonable, forward-looking statements are not guarantees of future performance and accordingly undue reliance should not be put on such statements due to the inherent uncertainty therein.

The Mineral Resource and Mineral Reserve figures referred to in this report are estimates and no assurances can be given that the forecast levels of gold will be produced. Such estimates are expressions of judgment based on knowledge, mining experience, analysis of drilling results and industry practices. Valid estimates made at a given time may significantly change when new information becomes available.

While it is believed that the Mineral Resource and Mineral Reserve estimates included in this report are well established, by their nature Mineral Resource and Mineral Reserve estimates are imprecise and depend, to a certain extent, upon statistical inferences which may ultimately prove unreliable. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. There is no certainty that Mineral Resources can be upgraded to Mineral Reserves through continued exploration.

Due to the uncertainty attached to Inferred Mineral Resources, it cannot be assumed that all or any part of an Inferred Mineral Resource will be upgraded to an Indicated or Measured Mineral Resource as a result of continued exploration.

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TWANGIZA GOLD MINE, DEMOCRATIC REPUBLIC OF THE CONGO

1 SUMMARY

SRK Consulting (UK) Limited ("SRK (UK)") was commissioned by Twangiza Mining SA (Twangiza Mining), a subsidiary of Banro Corporation, to independently review Twangiza Mining's December 31 2014 Mineral Reserve estimate which for the first time includes non-oxide material, increasing the Mineral Reserve and extending the Mine life to 14 years.

This report presents a compilation of information prepared by Twangiza Mining and presents findings from a reconciliation study completed as part of the review process and Mineral Reserve estimate.

This report provides:

- an update on the expansion and upgrade of the original processing plant to allow processing of non-oxide material at an annual throughput of 1.7 million tonne per annum (Mtpa);
- a summary of changes implemented as a result of production reconciliation with historical estimates;
- historical and forecast mining and processing operating costs based on the ability to process non-oxide material at the increased annual throughput;
- mine plan optimisation for the 1.7Mtpa processing operating costs, at a range of gold prices and mining operating costs;
- practical pit designs based on an optimal pit shell;
- a production schedule based on an economic cut-off grade;
- the tailings management facility plan to accommodate the increased Mineral Reserve tonnage and throughput rate;
- a financial model that incorporates the additional capital to be expensed during the mine life and assesses the sensitivity of the project to gold price fluctuation; and
- SRK (UK)'s review comments.

1.1 Project Overview

The Twangiza project is located at approximately 2˚52' South and 28˚45' East in the South Kivu Province of the Democratic Republic of the Congo (DRC), some 35 kilometres west of the Burundi Border and 45 kilometres to the south-southwest of Bukavu, the provincial capital.

Banro's properties all lie within the Kibara Belt, a Proterozoic intracontinental mobile belt situated between the Congo Craton in the west and the Tanzanian Craton in the east. Gold mineralisation at Twangiza is hosted by a folded package of mudstone and siltstone sediments and porphyry sills, confined by a doubly plunging anticlinal structure. Mineralisation is found along a 3.5 kilometre long, north trending corridor which hosts the two principal deposits of Twangiza Main and Twangiza North.

The Twangiza property consists of six exploitation permits totalling 1,156 square kilometres which are wholly-owned by Twangiza Mining, a subsidiary of Banro Corporation. These exploitation permits will expire in 2016 and are subject to renewal for consecutive 15 year periods.

The Twangiza Project poured its first gold in October 2011. The mine commenced with a refurbished plant designed primarily to process oxide material at 1.3Mtpa. Since commissioning, the plant has been improved and expanded to 1.7Mtpa and has been shown to be able to process harder non-oxide material.

1.2 Mineral Resource Statement

The Mineral Resources at Twangiza are based on a model originally prepared by SRK (UK) in 2009. The model has subsequently been updated under the supervision of Daniel Bansah of Twangiza Mining and Banro and has been reviewed by Martin Pittuck of SRK (UK). The revised model includes modifications to density values to account for a variance between the original model and tonnages mined to date, and modifying factors based on an analysis of historical production records.

The Mineral Resource estimate is reported according to the definitions and guidelines given in the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Standards on Mineral Resources and Mineral Reserves. The Mineral Resource Statement uses a cut-off grade of 0.4 g/t gold; it has been restricted to a pit shell which uses a USD1,600/oz gold price which is considered therefore to have reasonable prospects for economic extraction by open pit mining.

Table ES 1 below details the "Oxide" and "Non-Oxide" components of the Twangiza Mineral Resource estimate split by confidence category, at a cut-off grade of 0.4 g/t gold. The Mineral Resources are inclusive of the Mineral Reserves.

Table ES 1: Twangiza Mineral Resource Estimate By Confidence Category (December 31, 2014)

OXIDE MINERAL RESOURCE CATEGORY	**TONNES (Mt)**	**GRADE(g/t Au)**	**GOLD OUNCES (Moz)**
MEASURED	3.72	2.30	0.28
INDICATED	8.76	1.88	0.53
MEASURED AND INDICATED	**12.48**	**2.02**	**0.81**
INFERRED	1.34	1.32	0.06
NON-OXIDE MINERAL RESOURCE CATEGORY	**TONNES (Mt)**	**GRADE(g/t Au)**	**GOLD OUNCES (Moz)**
MEASURED	3.80	2.23	0.27
INDICATED	93.00	1.40	4.18
MEASURED AND INDICATED	**96.80**	**1.43**	**4.45**
INFERRED	11.65	1.12	0.42

NB: Any apparent errors are due to rounding and are therefore not considered material to the estimate

1.3 Mineral Reserve Statement

Mineral Reserves were estimated by Twangiza Mining under the supervision of Daniel Bansah and reviewed by a team from SRK (UK) led by Martin Pittuck who is a Qualified Person as such term is defined in National Instrument 43-101. The Mineral Reserve Statement is reported in accordance with National Instrument 43-101 requirements.

The Mineral Reserves stated in 2009 were restricted to oxide material; oxide mining commenced in 2011 and in recent years some non-oxide material has been increasingly blended into the plant feed averaging 20% in the second half of 2014.

The Mineral Reserves given in Table ES 2 below have now increased due to the addition of non-oxide material; they are contained in a practical pit design and they include the Valley Fill material.

Table ES 2: Twangiza Mineral Reserve Estimate (December 31, 2014)

CATEGORY	TONNES (Mt)	GRADE(g/tAu)	GOLD (Moz)
PROVEN	7.47	2.41	0.58
PROBABLE	14.91	2.22	1.06
PROVEN + PROBABLE	**22.38**	**2.28**	**1.64**

1.4 Capital Cost Summary

Twangiza Mining will need to invest in expanding its waste containment capability. Metago Environmental Engineers, who designed the current tailings management facility (TMF), have evaluated accelerated wall raising costs on the existing TMF and are designing and costing an additional TMF to supplement the existing designed tailings capacity.

Table ES 3 below summarizes the estimated capital costs associated mainly with mining operations and process plant upgrade to meet the requirements of the blend feed at a throughput of 1.7Mtpa.

Table ES 3: Capital Cost Summary

ITEM	COST (USD million)
CAPITALISED EXPENDITURE	
MINING – SUSTAINING CAPITAL	25.5
PROCESSING – SUSTAINING CAPITAL	6.4
TMF CONSTRUCTION	46.0
TAILINGS – SUSTAINING CAPITAL	30.5
GENERAL & ADMINISTRATION - SUSTAINING CAPITAL	20.6
BANRO FOUNDATION	1.2
TOTAL – CAPITALISED EXPENDITURE	**130.2**

1.5 Operating Cost Summary

The operating costs in Table ES 4 below were estimated and incorporated into the financial analysis. Estimates have been based on a ‘zero-based’ cost analysis following review of 2014 historical costs and implementation of several cost saving measures.

Table ES 4: Summary of LoM Operating Costs

ITEM	USD million / annum	USD / t processed	USD / oz poured
MINING	9.53	5.96	107
PROCESSING	32.1	20	361
G&A	20.6	12.9	232
TOTAL OPERATING COSTS	**62**	**39**	**699**

1.6 Financial Analysis

The cash flow model for the Twangiza project summarised in Table ES 5 below is based on the 31 December 2014 Mineral Reserve. It assumes a base case gold price of USD1,200 per ounce and a 5% discount rate. The financial model also reflects the fiscal aspects of the mining convention governing the Twangiza project, which includes a 100% equity interest and a 10 year tax holiday from the start of production. An administrative tax of 5% for the importation of plant, machinery and consumables has been included in the projected capital and operating costs.

Table ES 5: Financial Analysis Summary

ITEM	UNIT	AMOUNT
LIFE OF MINE GOLD PRODUCTION	koz	1,246
PRODUCTION PERIOD	years	14
ANNUAL GOLD PRODUCTION FOR FIRST 5 YEARS	koz	109
TOTAL CAPITAL COSTS	USD/oz	104
ALL IN COSTS	USD/oz	888
POST-TAX NET PRESENT VALUE	**USD million**	**285**
NET CASH FLOW AFTER TAX AND CAPEX	**USD million**	**395**

1.6.1 Sensitivity analysis

A sensitivity analysis was performed on the after tax profits by varying the gold price between USD1,000 and USD1,600 per ounce. The results are given in Table ES 6.

Table ES 6: Cash Flow Sensitivity

GOLD PRICE (USD/oz)	NET PRESENT VALUE (USD million)						
	1600	1500	1400	1300	1200	1100	1000
NPV 0.0%	847	734	621	508	395	282	169
NPV 5.0%	612	530	449	367	285	203	122
NPV 8.0%	516	447	378	310	241	172	103
NPV 9.5%	476	413	350	286	223	159	96
NPV 10.0%	464	402	341	279	217	156	94
NPV 12.5%	411	356	302	247	193	139	84
NPV 15.0%	366	318	270	222	173	125	77

1.7 Conclusions and Recommendations

SRK (UK) has reviewed the technical and economic work presented by Twangiza Mining and has assisted in the technical review of historical production records at the mine.

Overall the Mineral Resource base is considered to be well known; the oxide and non-oxide resources are modelled to an appropriate level of confidence for estimation of Mineral Reserves. Adjustment of the resource model densities has been implemented following a reconciliation of the block model with historical production which also gave support to the modifying factors used for estimating Mineral Reserves.

Mineral Reserves are contained within open pits which have been appropriately designed, however a redesign is recommended to bring old designs closer to more recent optimisations. Of particular note is the addition of non-oxide material to Mineral Reserve following the completion of a plant upgrade and capacity increase; this has allowed some 12.8 Mt of non-oxide ore to be added the reserve. The non-oxide material has very variable metallurgical recoveries, the detailed lithological and weathering models ensure accurate estimation of recoverable gold grade within the deposit and a cut-off grade is applied on this basis. The various ore types currently appear in different proportions in the mining schedule on an annual basis and SRK (UK) recommends further smoothing of the mining schedule to even this out.

The process plant has not worked at design capacity historically, mainly due to a shortage of funding. The plant has now been refurbished and SRK (UK) considers that it will be capable of operating at 1.7 Mtpa to fulfil the mine plan presented in this report as long as reagents, consumables and spares are adequately funded and supplied in good time.

The increased Mineral Reserve requires an increased tailings storage capacity. The existing facility is not cost effective and plans to supplement this capacity with a new facility are being developed.

Compared with 2014 costs, the annual and unit operating costs in the mine plan are lower primarily as a result of

- plant upgrade investment coming to an end,
- diesel fuel price reductions,
- a number of cost saving measures with suppliers and contractors that have been or will be implemented, and
- switching to hydroelectric power

The cost savings are based on plans which are at an early stage of implementation and require confirmation in practice. SRK (UK) is confident that if cost savings are made, then the project NPV will be as presented in this report, however if the changes are not realised then the NPV may be considerably lower.

Regarding the lower power costs that are anticipated from a hydroelectric scheme; these savings will only be realised after the hydroelectric facility has been built, however investors and contractors have yet to be identified.

There are a number of capital items to be incurred in the mine pan, the most significant of which is for tailing management facility construction; SRK considers the cost estimate to be in line with tailings construction cost metrics currently incurred on site and to be broadly in line with some design studies completed some years ago. These designs, costs and construction schedules need to be updated to address the specific requirement of the current mine plan. The capital costs are substantial and SRK considers that the final realised cost will be within the means of the project’s cash flow.

The Twangiza North pit area and the area identified for the new tailings dam both require resettlement of dwellings and relocation of artisanal mining activity. Whilst Twangiza Mining has a good track record of achieving this at Twangiza Main pit, the new areas may yet present challenges; SRK (UK) recommends that these activities are planned in good time in order to avoid significant delays to the mine plan.

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2 INTRODUCTION

SRK Consulting (UK) Limited, ("SRK (UK)") was commissioned by Twangiza Mining SA (Twangiza Mining) to independently review its 31 December 2014 Mineral Reserve estimate. SRK helped review over three years of production data in a detailed reconciliation study which has been used to inform the forecast production schedule and to assess the ability of the plant to process non-oxide material.

This report presents a compilation of information prepared by Twangiza Mining and its consultants and SRK's opinions on core aspects of the Mineral Reserve. Additional details for background information can be found in the "Updated Feasibility Study NI43-101 Technical Report, Twangiza Gold Project, South Kivu Province, Democratic Republic of Congo ("2009 Feasibility Study"); a copy of which can be obtained from SEDAR at www.sedar.com.

The Qualified Persons (within the meaning of National Instrument 43-101 ("NI43-101")) for the purposes of this report are Martin Pittuck and David Pattinson of SRK (UK) and who are independent of the project, and Daniel Bansah of Twangiza Mining and Banro. All the Qualified Persons visited the Twangiza mine site between 12th and 15th March 2015.

Twangiza Mining has warranted in writing that it has openly provided all material information to SRK (UK), which, to the best of its knowledge and understanding, is complete, accurate and true, having made due enquiry. SRK (UK) is not aware of any current or pending litigation or liabilities attached to Twangiza Mining.

3 RELIANCE ON OTHER EXPERTS

SRK UK has reviewed production data and mine planning information provided by Twangiza Mining. The environmental, fiscal and legal data has been provided directly by Twangiza Mining and SRK UK has received verbal assurance from Twangiza Mining that none of these aspects presents an impediment to achieving the production outlined in this report.

The modified Tailings Management Facility (TMF) plan was provided by Daniel Bansah of Twangiza Mining; the plan is derived from previous work completed by Metago (now SLR Consulting) on behalf of Twangiza Mining.

4 PROPERTY DESCRIPTION AND LOCATION

The property is located in the South Kivu Province of the Democratic Republic of the Congo (DRC), centred at approximately 2˚52' South and 28˚45' East, roughly 35 kilometres west of the Burundi Border and 45 kilometres south-south-west of Bukavu.

4.1 Mineral Tenure

In April 2002, the Government of the DRC formally signed an agreement which entitled Twangiza Mining to hold a 100% interest in the Twangiza Property under a revived mining convention which expires in March 2027 (subject to extension under the new DRC Mining Code).

The exploitation permits give Twangiza Mining exclusive rights to carry out exploration, development, construction and exploitation works within the perimeter over which they have been granted.

The six exploitation permits, or Certificat / Permis d'Exploitation (PE), covering a total area of 1,156 square kilometres define the Twangiza Property for which Twangiza Mining has exclusive mining rights; these are listed below and shown in Figure 4-3. The exploitation permits are 100% owned by Twangiza Mining, a subsidiary of Banro Corporation.

PE40 – Concession No 92
PE41 – Concession No 91
PE42 – Concession No 90
PE43 – Concession No 89
PE44 – Concession No 88
PE68 – Concession No 66

The property boundaries are located by co-ordinates provided with each exploitation permit. Twangiza Mining has had no need to physically beacon the boundaries as the mineralized zone and areas affected by the mining activities are located well within the boundaries defined by PE42.

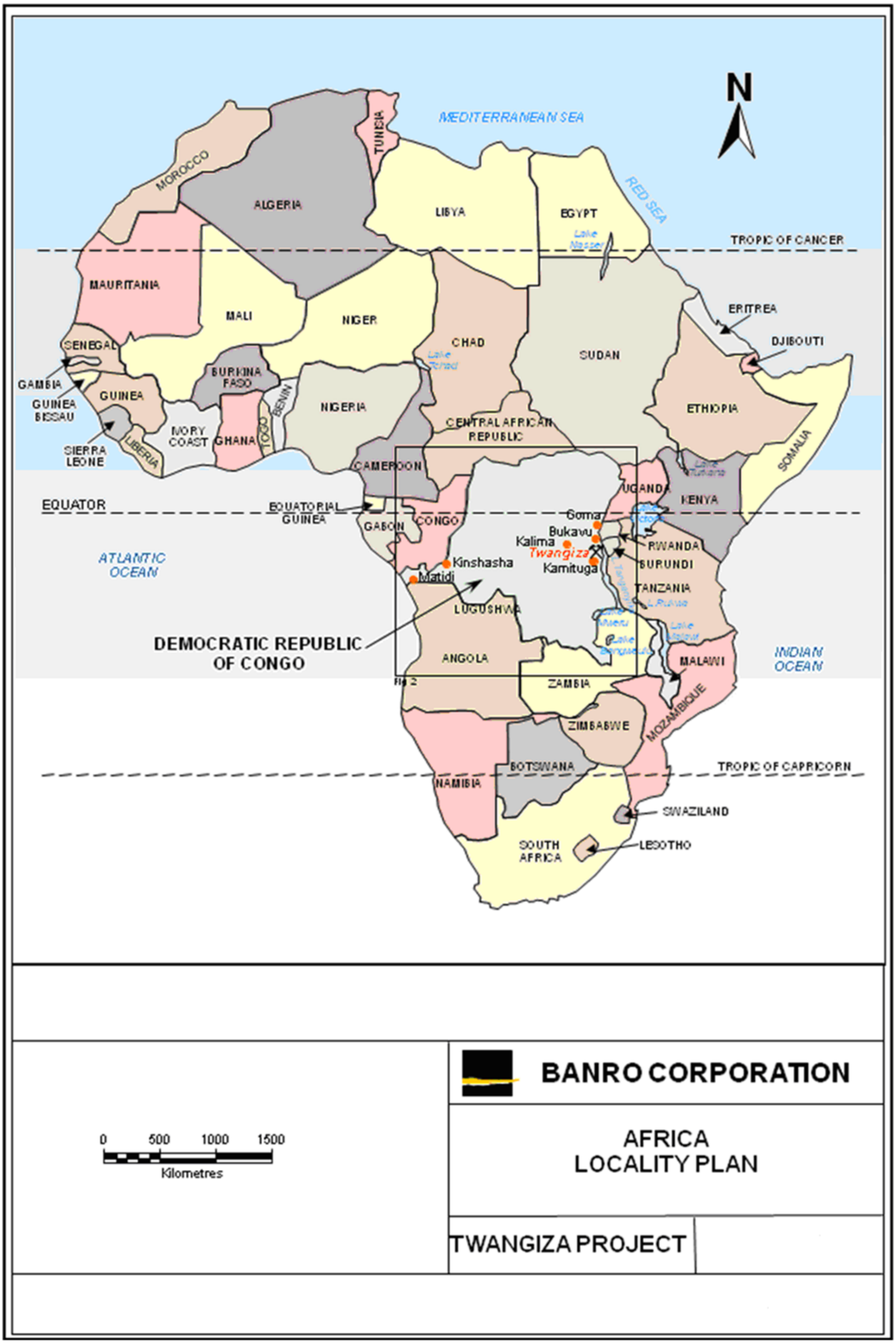


Figure 4-1: Africa Locality Plan

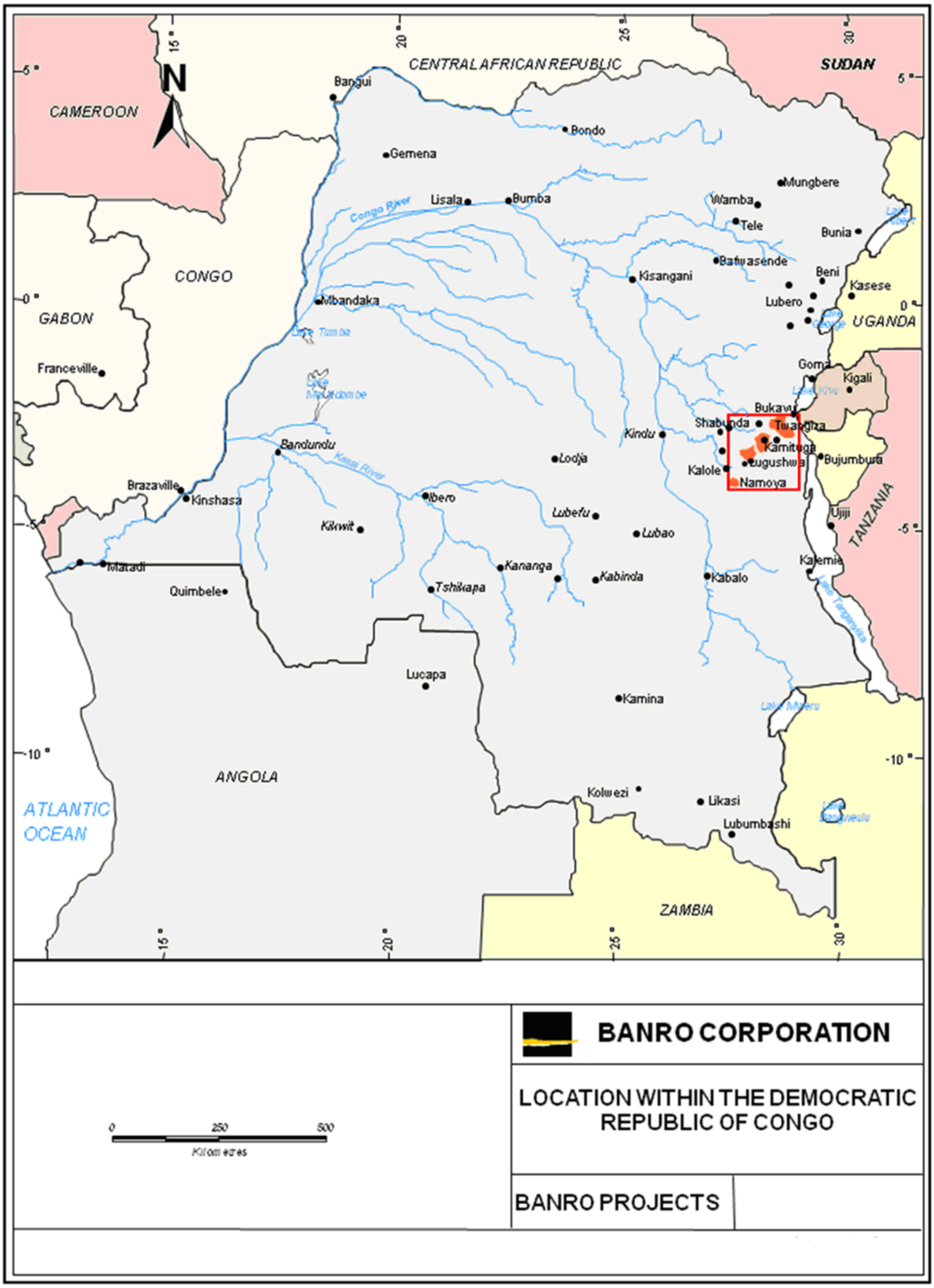


Figure 4-2: Location of the Property in Democratic Republic of the Congo

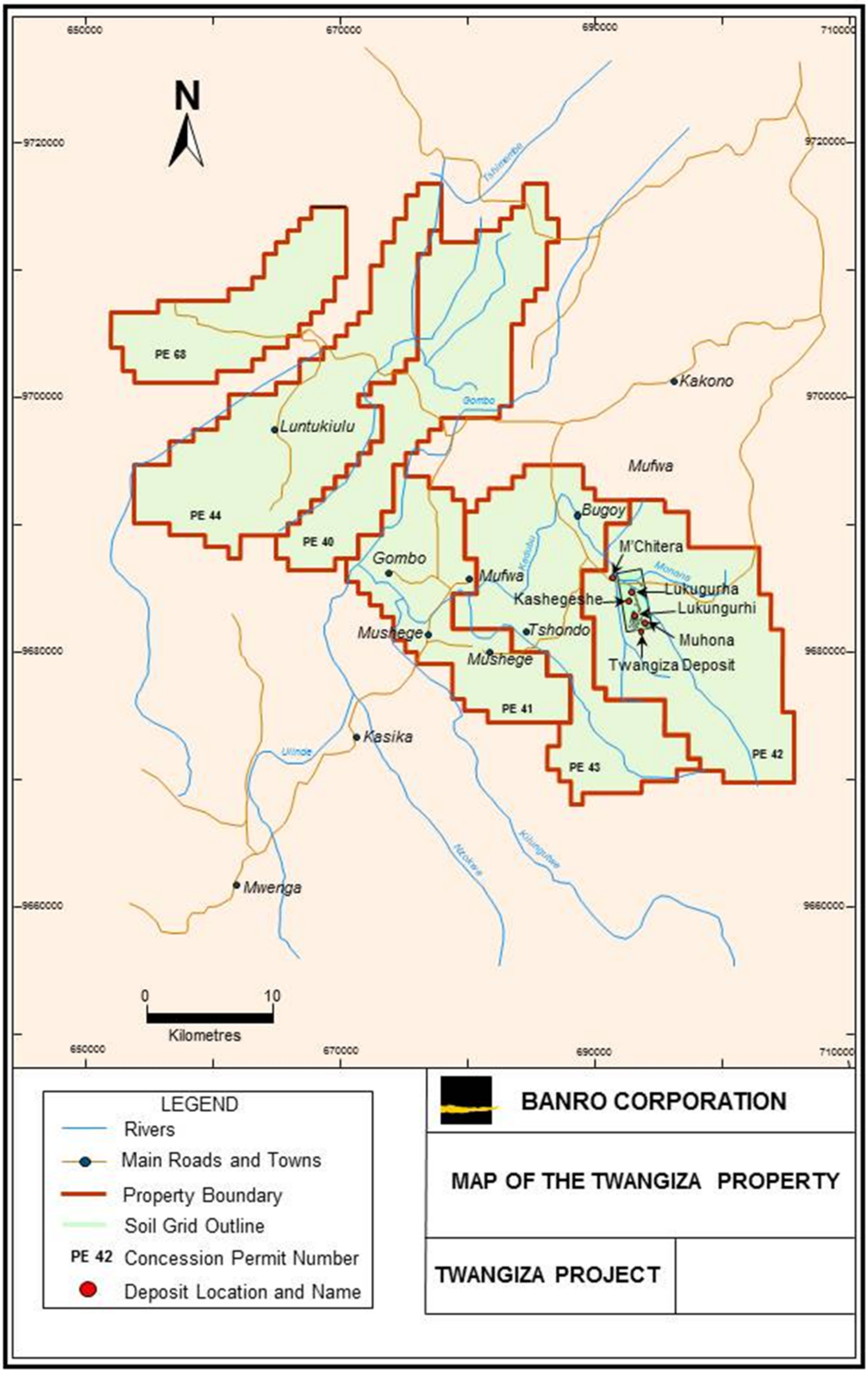


Figure 4-3: Map of the Twangiza Property

4.2 Natural and Existing Features

Details of the known mineralized zones, mineral resources and mineral reserves are included within Section 7 of this report.

During the colonial era there was small scale hard-rock mining for tin, alluvial gold and tungsten diggings. However, these sites (all outside Twangiza Mining's "mining area") have now become over-run by artisanal mining activity and there is little evidence of the earlier activity.

With the exception of informal artisanal activity there are no current or historical formal mine workings, tailings ponds or waste dumps located on the Twangiza Property other than those belonging to the operation described in this report

4.3 Royalties and Other Payments

A royalty calculated at 1.0% of revenue is paid to the DRC government.

4.4 Environmental Liabilities

Twangiza Mining will be liable to the DRC government for any damage to the environment resulting from a breach of the requirements of the DRC Mining Code, approved environmental impact statement (EIS) or associated environmental management plan of the project (EMPP).

More information regarding the status of the EIS and EMPP is included in Section 20 of this report.

On the basis of limited monitoring data and observations in the field, the following potential liabilities accruing from past mining (artisanal mining) and prospecting activities on site may occur:

Landslides due to destabilization of slopes;

Twangiza river: denuded deposits of tailings along 3-4 km stretch of riverbed, may be entrained by floodwaters, leading to high suspended sediment loads in downstream river reaches and resulting in negative impacts on aquatic biota and human consumption;

Mercury used by artisanal miners may be adsorbed in sediments and may be mobilized and transported in river water should acid rock drainage occur. However, this proposition has not been tested and proven. Acid rock drainage potential is indicated in preliminary tests, but has not been confirmed.

4.5 Required Permits and Approvals

The permits and approvals required to conduct the work at the Twangiza property are specified in the DRC Mining Code. Such permits and approvals have been obtained.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Topography, Elevation, Vegetation and Seismic Activity

Twangiza is situated in a mountainous area with deeply incised valleys with slopes typically greater than 30^{o}, forming a dendritic drainage pattern. The mining area occupies a steep ridge running north/south between two fast-flowing rivers, which join just to the north of the mine. Elevation in the area ranges from 1500 m to 2400 m above sea level.

Vegetation on the Twangiza property is a mosaic of transformed, agricultural plots and woodlots of cypress and eucalyptus, and montane grassland. One small, 2.18 ha patch of indigenous forest remains to the east of the mine area, the Lusirwe sacred forest.

Due to its location within the western arm of the Rift Valley system, the property is subject to seismic activity. Detailed discussion of the seismic hazard potential within the Twangiza area is included the 2009 Feasibility Study.

5.2 Means of Access

Twangiza Mining's offices are in Bukavu, the capital city of South Kivu Province some 45 kilometres north-northeast of the Twangiza Property. Bukavu has an airport, Kavumu, however access by road from Rwanda is the current preferred route for international access.

Road access from Bukavu to the Twangiza Property is possible by travelling some 55km on the recently upgraded N2 National Road and then 30km on the recently widened and upgraded Twangiza access road. The journey time is 2.5 hours during the dry season and extends to 4 hours under wet conditions. The property is also serviced by a helicopter and the journey between Bukavu and Twangiza is some fourteen minutes.

5.3 Climate

The climate at Twangiza can be classified as tropical to sub-tropical; the wet season falls between September and April and the main dry season is from May to August. Due to its close proximity to the equator, Twangiza experiences daylight and night hours that are almost equal, with daylight lasting between 6am and 6pm. The relative humidity generally exceeds 85 % and the mine is often in cloud.

Twangiza has an average annual rainfall of 1,796 mm. Rain generally occurs as soft, lengthy rainfall in the mid to late afternoons, but violent thunderstorms are also frequent. For the period October 2006 to December 2014 the highest monthly rainfall recorded for Twangiza was 357.5 mm in December 2011 and the lowest total monthly rainfall was in March 2008 (see Table 5-1).

The on-site weather station at Twangiza recorded data for the period October 2006 to December, 2014. The average temperature measured on-site is 18°C with a maximum temperature of 27.7°C, measured during March 2007, and a minimum temperature of 8.6°C, measured during July 2008.

The prevailing north-easterly wind direction for Twangiza is relatively consistent throughout the year. The average annual wind speed for all hours is 4.33 m/s.

Table 5-1: Monthly Rainfall (mm) For Twangiza (October 2006 to December 2014)

YEAR	J	F	M	A	M	J	J	A	S	O	N	D
2006	-	-	-	-	-	-	-	-	-	99*	238	238
2007	275	118	109	144	40	134.6	22.4	24.4	111	37.2*	177.4	203.2
2008	164.2	140	4.6*	86.2	61.8	53.8	27.6	12.2	29.2	103*	-	-
2009	-	-	-	-	63.6	14.8	0.4	12.4	112	73.4	162.2	-
2010	-	202.6	207.1	73	122.4	18.8	5.2	7.2	98	124.4	154.8	112.8
2011	299.4	87.1	262.9	48.5	59.7	92.8	36.3	21.6	82.7	107.4	264.5	357.5
2012	151.5	177	137.6	141	81.3	13.7	34	130	93	161	79.4	151
2013	197.9	123.6	282.4	141.6	11.8	0.2	0	39.5	213.8	88.4	248.7	215
2014	257.3	208.7	223.6	84.9	10.9	50.2	0	28.2	85.7	196.7	187.6	154

** Incomplete data set*
- No data

5.4 Surface Rights and Available Local Infrastructure

Twangiza Mining has mining rights to the Twangiza mine. The Twangiza Property is remotely located and there is no existing supply of power suitable for the Project requirements. A diesel-generator power plant has been established to provide the power required to the Twangiza Project. Twangiza Mining plans to eventually take power from a hydroelectric scheme due to come on line in the next few years.

The topography in the area is challenging. Water containment dams and a tailings disposal facilities can be placed in nearby valleys of limited dimensions or in wider river plains within several km of the mine.

The local workforce consists primarily of subsistence farmers and artisanal miners. In addition the communities contain a number of skilled workers (i.e. artisans - builders, carpenters, electricians, plumbers etc).

6 HISTORY

The earliest recorded work programs within the Twangiza Property consisted of alluvial mining for tin and gold as early as the 1930's along prominent rivers and creeks by Mines des Grandes Lacs (MGL). MGL began exploration for in-situ resources in 1957 and followed alluvial gold deposits upstream from the Mwana River to the present day Twangiza deposit.

MGL tested the Twangiza deposit through 8,200 metres of trenching and 12,100 metres of adits (20 metre by 20 metre grid) on seven levels (Levels 2100 to Level 2220).

In 1974 to 1976, Charter Consolidated Limited ("Charter") undertook an evaluation program of the Twangiza area in order to verify the results obtained by MGL and to look for possible extensions of the deposit. Work also included metallurgical studies.

From 1982 to 1984, SOMINKI undertook a feasibility study which was completed by ABAY, a Belgian consulting company and in 1988 the Northern Queensland Company assessed the deposit and generated some financial models. A report was prepared by Billiton in 1989 for SOMINKI and submitted to the Ministry of Planning for tax exoneration purposes.

In January 1996, Banro Resource Corporation's (now Banro Corporation) wholly owned subsidiary, African Mineral Resources Inc. ("AMRI"), in conjunction with its joint venture partner Mines D'Or du Zaire ("MDDZ"), completed the purchase of the outstanding privately held shares of SOMINKI. The joint venture partners controlled 72% (AMRI - 36%, MDDZ - 36%) of SOMINKI, with the remaining 28% held by the Government of Zaire (DRC). Banro subsequently acquired MDDZ's 36% interest in SOMINKI in December 1996.

In early 1997, Banro, SOMINKI and the government of the DRC ratified a new 30 year mining convention that provided for SOMINKI to transfer its gold assets to a newly created company. Société Aurifère du Kivu et du Maniema, SARL ("SAKIMA") was incorporated to acquire the assets of SOMINKI as stipulated in the new mining convention. Banro consolidated the information and from August 15, 1997, to April 15, 1998, undertook a field exploration program managed by CME and Company (CME).

In addition to this asset transfer, the new mining convention included a ten year tax moratorium from the start of commercial production, the ability to export all gold production, the ability to operate in US currency, the elimination of import duties and title confirmation for all of the concessions. The new mining convention provided for Banro to control 93% of SAKIMA with the remaining 7% held by the Government of the DRC as a net carried interest.

In July 1998, President Laurent D. Kabila issued presidential decrees which, amongst other things, effectively expropriated SAKIMA's gold assets. Banro initiated arbitration proceedings against the Government of the DRC seeking compensation for the expropriation of the assets.

In April 2002, the Government of the DRC formally signed a settlement agreement with Banro. The agreement called for, among other things, Banro to hold a 100% interest in the Twangiza Property under a revived mining convention which expires in March 2027 (subject to extension under the new DRC Mining Code).

Additional information regarding the Twangiza Property with respect to history is set out in the 2009 Feasibility Study.

7 GEOLOGICAL SETTING AND MINERALISATION

7.1 Regional Geology

The Twangiza Property is located in the northern half of the Great Lakes sub-province of High Africa, one of the world’s principal Precambrian orogenic-metallogenic provinces, see Figure 7-1. Twangiza lies within the Kibara Belt, a Proterozoic intracontinental mobile belt situated between the Congo Craton in the west and the Tanzanian Craton in the east. The belt trends in a NNE-SSW direction for over 2,000 km from Katanga to Lake Victoria, and attains its maximum width of about 500 km slightly to the north of the Twangiza-Namoya area.

The belt has a long and complex evolution, stretching from the Palaeoproterozoic prior to the Eburnean orogeny, through to the Neoproterozoic and the Pan African event. The belt is dominated by clastic sedimentary rocks with minor carbonates and volcanics, which have been intruded by granitoids, mafics and alkaline complexes.

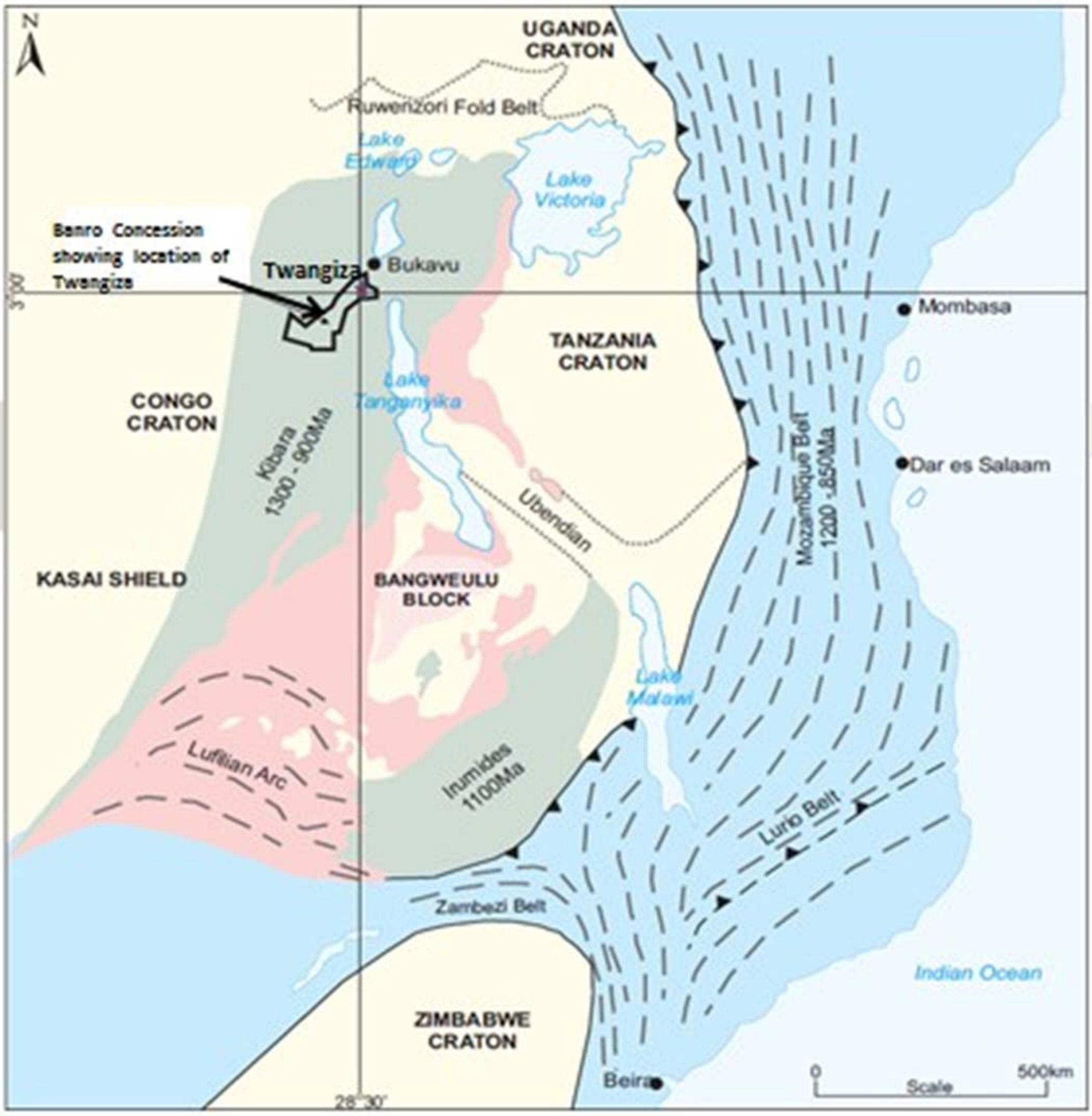


Figure 7-1: Regional Setting of The Kibara Belt

7.2 Property Geology

The concessions-scale geology can be divided into three distinct litho-structural terrains as illustrated in Figure 7-2. The eastern terrain is characterised by folded, broadly N-S trending Neoproterozoic sediments, which are part of the Itombwe synclinorium, a regional-scale fold which extends southwards from the Twangiza area for about 150 km. The western domain has a distinct NW-SE tectonic grain, and is believed to be Palaeoproterozoic in age. The third domain occurs in the north, where recent basalts blanket the Proterozoic rocks.

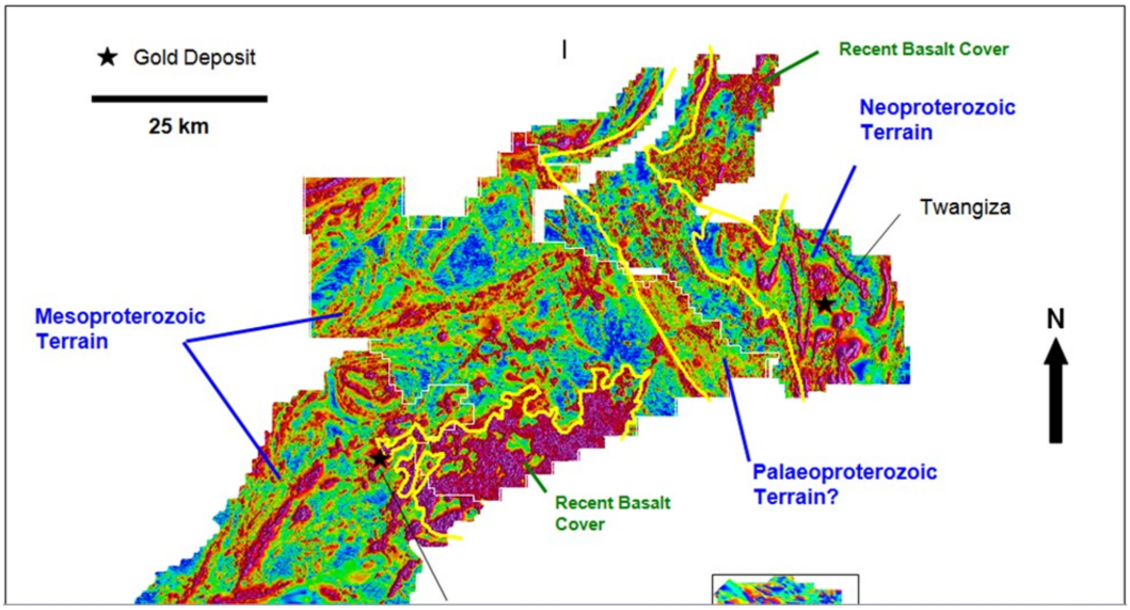


Figure 7-2: Magnetic Image Showing Litho-Structural Domains Of The Twangiza Concession

The sediments in the Neoproterozoic terrain are weakly metamorphosed. The dominant lithology is mudstone, often with a significant amount of carbonaceous material; units of siltstone are commonly interbedded with the mudstone. Quartz wacke and sandstone occur as thin beds or lenses. A characteristic feature of the Twangiza area is the presence of a conglomerate consisting of clasts of granite, mudstone and siltstone supported by a matrix of dark grey silty mud. It frequently contains a significant amount of detrital magnetite, and forms the relatively highly magnetic unit that clearly defines the geometry of the concession-scale folds in the magnetic images.

In the vicinity of the Twangiza Mine, the Neoproterozoic sediments have been intruded by porphyritic sills, ranging in thickness from less than 1 m to over 50 m. The sills have undergone extensive hydrothermal alteration and are thought to be part of a suite of alkaline intrusives emplaced at about 750 Ma.

The Neoproterozoic terrain at Twangiza is characterised by a series of N-S trending, concession-scale folds, which plunge to the north. These folds vary from being open to almost isoclinal, although the average limb dips are usually between 50° and 80°. Smaller-scale folds, probably parasitic to the larger structures, are commonly seen on a prospect scale; they display plunges to the north and south, or are doubly-plunging like the fold hosting the Twangiza deposits. The folding is considered to have developed in response to E-W compression in the Pan African orogeny at about 550 Ma.

Faulting in the Neoproterozoic terrain is common, the main trends being NE-SW to E-W. In addition, zones of shearing and/or brecciation have been mapped sub-parallel to the fold axes at several prospects, and may have had a control on the mineralisation.

7.3 Weathering

The rocks in the area have been affected by tropical weathering resulting in degradation of the original rock fabric down to a depth 100m; in the deposit area where most data is available the weathering is particularly intense in the top 30m.

7.4 Mineralisation

The Twangiza Main ore body consists of a wide (up to 200 m) zone of pervasively altered mudstone, siltstone and porphyry sills, with abundant sulphidic veins. The veins form a complex irregular network, although veining parallel to bedding is relatively common. Hydrothermal fluids have exploited both the fracture system which developed during folding due to competency contrasts between the lithologies, and dilational zones between bedding planes to form saddle reefs.

The style of mineralisation in the sediments and sills varies, but can be sub-divided into two main types as discussed below:

Mineralisation in the sills is characterised by the presence of pyrite and arsenopyrite (approximately 65% pyrite: 35% arsenopyrite). The total abundance of sulphide is variable, averaging about 3% of the rock, but locally comprising up to about 30% of a 1 m sample. There is a positive correlation between grade and sulphide content. The sulphides occur in a variety of habits: (a) disseminated crystals, (b) stringers, (c) coarsely crystalline veins up to 10 cm in width, but usually 1 – 3 cm across, and often with intergrown quartz, and (d) irregular massive patches. Further details can be found in Section 13.4.1.

The quantity of sulphides disseminated in the mineralised sediments is generally lower, they are generally finer grained and are more common in the relatively porous siltstone units. The sulphide veins in the sediments generally contain more quartz, either intergrown with the pyrite and arsenopyrite, or forming borders to the veins.

In the oxidised zone, the veins in both the porphyry and sediments have weathered to limonite-silica intergrowths. This limonite-silica veining is a common feature of the mineralisation in outcrop. Limonite-filled boxworks, and irregular limonite patches and coated vugs have formed due to oxidation of the disseminated sulphides and patches.

Hydrothermal alteration associated with the gold mineralisation has formed three broad assemblages.

Additional information regarding the Twangiza property with respect to geological setting and mineralization is set out in the 2009 Feasibility Study.

8 DEPOSIT TYPES

The spatial association of gold deposits with Sn-W mineralisation in the Kibara belt has led to the suggestion that the gold bearing fluids were also related to the intrusion of the tin-bearing G4 granites. However, the rocks which host the Twangiza deposit are considered to be Neoproterozoic in age, and therefore post-date the G4 granites (c. 975 Ma).

It is proposed that the Twangiza mineralisation (and possibly the other gold deposits in the Kibara belt) are rather related to fluids derived from the devolatisation of the lower crust during the Pan African orogeny at about 550 – 520 Ma.

The Pan African orogeny at about 550 Ma, involved E-W compression leading to deformation of the Neoproterozoic sediments and sills into N-S trending folds. Auriferous fluids were focused into these structural traps; at Twangiza Main, the most important traps were the low-pressure hinge zones of anticlinal folds, see Figure 8-1.

The feeder structures were probably sub-vertical, limb-parallel structures or “limb shears”. Mineralisation would be expected to deteriorate down the fold limbs away from the fold closure and away from the “sill zone” where the stratigraphy is relatively homogeneous. Twangiza North hosts deeper, stratabound, mineralisation where the feeder structures have intersected relatively reactive, internally fractured sills, resulting in a sharper mineralised boundary.

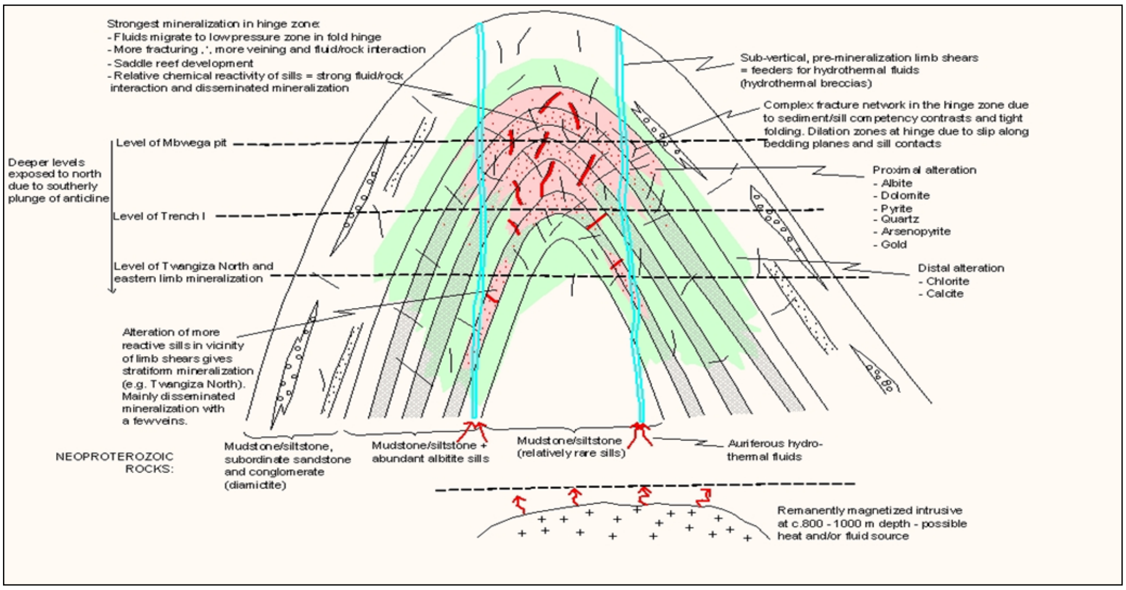


Figure 8-1: Deposit Geology

9 EXPLORATION

Exploration on the Twangiza Property has been divided into historical exploration and the recent work carried out from October 2005 to December 2014. The recent exploration is subdivided into five phases: October 2005 – December 2006, January 2007 – November 2008, January 2009 – December 2010, January 2011 – December 2012 and January 2013 – December 2014.

9.1 Historical Exploration

There have been three major field exploration programs on the Twangiza Property prior to 2005.

The first was between 1957 and 1966 by MGL and consisted of the driving of approximately 12,100 metres of adits and 8,200 metres of trenches at the Twangiza deposit. A total of 17,400 channel samples were collected at two metre intervals from both the trenches and adits.

Secondly, from 1974 to 1976, Charter Consolidated Limited undertook an evaluation program of the Twangiza area in order to verify the results obtained by MGL and to look for possible extensions to the mineralisation. Soil sampling was conducted over a 4.6 square kilometre area to the north of the Twangiza deposit. Anomalous soil samples were tested by 11 pits, 6 trenches and 5 adits. Work also included the re-sampling of three MGL adits (Levels 2100, 2130, and 2220).

The third historical program was undertaken by Banro between August 15, 1997 and April 15, 1998. The program was managed by CME and consisted of:

- Topographical surveying, LANDSAT acquisition and interpretation, Helicopter-supported airborne magnetic survey.
- Detailed geological mapping and rock sampling, grab samples and channel samples from 16 adits were taken and after sample preparation at Banro's on site sample preparation laboratory were sent to Acme Analytical Laboratory in Vancouver, Canada for analysis;
- Petrographic studies and density testing was performed on behalf of Banro by CME

9.2 Recent Exploration (October 2005 – December 2006)

Banro resumed its exploration programme at Twangiza after the Congolese government had established control and authority in the area in October 2005.

9.2.1 Soil geochemical programme

A soil geochemical programme designed to test the immediate northern, eastern, western and southern extensions of the known Twangiza mineralisation was completed in 2006. The 7 km soil geochemical grid had its base line orientated along the hinge of the anticline at 350º. Soil sampling was undertaken at 40 m intervals on lines spaced at 80 m. The baseline origin for the soil geochemical grid was pegged at UTM coordinate 9682698.2N / 693500.5E, which corresponds to a local grid coordinate of 10000N/20000E.

9.2.2 Trenching programme

A trenching programme was initiated to test the gold-in-soil geochemical anomalies and the continuity of mineralisation on the northern extension of the Twangiza Main deposit, as well as the southern and northern extensions of the Lukungurhi workings. A total of 785 channel samples were collected from 1.159 m of trenching.

9.2.3 Prospect scale mapping

A detailed mapping project was carried out in the Twangiza artisanal workings in order to gain a better understanding of the geology and mineralisation controls and to verify and compliment the diamond drilling data. The study reviewed all aspects of the geology including lithology, structure and alteration.

9.3 Recent Exploration (January 2007 – November 2008)

9.3.1 Geophysical exploration

Airborne magnetic and radiometric surveys were completed over the entire Twangiza property, utilising a flight line spacing of 100 m with tie lines at 1000 m intervals; several targets were identified for follow-up work.

9.3.2 LIDAR survey

LIDAR to create accurate topographic maps of the region. In addition, colour aerial digital photography has been rectified to create accurate orthophotos.

9.3.3 Additional regional work

During late 2007 Banro began exploration of the Twangiza property outside the main trend. The following targets were investigated:

Mufwa

Located 13 km northwest of the Twangiza Main deposit, Mufwa is the focus of intense artisanal activity, where miners are exploiting a series of quartz veins within mudstone.

Kaziba

Located 11 km east of the Twangiza Main deposit, the Kaziba target was discovered towards the end of 2008. Sulphide-associated, disseminated mineralisation similar in style to that at Twangiza Main, occurs within mudstones and siltstones on the western limb of a northerly plunging anticline. Exploration during 2008 consisted of mapping and sampling of artisanal workings, and soil sampling of a 2 x 1 km area around the workings on an 80 x 40 m grid. A total of 290 rock samples and 573 soil samples were collected. The data indicates the presence of gold mineralisation over a strike of 250 m, potentially with a thickness of up to 30m.

Tshondo

Preliminary work at Tshondo, an old colonial discovery located 9 km west of Twangiza Main, indicates that gold is associated both with quartz veins and within the surrounding hydrothermally silicified mudstone and siltstone. The mineralisation is associated with the axis of a northerly plunging syncline. Artisanal mining is focusing on the relatively high-grade quartz mineralisation over a strike of 500 m. Soil geochemical sampling in tandem with rock chip sampling, trenching and adit mapping and sampling was undertaken in late 2010. Analytical results are awaited to warrant further work comprising diamond drilling.

Radiometric anomaly

This prospect is approximately 5 km west of Twangiza Main, and was targeted due to the presence of a coincident uranium-thorium radiometric anomaly similar to that at Twangiza, located on a well-defined anticline axis. Work during 2008 comprised a programme of stream sediment sampling (117 samples) and regional mapping. The stream results define three anomalous areas with values of up to 1,840 ppb Au.

Southern anomaly

This target is located 10 km south of Twangiza Main. It is situated on the axis of a tightly folded syncline, and is associated with a coincident uranium-thorium radiometric anomaly. Historical records indicate that alluvial gold was exploited in colonial times, and alluvial artisanal mining is still carried out locally. The 2008 programme comprised stream sediment sampling (185 samples) and regional mapping. Anomalous values of up to 470 ppb Au will be followed up initially by soil sampling.

9.4 Recent Exploration (January 2009 – December 2010)

During this period, exploration focused on mineralized structures flanking the Twangiza Main orebody and on geochemical sampling to the west, north and east of the original 7 x 2 km grid. Regional work continued at Kaziba and the Radiometric Anomaly, and also at Ntula, a new discovery in the north of the Twangiza concession.

9.4.1 Twangiza West Zone

This zone lies immediately to the west of the Twangiza Main deposit. Work during 2009 included surface mapping and rock chip sampling, and extensive auger drilling.

9.4.2 Twangiza East Zone

Mineralisation to the east of the Twangiza Main ore body was tested by extensive auger drilling (221 holes, 866 m).

9.4.3 Valley Fill

An extensive pitting program was completed aimed at estimating the gold resource of the Mwana River Valley Fill material, a total of 44 pit were completed along 17 traverses. This was followed by 36 auger drill holes totalling of 119.15 m.

9.4.4 Geochemical surveys

Soil sampling, rock chip sampling and mapping coverage was expanded.

9.4.5 Regional work

Exploration continued at Kaziba, Radiometric Anomaly and Ntula.

9.5 Recent Exploration (January 2011 – December 2012)

During this period, exploration focused on infill drilling at Twangiza East and West flanks to provide a better understanding of these deposits. Regional work focused on soil sampling, rock chip sampling and mapping at Ntula, Luntukulu and Kaziba.

9.6 Recent Exploration (January 2013 – December 2014)

During the period 2013 – 2014, the company reduced its exploration activities in the Twangiza concession to grassroots activities focused on the Ntula – Mufwa regional corridor and prospects around Luntukulu area. In 2014, there was further reduced exploration focussed on the Mufwa and Kadubo Prospects.

10 DRILLING

Due to the extreme topography at Twangiza, resource definition drilling was supported by an A-Star 350 B2 helicopter for moving drilling rigs, materials and personnel from site to site. The majority of drilling has been diamond drilling focussing on Twangiza Main and North deposits as summarised in Table 10-1 and shown in Figure 10-1.

Table 10-1: Drilling Summary

	Diamond Drilling		RC Drilling	
	number	length (m)	number	length (m)
Twangiza North	164	31,373	4	438
Twangiza Main	169	40,427	15	2,936
Total	333	71,800	19	3,374

The Mineral Resource model in this report uses only the resource drilling; it has not been updated with the grade control drilling undertaken since the mining started.

10.1 Drilling (September 1997 – March 1998)

A total of 20 diamond drilling holes covering, 9,122 metres, 8,577 samples, HQ and NQ core size was completed between September 4, 1997 and March 9, 1998. The drilling covered an 800 m strike length of mineralisation within the hinge of the Twangiza Anticline, with holes drilled at different orientations.

Drilling was performed by Rosond International Limited of South Africa utilizing two Longyear 38 drill rigs with a maximum depth potential of 600 m.

10.2 Drilling (February 2006 – May 2008)

Four drill rigs were deployed at the Twangiza property, with two additional rigs mobilized in 2008.

A total of 17,037.34 metres of diamond drilling involving 71 holes were completed between February and December, 2006. In January 2007, a major drilling campaign commenced with the aim of converting the remaining Inferred Mineral Resources at Twangiza Main into the Indicated and Measured categories and to identify additional Inferred Mineral Resources particularly at Twangiza North.

A total of 61 resource holes were drilled at 40 m centres to infill the holes drilled in 1997/98 with the objective of upgrading the Inferred Mineral Resources to the higher confidence Measured and Indicated Resources. In addition, 39 exploration holes were drilled to test the Twangiza North geochemical anomaly.

The remainder of the programme between May 2007 and May 2008 consisted of a total of 24,231.3 metres of PQ, HQ and NQ diamond drilling involving 116 holes. The focus of the drilling was to infill the drilling grids and potential resources within the Twangiza North area of the deposit. 98 holes were drilled at 40 m centres to infill the holes drilled in 2006 and early 2007 in the Twangiza North deposit and 18 holes were drilled in Twangiza Main.

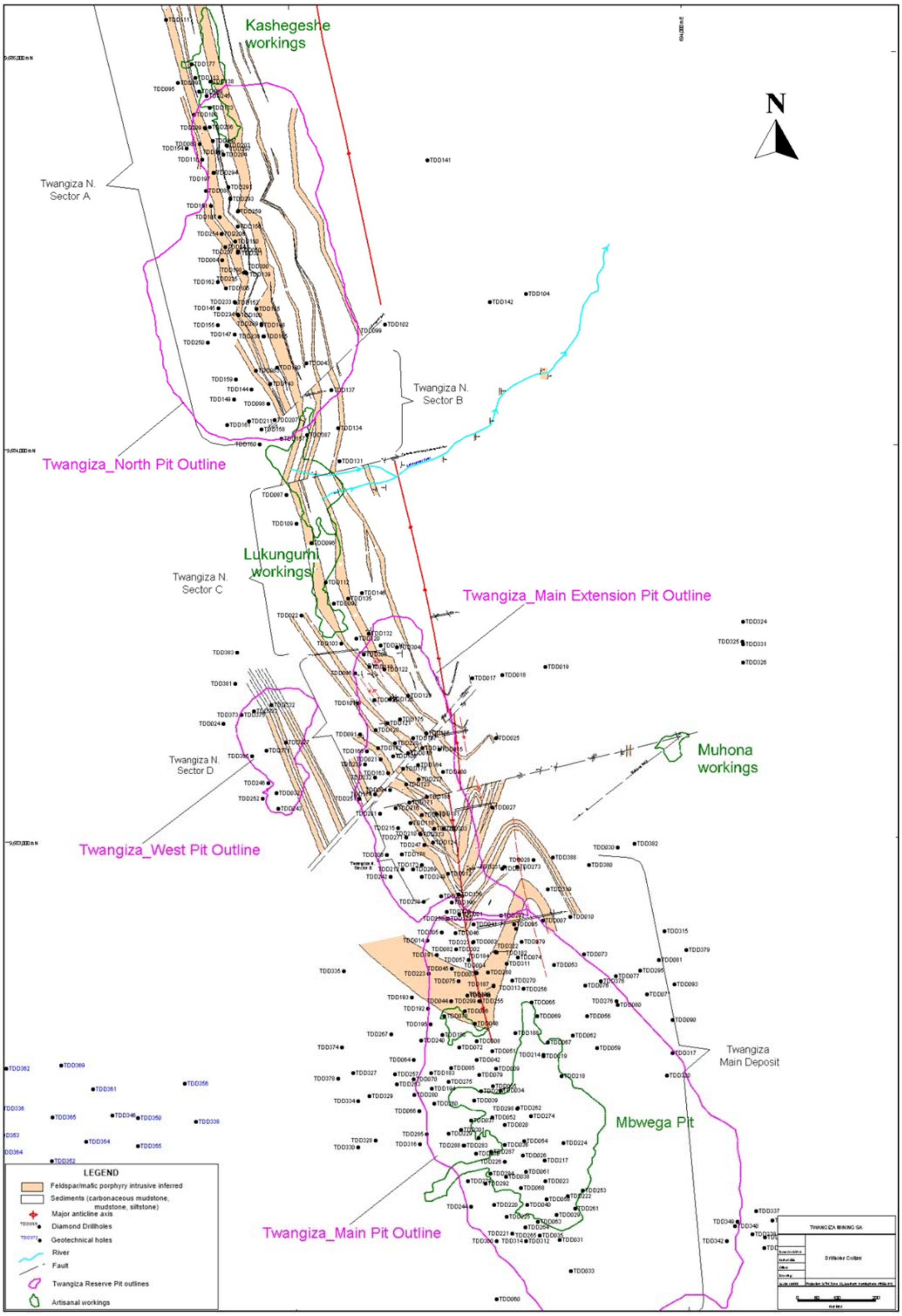


Figure 10-1: Plan of Resource Drillhole Locations

The diamond drilling was performed by Geosearch International Limited of South Africa utilizing four portable CS1000 and two Longyear 38 drill rigs with a maximum depth capability of 600 metres.

All drill hole collars were surveyed with RTK GPS equipment. Drill hole collar azimuths and inclinations were established at surface by using hand held compasses. Down-hole surveying of drill holes utilized a Reflex Single Shot or Flexit instrument, which measures both azimuth and inclination of the hole. All drill core was orientated. Orientation was carried out by the "Spear" method or the Ezy Mark system. The majority of the drill holes were drilled to the east on an azimuth of between 70 - 80°, and at inclinations of between 50 – 55°. A portion of the programme has been drilled in the opposite direction on an azimuth of 260° to improve the definition of the 3D wireframes.

10.3 Drilling (May 2008 – November 2008)

102 diamond drill holes totalling 21,952.26 metres of PQ, HQ and NQ were completed between May 2008 and November 2008. Drilling tested the mineralised interpretation at depth with the aim of increasing confidence in the previous estimates, particularly in Twangiza Main. Resource holes were drilled on 40 m centres to infill the holes drilled during previous campaigns with the objective of upgrading the Inferred Mineral Resources to the higher confidence Measured and Indicated Mineral Resources and improving the estimation at depth. The same drilling procedures were used in terms of rig set-up and rig movement as in the previous drilling campaign.

The majority of the holes in the programme were drilled to the east on an azimuth of between 70 - 80° at dips of between 50 – 55°. A portion of the programme has been drilled in the opposite direction on an azimuth of 260° to improve the definition of the 3D wireframes.

10.4 Drilling (December 2008 – December 2012)

138 diamond drill holes, 61 RC holes and 1,515 Auger holes totalling 12,565.71 metres, 8,101 metres and 4,424.21 metres respectively were completed between December 2008 and December 2012.

Diamond drilling was mainly used for infill drilling at Twangiza East (7 holes totalling 747.91metres) and Twangiza West (28 holes totalling 2,638.45 metres) zones and also to further test the Ntula (29 holes totalling 3,861.46 metres) and Lukugurhi (4 holes totalling 442.15 metres) mineralisation. Some 90 diamond drilling holes totalling 3,613.94 metres were also undertaken for geotechnical and electrical study of the Tailings Dam and the plant site areas.

A total of 3 Reverse Circulation holes (375 metres) had been completed in this zone by end of year 2010. RC and Auger drilling undertaken mainly focused on exploration even through some RC holes were drill for ground water monitoring.

The majority of the diamond and RC drill holes in the programme were drilled to an azimuth of between 70 - 80° at dips of between 50 – 55°. A portion of the programme has been drilled in the opposite direction on an azimuth of 260° to improve the definition of the 3D wireframes. All auger drill holes were drilled at a vertical angle as well as diamond and RC holes drilled for geotechnical study and ground water monitoring.

10.5 Drilling (January 2013 – December 2014)

Due to the significant scaling down in exploration activities, there was no resource definition drilling undertaken in this period. 20 auger holes totalling 43.6 m were drilled at Ntula in the regional exploration area for target generation.

2 resource holes (183.9 m) were drilled in the Twangiza main pit to test the extension of the main pit mineralisation in 2014. Also 9 geotechnical holes totalling 316 m were drilled for geotechnical studies within the Twangiza main pit in 2014.

10.6 Results

A summary of significant mineralised intersections can be found in Appendix II of the 2009 Feasibility Study.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 SAMPLING METHOD AND APPROACH

11.1.1 Soil Geochemistry

Approximately 4 to 6 kg of soil was taken from each sample site, below the upper soil horizon containing vegetative matter. The average sample depth was 35 cm. Soil samples were collected and placed in a plastic sample bag. A wet strength sample tag with a unique sample number was placed in the bag.

11.1.2 Trench Samples

Continuous channel samples were taken from the floor of all trenches that intersected saprolite or weathered bedrock. Each channel was approximately 10 centimetres wide and 5 centimetres deep. Sample intervals were determined by geological features: in homogeneous rock, the maximum sample interval was 1 metre, and the minimum sample interval employed was 0.3 m. Veins, altered zones, or distinct geological units were sampled so that the contacts were a standard 2 cm within the sample boundaries. Sample weights varied between 3 and 6 kilograms.

11.1.3 Stream Sediment Samples

The stream sediment sampling technique employed by Twangiza Mining involved collecting the fine sediment fraction from the top of the stream bed. The geologist took at least six 250 g sub-samples from within 10 m upstream and downstream of the starting point. The sub-samples were all placed in the same plastic sample bag, taking care to remove pebbles and organic debris; any excess water was decanted. The bag was then sealed with a cable tie and placed inside a second plastic bag. The sample ticket was placed inside the second bag, and the second bag sealed.

11.1.4 Drill Core Samples

Twangiza Mining has internal guidelines and documentation which relate to the sampling of drill hole core samples. SRK (UK) has reviewed the procedures and finds the methodology used to be acceptable for the collection of representative samples. The entire length of each drill hole was sampled, sample intervals were determined by geological features. In homogeneous rock, the maximum sample interval was 1 m. The minimum sample interval was 0.3 m. 1 m of split HQ core provides approximately 4.3 kg of material for analysis. Samples were cut at site and shipped to Banro's Bukavu sample preparation laboratory for processing. The sample pulps were then sent to the SGS laboratory in Mwanza, Tanzania to be analysed for gold by 50 g fire assay.

11.1.5 Auger Samples

Holes were drilled using "window sampling" auger tubes, powered by an Atlas Copco percussion hammer. The diameter of the upper sample was 64 mm, progressively reducing with depth to 49 mm, 39 mm and 29 mm. The maximum hole depth was 7 m. Samples were logged and photographed whilst in the window sampling tube, and then removed from the tube and bagged for gold analysis. The logging and sampling principles employed were the same as those used for diamond drill holes.

11.2 Sample Preparation and Analysis

11.2.1 Sample Preparation

During the resource definition drilling phase of work, a DRC subsidiary of Banro ran its own sample preparation facility in Bukavu, DRC using its own full-time employees. ALS Chemex, Johannesburg built the sample preparation facility and trained Banro staff. The facility was commissioned in September 2005. Analysis of samples was undertaken by the SGS laboratory in Mwanza, Tanzania while Genalysis in Western Australia served as the umpire laboratory. Both SGS and Genalysis are internationally (NATA) accredited and utilize conventional sample preparation, sample analysis and associated quality control protocols.

The preparation of soil samples is independently carried out to avoid possible contamination from the higher-grade trench, adit or core samples.

Samples were delivered to the Banro sample preparation facility in large white bags that hold between 20 and 30 kg of samples. Each shipment between the field and Bukavu had a covering dispatch form that was filled out in triplicate. Two copies were sent to Bukavu with the samples and one remained at the project site. If there was any discrepancy between the sample numbers and/or the number of samples recorded on the sample sheets and those samples physically received at the Bukavu sample preparation laboratory, the problem was immediately dealt with via HF radio communications and a reconciliation report was sent by mail to the Senior Project Geologist.

The in-house sample preparation facility is a containerised laboratory specially designed by ALS Chemex but managed by a subsidiary of Banro with periodic laboratory audits carried out by external consultants.

The in-house sample preparation facility comprises an electric oven, two jaw crushers, three disc pulverisers and air compressor system all assembled in one ‘20 footer’ and one ’40 footer’ steel container.

All samples received from the field were sorted and oven dried in labelled steel pans to optimise the resident drying time. Using the jaw crushers, all adit, trench and drill core samples were crushed to 80% passing a 2 mm screen. The crusher was thoroughly cleaned between any two samples. After every 10th sample, the crusher was flushed with barren granite, and the pulveriser was cleaned with similar material between each sample. The cleaning process was enhanced by the use of compressed air after each sample.

The crushed sample was split using a riffle splitter to produce 800-1,500 g of material, which was pulverised using B2000 Low Chrome Bowls for 90 to 300 seconds depending on the hardness of the sample. The samples were pulverised to 90% passing a 75 micron screen.

The sample preparation laboratory has organised areas/shelves designed for the storage of coarse and pulp rejects such that the samples can be retrieved in a reasonable amount of time.

Pulp samples, of approximately 150 g, are placed in brown packet envelopes, which in turn are placed in a rectangular cardboard box that holds approximately 20 pulp samples. These boxes are shipped in batches to the assay laboratory.

11.3 Laboratory Analysis

Prior to 2005, limited adit re-sampling by CME using ACME in Vancouver confirmed the repeatability of the original MGL adit sample assay results determined by their laboratory in Kamituga.

Three laboratories have been used by Banro for sample assaying since the commencement of exploration in Twangiza in October 2005. The initial soil geochemical samples, adit re-samples and trench samples were analysed by ALS Chemex, Johannesburg at which time SGS-Mwanza served as the umpire laboratory.

For resource drilling Banro used SGS-Mwanza as the principal analytical laboratory and Genalysis-Perth as the umpire laboratory.

All gold analyses have been carried out using conventional 50 g charge fire assay with atomic adsorption spectrography (AAS) finish. The three laboratories involved have carried out internal checks, which in the case of SGS-Mwanza are detailed in the section on quality control.

11.4 Quality Control Procedures

In order to monitor the integrity of the sample preparation and analytical data screen tests of crushed (5%) and pulverized (10%) samples were routinely carried out to monitor the particle size and percentage passing the 2 mm and 75 micron screens respectively.

To provide a measure of accuracy, precision and confidence, a range of international reference materials, duplicates and blanks were routinely but randomly inserted into each batch of samples at a frequency of 12%. Blank samples were inserted during the routine crushing and pulverising processes. Blanks were inserted into sample batches at a frequency of 1 in 50 and a crush duplicate split was submitted 1 in 50. Standard reference materials were inserted at a frequency of 4 in 50. International reference materials were predominantly sourced from Rocklabs Limited, New Zealand and occasionally from Geostats Pty Limited, Australia.

11.5 Assessment of Quality Control Data

Quality control procedures have been implemented in all stages of the sample preparation and analytical process. The quality control database is currently stored in a series of Century System database which were provided to SRK (UK) for analysis. The quality control work included the insertion of international reference samples, inter-laboratory checks, sample preparation laboratory duplicates, blanks, and the analytical laboratory's internal checks. These are all described in detail in the following sections.

11.5.1 Certified Reference Material

Four certified reference material samples (standards) were inserted in each batch of 50 samples. The standards were sourced from Rocklabs Limited, New Zealand and a few came from Geostats, Australia. The standard samples are in pulp form and are supplied in plastic containers of 2.5 kg each, of both oxide and sulphide material with variable grade ranges covering the expected grade range for the Twangiza deposit. A total of seven different grade ranges of standards have been randomly inserted into batches of samples submitted to the analytical laboratory during the latest drilling programme; these cover the entire grade range and include both oxide and sulphide material.

The standards are randomly inserted using the same quantity and sachets as the laboratory pulps, making them difficult to be detected by the analytical laboratory. Statistical assessment of the results is completed routinely using Twangiza Mining's internal guidelines and not using the parameters provided as part of the Rocklabs Quality Control package. The mean assay values returned by the laboratory for all certified reference material in relation to their respective values are considered to be within acceptable limits. Results reporting out of assigned limits trigger a re-assay of that batch of samples.

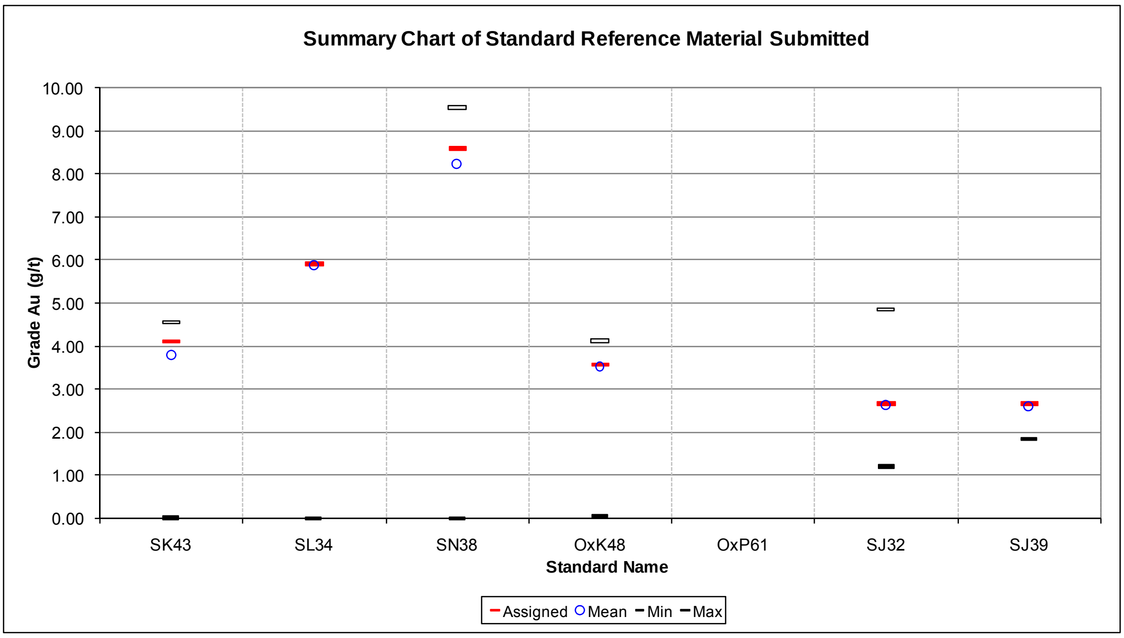


Figure 11-1: Summary of Returned Assays per Standard

Table 11-1: Statistics of Results of Standard Certified Sample Submissions

STD REFID	REF VALUE (g/t Au)	NUMBER OF SUBMISSSIONS	MIN (g/t Au)	MAX (g/t Au)	MEAN (g/t Au)	STD DEV	CoV
SP37	18.14	26	16.9	20.40	17.75	0.76	0.04
SN38	7.92	33	0.01	9.38	7.92	2.21	0.28
SL51	5.909	44	4.24	6.67	5.85	0.35	0.06
SL34	5.86	400	0.01	15.00	5.86	0.73	0.12
SK52	4.107	42	0.41	4.54	4.01	0.61	0.15
SK43	4.00	59	0.09	4.44	3.90	0.64	0.16
SK33	4.00	479	0.02	4.45	3.97	0.25	0.06
SI54	1.761	38	1.63	2.00	1.80	0.09	0.05
SI42	1.761	54	1.03	1.99	1.78	0.12	0.07
SH41	1.344	6	1.16	1.35	1.28	0.06	0.05
SG40	0.976	15	0.83	1.11	0.98	0.06	0.06
SE44	0.606	42	0.55	0.65	0.61	0.02	0.03
OXP61	14.92	35	13.9	15.80	15.00	0.39	0.03
OXN62	7.706	42	6.76	8.33	7.77	0.31	0.04
OXL63	5.865	8	5.62	6.27	5.94	0.21	0.04
OXL40	1.82	33	1.05	1.99	1.85	0.15	0.08
OXK69	3.583	31	3.24	3.73	3.58	0.10	0.03
OXJ68	2.342	55	2.22	2.55	2.35	0.07	0.03
OXJ64	2.366	36	2.18	2.72	2.35	0.09	0.04
OXI54	1.868	8	1.78	1.87	1.83	0.03	0.02
OXI 67	1.817	63	1.52	2.05	1.83	0.07	0.04
OXH55	1.282	13	1.05	1.35	1.28	0.09	0.07
OXH52	1.31	72	0.01	1.58	1.27	0.24	0.19
OXG60	1.07	8	0.99	1.16	1.07	0.06	0.06
OXG46	1.037	5	0.97	1.09	1.03	0.05	0.05
OXF65	0.805	39	0.74	0.94	0.83	0.03	0.04
TOTAL		1686					

A review of the charts shows the laboratory historically produced accurate assays, however in the later stage of the drilling programme the results were more variable and there was a low bias but grades remain within acceptable limits. The limits shown on the charts are based on the certified standard deviation with the upper and lower limits set at three times the standard deviation value. A summary of the standard used in submissions from the latest round of drilling are shown in Figure 11-1 and Table 11-1.

11.5.2 Inter-laboratory check assays

Twangiza Mining completed inter-laboratory check assays with the same assay techniques on a routine basis. Sample pulp duplicates were sent at the end of every quarter in batches of between 80 and 100 samples for analyses at both SGS-Mwanza and Genalysis-Perth. The pulp samples cover a representative grade range.

Summary statistics for the two datasets have been determined and the Mean Relative Differences (MRD) of the SGS and Genalysis result have been calculated. Above 1.0 g/t Au there is no significant bias with the mean relative difference (MRD) reporting between -2.6 % and 4.3 %.

11.5.3 Duplicate coarse split

Before the 2009 Feasibility Study, a total of 1,377 split duplicates were included in routine sample batches. These are statistically reviewed in Table 11-2 below. After the 2009 Feasibility Study a total of 158 split duplicate were submitted, these are summarised in Table 11-3.

The data is plotted in Figure 11-2. There is generally close agreement although some pairs show considerable variance indicating coarser gold in a minor proportion of the samples or some potential for sample swaps during submission and analysis.

Table 11-2: Coarse Split Sample Pairs pre FS

	ALL SAMPLES		>0.5g/t Au	
	ORIGINAL g/t Au	DUPLICATE g/t Au	ORIGINAL g/t Au	DUPLICATE g/t Au
NUMBER OF PAIRS	1377	1377	274	274
MINIMUM	0.004	0.005	0.50	0.01
MAXIMUM	47.70	44.20	25.3	22.4
MEAN	0.57	0.55	2.07	1.97
STANDARD DEVIATION	2.39	2.43	2.80	2.53

Table 11-3: Coarse Split Sample Pairs Post FS

	ALL SAMPLES		>0.5g/t Au	
	ORIGINAL g/t Au	DUPLICATE g/t Au	ORIGINAL g/t Au	DUPLICATE g/t Au
NUMBER OF PAIRS	158	158	16	16
MINIMUM	0.005	0.005	0.52	0.51
MAXIMUM	2.99	3.15	2.99	3.15
MEAN	0.18	0.17	1.31	1.20
STANDARD DEVIATION	0.45	0.41	0.89	0.79

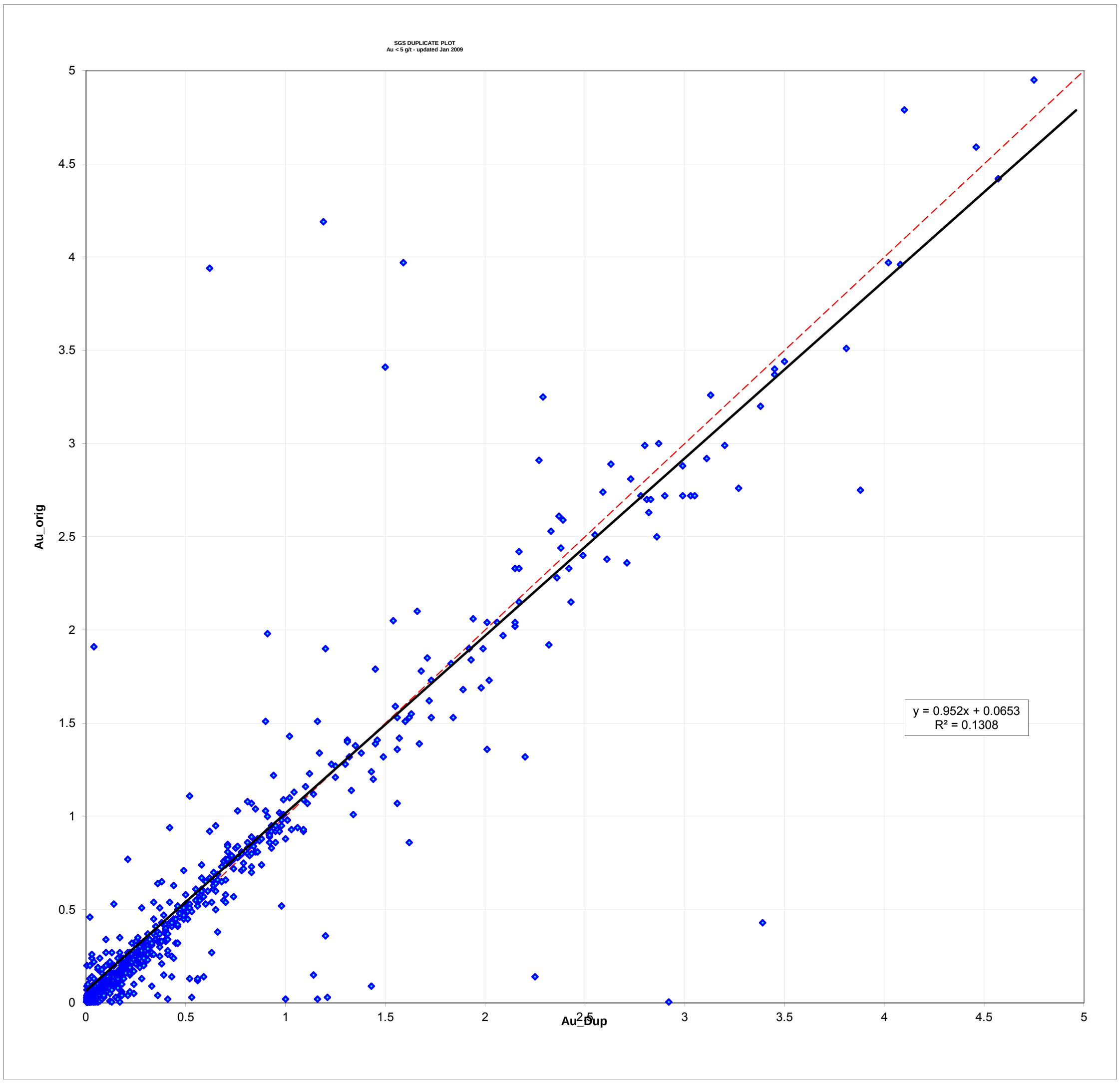


Figure 11-2: Original vs. Coarse duplicate splits - all samples

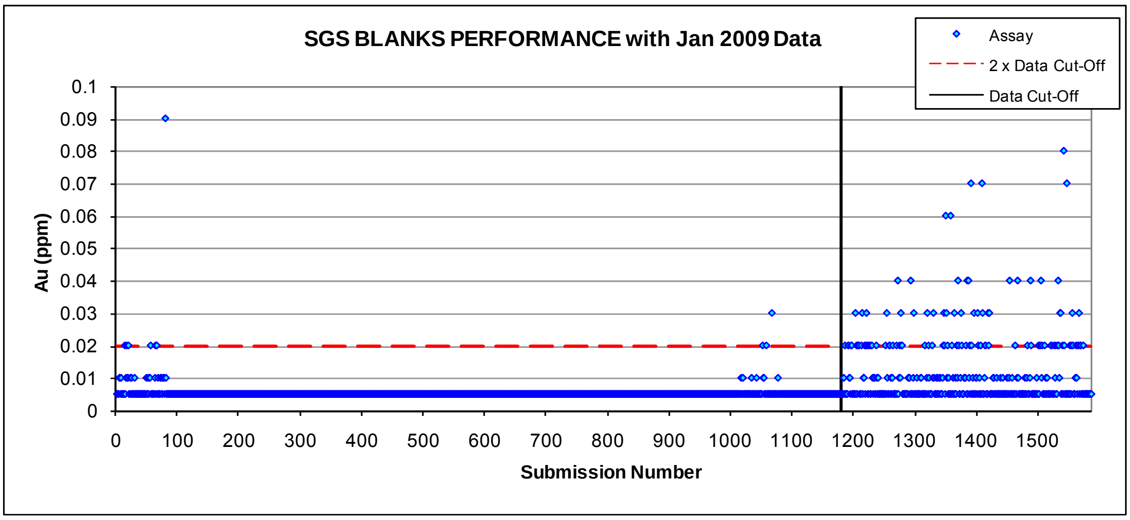


Figure 11-3: Summary of All Blank Submissions to SGS Mwanza

11.5.4 Blank samples

The blank material is made from granite and was purchased from ALS Chemex - Mwanza laboratory. There were 1587 blank sample submissions which returned a mean value of 0.007 g/t; these are shown in Figure 11-3. The more recent results suggest there was a minor contamination issue; the grouped nature of slightly high results may indicate periods in which the routine cleaning the equipment between samples was not undertaken thoroughly. Overall this is not considered to be an issue for the data quality.

12 DATA VERIFICATION

12.1 Database and Data Quality

Table 12-1 below shows the data type and amount available for geological modelling and Mineral Resource Estimation.

Table 12-1: Data Type and Amount of Data Employed In Current Geological Modelling

DATA TYPE	NUMBER	TOTAL LENGTH	NO OF SAMPLES
HISTORICAL TRENCHES (MGL)	71	8,323	4,088
HISTORICAL ADIT (MGL)	202	13,207	13,716
1997/98 DIAMOND HOLES (CME)	20	9,122	8,557
RE-SAMPLED ADITS (CME)	16	1,689	1,613
FOLLOW-UP	6	966.4	624
FOLLOW-UP (PHASE 1) DD HOLES	33	6,421	6,881
FOLLOW-UP (PHASE 2) DD HOLES	38	10,616	11,194
FOLLOW-UP (PHASE 3) DD HOLES	29	6,835.78	7,602
FOLLOW-UP (PHASE 3 cont) DD HOLES	116	22,114	24,534
FOLLOW-UP (PHASE 4 cont) DD HOLES	102	21,952.26	20,004
FOLLOW-UP (PHASE 5)	138	12,557	8,676
FOLLOW-UP (PHASE 5 Cont.)	11	499.90	186

12.2 Adit Check Sampling

In the 1960’s approximately 13,207 metres of adits were developed and sampled by MGL. A programme of data verification was completed by CME by re-sampling 1,613 of the original 13,716 adit channel samples recorded by MGL. In total, 16 adits (including crosscuts) were rehabilitated, mapped and channel sampled at one metre intervals. A total of 1,613 channel samples were collected from both walls of the adits. A graphical illustration of gold results for the MGL and CME sampling is provided in Table 12-1.

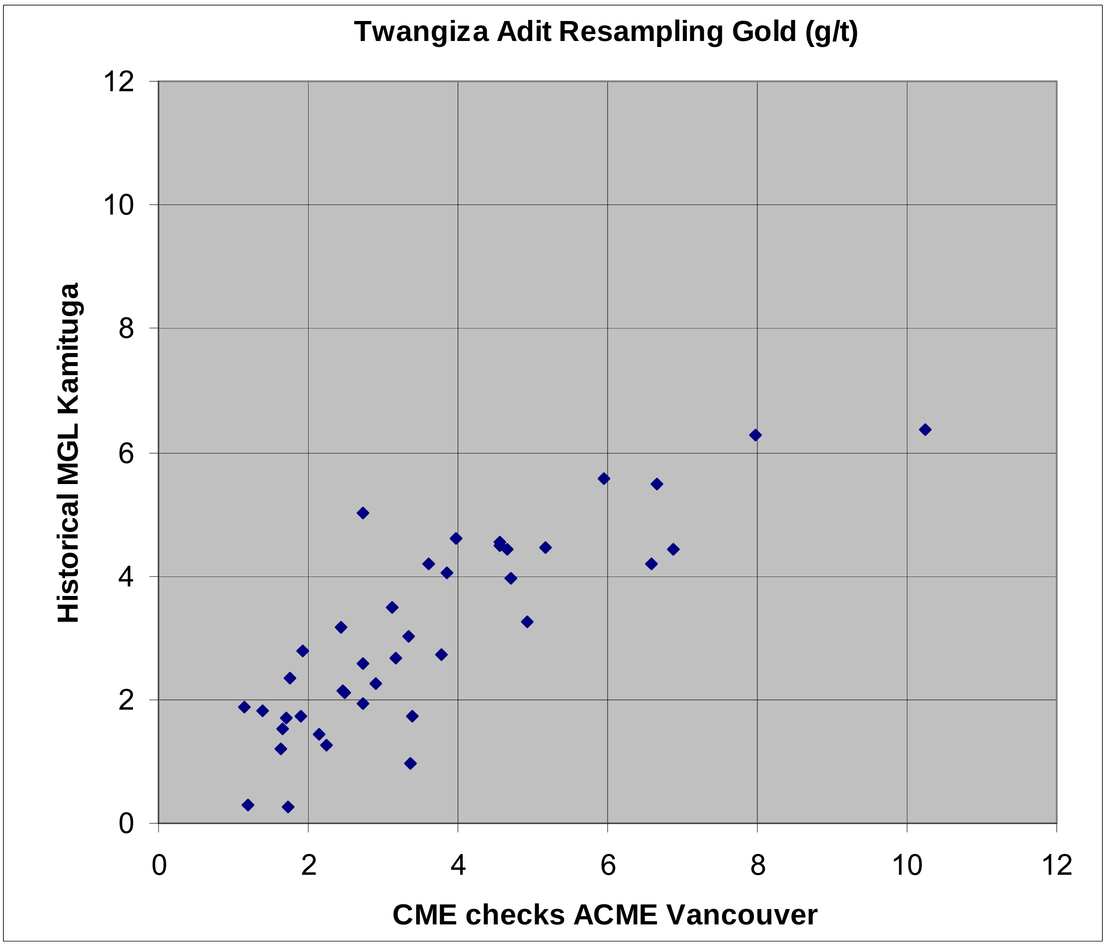


Figure 12-1: Comparison of Acme Vancouver vs. MGL Kamituga Laboratories

Composites of significant intersections using a 1 g/t Au cut-off based on CME's re-sampling are presented in Table 12-2 below.

Table 12-2: Adit sampling - statistics of composites

COMPANY	NO OF COMPOSITES	MIN (g/t Au)	MAX (g/t Au)	AR. MEAN (g/t Au)	STD. DEV (g/t Au)	WEIG-MEAN (g/t Au)
MGL	40	0.25	6.37	3.05	1.57	3.76
CME	40	1.15	10.24	3.59	1.99	3.45

MGL and CME results over the same interval compare reasonably well; overall CME results are slightly higher than the MGL results.

12.3 Data Validation

The primary sample data files were validated in several ways:

- visually checking summary statistics of all data fields to ensure they do not exceed reasonable limits and unrecognized rock codes;
- passing the data through a general validation process that identifies, amongst others duplicate samples, reversed and over-length intervals as well as overlapping sample limits; and
- plotting the samples in plan and in section and comparing the sample locations and assay values.

A number of transcription errors were originally found in the database due to sample numbers entered in the gold column. The error was reported to Twangiza and the files corrected accordingly.

SRK (UK) has reviewed logging procedures on site and the electronic files provided by Twangiza at the time of the resource estimate. SRK (UK) noted the lower quality in the returned assays during the third quarter of 2008 but nevertheless considers the data to be valid and therefore suitable for the Mineral Resource estimate.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Background

Scoping study metallurgical tests were conducted on oxide, transition and fresh, non-refractory, ore types from the Twangiza Main and North ore bodies. The testwork evaluated a number of process options to recover gold including flotation, gravity and cyanidation processes. Comminution tests were also performed. The scoping study results were used to develop the initial process flow sheet for the Twangiza ore body. The scoping study testwork results are contained in the following reports:

- September, 2007, SGS, “Laboratory Testwork: Scoping Study – Banro Twangiza Project (Democratic Republic of Congo)”, Report # MET 07/U82 TWA;
- October 5, 2006, SGS, “Procedural Diagnostic Leach Appraisal on 11 Gold Bearing Ore Samples”, Report No. MET 06/S16;
- April 20, 2007, SGS, “Bulk Rougher Flotation Testwork on Twenty Three Gold Ore Samples”, Report No. Flotation 07-412;
- July 3, 2006, SGS, “Flotation Testwork on a Gold Bearing Ore Blends from Banro-Twangiza Mine”, Report No. Flotation 07/132;
- June 28, 2007, SGS, “Mineralogical Characterization and Gold Deportment Study on Five Gold Ore Samples from the Twangiza Gold Deposit, DRC”, Mineralogical Report No. MIN 0507/066;
- October 3, 2007, SGS, “Bio-oxidation Test Program on Twangiza Concentrate”, Report No. BIOMET 07/06;
- October, 2007, Geobiotics, “Phase 1, Report for GEOCOAT® Amenability Test”; and
- April 22, 2008, SGS, “Cyanide Destruction Testwork on Twangiza North and Main Composite Middlings and Tailings”, Report No. BIOMET 08/09.

As part of the pre-feasibility study SENET initiated and supervised additional metallurgical testwork to further investigate and improve gold recoveries and reagent consumptions. These results were incorporated in to the pre-feasibility study. A list of the reports prepared by a number of laboratories is provided below:

- April 22, 2008, SGS, “Cyanide Destruction Testwork on Twangiza North and Main Composite middlings and tailings”, Report # BIOMET 08/09;
- May 26, 2008, SGS, “A General Mineralogical and Gold Deportment Study on two Gold bearing ore Samples from the Twangiza Project”, Report # MIN 0208/038a;
- June 2008, Knelson Concentrators Africa (Pty) Ltd, “Extended Gravity Recoverable Gold (EGRG) Testwork on Two Ore Samples from the Twangiza Project submitted by SENET (Pty) Ltd” Report No. SENET/03/08;
- July 2008, Knelson Concentrators Africa (Pty) Ltd, “Modelling the Proposed Gravity Gold Recovery Circuit at the Twangiza Project” Report No. SENET/03.2/08; and
- July 2008, Paterson and Cooke, “Twangiza Project: Results of Bench-Top Thickening and Rheology Testwork”, Report No. SEN-TWA-8144R01 Rev 0.

Further metallurgical tests were conducted to raise the level of confidence to feasibility level and results of these tests are contained in the reports listed below:

- July 4, 2008, SGS, “Metallurgical Test Programme for Gold Bearing Ore Sources (Twangiza Non-Refractory FP Samples”, Report No. Met/08/131 Rev2;
- August 07, 2008, SGS, “Cyanide Destruction Testwork on Twangiza Main Fresh and Transitional Composite Middlings and Tailings”, Report No. BIOMET 08/15;
- August 08, 2008, SGS, “Cyanide Destruction Testwork on Twangiza North Fresh and Transitional Composite Middlings and Tailings”, Report No. BIOMET 08/18;
- August, 2008, Knelson Concentrators Africa (Pty) Ltd, “Extended Gravity Recoverable Gold (EGRG) Testwork on Two Main Ore Body Fresh and Transition Ore Samples from the Twangiza Project submitted by SENET (Pty) Ltd” Report No. SENET/05/08;
- August, 2008, Knelson Concentrators Africa (Pty) Ltd, “Modelling the Proposed Gravity Recovery Circuit at the Twangiza Project Main Ore Body Hard Ores” Report No. SENET/05.1/08;
- August, 2008, Knelson Concentrators Africa (Pty) Ltd, “Extended Gravity Recoverable Gold (EGRG) Testwork on Two North Ore Body Fresh and Transition Ore Samples from the Twangiza Project submitted by SENET (Pty) Ltd” Report No. SENET/06/08;
- August, 2008, Knelson Concentrators Africa (Pty) Ltd, “Modelling the Proposed Gravity Recovery Circuit at the Twangiza Project North Ore Body Hard Ores” Report No. SENET/06.1/08;
- November 20, 2008, SGS, “Metallurgical Test Programme: SENET Project – Twangiza Oxide”, Report No. Met07/BB27;
- December 10, 2008, Mintek, “Twangiza Comminution Test Work Report”, Report No. MPC-584;
- January, 2009, Orway Mineral Consultants (OMC), “Twangiza Gold Project Comminution Circuit Design”, Report No. 8282; and
- February 10, 2009, SGS, “Metallurgical Test Programme: SENET Project – Twangiza Non-Refractory Variability Samples”, Report No. Met07/BB27.

The summarised findings of the testwork are given in Table 13-1.

Table 13-1: Ore Characteristics

ITEM	UNIT	OXIDE		TRANSITIONAL	
		MAIN	NORTH	MAIN	NORTH
ORE CHARACTERISTICS					
ORE HEAD GRADE (AU)	g/t	2.42	2.1	1.52	2.36
SPECIFC GRAVITY	t/m^3	2.79	2.88	2.79	2.99
UNCONFINED COMP STRENGTH	MPa	14.88	8	22.63	10.56
BOND CRUSHER WORK INDEX	kWh/t	3.1	2.7	9.41	6.54
BOND CRUSHER WORK INDEX	kWh/t	4.3	3.6	12.1	8.1
ROD MILL WORK INDEX	kWh/t	6.8	5.3	8.06	4.05
BALL MILL WORK INDEX	kWh/t	3.2	4.23	8.61	6.88
ABRASION INDEX	Ai	0.0078	0.034	0.196	0.1932
LIFE FACTOR		9.16	10.87	2.19	2.19
JK TECH PARAMETERS					
A		85.9	85.9	55.1	65.6
b		1.25	1.25	2.49	5.22
T_a		1.14	1.14	0.96	3.38
GRAVITY					
GRAVITY RECOVERY	% of h/g	16.5	36.4	11.8	41.6
GRAVITY CONCENTRATION RATIO		1,100	2,500	1,100	1,100
CONC.MASS AS % OF FEED TO CONC	%	0.05	0.05	0.05	0.05
INTENSIVE LEACH DISSOLUTION		98	98	98	9
CIL					
LAB LEACH DISSOLUTION	%	91.1	89.5	80.0	91.7
EFFICIENCY FACTOR	%	98.0	98.0	98.0	98.0
CIL DISSOLUTION	%	89.3	87.7	78.4	89.9
LEACH SOLIDS FEED % m/m	%	35	35	42	42
CYANIDE CONSUMPTION	kg/t	0.06	0.63	0.98	0.65
LIME CONSUMPTION AS 100%	kg/t	4.31	4.70	2.26	1.27
TOTAL LEACHING RESIDENCE TIME	hrs	15	15	24	24
CARBON TO GOLD LOADING RATIO		2,500	2,500	1,750	1,750
GOLD PRODUCTION					
OVERALL RECOVERY	%	90.2	91.2	79.5	93.2
DETOXIFICATION					
REQUIRED RESIDENCE TIME	hr	2	2	2	2
RESIDUAL CYANIDE IN CIL TAILS	ppm	90	90	90	90
LIME CONSUMPTION AS 100%	kg/t CaO	0.06	0.06	0.06	0.06
SMBS USAGE RATE / RESIDUAL CYANIDE	g/g NaCN	4.44	4.18	4.57	4.62
$CUSO_4$USAGE RATE / RESIDUAL CYANIDE	g/g NaCN	0.48	0.48	0.11	0.15

13.2 Sample Selection

All samples were selected and supplied by Twangiza according to SENET's requests.

13.3 Review of Scoping Study Metallurgical Testwork

This section summarises the results of the metallurgical tests performed by SGS in 2007. Sample sourcing and make-up test methods and detailed results are contained in the reports from SGS referenced above and are part of the scoping study technical report produced by SENET dated 13th September, 2007 and entitled "Preliminary Assessment NI 43-101 Technical Report, Twangiza Gold Project, South Kivu Province, Democratic Republic of Congo".

The main findings from the SGS scoping study metallurgical tests were as follows:

- Bond Ball Mill Work indices (BBWi) obtained ranged from about 3.2 – 13.2 kWh/t metric. It was noted that the oxides were in the in the very soft category and transition and fresh were in the medium band;
- The abrasion results ranged from 0.0096 to 0.66 which showed that the oxides were in the low abrasive category whilst the transition and fresh was in the medium to high abrasive classes;
- Diagnostic leach tests indicated that gold in oxides was amenable to direct cyanidation while transition and fresh samples displayed that some of the gold could easily be recovered with direct cyanidation technique and some could hardly be recovered using this method;
- Mineralogical investigations on transition and fresh samples revealed that a fair proportion of the "not so easy" to recover gold was associated with arsenopyrite and pyrite and that only 30-60% of the gold would be amenable to direct cyanidation, a phenomenon that was proved in the later stages of the scoping study testing;
- The oxides (main and north) responded well to gravity and leach recovery testwork with overall recoveries of 90% being attained;
- Bottle-roll tests conducted on transition and fresh ore samples with excess cyanide indicated that there were distinct lithologies within each ore type and each lithology responded differently to direct cyanidation. Typically the gold dissolution achieved was:
    - Transition FP – 81%
    - Transition CMS – 38%
    - Fresh FP – 91.8%
    - Fresh CMS – 54%

These results led to the investigation of an alternative processing methods utilising flotation followed by separate treatment by cyanidation of the flotation concentrates and tailings of all lithologies. In general the following results were achieved:

- Generally >95% of the gold was recovered into the concentrates in all instances;
- The mass yield to flotation concentrates were generally high; ranging between 30 and 50%. In instances where low mass yields were obtained (10-15%), low gold recoveries were also noted;

- For CMS, it was noted that there was no significant improvement in gold dissolution from concentrates and flotation tails compared to dissolutions obtained on the “as Received” samples; and
- For FP, it was noted that gold dissolution remained the same from the leaching of the concentrates and flotation tailings compared to dissolutions obtained on the “as Received” samples.

Based on poor gold recoveries, seen for both flotation concentrates and tailings from CMS material, bio-oxidation was investigated as an alternative treatment route. Bio-oxidation resulted in a sulphide oxidation of 95% after 29 days and final gold recovery of 88% from the oxidised residue. This represented a 30% improvement in gold recovery over direct cyanidation. While the tests demonstrated that the ore was amenable to bio-oxidation, the high mass pulls (between 30 and 50%) coupled with the long residence time meant that heap leach of the concentrates using a process called GEOCOAT® was the only viable option. However, due to real estate limitations at the Twangiza site this option was “shelved”.

13.4 Review of Pre-Feasibility Study Metallurgical Testwork

Following a review of the scoping study metallurgical test results, a test programme for the pre-feasibility study was implemented and was divided into two phases, namely:

Phase 1: Optimization tests aimed at establishing optimum parameters for gold recovery and characterization of the comminution properties;

Phase 2: Variability testwork aimed at testing the optimum conditions / parameters on variability samples.

The following is a summary of the pre-feasibility study testwork findings and detailed results are contained in the reports from SGS, Knelson Concentrators Africa and Paterson and Cooke referenced above and are part of the prefeasibility study technical report produced by SENET dated August 13, 2008 and entitled “Pre-feasibility Study NI43-101 Technical Report, Twangiza Gold Project, South Kivu Province, Democratic Republic of Congo”.

13.4.1 Mineralogy

The mineralogical testwork was conducted on the main and north oxide composite samples. No additional mineralogy was conducted on transition and fresh ore types as the work conducted during scoping study phase was deemed to be adequate. The main findings were:

- The Twangiza north oxide ore is characterised by coarse (mostly >75μm ECD) liberated gold (>99%) and gravity gold recovery could be an option for this type of ore; and
- Twangiza main oxide ore is characterised by finer-grained gold (all <75μm ECD) with a substantial proportion locked in gangue minerals (~20%) which could potentially exhibit recovery problems. Both Twangiza main and north oxide ore contained considerable amounts of clay minerals that could be detrimental to different parts of the process such as mill selection, thickening and equipment sizing such as pumps, agitators and inter-stage screens.

13.4.2 Comminution results of composite samples

The following are main findings from the comminution tests performed on the composite samples:

- The Bond Rod Mill Work Indices (BRWi) for main oxide and north oxide was classified as soft;
- The abrasion indices (Ai) for the oxides were all <0.05 which classified them as very low abrasive ores. The transition and fresh abrasion indices were classified as moderately abrasive. This indicates that medium grinding media consumptions and liner wear will be expected;
- The Bond Crushability Work Indices was only performed on the oxide and showed the ore to be very soft; and
- The UCS results for the oxides indicated that the ore is very soft (<50MPa) and should be treated using a mineral sizer.

13.4.3 Comminution results of variability samples

Variability comminution testwork included BBWi determinations only and the results are summarized below.

The BBWI results showed that the main and north oxides exhibited very soft to soft behaviour (1.8-7.4kWh/t);

The main transition FP non refractory samples were in the medium hardness category (9.1-14.5 kW/h) and north transition FP non refractory in the medium to hard class (10.7-18.2 kWh/t);

The Bond ball work indices of main fresh FP non refractory indicated medium hardness (10.8-13.4 kWh/t) whilst the north fresh FP non refractory showed a range of medium to hard (14.3-16.2 kWh/t).

The results obtained from the variability testwork showed that the ore is variable in nature and this should be taken into account when designing the comminution circuit.

13.4.4 Gold recovery testwork

Optimisation gold recovery testwork was performed on the main and north oxide composite samples during the scoping study. The optimum conditions obtained during these tests were used to evaluate the variability of the ore.

No optimisation gold recovery tests were performed on the non-refractory transition and fresh samples during the scoping study. It was therefore necessary to perform these tests during the pre-feasibility study.

13.4.5 Gravity and intensive cyanidation testwork (oxides only):

Twangiza main oxides displayed less gravity recoverable gold than Twangiza north oxides which is in line with mineralogical investigations. However in both cases, the recoveries warranted the installation of a centrifugal concentrator to recover the free gold ahead of CIL.

Twangiza main oxides obtained a GRG (gravity recoverable gold) recovery of 28.7% which equated to a predicted plant gravity recovery of 20.9%.

Twangiza north oxides achieved a higher GRG recovery of 49.7% which equated to a predicted plant gravity recovery of 43.4%.

These gravity recoveries, obtained for the oxides, were in line with the recoveries obtained during the scoping testwork phase.

Intensive cyanidation testwork, performed on the gravity concentrate gave gold dissolution >95% in less than 24 hours irrespective of the concentrate grades. This is an indication that concentrates will be easily leached during plant operations. An intensive cyanidation reactor followed by electrowinning was therefore considered as part of the gravity gold recovery system.

13.4.6 Variability leach testwork (oxides only)

The calculated head grade of main oxide varied from 1.99 g/t to 5.04 g/t with minimum and maximum recoveries of 83.66% and 88.51%. This showed that this ore body, when leached, obtained similar recoveries irrespective of head grade.

Cyanidation of north oxide variability samples with head grades ranged from 0.74 g/t to 1.97 g/t resulted in gold dissolutions between 70.12% and 89.77%. The varying range in the dissolution demonstrated that this ore is highly variable which could be an indication of presence of some refractory material in the oxides.

The sodium cyanide consumption for the oxides was generally less than 1 kg/t which was much higher than that obtained during the scoping study which could be an indication of the variable nature of cyanide consumers associated with the oxide ore. Lime consumption was generally very high however still comparable to what was achieved during the scoping study, for all samples that were tested.

13.4.7 Cyanide Detoxification (Oxides Only)

Bench scale cyanide destruction testing was performed on composite samples of oxide leach tailings generated from metallurgical testwork.

The relative effectiveness of ferrous sulphate, sodium metabisulphite, sodium metabisulphite plus copper, hydrogen peroxide and alkaline chlorination at removing or destroying soluble cyanide forms in the tailings were compared and results showed that the alkaline chlorination and ferrous sulphate gave the best cyanide destruction results. However this method was not selected due to environmental concerns. The use of sodium metabisulphite and copper was chosen as the effective method, with 82% cyanide destruction.

13.4.8 Transition and fresh (main and north) optimisation recovery testwork

Bulk gravity tests were performed on main and north (transition and fresh) non-refractory composite samples to produce middlings and tailings for the leach optimisation testwork and gravity concentrate for the intensive cyanidation testwork. The main findings were:

Gravity - The composite samples showed gravity recoveries varying from 4.5 to 21%, with mass yields ranging from 0.33 to 0.5%. These test results are inconclusive and additional EGRG (Extended Gravity Recoverable Gold) tests were scheduled to be performed during the feasibility testwork phase;

Intensive Cyanidation - Intensive cyanidation tests were performed on the primary gravity concentrates and relatively high gravity concentrate dissolutions (>90%) were obtained for the transition main and north samples. Approximately 85% gold dissolution was obtained for the fresh main and north gravity concentrates which could be an indication of the presence of some refractory material in the concentrates. Overall, the concentrates responded well to intensive cyanidation;

Preg-robbing - The preg-robbing test results showed that the transition and fresh ores displayed preg-robbing characteristics which is in line with the mineralogical findings conducted during scoping study phase. All further tests were therefore conducted using the CIL treatment route;

Effect of Grind - It was noted for all ore types that there was an increase in dissolution with finer grinding and an optimum grind of 80% -75 µm was chosen which is in line with the selected grind for oxides. This grind was used for all subsequent testwork;

Effect of cyanide addition - Transition and fresh (main and north) recoveries increased with increase in cyanide addition from 1.0 to 3.0 kg/t. However the increase in cyanide consumptions and hence costs, erodes the benefits associated with extra recovery. It was recommended that cyanide addition rates be maintained at 2 kg/t; and

Effect of Time and Oxygen - a small increase in dissolution was noted with an increase in leach time above 24 hours. However, 24 hours was selected as the leach time in order to reduce the number and size of the leach tanks.

13.4.9 Settling and viscosity

Settling testwork was conducted by Paterson and Cooke on oxides, transition and fresh ore samples obtained from Twangiza main and north ore bodies. The detailed report is entitled "July, 2008, Paterson and Cooke, "Twangiza Project: Results of Bench-Top Thickening and Rheology Testwork", Report No. SEN-TWA-8144R01 Rev 0". The main findings were:

- The main and north oxide mill feed slurries exhibited similar rheological characteristics and was classified as pastes at mass solids concentrations greater than 52% w/w. Slightly higher yield stress and viscosity values were achieved for the north oxide and it was found that due to the high viscosities, the addition of viscosity modifiers to the milling circuit might be required; and
- Whilst a thickener has not been included in the Twangiza design, flocculant addition and thickener sizing information is included in the Paterson and Cooke report should the results be required.

13.5 Review of Feasibility Study Metallurgical Testwork

Following the review of the scoping study and the pre-feasibility study metallurgical test results, SENET proposed a test program for the feasibility study with the following aims and objectives:

- To characterise the ore body further with respect to its comminution characteristics by performing SAG mill tests to support the flow sheet developed during the pre-feasibility study or to develop a new flow sheet for the feasibility study; and
- To determine the variability of the ore by using the optimum parameters achieved during the scoping and pre-feasibility testwork phases for gold recovery.

The results for the EGRG tests and cyanide detoxification tests, performed on main and north transition and fresh composite samples during the pre-feasibility testing phase, were only available after completion of the pre-feasibility study and are included as a summary in this section.

13.5.1 Gravity recovery – main and north transition and fresh samples

Knelson Africa was commissioned to perform GRG tests and simulation of the results using KCMOD*Pro model to predict circuit recovery and the result and their findings is contained in four reports noted below:

- August, 2008, Knelson Concentrators Africa (Pty) Ltd, “Extended Gravity Recoverable Gold (EGRG) Testwork on Two Main Ore Body Fresh and Transition Ore Samples from the Twangiza Project submitted by SENET (Pty) Ltd” Report No. SENET/05/08;
- August, 2008, Knelson Concentrators Africa (Pty) Ltd, “Modelling the Proposed Gravity Recovery Circuit at the Twangiza Project Main Ore Body Hard Ores” Report No. SENET/05.1/08;
- August, 2008, Knelson Concentrators Africa (Pty) Ltd, “Extended Gravity Recoverable Gold (EGRG) Testwork on Two North Ore Body Fresh and Transition Ore Samples from the Twangiza Project submitted by SENET (Pty) Ltd” Report No. SENET/06/08;
- August, 2008, Knelson Concentrators Africa (Pty) Ltd, “Modelling the Proposed Gravity Recovery Circuit at the Twangiza Project North Ore Body Hard Ores” Report No. SENET/06.1/08.

The results are summarized in the Table 13-2.

Table 13-2 Predicted Plant GRG Recoveries

ORE TYPE	GRG VALUE (%)	PREDICTED PLANT GRG RECOVERY (%)
MAIN TRANSITION	16.7	11.8
MAIN FRESH	11.6	6.6
NORTH TRANSITION	53.8	41.6
NORTH FRESH	36.1	26.5

Free gold recovery from the Twangiza main, transition and fresh ores, is problematical for gravity recovery as the GRG values are low, the F_{80} is coarse and the free gold is finely disseminated. These factors result in a difficult and low-value opportunity for gravity recovery of gold from these ores as can be seen by the results in Table 13-2.

In contrast, the Twangiza north transition and fresh ore samples, exhibited a more ideal situation for gravity recovery as the GRG values were high. Consequently, a gravity recovery stage was included for the Twangiza process plant.

13.5.2 Cyanide destruction

Bench scale cyanide destruction testing was performed on samples of leach tailings generated from metallurgical testwork. The samples were as follows:

- Main Transition ore.
- Main Fresh ore.
- North Transition ore.
- North Fresh ore.

SGS performed the cyanide destruction tests and the results and findings are contained in two reports listed below:

- August 07, 2008, SGS, “Cyanide Destruction Testwork on Twangiza Main Fresh and Transitional Composite Middlings and Tailings”, Report No. BIOMET 08/15; and
- August 08, 2008, SGS, “Cyanide Destruction Testwork on Twangiza North Fresh and Transitional Composite Middlings and Tailings”, Report No. BIOMET 08/18.

Two methods were used on the transition and fresh samples; sodium metabisulphite plus copper, and Caro’s Acid for removing or destroying soluble cyanide forms in the tailings. A summary of WAD Cyanide removal and reagent consumptions are shown in Table 13-3.

The WAD (Weak Acid Dissociable) cyanide removal efficiency for all samples was very low and this was due to the high WAD CN in the feed to the detoxification. The WAD cyanide removed ranged from 218 to 594 ppm which is higher than what will be expected in the plant. The cyanide destruction process will be designed to be run on a continuous basis but was performed batch wise at the lab and is less efficient and tend to require more reagents for efficient cyanide removal. The reagent consumptions are therefore exaggerated and should be lower.

Table 13-3: Summary of Cyanide Destruction Tests

TEST METHOD	UNIT	MAIN		NORTH	
		TRANS.	FRESH	TRANS.	FRESH
SMBS and $CuSO_4$					
START WAD CN (FEED TO TAILINGS)	ppm	744	793.6	345	322
END WAD CN (AFTER DETOX)	ppm	150.4	288	72	104
WAD CN REMOVED	ppm	594	506	273	218
WAD CN REMOVAL EFFICIENCY	%	79.78	63.71	79.13	67.7
g (SMBS) : g (WAD CN)	g:g WAD	4.57	5.73	4.62	5.41
g ($CuSO_4$) : g (WAD CN)	g:g WAD	0.11	0.15	0.19	0.25
CARO'S ACID					
START WAD CN (FEED TO TAILINGS)	ppm	744	793.6	345	322
END WAD CN (AFTER DETOX)	ppm	240	241.6	104	152
WAD CN REMOVED	ppm	504	552	241	170
WAD CN REMOVAL EFFICIENCY	%	67.74	69.56	69.86	52.8
g (CARO'S ACID) : g (WAD CN)	g:g WAD	57.9	6.2	56.2	74.3
g ($CuSO_4$) : g (WAD CN)	g:g WAD	0.1	-	-	-

13.5.3 Comminution appraisal

Comminution testwork was performed in two phases. The first phase involved performing the tests on individual composites of the transition and fresh ores types, main and north ore bodies. These results were analysed and the data was provided for the second phase to Orway Mineral Consultants (OMC), an internationally recognised specialist in grinding circuit design, for interpretation and mill sizing by comparison against their existing database.

Mintek carried out full bench scale comminution tests that included the standard bond work index (BBWi) tests, Advanced Media Competency Tests (AMCT) and JKTech drop weight testing. A summary of the results are shown in the Table 13-4 and Table 13-5. The detailed results can be found in the report by Mintek, "December 10, 2008, Mintek, "Twangiza Comminution Test Work Report", Report No. MPC-584".

Table 13-4: Summary of Bond Work Index Tests

SAMPLE	BBWi (kWh/t)	BRWi (kWh/t)	BRWi : BBWi	BBWi CLASSIF.
MAIN FRESH	10.53	9.37	0.89	MEDIUM
MAIN TRANSITION	8.61	8.06	0.94	SOFT
NORTH FRESH	13.39	14.1	1.05	MEDIUM
NORTH TRANSITION	6.88	4.05	0.59	SOFT

Table 13-5: Summary of AMCT And JKTech Drop Weight Tests

SAMPLE	C_{Bi}	UCS	UCS CLASSIF IC.	JKMRC PARAMETERS			
				A	b	Axb	T_a
MAIN FRESH	9.8 ± 1.8	161.29	MEDIUM	88.6	0.68	60.2	0.69
MAIN TRANSITION	9.41 ± 0.99	22.63	SOFT	55.1	2.49	137.2	0.96
NORTH FRESH	9.08 ± 0.78	177.15	MEDIUM	79.4	0.55	43.7	0.15
NORTH TRANSITION	6.54 ± 0.7	10.56	SOFT	65.6	5.22	342.4	3.38

The four ores were characterised into two groups; main fresh and north fresh ores were considered medium hardness and main transition and north transition ores were soft. The results showed that the ores would not be amenable to fully autogenous milling. The BBWi:BRWi was less than 1.1, indicating that the ore has a tendency to break down into a size that can easily be handled by a secondary ball mill.

The bond crushability index results showed that all ore types have low crushability indices and are characterised as very soft (values <10kWh/t).

The low UCS values that were recorded for main transition and north transition ores indicated that they are soft and will fracture easily. Main fresh and north fresh ores were more competent and were classified as hard ores.

The main parameters from the JKTech testing, A*b, was consistent with the rest of the tests conducted and indicated the fresh main and north ores were more competent than the transition ores.

Comminution testwork results were forwarded to OMC for ore interpretation and modelling and was used for the design of the comminution circuit for Twangiza.

A summary of OMC's interpretation, findings and recommendations are as follows and detailed information can be found in the report by OMC referenced "January, 2009, Orway Mineral Consultants (OMC), "Twangiza Gold Project Comminution Circuit Design", Report No. 8282":

The ore interpretation showed that the ore had a low competency and showed lower fracture energies when compared to the database. This meant that Twangiza ore will not provide good lump media for AG and SAG grinding;

A combination of the JK parameters and OMC power modelling demonstrated that the North Fresh ore deposit exhibited the highest SAG and ball mill specific grinding energies and was used as the limiting factor for the design of the circuit;

OMC suggested that a SABC (SAG-Ball mill-Crushing circuit) and SS SAG (Single stage SAG mill) comminution circuit designs should be considered. The power at the pinion required for both options is as follows:

- SABC (4716 + 5070 kW) – 9 786 kW
- SS SAG – 9 900 kW

Higher throughputs will be expected when treating the oxide and transition material as they are softer. The maximum throughput will be limited by the total volume flow in the circuit as excessive water will be required to overcome the viscosity issue for the oxide ore;

The throughputs in the SABC circuit selection will be from 505 – 700tph and 505 – 625t/h in the SS SAG circuit;

The SABC mill sizes selected were as follows:

- SAG mill – 8.5mØ x 4.45mEGL (Grate discharge, variable speed 6.2MW motor)
- Ball mill – 6.1mØ x 9.07mEGL (Over-flow discharge, fixed speed 6.2MW motor)

The mill size in the SS SAG circuit was 9.75mØ x 6.38mEGL (Grate discharge, fitted with a 12MW variable speed drive).

The liner and ball consumptions were as follows as shown in Table 13-6.

Table 13-6: Liner and Ball Consumptions

ITEM	UNIT	SABC	SS SAG
SAG MILL			
BALL CONSUMPTION	kg/t milled	0.389	0.739
LINER CONSUMPTION	kg/t milled	0.077	0.146
BALL MILL			
BALL CONSUMPTION	kg/t milled	0.538	
LINER CONSUMPTION	kg/t milled	0.071	

Recommendations raised by OMC are shown below:

- The grinding circuit mill sizes should be reviewed once the preferred circuit configuration has been selected and expected feed blend has been defined from the mining schedule;
- The milling turndown scenarios need to be investigated when softer oxide and transition ores are treated; and
- Viscosity modifiers should be considered when treating oxide ore to ensure that higher milling densities can be used.

13.5.4 Recovery variability testing

The aim of the variability tests was to establish the degree of variability within the ore zones identified with respect to their metallurgical response using the optimum conditions determined during the scoping and pre-feasibility testwork phases as listed below.

Oxides – Grind of 80% -75µm, 24 hour leach time, pH of 10.5 and cyanide addition of 1kg/t;

Transition and Fresh - Grind of 80% -75µm, 24 hour leach time, pH of 10.5 and cyanide addition of 2kg/t.

A summary of the variability results is shown below and the detailed reports containing the results is contained in the following reports:

- November 20, 2008, SGS, “Metallurgical Test Programme: Senet Project – Twangiza Oxide”, Report No. Met07/BB27; and
- February 10, 2009, SGS, “Metallurgical Test Programme: Senet Project – Twangiza Non-Refractory Variability Samples”, Report No. Met07/BB27.

The results obtained during the variability tests showed that the Twangiza ore is variable in terms of gold recovery.

Oxide variability testwork

Gravity and leach variability testwork was performed during the prefeasibility and results showed that the samples were highly variable obtaining recoveries less than 90% which was originally obtained during the optimisation testwork phase. The lime consumptions were also very high. This led to SENET requesting more samples (main and north) for repeat variability tests to increase the confidence level of the results obtained. A summary of the results is shown below in Table 13-7 and Table 13-8.

Table 13-7: Main Oxide Variability Results

SAMPLE ID	GRG (g/t)	GRAV MASS PULL (%)	GRAV RECOV (%)	CIL FEED GRADE (g/t)	NaCN CONS. (kg/t)	CaO CONS. (kg/t)	CIL DISSOL. (%)	OVER. GOLD RECOV. (%)
#68	2.15	0.07	2.8	2.14	0.56	4.48	76.5	77.1
#69	5.49	0.04	1.2	5.59	0.65	5.14	86.6	86.8
#70	5.17	0.07	1.2	5.29	0.59	4.91	83.9	84.1
#71	3.46	0.06	3.4	3.54	0.48	4.98	97.1	97.2
#72	2.70	0.05	19.4	2.20	0.65	6.01	87.6	90.0
#73	3.55	0.07	1.8	3.62	0.56	5.58	90.4	90.5
#74	3.08	0.05	4.1	2.99	0.54	2.91	94.8	95.0
#75	2.43	0.06	1.3	2.48	0.68	4.04	93.9	94.0
#76	4.23	0.04	2.6	4.27	0.71	3.74	92.6	92.8
#77	6.70	0.07	3.1	6.46	0.85	5.99	73.5	74.3

Table 13-8: North Oxide Variability Tests

SAMPLE ID	GRG (g/t)	GRAV MASS PULL (%)	GRAV RECOV (%)	CIL FEED GRADE (g/t)	NaCN CONS. (kg/t)	CaO CONS. (kg/t)	CIL DISSOL. (%)	OVER. GOLD RECOV. (%)
#78	0.54	0.05	7.0	0.50	0.56	3.11	93.9	94.4
#79	4.56	0.08	28.3	4.26	0.85	5.37	88.8	92.0
#80	3.65	0.08	18.7	3.06	0.48	5.10	84.7	87.6
#81	3.29	0.09	19.3	2.78	0.83	5.88	84.9	87.8
#82	2.26	0.04	42.7	1.35	0.54	4.80	89.7	94.1
#83	1.56	0.05	27.9	1.01	0.48	3.28	57.5	69.3
#84	2.44	0.05	12.2	2.24	0.45	5.80	78.2	80.8
#85	11.66	0.06	25.8	8.76	0.83	6.24	91.5	93.7
#86	2.61	0.09	2.4	2.76	0.65	6.07	82.4	82.8
#87	11.13	0.05	13.9	9.67	0.62	5.44	85.7	87.7

The cyanide consumption decreased from the prefeasibility testwork and the lime consumption varied from 2.91 to 6.01 kg/t which was high but comparable to what was previously obtained. On average the recoveries were similar to the scoping and optimisation results. The varying range in the dissolutions obtained demonstrated that this ore is highly variable which could be an indication of the presence of some refractory material in the oxides. This applied to both the main and north ore bodies.

Transition and fresh variability testwork

The testwork was performed on the initial samples (labelled 'sack') that were sent together with additional samples requested to confirm results, recoveries and reagent consumptions. Recovery variability testwork was performed on fourteen main transition and fresh ore samples. Only one fresh sample was tested due to sample availability and eight north transition and five north fresh samples underwent leach variability testwork

The results showed that both the main and north ore bodies are highly variable with regards to reagent consumptions and gold dissolution. This could be due to the presence of refractory transition and fresh material in the samples that were submitted as was noticed with the oxides. The results are summarised in Table 13-9 and Table 13-10Table 13-9.

Table 13-9: Main Transitional and Fresh Variability Results

SAMPLE ID	CIL FEED GRADE (g/t)	NaCN CONS. (kg/t)	CaO CONS. (kg/t)	CIL DISSOL. (%)
MAIN TRANSITION SACK 8	6.80	1.56	2.69	89.9
MAIN TRANSITION SACK 9	2.06	0.98	7.46	73.5
MAIN TRANSITION SACK 10	4.32	1.42	7.42	82.2
MAIN TRANSITION SACK 11	4.82	1.56	4.22	64.7
MAIN TRANSITION SACK 12	2.90	1.56	2.86	63.0
MAIN TRANSITION SACK 23	2.47	1.25	3.19	55.7
MAIN TRANSITION SACK 24	1.54	1.27	2.86	77.0
MAIN TRANSITION SACK 25	2.69	1.56	2.52	75.0
MAIN TRANSITION SACK 26	3.14	0.64	2.42	85.0
MAIN TRANSITION SACK 27	1.90	0.72	1.24	95.3
MAIN TRANSITION 56	0.91	0.84	1.55	83.7
MAIN TRANSITION 57	3.08	0.98	1.68	76.6
MAIN TRANSITION 58	3.37	1.13	2.78	68.0
MAIN FRESH 59	0.92	0.69	0.95	69.4

Table 13-10: North Transitional and Fresh Variability Results

SAMPLE ID	CIL FEED GRADE (g/t)	NaCN CONS. (kg/t)	CaO CONS. (kg/t)	CIL DISSOL. (%)
NORTH TRANSITIONAL SACK 38	1.51	0.62	1.39	92.1
NORTH TRANSITIONAL SACK 39	3.62	0.71	1.09	79.5
NORTH TRANSITIONAL SACK 40	10.35	0.27	1.19	96.6
NORTH TRANSITIONAL SACK 41	3.56	0.71	0.79	92.1
NORTH TRANSITIONAL SACK 42	0.85	0.71	0.72	98.0
NORTH TRANSITION 60	0.73	0.65	3.11	89.1
NORTH TRANSITION 61	1.29	0.56	1.91	91.0
NORTH TRANSITION 62	2.72	0.68	3.15	81.8
NORTH FRESH 63	1.12	0.61	1.38	78.9
NORTH FRESH 64	1.50	0.67	1.35	78.4
NORTH FRESH 65	2.81	0.78	1.85	86.7
NORTH FRESH 66	1.36	0.58	1.35	79.4
NORTH FRESH 67	2.74	0.72	1.51	92.2

13.5.5 Transition CMS and fresh CMS refractory ore testwork

Head analysis transition CMS and fresh CMS refractory ores

Head analyses were completed on the Twangiza Transition CMS and the Fresh CMS composite samples. They were fire assayed for Au and Ag. The average gold content of the Transition CMS was 2.23g/t and 2.65 g/t for the fresh CMS. A portion of the head split was assayed semi-quantitatively using ICP for an additional 21 elements. The two samples were also analysed for total carbon, carbon speciation, total sulphur, sulphur speciation, arsenic and mercury. The multi-elemental analysis indicated that both the transition CMS and fresh CMS samples were high in arsenic (0.46% and 0.93%, respectively). The specific gravity (SG) of the two ore types were 2.82 and 2.99 for the transition and fresh ores respectively. The sulphur speciation assays showed that the fresh CMS ore body is a massive sulphide and the transition CMS is oxidised and still contains a significant amount of sulphide mineralisation. The carbon analysis illustrated that there are large amounts of preg-robbing material that have to be removed upfront during flotation prior to the leach. A pre-float step for graphitic carbon removal was therefore included. The results are summarised in Table 13-11 and Table 13-12.

Table 13-11: Head Assay Carbon and Sulphur Speciation

ELEMENT	UNIT	FRESH CMS	TRANSITION CMS
S (t)	%	10.20	5.61
S (2^-)	%	9.57	4.53
S (0)	%	<0.5	<0.5
C (t)	%	1.82	1.37
CO_3	%	2.11	0.76
C (org)	%	1.40	1.22
C (graph)	%	0.09	0.08

The flotation testwork was divided into the following stages:

- Scouting tests;
- Grind optimisation;
- Reagent optimisation;
- Rougher rate tests;
- Flotation test at optimum conditions;
- Bulk concentrates generation for leach testwork.

Scouting testwork

Scouting testwork was performed using a generic reagent suite used during the early stage of the testwork, to determine the mass pulls and recoveries that can be obtained to start the optimisation testwork. The results from the scouting testwork were not in line with what was obtained in the Pre-Feasibility Study as higher recoveries and mass pulls were expected. The Fresh CMS gold recovery was above 90 % at a mass pull of 30 %. A gold recovery of 75 % was achieved for Transition CMS at a mass pull of 15%. Optimisation testwork followed to maximise recovery.

Table 13-12: Full Elemental Analysis

ELEMENT	UNIT	FRESH CMS	TRANSITION CMS
Au	g/t	2.65	2.23
Ag	g/t	<1	<1
Hg	ppm	<3	<3
As	%	0.93	0.46
Al	%	6.20	5.80
Ba	ppm	140	210
Be	ppm	<1	<1
Ca	%	0.76	0.16
Cd	ppm	<10	<10
Co	ppm	32	32
Cu	ppm	<10	<10
Fe	%	10.9	7.9
K	%	0.24	0.32
Li	ppm	<10	<10
Mg	%	0.42	0.14
Mn	%	220	210
Mo	ppm	<10	<10
Na	%	3.9	3.5
Ni	ppm	52	54
P	ppm	310	330
Pb	ppm	33	<30
Sn	ppm	<20	<20
Sr	ppm	15	13
V	ppm	51	60
Y	ppm	13	17
Zn	ppm	16	41
Zr	ppm	100	150
Cr	%	<0.02	<0.02
Si	%	23	25
Ti	%	0.34	0.39

Two bulk composites, transition CMS and fresh CMS, were made up and each crushed to -1.7mm, using a large scale laboratory jaw crusher, and blended. These samples were used for the flotation and leach recovery testwork shown below.

Grind optimisation

The effect of flotation feed grind size on metallurgical performance for each of the composites is presented in Figure 13-1. The summarised results are shown in Table 13-13 and Table 13-14. The optimum grind chosen was 70% -75 µm. Grinding finer increased the mass pull but recovery decreased for the fresh CMS. A slight improvement in gold recovery for the transition CMS material was noted with a finer grinding from 70% to 80% -75 µm.

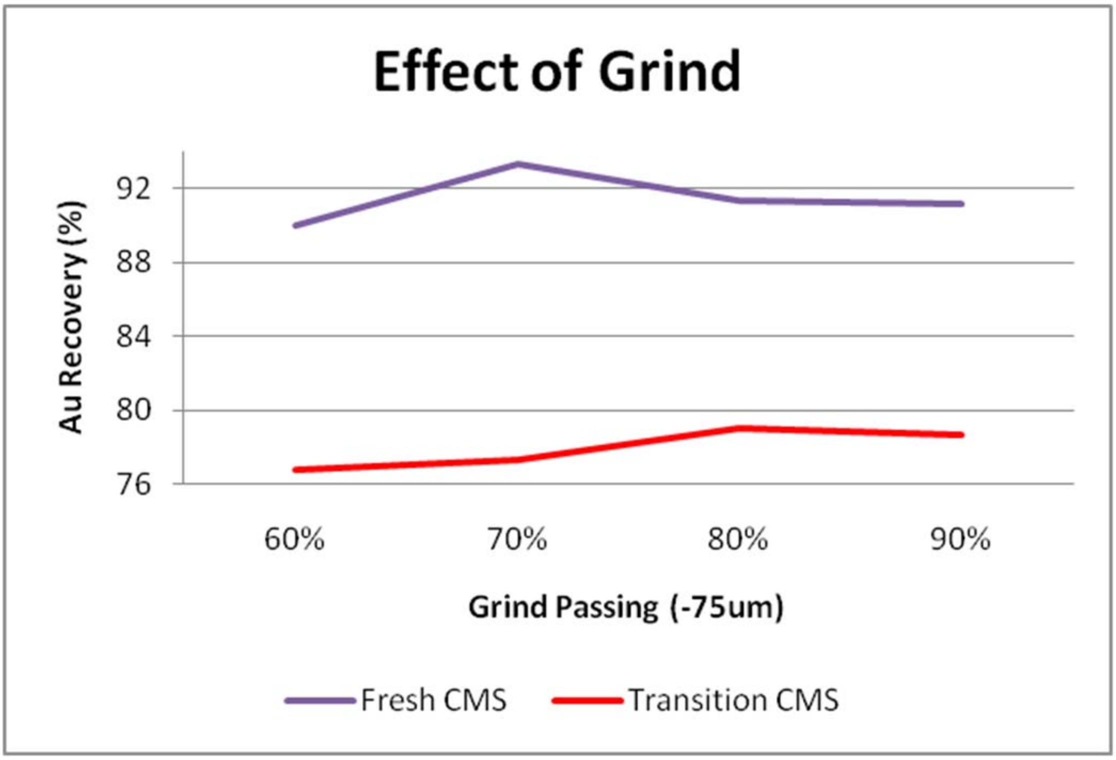


Figure 13-1: Effect of Grind on Recovery

Table 13-13: Grind Optimisation Results Summary - Fresh

GRIND -75µm	MASS PULL (%)	Au (g/t)	S (%)	S^{2-} (%)	Au REC. (%)	S REC. (%)	S^{2}- REC. (%)
60%	27.95	10.14	28.00	24.54	89.94	88.94	97.90
70%	32.58	7.53	29.80	22.14	93.34	90.97	89.17
80%	36.93	6.45	32.40	19.00	91.29	91.34	86.88
90%	36.91	6.32	30.23	13.87	91.13	86.04	78.44

Table 13-14: Grind Optimisation Results Summary - Transition

GRIND -75µm	MASS PULL (%)	Au (g/t)	S (%)	S^{2-} (%)	Au REC. (%)	S REC. (%)	S^{2}- REC. (%)
60%	14.15	14.46	30.20	24.90	76.8	74.55	80.41
70%	15.00	14.33	30.39	27.90	77.35	80.85	91.62
80%	16.40	12.29	35.75	24.02	79.02	85.18	87.88
90%	19.93	9.49	22.38	20.95	78.68	84.01	86.27

Reagent optimisation

Reagent optimisation testwork was performed on fresh CMS by varying the collector and secondary collector addition and keeping the copper sulphate (activator) the same at 100g/t at natural pH. It was found that a secondary collector had to be used to maximise the recovery of all sulphides. The results for the fresh CMS reagent optimisation tests by varying the collector addition are shown in Table 13-15. An increase in the collector addition increased the mass pull and improved the gold recovery slightly.

Table 13-15: Reagent Optimisation at Natural pH

REAGENTS	MASS PULL (%)	Au RECOVERY (%)	S^{2-} RECOVERY (%)
100g/t PAX and 50g/t Aero407	34.82	88.86	89.45
200g/t PAX and 100g/t Aero407	39.44	90.99	91.64
300g/t PAX and 150g/t Aero407	45.87	93.08	95.86
400g/t PAX and 200g/t Aero407	46.70	93.85	95.59

Tests were performed at acidic pH, Table 13-16, to determine the effect it will have on the mass pull and recovery when the collector addition was reduced. It was found that the reduction in the collector addition adversely affected gold recovery. More acidic conditions were detrimental to both gold and sulphide recovery.

Further reagent optimisation scouting testwork was performed to determine the impact that different collectors would have on mass pull and recovery. The best results showed that PAX will have to be used with a secondary collector, Aero 3302, Table 13-17.

Table 13-16: Reagent Optimisation at Acidic pH

REAGENTS	MASS PULL (%)	Au RECOVERY (%)	S^{2-} RECOVERY (%)
200g/t PAX and 100g/t Aero407	33.12	92.24	92.58
100g/t PAX and 50g/t Aero407	36.59	90.51	91.98
50g/t PAX and 25g/t Aero407	32.37	86.30	89.84

Table 13-17: Reagent Optimization by Varying Collector Type

REAGENTS	MASS PULL (%)	Au RECOVERY (%)	S^{2-} RECOVERY (%)
200g/t PAX and 100g/t Aero 3418A	36.16	91.33	91.79
200g/t PAX and 100g/t Senkol 295	37.49	93.6	92.83
200g/t PAX and 100g/t Aero 3302	38.39	95.00	92.38
200g/t Aero 3418A only	34.94	92.94	90.63
200g/t PAX and 100g/t Aero 407	41.59	92.75	91.83

Rougher rate tests

A rougher rate test was performed using the optimum reagent suite on the fresh and transition CMS and it was found that most of the gold was recovered in the first 26 minutes. A total gold recovery of 93% was obtained for fresh ore and was in line with the recoveries obtained previously. A poor recovery of 82% was achieved for transition ore and will be reviewed at a later stage by performing mineralogical investigations of the feed and tailings. It was noted however that the flotation test on the transition CMS was a success with respect to the sulphide recovery as a recovery of 93% was achieved. A summary of the results is presented in Table 13-18 and Table 13-19.

Table 13-18: Rougher Rate Tests - Fresh

ITEM	TIME (min)	MASS PULL (%)	Au RECOVERY (%)	S^{2-} RECOVERY (%)
SULPHIDE CONCENTRATE 1	26	35.6	88.34	85.77
SULPHIDE CONCENTRATE 2	6	2.95	3.56	6.38
SULPHIDE CONCENTRATE 3	8	1.51	0.61	1.53
SULPHIDE CONCENTRATE 4	10	0.97	0.37	0.69
CUMULATIVE/TOTAL	50	41.03	92.88	94.37

Table 13-19: Rougher Rate Tests - Transitional

ITEM	TIME (min)	MASS PULL (%)	Au RECOVERY (%)	S^{2-} RECOVERY (%)
SULPHIDE CONCENTRATE 1	26	17.58	79.49	89.24
SULPHIDE CONCENTRATE 2	6	1.75	1.37	1.92
SULPHIDE CONCENTRATE 3	8	1.98	0.87	1.02
SULPHIDE CONCENTRATE 4	10	1.65	0.66	0.64
CUMULATIVE/TOTAL	50	22.96	82.39	92.82

Bulk concentrate generation

A bulk concentrate was generated using the optimum conditions obtained from all the flotation tests performed to date for use during the leach optimisation testwork. A summary of the results is shown in Table 13-20.

Table 13-20: Bulk Concentration Results

MASS PULL (%)	Au RECOVERY (%)	S^{2-} RECOVERY (%)
56	95	98

13.6 Review of Comminution Circuit Parameters to Process 1.7 Mtpa

13.6.1 Background

SENET requested OMC to undertake a comminution equipment evaluation for the Twangiza Project.

The circuit configuration for processing of 1.3 Mtpa Oxide ore using a tertiary crushed feed followed by two stage ball milling had been reviewed previously. The use of a scrubber to remove the clay from the primary crushed ore prior to the secondary crushing had been included.

The following equipment was selected:

- Scrubber - 3.0 m Ø x 8.0 m EGL, 300 kW,
- Secondary cone crusher – Kawasaki KM2513G, 185kW,
- Tertiary cone crusher – Kawasaki KM1213G, 185kW,
- Ball mill 1 - 3.81 m x 5.03 m EGL, 1,300 kW,
- Ball mill 2 - 3.23 m x 3.04 m EGL, 550 kW.

13.6.2 Summary of investigation by Orway Mineral Consultants

The maximum power achievable with the selected scrubber is 1.09 kWh/t at a 275 t/h feed rate. This is expected to be sufficient for most ores. The design is based on the North oxide ore characteristics (BBWi = 4.3 kWh/t). The North Transition ore is 60% more competent (BBWi = 6.88 kWh/t) and throughput will be affected by this material. Modelling showed that the circuit is very sensitive to ore characteristics and that this should be managed actively during operation. Feed particle size distributions have been sourced from the OMC database for modelling.

Following the recommendations by OMC after expansion, a number of key modifications were implemented to counter the effect of size distribution and that of clay balls.

The grind in the ball mills is sensitive to the feed size fraction; the coarser the feed, the coarser the grind which results in a poor recovery. In-pit crushing and screening operations have been added to the circuit to assist the plant crushing system to obtain -10mm material to feed the ball mills.

The clay balls proportion depends on the proportion of clay in the ore body. The deeper mining is done, the less clay balls are formed and the easier the operations. The Plant crushing system alone could not cope with the amount of clay balls. With additional capacity provided by the in-pit crushing and screening, over 50%of clay balls by-pass completely the Plant crushing system and report directly into the ball mills where they are dealt with by using a varying proportion of steel balls and dilution rates.

Active management of the closed side setting of the crushers and the closing screen panels may be required. Accurate measurement (including provision for calibration) of the -2mm feed to the mill sump will be required for process control purposes. The mills are capable of treating the expansion throughput at the selected design ore characteristics and still maintain typical design margins. Full details of the review can be found in the report no. 8623.30 Rev 1 titled Twangiza Gold Project – 1.7 Mtpa Throughput Option – Oxide Only by OMC.

14 MINERAL RESOURCE ESTIMATE

14.1 Introduction

A Mineral Resource estimate was originally produced by SRK (UK) for the 2009 Feasibility Study which was undertaken by Martin Pittuck and Benjamin Parsons of SRK (UK), both of whom signed as Qualified Persons. A modified version of the same resource model was provided by Twangiza Mining and reviewed by SRK (UK) for the 2011 Phase 1 Economic Assessment compiled by SENET; Martin Pittuck signed off the resource as Qualified Person.

The relatively minor modifications to the resource model since 2009 have been prepared in house by Twangiza Mining's team under the supervision of Daniel Bansah (Head of Projects and Operations) who is also a Qualified Person. The interim modifications have involved:

- addition of minor deposits (Twangiza West and East);
- conversion of Valley Fill deposit from Inferred to Indicated;
- creation of diluted grade and recoverable grade variables;
- changes to topographic survey to account for mining depletion; and
- revisions to the density model in light of production reconciliation results.

All modifications have been reviewed and approved by SRK (UK).

14.2 Approach

The adit, drill hole and trench data were originally plotted on level plans and vertical sections; lithological and mineralisation interpretations were then outlined on vertical sections and flitch plans by Twangiza Mining supplied to SRK (UK) for creation of 3D geological and mineralised wireframes.

The 3D model continuity was assessed by SRK (UK) in Leapfrog mining software (Leapfrog). The mineralised domains were largely based on a geological cut-off of 0.3 g/t Au but were also formed so as to ensure continuity between sections and flitch plans. This process was repeatedly verified until a robust 3D model of the mineralised zones had been created.

The statistics and geostatistics for the main deposits were completed in Isatis. Datamine was used for block modelling and grade estimation. The resource data, survey data and block models are housed in the Datamine software (Datamine).

14.3 Density determinations

14.3.1 Feasibility Study Data collection and analysis

The initial bulk density testwork was performed on behalf of Twangiza Mining by CME in 1998 on a variety of rock types; a total of 165 samples were assessed by CME. Twangiza Mining subsequently undertook more density determinations from drill core; a total of 2,031 relative density determinations were undertaken by Twangiza Mining.

Twangiza Mining's density determinations were undertaken at the following intervals in all drill holes:

- Every 2 meters outside mineralized zones.
- Every 1 meter within mineralized zones

The following method was employed:

- The geologist selects samples and marks each sample position with an aluminium tag. The borehole number and depth are also written on the selected piece of core with a marker pen. In order to avoid bias when taking samples, the first piece of solid core weighing over 200 g after the meter mark is selected.
- The depth of each sample and rock type is recorded by the geologist.
- The weight of the core sample is recorded (Weight 1)
- The sample is checked for porosity by placing the sample in water and recording the increase in weight over a 3-minute period (Weight 2). If the sample absorbs more than a gram of water it is treated as porous.
- The sample is dried in an oven at 105°C for 30 minutes, and then re-weighed (Weight 3). This enables the moisture content of the air dried core to be calculated:

$$\text{Moisture Content (\%)} = \frac{(\text{Weight 1} - \text{Weight 3}) \times 100}{\text{Weight 1}}$$

- If the sample is porous it is coated with varnish using a brush, ensuring all cavities and irregularities are coated; the dry sample is weighed whilst suspended in water (Weight 4) allowing the dry density to be determined:

$$\text{Dry Density } (t/m^3) = \frac{\text{Weight 3}}{(\text{Weight 3} - \text{Weight 4})}$$

The density determinations were statistically reviewed according to lithological type and position within the weathering profile; a summary of block model dry density values used in the Feasibility Study is given in Table 14-1.

Table 14-1: Feasibility Study Dry Solid Densities

	DRY DENSITY (g/cm^3)		
MATERIAL TYPE	PORPHYRY	SEDIMENT	WASTE
UPPER OXIDE	1.8	2.1	2
LOWER OXIDE	2.15	2.05	2.15
TRANSIT. (SOFT)	2.35	2.4	2.35
TRANSIT. (HARD)	2.75	2.65	2.75
FRESH	2.85	2.7	2.85

14.3.2 Pit Density Determinations

During the course of the reconciliation study undertaken as part of this technical report, a number of check density samples were taken from the Twangiza open pit. In summary these comprised 5 samples taken from small pits and 1 bulk sample.

Pit samples were taken by digging a hole approximately 15cm by 15cm by 10cm deep to provide a sample volume of 2,400 cm^3 on average. The volume of each sample was determined by assessing the weight of sand required to fill the each sample pit and then calculating the volume of sand in the pit by dividing the weight of the sand by the known bulk density of the sand (1.25 g/cm^3).

A 100g subsample of each pit sample was weighted whilst wet and then reweighed after drying overnight in an oven at 105°C; this allowed moisture content to be determined.

Each pit sample was weighed as soon as it had been extracted; this provided a wet weight which was then corrected by the moisture content to determine its dry weight. The dry density of each sample was calculated by dividing the dry weight by the volume of the pit. A summary of the pit samples is given in Table 14-2.

Table 14-2: Pit Density Determinations

	WHITISH GREY MUDSTONE			REDDISH BROWN MUDSTONE	
	No.1	No.2	No.3	No.5	No.6
Wet Density (g/cm^3)	2.15	1.88	2.13	1.90	1.95
Moisture Content (%)	10%	18%	11%	25%	20%
Dry Density (g/cm^3)	1.92	1.54	1.90	1.43	1.56

14.3.3 Bulk Sample Density Determinations

During the course of the reconciliation study undertaken as part of this technical report, the Twangiza Mining team used a bulk sample method to determine density of material routinely mined from the open pit. A block of ground, approximately 5m by 5m by 2.5m deep was excavated and hauled directly to the crusher. The excavation was surveyed and determined to have a volume of 67 m^3. The mill weightometer reading was taken before the trucks tipped this material into the crusher and after the material had travelled up the belt over the weightometer.

A sub sample of the material was taken weighed whilst wet and then weighted again after drying to determine the moisture content.

The weightometer recorded 134 tonnes of wet material (giving a wet density of 2.0 g/cm^3) and this was then corrected for moisture content (found to be 23.8%) to give a dry density of 1.53 g/cm^3.

14.3.4 Revised Density Model

Density in the ore is affected by weathering intensity, hence modelling upper oxide, lower oxide, soft, transition, hard transition and fresh weathering layers in the Mineral Resource model. The recent density work suggests that very close to surface the density of the ore is lower than previously estimated for the upper oxide.

At Twangiza Main, it is proposed by Twangiza Mining and SRK that ore density has also been affected by material collapsing into underground mining voids which were probably accessed from the CME adits. The large artisanal open pit (Mbwega Pit) was originally thought by Twangiza Mining and SRK to be the only artisanal mining site and the sole source of the Valley Fill deposit. Mbwega Pit featured in the topographic survey used in the Feasibility Study and the material mined from there did not add to any resource models. However, collapsed underground workings are now thought to exist, causing zones of broken material with low density but of sufficient bulk so as not to be recognized as mine workings in the pit.

The Feasibility Study density sampling protocol required solid core samples to be taken which is common practice but it results in low density material being under-represented in the core samples used to estimate density, this is normally accounted for by rounding down density estimates in oxide. However, in light of operational information, the bias in the original estimate is now thought to be more considerable and so has been specifically addressed in this technical report.

To reflect the most intense weathering near surface that was under-represented in core density sampling, the block model within 15m of the original topographic surface was reduced from 2.05 t/m^3 to 1.80 t/m^3 at Twangiza Main and to 1.89 t/m^3 at Twangiza North. This affects approximately quarter to half of the material in the oxide resource.

To reflect the collapsed material that was under-represented in core density sampling, a 20% discount was applied to oxide ore blocks in the part of the Twangiza Main Pit closest to the CME adits, which is where underground mining is assumed to have taken place. This affects approximately 15% of the Mineral Reserve.

The value of 20% is based on the reconciliation study in which a historical tonnage shortfall was recorded at the plant. Many possible explanations were considered, not all were studied through to a satisfactory conclusion but the decision was taken to assume the balance of adjustments should be attributed ore density.

The resultant adjusted ore densities are given in Table 14-3.

Table 14-3: Summary of 2015 Revised Dry Density Values

		MAIN DENSITY (g/cm^3)			NORTH DENSITY (g/cm^3)			ALL HARDNESS (MPa)		
MATERIAL TYPE	**Resource Category**	**PORPHYRY**	**SEDIMENT**	**WASTE**	**PORPHYRY**	**SEDIMENT**	**WASTE**	**PORPHYRY**	**SEDIMENT**	**WASTE**
UPPER 15m OXIDE	**Measured**	1.45	1.45	1.45						
	Indicated & Inferred	1.80	1.80	1.80	1.89	1.89	1.89	15	15	15
UPPER OXIDE	**Measured**	1.45	1.69	1.61	1.80	2.10	2.00	15	15	15
	Indicated & Inferred	1.80	2.10	2.00	1.80	2.10	2.00	15	15	15
LOWER OXIDE	**Measured**	1.73	1.65	1.73	2.15	2.05	2.15	35	35	35
	Indicated & Inferred	2.15	2.05	2.15	2.15	2.05	2.15	35	35	35
TRANSIT. (SOFT)	**Measured**	1.89	1.93	1.89	2.35	2.40	2.35	60	60	60
	Indicated & Inferred	2.35	2.40	2.35	2.35	2.40	2.35	60	60	60
TRANSIT. (HARD)	**Measured**	2.21	2.13	2.21	2.75	2.65	2.75	175	175	175
	Indicated & Inferred	2.75	2.65	2.75	2.75	2.65	2.75	175	175	175
SULPHIDE	**Measured**	2.29	2.17	2.29	2.85	2.70	2.85	210	210	210
	Indicated & Inferred	2.85	2.70	2.85	2.85	2.70	2.85	210	210	210

14.4 Descriptive statistics of assay data

Summary statistics have been calculated for the different sampling phases and have been compiled in Table 14-4 below. The results of the analysis were compared for similarities and differences to determine whether the assay values could be combined for the modelling and estimation processes.

Table 14-4: Summary of Raw Statistics per Sampling Phase

DESCRIPTION	NO	MIN (g/t Au)	MAX (g/t Au)	MEAN (g/t Au)	VAR.	CoV
RECENT (2011 – 2014) DD HOLES	**2,658**	**0.005**	**18.8**	**0.19**	**0.61**	**4.14**
RECENT (2009 – 2010) DD HOLES	40,708	0.01	63.4	0.81	4.26	2.56
RECENT (PHASE FS INFILL) DD HOLES	21,148	0.01	58.4	0.78	4.93	2.86
RECENT (PHASE 3 PFS) DD HOLES	24,534	0.01	121.00	0.36	3.75	5.44
FOLLOW-UP DRILL HOLE SAMPLES	25,677	0.01	280.00	0.70	7.74	3.95
1997/98 DRILLING SAMPLES	8,884	0.01	310.28	0.95	16.79	4.31
ALL TRENCH	4,660	0.01	93.80	2.61	15.15	1.49
1997/98 ADIT SAMPLES	1,610	0.01	52.44	3.06	18.95	1.42
HISTORICAL ADIT SAMPLES	13,328	0.01	157.70	3.56	25.61	1.42

The mean values of the data presented in the table above indicate that the trench and adit samples have significantly higher mean grades, which is due to the fact that they are preferentially located in the oxide material. The drill hole data which tends to have been lower mean grades have been drilled into all four material types (upper oxide, lower oxide, transition and sulphide) but preferentially into the transition and sulphide material and include both ore and waste sampling. More recent drilling has a lower average grade owing to targeting the narrower and deeper intersections in Twangiza North thus incurring a greater proportion of waste samples.

14.5 Geological Modelling

14.5.1 Geological wireframes

In the Feasibility Study, metal recoveries from different rock units were highly variable, especially within the transitional zone; there were good recoveries from the porphyry material and lower recoveries in the sediment material. With this in mind, care was taken to model the lithologies in the resource block model.

The geological aspects considered during interpretation were lithological and structural with respect to anti-formal axis and some minor offset faulting. SRK (UK) reviewed the geological data and concluded that the following geological factors should be included in the 3D block model:

- Separation of the rock-codes into porphyry and sediment rock types;
- Construction and extension of fault planes which impact on the mineralisation cutting the deposit;
- Creation of oxidation surfaces to include upper and lower oxide, soft and hard transitional material and top of fresh rock;

Evidence of a synclinal structure to the east of the pit wall between X 693,525 – 693,700 and Y 682,800 – 683,000, which due to its proximity to the potential final pit limits, will influence the geotechnical issues.

The average distance between section lines is approximately 40 – 50 m within the main drilling increasing to approximately 150 m at the edge of the deposits. To ensure waste blocks were assigned the correct oxidation code for mine planning, interpretations were extended to the east and west beyond the current drilling information.

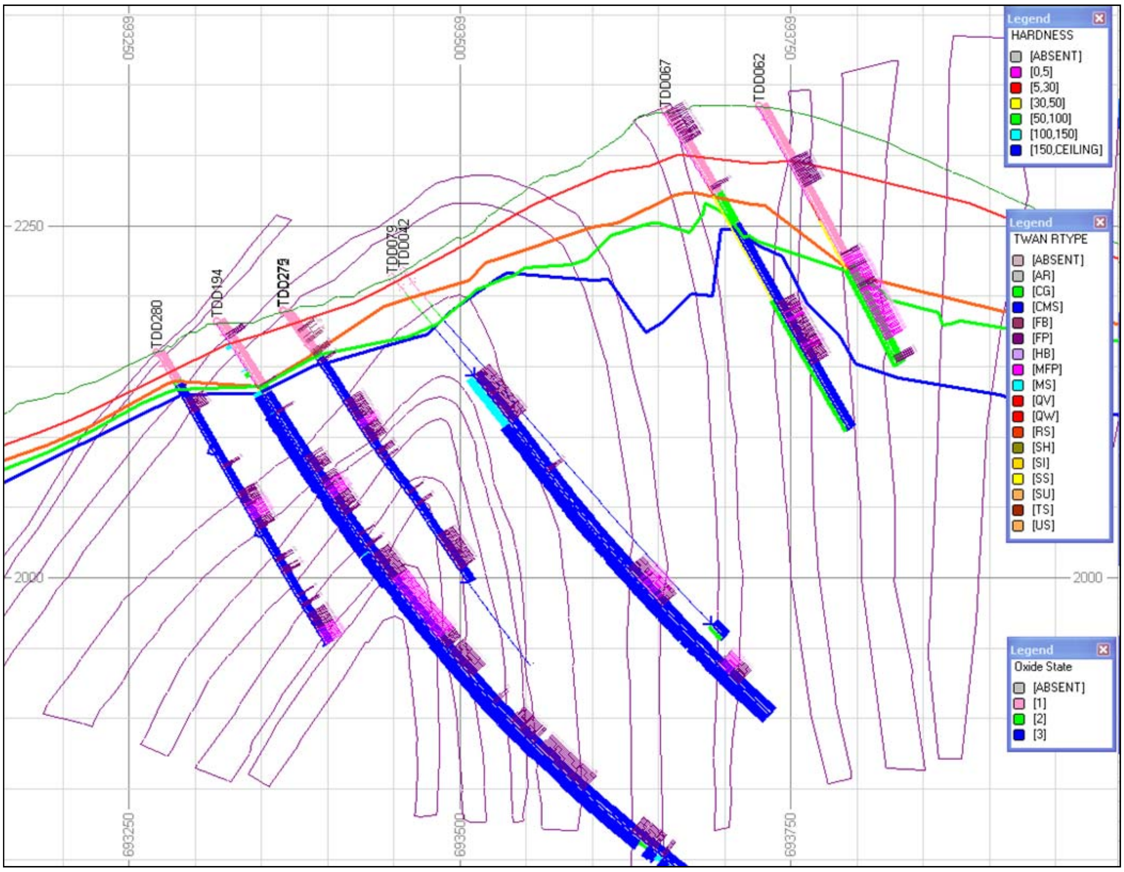


Figure 14-1: Cross-Section Showing Digitised Interpretation of Oxide Contacts & Lithology

The anticline hinge zone plunges to the south and the interpretation progresses northwards through the deposit the mineralised part of the hinge zone daylights above the current topography. At this point the interpretation has been changed from a single unit to separate east and west mineralised limbs of the anticline. In general, the anticline becomes more isoclinal with depth.

The location of North East – South West trending faults and East – West trending faults in the northern portion of the Twangiza Main deposit plays an important role in the structure of the anticline. Interpretation of the faults was supplied by Twangiza Mining and extended to intersect the main geological model by SRK (UK). SRK (UK) also imported bedding measurements into the database in an attempt to improve the interpretation of the dips within the porphyry units.

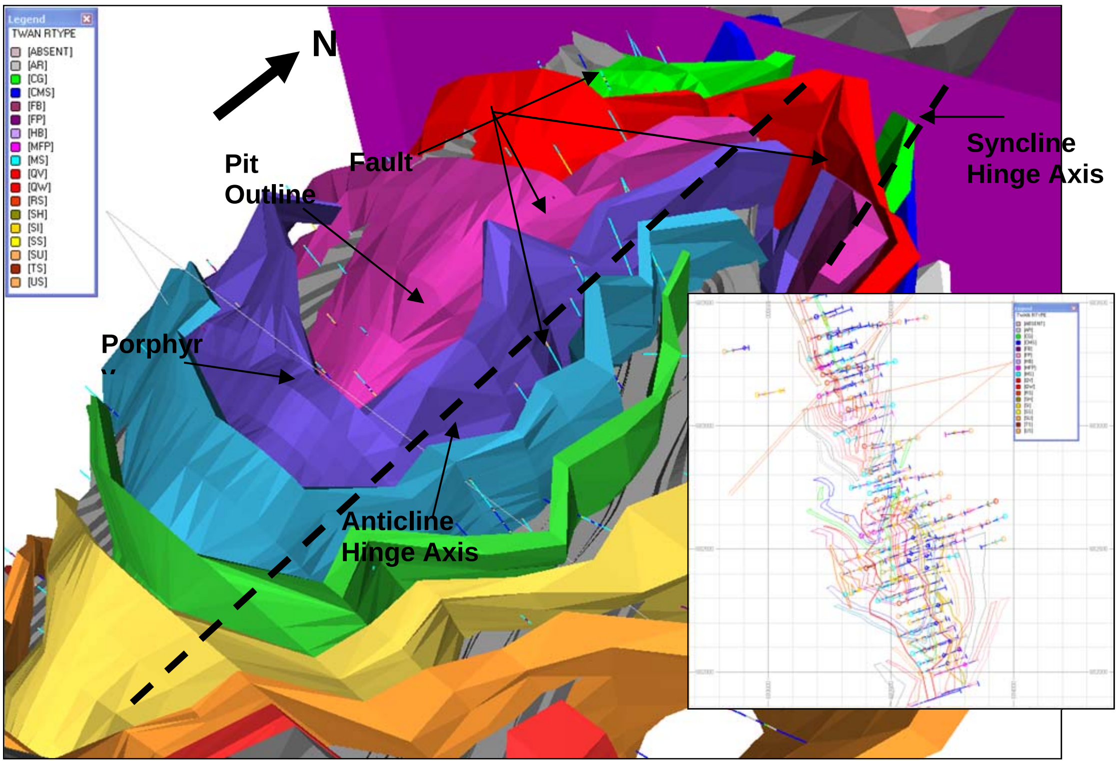


Figure 14-2: 3D Screenshot of Porphyry Wireframe at Twangiza Main

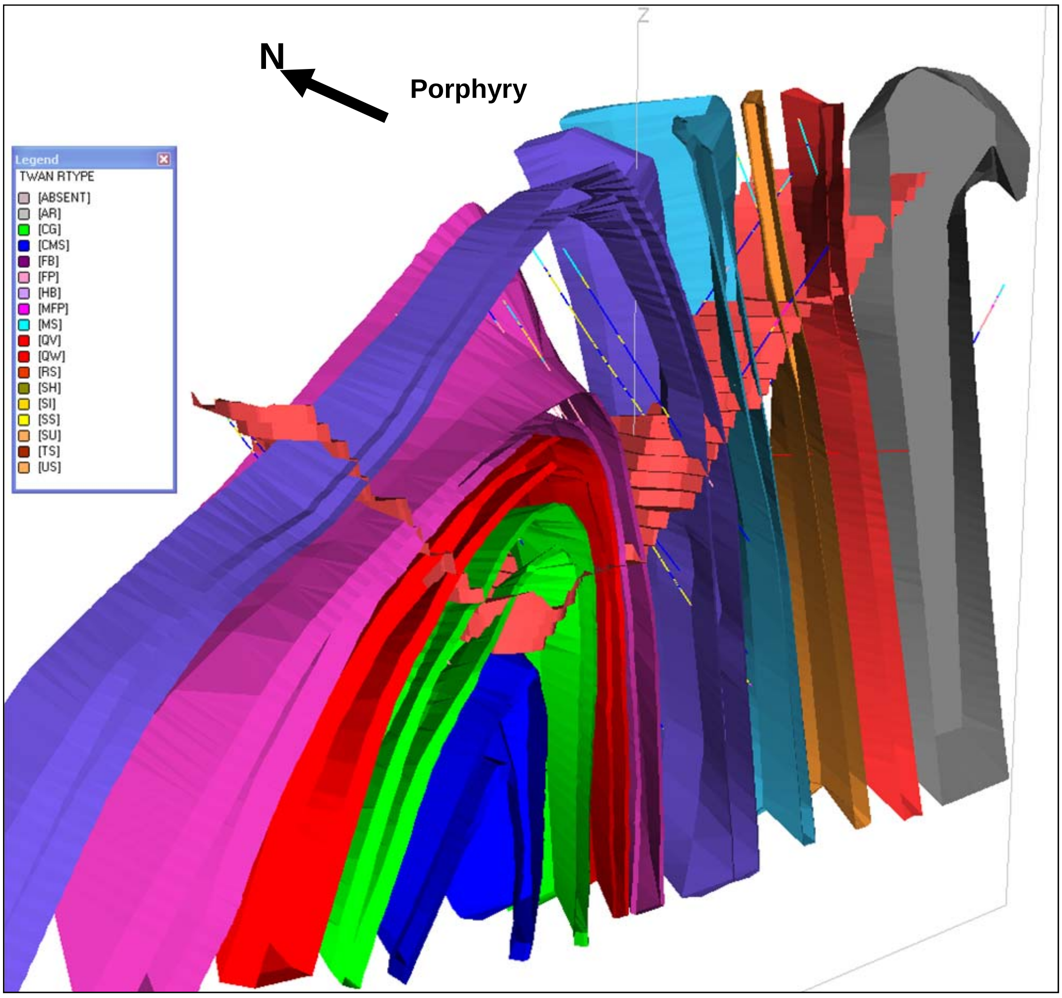


Figure 14-3: 3D Screenshot showing cross-section of porphyry wireframe

14.5.2 Mineralisation wireframe

The mineralised domain perimeters were defined in the light of the available geological knowledge and using a 0.3 g/t Au iso-surface. Plan interpretations created by Twangiza Mining were also used to colour code the different mineralised lodes. Depth extrapolation beyond the limits of drilling was carried out to ensure consistency in shape and orientation, this is reflected in the classification of resource.

To ensure continuity in the mineralised interpretation Leapfrog has been utilised to identify consistent mineralised lenses. Semi-variograms and search ellipses have been generated which are rotated into the general dip and strike of the deposit. Different anisotropies were tested until the most continuous shape was created. The leapfrog wireframe created agreed with and improved the historic interpretations of the mineralised zone and therefore was used as a guideline during the digitisation of the mineralised strings on each vertical drilling section. This technique provides increased confidence in the potential ore / waste contacts between drill holes and in areas of relatively low sampling density.

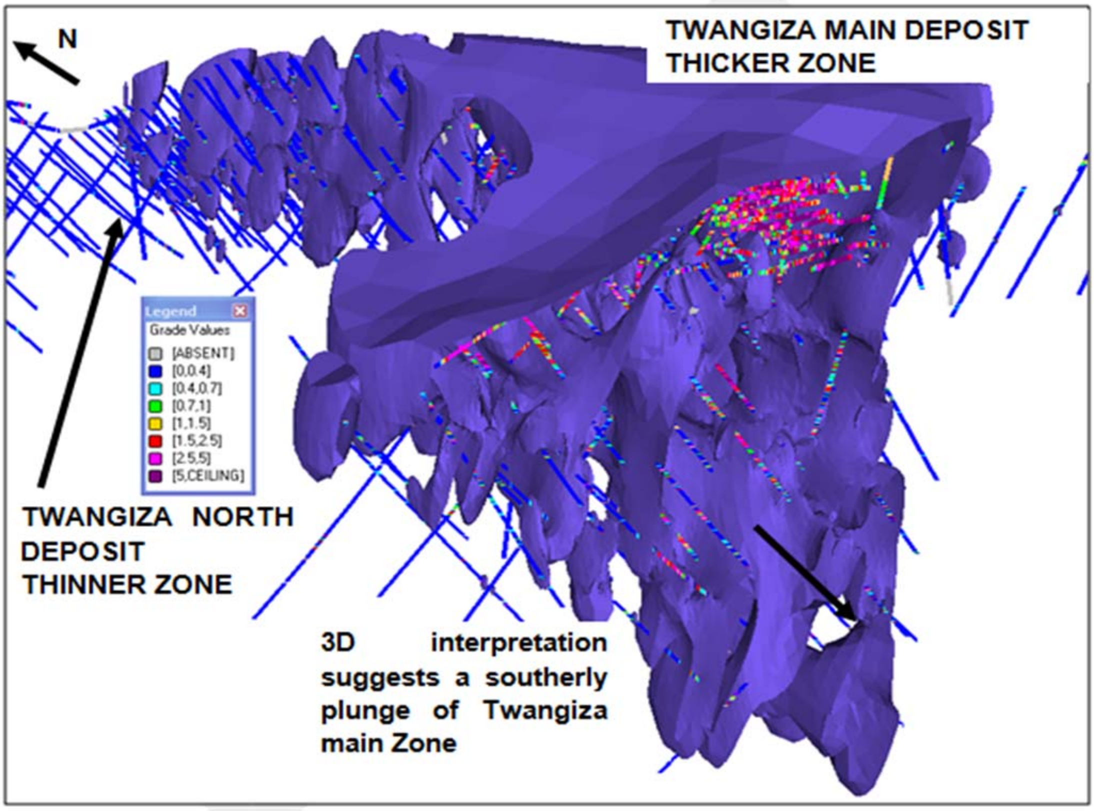


Figure 14-4: 3D Screenshot 0.3 g/t Au Leapfrog Iso-Surface

The subdivision of deposit into zones is based on observed consistency of mineralisation. Two different styles of mineralisation are present with the Twangiza deposit and can be summarised using the following characteristics:

The Twangiza Main deposit is up to 300 m wide in places, width increases towards surface within the transitional and oxide zones. At depth the mineralisation zones tend to be narrower and contain zones of low grade material (i.e. less than 0.5 g/t Au).Twangiza East, West and main extension mineralisation style is same as that of the Main.

The East and Western flanking structures are considered as part of the eastern and western limbs of the main Twangiza anticlinal structure as they bear similar characteristics.

The Twangiza North deposit comprises thinner zones (10 – 30 m wide) which are mainly confined to the porphyry units and which have sharp contacts between ore and waste material.

The mineralisation contact is more distinct at Twangiza North than the more diffuse distribution at Twangiza Main, therefore a hard boundary at 0.4 g/t Au has been used at Twangiza North. In the upper portions of the Twangiza Main zone the contacts are more difficult to define and the mineralisation appears to be wider. A statistical analysis of the alteration pattern associated with the mineralisation and general visual appraisal suggests a natural cut-off at 0.3 to 0.5 g/t Au.

The Twangiza North style of mineralisation is a series of thinner zones. The main characteristics of the Twangiza North style ore bodies are the strong relationship between the mineralised material and the lithology. The close relationship again stresses the requirement for an accurate geological model as changes in the geology could cause changes in the mineralisation model. The Twangiza North style mineralisation begins to the north of a faulted zone (northing 682800 N), beyond which mineralisation is limited to the porphyry. The figure above shows the current interpretation of the geometry and situation of each of the mineralised domains.

The mineralised domains were assigned a numeric code. Attempts were made to follow the strike trends of the major zones and to limit the number of mineralised zones (lodes). The main mineralisation zone has been defined as domain 2. The domains were stored in the model under the field KZONE as summarised in Table 14-5 below.

Table 14-5: Summary of Kriging Zones Used in the Latest Block Model

KZONE	DESCRIPTION
KZONE 1	UPPER OXIDE (CAP)
KZONE 2	MAIN MINERALISATION ZONE
KZONE 3 - 6	SECONDARY MINERALISED ZONES

14.5.3 Geological block model

A 10 x 10 x 5 m prototype parent block was created with sub-blocking allowed along the boundaries to a minimum of 2.5 m along strike and 1 m across strike and in the vertical direction. Further sub-blocking was used at surface.

Table 14-6: Details of Block Model Dimensions For Geological Model

BLOCK EDGE	ORIGIN	BLOCK SIZE	NO OF BLOCKS
MIN X	692,000	10	250
MIN Y	681,500	10	450
MIN Z	1,500	5	220

Using the wireframes described above, a series of codes were developed to describe each of the major geological properties of the rock types; these were used for the pit optimisation exercise. Table 14-7 summarises additional fields created within the geological model and the codes used.

Table 14-7: Summary of Fields Used For Flagging Different Geological Properties

FIELD NAME	CODE	DESCRIPTION
LITH1	100	PORPHYRY
	101	SEDIMENT
OXTRAN	1	UPPER OXIDE
	2	LOWER OXIDE
	3	TRANSITIONAL
	4	FRESH ROCK
SRKZONE	1	TWANGIZA MAIN
	2	TWANGIZA NORTH
HARD	1	SOFT
	2	HARD
HARDNESS	VARIABLE (MPa)	AS DESCRIBED IN Table 14-3

Further alphanumeric codes described in Table 14-8 and Table 14-9 were stored in the new field “ROCK”. The model is split into two components based on a northing of 682,800 N, marking the change from Twangiza Main type mineralisation to Twangiza North mineralisation.

Table 14-8: Rock Codes Used in Twangiza Main

	MEASURED		INDICATED		INFERRED	
	PORPHYRY	SEDIMENT	PORPHYRY	SEDIMENT	PORPHYRY	SEDIMENT
OXIDE 1	1001	1011	1002	1012	1003	1013
	MMPO1	MMSO1	MDPO1	MDSO1	MFPO1	MFSO1
OXIDE	1021	1031	1022	1032	1023	1033
	MMPO	MMSO	MDPO	MDSO	MFPO	MFSO
TRANSITION	1041	1051	1042	1052	1043	1053
	MMPT	MMST	MDPT	MDST	MFPT	MFST
SULPHIDE	1061	1071	1062	1072	1063	1073
	MMPS	MMSS	MDPS	MDSS	MFPS	MFSS

Table 14-9: Rock Codes Used in Twangiza North

	MEASURED		INDICATED		INFERRED	
	PORPHYRY	SEDIMENT	PORPHYRY	SEDIMENT	PORPHYRY	SEDIMENT
OXIDE 1	2001	2011	2002	2012	2003	2013
	NMPO1	NMSO1	NDPO1	NDSO1	NFPO1	NFSO1
OXIDE	2021	2031	2022	2032	2023	2033
	NMPO	NMSO	NDPO	NDSO	NFPO	NFSO
TRANSITION	2041	2051	2042	2052	2043	2053
	NMPT	NMST	NDPT	NDST	NFPT	NFST
SULPHIDE	2061	2071	2062	2072	2063	2073
	NMPS	NMSS	NDPS	NDSS	NFPS	NFSS

14.6 Topography, oxide/transition sub-models

A LIDAR survey was used to create a digital terrain model within Datamine to represent the original topography which included the artisanal open pit in the oxide zone of the Twangiza Main deposit.

This has been updated by the end December 2014 mine survey to account for all depletion to date.

The oxide and transitional models were created by linking the cross-sectional interpretations to form single surfaces within Datamine. These surfaces have been used during the zoning process of samples and in creating the final block model.

14.7 Statistical analysis of the mineralised data

14.7.1 Selection of composite lengths for statistics

Sample lengths are varied as shown in Figure 14-5. To ensure no bias exists in the compilation of the statistics and geostatistics a standard composite 2.0 m length was used.

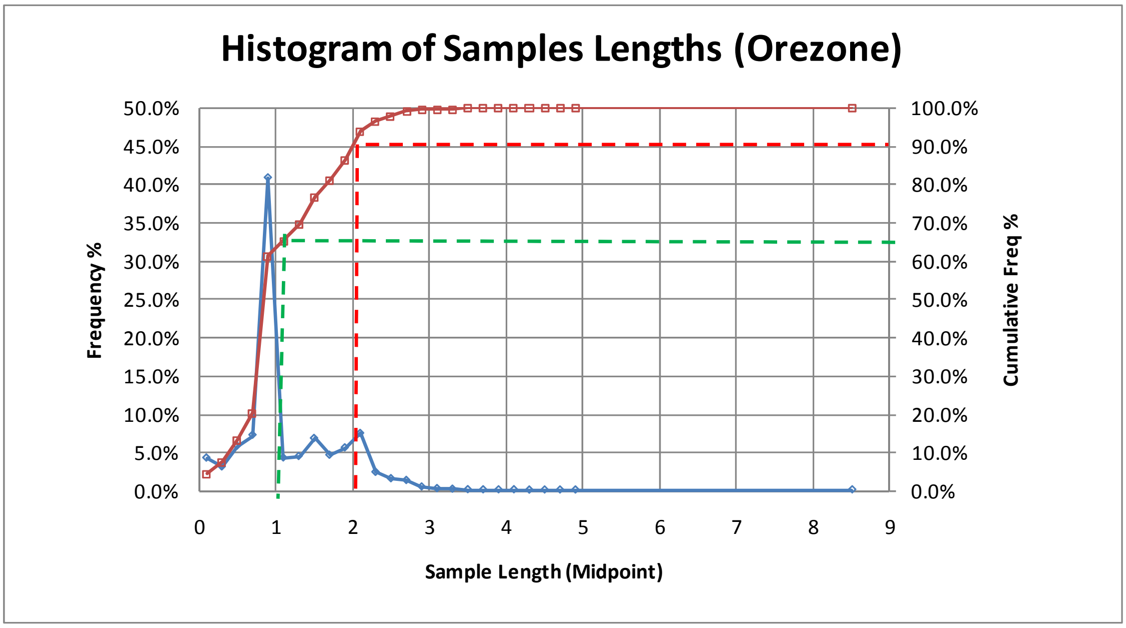


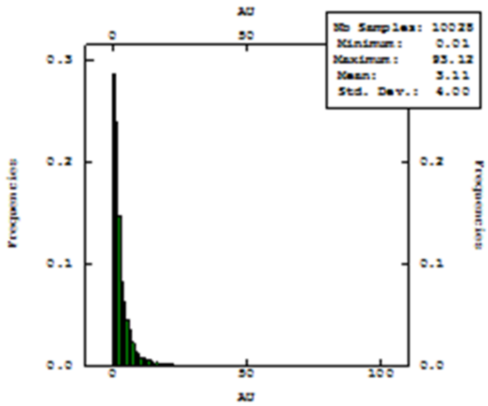
Figure 14-5: Histogram of Sample Lengths In Twangiza Database

14.7.2 Summary statistics and histograms

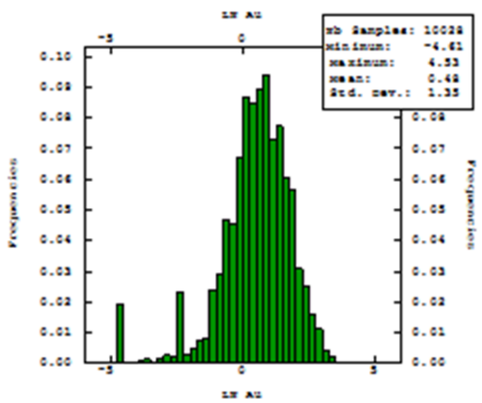
Each of the individual zones was assessed independently as illustrated in Table 14-10 from the summary statistics. On reviewing the summary statistics SRK (UK) took the decision to combine the different deposit areas and to only subdivide zones based on the oxidation state.

Table 14-10: Summary Statistics of 2m Composites

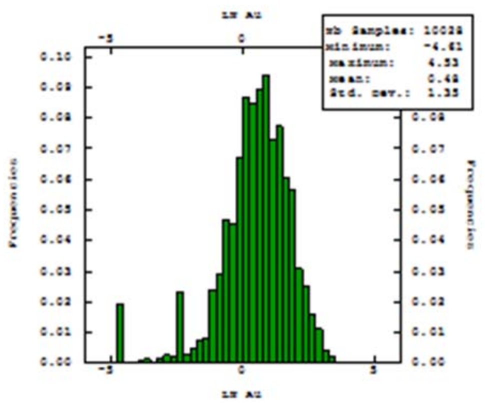
FIELD	N	MIN	MAX	MEAN	VAR	CoV	DESCRIPTION	FIELD
Au	28,487	0.01	157.82	2.51	14.28	1.50	ALL 2m SAMPLES	Au
Au	10,028	0.01	93.12	3.11	15.99	1.29	UPPER OXIDE	Au
Au	4,772	0.01	69.00	3.13	15.62	1.26	LOWER OXIDE	Au
Au	4,544	0.01	157.82	2.78	23.87	1.76	TRANSITIONAL	Au
Au	9,143	0.01	60.54	1.41	5.08	1.60	SULPHIDE	Au
Au	10,034	0.01	93.12	3.11	15,99	1.28	KZONE 1	Au
Au	15,369	0.01	157.82	2.39	14.36	1.59	KZONE 2	Au
Au	787	0.01	9.66	1.07	1.76	1.24	KZONE 3	Au
Au	2,175	0.01	60.54	1.23	6.71	2.10	KZONE 4	Au
Au	11	0.39	1.80	0.89	0.23	0.54	KZONE 5	Au
Au	111	0.03	9.31	1.60	2.69	1.03	KZONE 6	Au
Au	405	0.01	10.21	0.87	1.57	1.45	TW OX	Au
Au	363	0.01	6.99	0.47	0.69	1.75	TW TR	Au
Au	430	0.01	15.16	0.99	1.83	1.37	TE OX	Au
Au	22	0.15	3.17	0.86	0.51	0.82	TE TR	Au



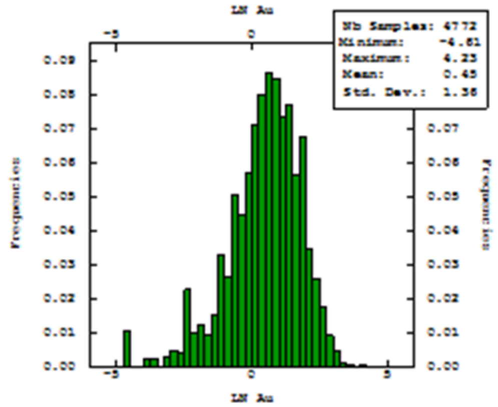
(a) Upper Oxides



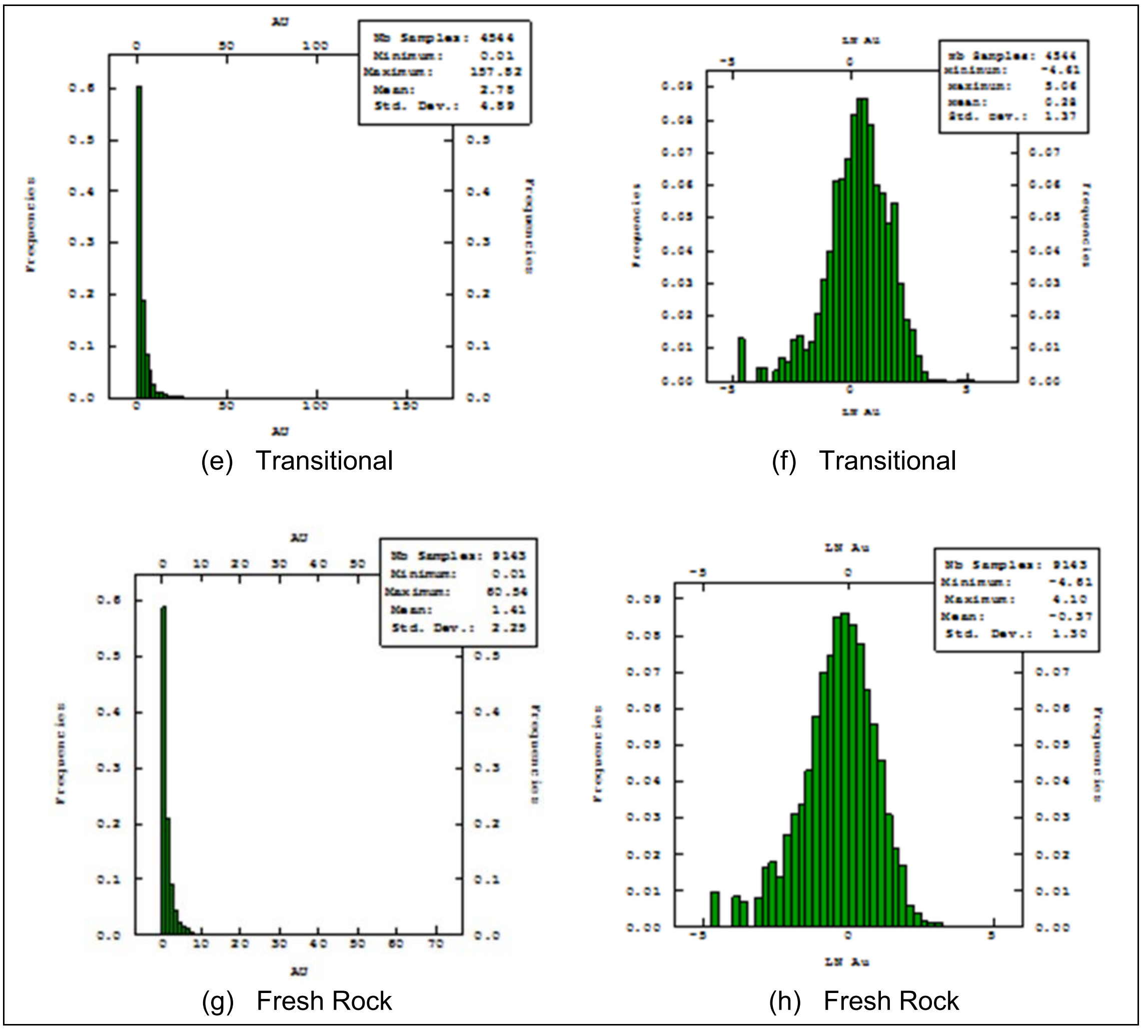
(b) Upper Oxides



(c) Lower Oxides



(d) Lower Oxides



(e) Transitional (f) Transitional

(g) Fresh Rock (h) Fresh Rock

Figure 14-6: Comparative Histograms Per Oxidation Domain

The statistical distributions for each of the individual zones display similar properties and have distributions which tend towards log normal. The histograms and log histograms per domain are shown in Figure 14-6 below. Descriptive statistics were calculated and statistical graphs produced in both real and log space as a measure of assessing:

- The population characteristics of the mineralised grade distribution;
- Confirmation of the statistical domains, and possible combining of zones for geostatistics;
- The need, if any, to apply a top-cut during grade interpolation.

14.7.3 High grade capping

Plots of the composite assay grades against the cumulative mean and cumulative CoV were produced for each of the sample types within the different oxidation domains, an example is given in Figure 14-7. The plots were used to distinguish the grade at which the cumulative totals CoV becomes extreme. Using this methodology top-cuts were defined for each domain as summarised in Table 14-11.

Furthermore, log-probability plots have been checked to ensure the top-cuts applied are applicable and the spatial occurrence of the extreme cut gold values was visually verified to determine if they formed discrete zones.

Table 14-11: High-Grade Capping

DOMAIN	GRADE CAPPING (g/t)
UPPER OXIDE	20
LOWER OXIDE	25
TRANSITION	20
SULPHIDE	10

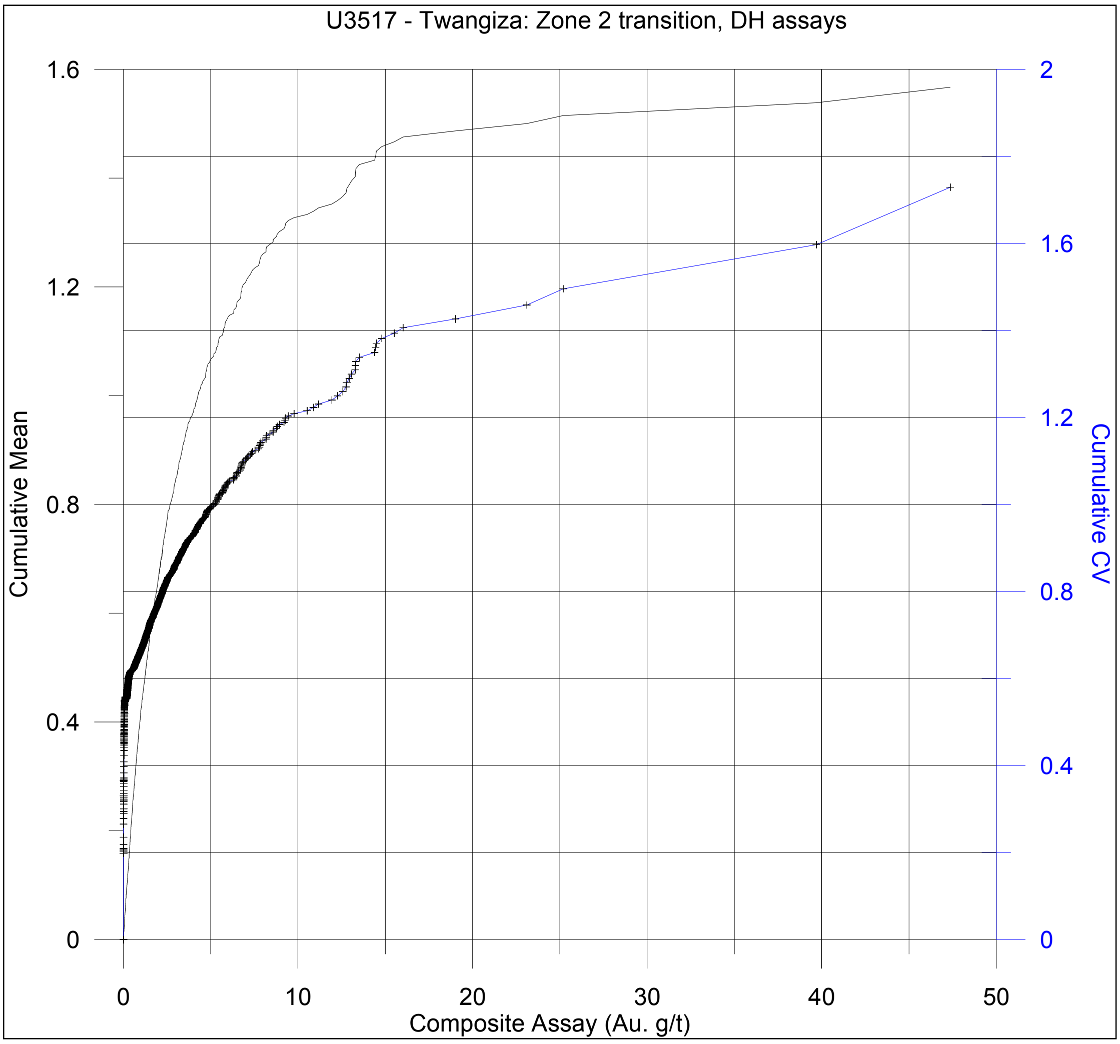


Figure 14-7: Example Plot of Cumulative Mean and Covariance

14.8 Geostatistical analysis

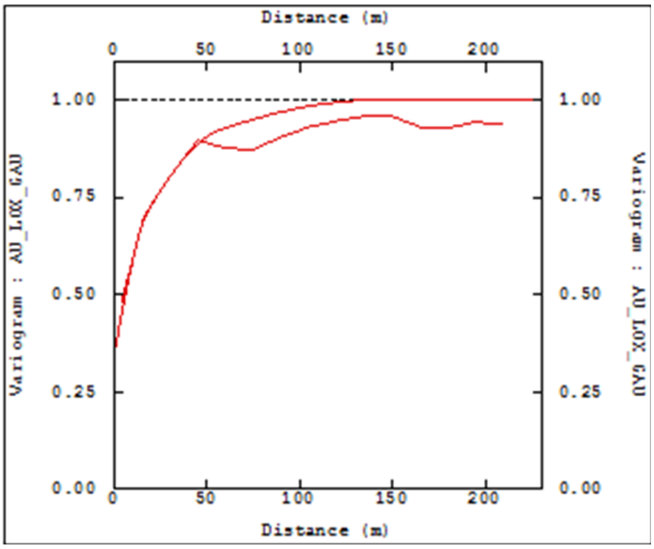
Geostatistical analysis was carried out on the selected composite samples and for the various material types (upper oxide, lower oxide, transition and sulphide). Initially variographic analysis was completed to establish any directional anisotropy. Based on the results of the semi-variograms the search ellipse and the kriging parameters were optimised.

14.8.1 Semi-variograms

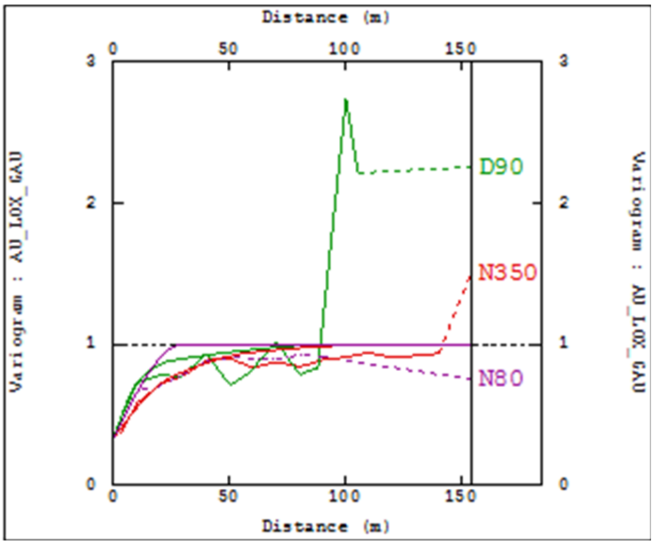
Semi-variograms were estimated for each of the four oxidation domains described earlier. Initially the data was transformed using the Discrete Gaussian transformation algorithm. Semi-variogram models were produced in this transformation for the experimental variograms and these models were then back transformed into the untransformed data space for use in the kriging routine.

Variograms were produced in the three principal directions along strike, down dip and across the strike. In addition to these, an omni-directional down-hole variogram was produced, and used to determine the nugget variance which was applied to the directional variograms. Figure 14-8 gives an example of the down hole and directional Gaussian variograms produced for the Lower Oxide domain.

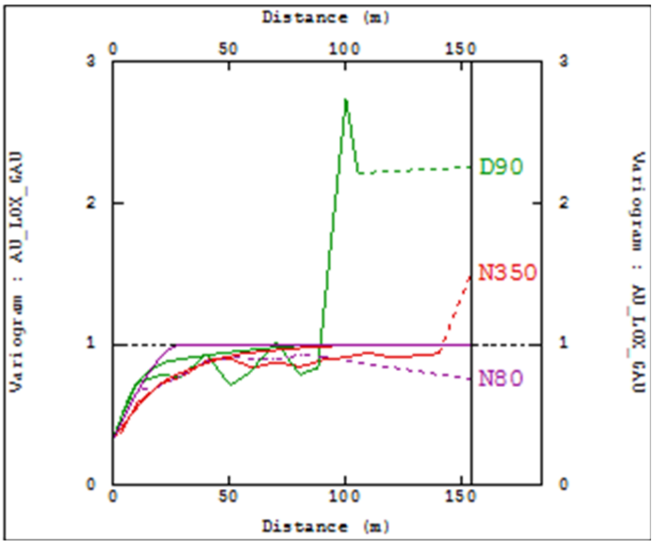
The best variograms were produced from the along strike direction. Those produced for the down dip directions were affected by the limited data extent in this direction. The back-transformed Gaussian variograms for the Lower Oxide domain are also included below. The final semi-variograms per zone can be found in Appendix V of the 2009 Feasibility Study.



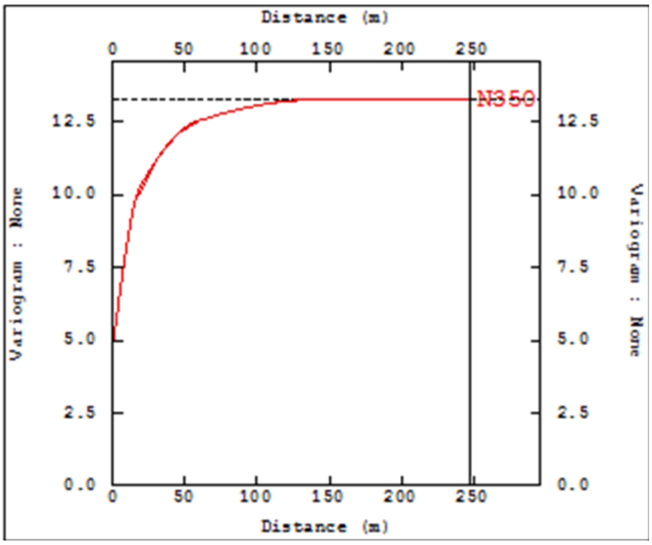
(a) Down Hole Gaussian SV, LO domain



(b) Directional Gaussian SV, LO domain



(c) Back-transformed Down Hole Gaussian SV, LO domain



(d) Back-transformed Directional Gaussian SV, LO domain

Figure 14-8: Summary of Gaussian Transformed Variograms For Lower Oxide Zone

The back-transformed Gaussian variogram parameters determined for each of the domains are included in Table 14-12. All zones are characterised by a nugget variance in the order of 40 – 45% of the sill value, while the fresh rock semi-variograms display a nugget variance in the order of 56%, which indicates a relatively high degree of variability.

Table 14-12: Back-Transformed Gaussian Variogram Parameters

VARIOGRAM PARAM.	UPPER OXIDE	LOWER OXIDE	TRANSITION	SULPHIDE
Co	5.15	5.96	4.39	1.52
C1	3.26	2.45	2.27	0.47
a1 – ALONG STRIKE (m)	12	25	25	20
a1 – DOWN DIP (m)	13	12	10	15
a1 – ACROSS STRIKE (m)	20	30	10	20
C2	2.56	3.07	2.39	0.73
a2 – ALONG STRIKE (m)	55	60	30	80
a2 – DOWN DIP (m)	40	25	80	60
a2 – ACROSS STRIKE (m)	50	30	50	32
C3	1.29	1.81	1.89	-
a3 – ALONG STRIKE (m)	160	120	160	-
a3 – DOWN DIP (m)	150	100	80	-
a3 – ACROSS STRIKE (m)	60	30	60	-
NUGGET EFFECT (%)	42.10	60.94	40.13	55.88

14.8.2 Block estimation

Grade estimation was performed using Ordinary Kriging routines within the Datamine software package. A quantitative Kriging Neighbourhood Analysis (QKNA) exercise was completed in order to optimise the kriging parameters; this was completed within the Isatis software package. Each of the four oxidation domains was optimised individually with the search ellipses rotated into the plane of the ore body domain to take account of the anisotropy identified during the semi-variogram analysis.

The Twangiza North domain search was oriented with the major (x) search axis striking towards 340° and dipping 75° to the west. A second rotation with the major (x) search axis striking towards 340° and dipping 45° to the west, has also been tested as there was evidence within the Twangiza Main deposit of high-grades running in this orientation. The general anisotropy defined in the Geostatistical studies of the main zones were used to define the orientated search ellipses for these zones as they are consistent with the interpreted geology of the deposit. Dynamic anisotropy was employed in the estimation process for Twangiza East and West estimation.

In general the results displayed good slopes of regression (i.e. greater than 0.8) within the well informed areas for all scenarios. A summary of the final parameters selected are shown in Table 14-13 below.

A minimum of 4 and a maximum of 18 composites were used to estimate a block in the first pass for all zones. This represents an increase from a minimum of 2 and a maximum of 12 composites used in the previous model. The block discretization for the interpolations was 4 x 4 x 4.

Table 14-13: Search Radius For Pass 1

PASS 1 PARAMETERS	UO DOMAIN	LO DOMAIN	TR DOMAIN	FR DOMAIN
RADIUSX (m)	130	130	130	80
RADIUS Y (m)	100	100	80	60
RADIUS Z (m)	20	20	20	20
MIN. NO. OF SAMPLES	4	4	4	4
NO. OF SECTORS	4	4	4	4
MAX NO. OF SAMPLES	18	18	18	18
DESCRITIZATION	4x4x4	4x4x4	4x4x4	4x4x4

The first pass employed a search ellipse which equates to approximately the semi-variogram range. The longest dimension of the ellipse was approximately equal to one of the range of the variogram. Approximately 90 % of the blocks within the current economic pit were estimated using the first search range

Axis multipliers were set at 2 and 3 for the second and third search volumes. The number of samples used was increased in the second search volume to produce more averaged grades.

The compiled block model was checked within the Datamine software package for missing or duplicated estimates to ensure there is no double accounting of ore tonnage.

14.9 Mineral Resource Classification

The definitions provided in the following section are taken from the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Standards on Mineral Resources and Reserves.

A Mineral Resource is a concentration or occurrence of diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals in or on the Earth's crust in such form and quantity and of such a grade or quality that it has "reasonable prospects for economic extraction." The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge.

The term "reasonable prospect for economic extraction" implies a judgement (albeit preliminary) by the Qualified Person in respect of the technical and economic factors likely to influence the prospect of economic extraction, including the approximate mining parameters. In basic terms the code highlights that the Mineral Resource is not simply an inventory of all mineralisation drilled or sampled, regardless of cut-off grade, likely mining dimensions, location and continuity. The Mineral Resource is therefore the portion of the Mineralised Block Model which under assumed and justifiable technical and economic conditions in whole or in part may become economically extractable.

14.9.1 Geological complexity

As highlighted above, the oxidation and lithological model at Twangiza is important in terms of metal recoveries of the different units. The porphyry units provide reasonably large continuous zones which have been modelled with a reasonable level of confidence within well drilled areas. The geological knowledge, detailed interpretation and good data density in well drilled areas and areas with adits, have allowed the resource to be classified with high confidence in some places.

14.9.2 Quality of data used in the estimation

Twangiza Mining used best methods of sampling and sample preparation. Twangiza Mining has conducted a systematic process of sample preparation within clean facilities, which is well documented. The systems include regular insertion of blanks, standards and duplicates in all sample submission to the laboratory in Mwanza. In general, the results from the QA/QC programme indicate acceptable levels of errors are achieved at the laboratory.

14.9.3 Results of the geostatistical analysis

Based on the variograms, the QKNA was undertaken and the results showed a good slope of regression within the well informed and reasonably informed blocks. The slope of regression values, however, reduced in areas of limited sampling where the sample support is reduced.

14.9.4 Classification Method

The classification was carried out using a combination of drill hole spacing, slopes of regression, geological and wireframe confidence. Classification was applied to the model by digitizing areas on 80m spaced vertical sections based on

- Measured Mineral Resource consists of kriged model blocks which have been interpolated by data within 20m and are limited to the areas surrounding the adit sampling within the Twangiza Main proportion of the deposit, with extensions of 10m–20m below the deepest adit;
- Indicated Mineral Resources are those kriged blocks which have been interpolated by adit and drill hole data using an average drill hole spacing of 40 x 40m within the search area. A minimum number of points used to estimate a block grade has also been reviewed as have the search volumes and the slope of regression; and
- Inferred Mineral Resources are model blocks lying outside the first search area estimated by OK and those blocks which are deemed to be poorly informed and therefore have a low associated slope of regression. The down-dip extensions of the Inferred Mineral Resources have been limited to 50m–100m from the Indicated boundary and to remove interpretations based on a single drill hole spacing of 150m.

Areas which were modelled as part of the geological model but which fell outside of the Inferred boundary limits were flagged in the model as areas for potential additions to the Mineral Resources. These areas require additional drilling before being classified as a Mineral Resource.

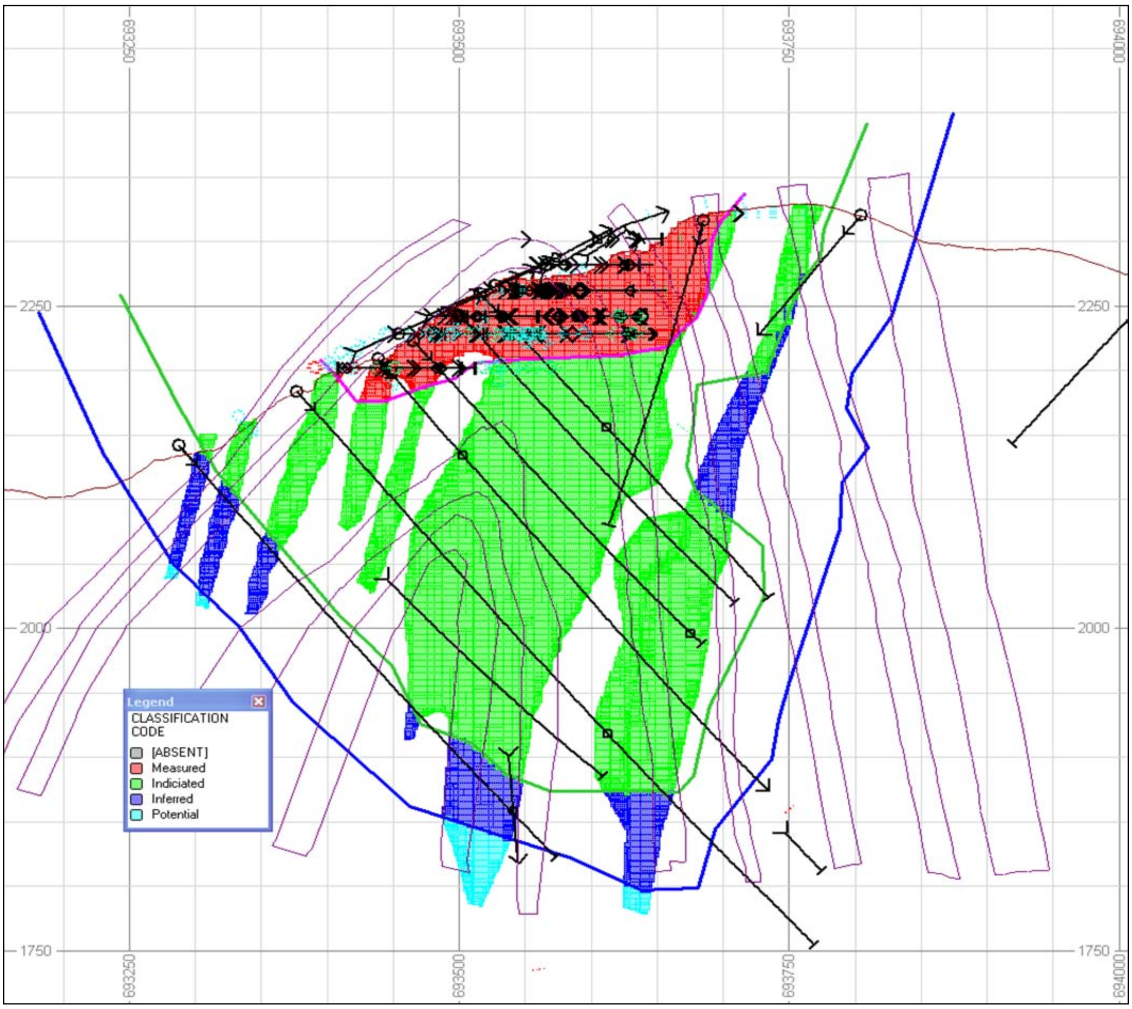


Figure 14-9: Typical Section Displaying Classification

14.10 Valley Fill model

The Mwana River Valley Fill deposit comprises tailings that resulted from historical artisanal mining and washing of oxide ore in the Mbwega pit at Twangiza Main. The deposit runs to the west and northwest of Twangiza Main as shown in Figure 14-10.

To build on the Inferred Mineral Resource in the 2009 Feasibility Study, further sampling has been undertaken in the last quarter of 2009 and the end of 2010 to obtain a better understanding of the grade distribution of the gold, average gold grade and the thickness of the deposit.

50 auger holes were drilled to a maximum of 7m length, totalling 161m generating 113 samples. 14 pits up to 4.2m deep totalling 35.1 m were dug further downstream generating 50 samples. Sample spacing is approximately 100m along strike by 40m across sections and covering a total area of some 260,000m^2.

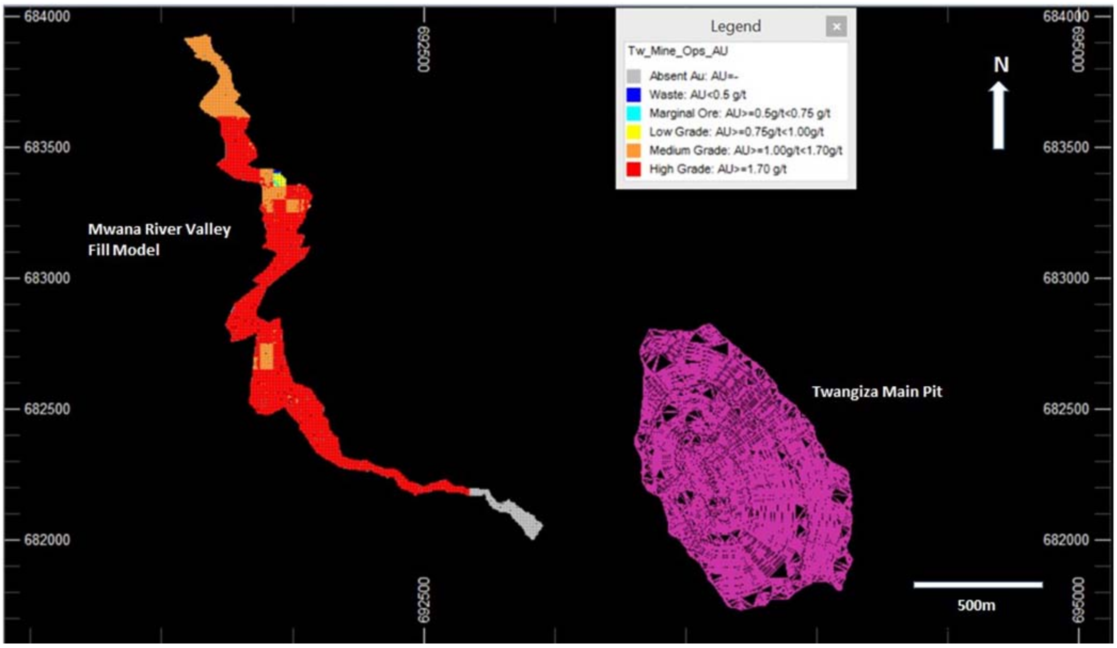


Figure 14-10: Plan Showing Valley Fill and Twangiza Main

Based on the sampling, sectional interpretations were digitized along the auger and pitting lines and the perimeters were linked together to generate a 3D wireframe in Datamine Studio software. The total volume of the model was 1,876,157m3 and the model had an average thickness of 7m. A 3D block model was created from the wireframe as summarised in Table 14-14.

Table 14-14: Valley Fill Block Model Parameters

Model Parameters	Block Dimension	Min	Max	Range
X	50	691580	692980	1400
Y	50	681950	683950	2000
Z	2	1850	2000	150

Samples were composited to 2m and composited sample grades were used to interpolate grade into the blocks using ordinary Kriging to generate a 3D resource model. A density factor of 1.65t/m^3, determined by weighing dried samples from known volumes, was applied for tonnage estimation.

By the end of 2013, 0.33 Mt of Valley Fill material grading 3.13 g/t Au for 0.03 Moz of gold had been mined from the upstream part of the deposit which was removed to allow construction of the TMF1A dam wall and avoid sterilisation. At the end of this mining activity the mined out area of approximately 42,810m2 was surveyed and the outlines were used to deplete the Valley Fill Mineral Resource model. During the mining process some areas were mined to a depth of up to 10m.

Between February and March 2014, some 17 additional pits were sampled in the Mwana river bed. A total of 46 samples were collected and assayed and the results were used to update the Valley Fill Mineral Resource model. This additional sampling along with the good agreement of the mined grades enabled the Valley Fill Mineral Resource to be upgraded to the Indicated category and to be included in the Mineral Reserve.

The remaining Valley Fill Mineral Resource reported at 0.40g/t insitu Au cut-off is 2.18 Mt at an average grade of 1.93g/t for 0.13 Moz gold. Whilst the grades further downstream are relatively low, being further from the point of deposition, all the block grades in the resource model are above cut-off. The Valley Fill resource contributes 2% of the Measured and Indicated Mineral Resource at Twangiza.

14.11 Mineral Resource Statement

An updated Mineral Resource estimate as at end December of 2014, has been prepared as reported in Banro's press release dated June 08, 2015. As a basis for the update, the Mineral Resource model was modified to reflect changes to material densities (mainly upper oxide zone), artisanal mining, topography and mining depletion as determined by an historical reconciliation review and survey update as at end December 2014. The estimate had a 0.40 g/t gold cut-off grade applied and was constrained to a USD1,600/oz gold price optimum pit shell, which was considered an appropriate price for this purpose, being 33% above the prevailing forecast gold pricing.

The updated resource reflects the remaining resource at the end of year 2014 after all depletions due to mining. Table 14-15 below details the "Oxide" and "Non-Oxide" components of the 2014 Twangiza Mineral Resource estimate, split by confidence category, reported at a cut-off grade of 0.40 g/t gold.

Table 14-15: Mineral Resource Estimate as at December 31, 2014

OXIDE MINERAL RESOURCE CATEGORY	Tonnes (Mt)	Grade (g/t Au)	Gold (Moz)
MEASURED	3.72	2.3	0.28
INDICATED	8.76	1.88	0.53
MEASURED AND INDICATED	**12.48**	**2.02**	**0.81**
INFERRED	1.34	1.32	0.06
NON-OXIDE MINERAL RESOURCE CATEGORY	**Tonnes (Mt)**	**Grade (g/t Au)**	**Gold (Moz)**
MEASURED	3.8	2.23	0.27
INDICATED	93	1.4	4.18
MEASURED AND INDICATED	**96.8**	**1.43**	**4.45**
INFERRED	11.65	1.12	0.42

NB: Any apparent errors are due to rounding and are therefore not considered material to the estimate

The details of the Twangiza Mineral Resource by pit, material type (oxidation state) and confidence category are outlined in Table 14-16.

Table 14-16: Mineral Resource Estimate by Pit, Material Type and Confidence Category as at December 31, 2014

PROSPECT	MATERIAL	MEASURED MINERAL RESOURCES			INDICATED MINERAL RESOURCES			INFERRED MINERAL RESOURCES		
		Tonnes (Mt)	Average Grade (g/t)	Gold Content (Moz)	Tonnes (Mt)	Average Grade (g/t)	Gold Content (Moz)	Tonnes (Mt)	Average Grade (g/t)	Gold Content (Moz)
Twangiza Main	Oxide	3.14	2.54	0.26	2.15	1.43	0.10	0.39	0.93	0.01
	Transition	3.58	2.25	0.26	9.55	1.63	0.50	0.76	0.96	0.02
	Fresh	0.22	1.80	0.01	75.96	1.30	3.18	8.76	1.15	0.32
Twangiza Main Sub Total		**6.94**	**2.37**	**0.53**	**87.66**	**1.34**	**3.78**	**9.91**	**1.13**	**0.36**
Twangiza North	Oxide				1.75	2.63	0.15	0.00	1.24	0.00
	Transition				2.12	2.21	0.15	0.04	1.53	0.00
	Fresh				1.57	2.43	0.12	0.01	3.04	0.00
Twangiza North Sub Total					5.44	2.41	0.42	**0.05**	**1.86**	**0.00**
Twangiza Central	Oxide	0.01	1.16	0.00	2.11	1.86	0.13	0.41	2.08	0.03
	Transition				1.19	1.80	0.07	0.42	2.01	0.03
	Fresh				1.57	2.43	0.12	0.01	3.04	0.00
Twangiza Central Sub Total		**0.01**	**1.16**	**0.00**	**4.87**	**2.03**	**0.32**	**0.85**	**2.06**	**0.06**
Twangiza East	Oxide							0.07	0.75	0.00
	Transition									
	Fresh									
Twangiza East Sub Total								**0.07**	**0.75**	**0.00**
Twangiza West	Oxide	0.00	0.00	0.00	0.58	1.11	0.02	0.46	0.95	0.01
	Transition	0.00	0.00	0.00	0.36	1.26	0.02	0.56	0.81	0.01
	Fresh	0.00	0.00	0.00	0.67	1.33	0.02	1.08	0.77	0.03
Twangiza West Sub Total		**0.00**	**0.00**	**0.00**	**1.61**	**1.11**	**0.06**	**2.10**	**0.82**	**0.06**
Valley Fill					**2.18**	**1.93**	**0.14**			
Stockpile		**0.57**	**1.07**	**0.02**						
TOTAL		**7.52**	**2.27**	**0.55**	**101.76**	**1.44**	**4.71**	**12.99**	**1.14**	**0.48**

15 MINERAL RESERVE ESTIMATE

15.1 Geotechnical

The current pit optimisation and design applied for estimation of the updated Mineral Reserve relies on the geotechnical analysis completed as part of the PFS.

A full description of the PFS geotechnical investigation and slope stability analysis is given in Section 16.1 of the SRK (UK) report entitled “Pre-Feasibility Study NI43-101 Technical Report, Twangiza Gold Project, South Kivu Province, Democratic Republic of Congo”, dated August 13, 2008.

Based on the slope parameters derived by SRK (UK) for the PFS, the following overall slope angles were used for the pit optimization taking into consideration ramp sections:

- Overall slope angle in soil/saprolite material of 27.5 to 30°;
- Overall slope angle in oxide material of 34 to 38°;
- Overall slope angle in transitional material of 41 to 42°; and,
- Overall slope angle in fresh material of 53°.

These parameters assume dry mining conditions.

15.2 Open pit optimization

15.2.1 Introduction

The 2015 open pit optimisation assumes a 1.7Mtpa throughput rate and is based on the resource block model generated by Twangiza Mining as presented in this report.

A conventional open pit shovel and truck method will be used for the mining of sufficient ore to supply 1.7Mtpa of ore throughput. The mining functions of the operation will be owner mining as per the current practise.

The Whittle process requires various input data including the resource block model, unit costs and other physical parameters such as the slope angles at which the pit can be mined. Unit costs specific to the Twangiza operation were determined by Twangiza Mining from a zero based budgeting exercise and an analysis of historical costs.

The open-pit optimization study was performed using the Whittle/Gemcom Four-X Analyser (Whittle 4X) software package to provide guidance to the potential economic final pit geometries. Whittle 4X compares the estimated value of the individual mining blocks at the pit boundary versus the cost for waste stripping. It establishes the pit walls where the ore revenue and waste stripping cost balance for maximum net revenue.

The optimum pit was considered by Twangiza Mining to be sufficiently similar to the pit designs generated in early 2014 so as not to require a re-design. The ore/waste tonnages in the design pits have been calculated and scheduled to determine the ore production and the waste stripping requirements.

The following sections describe the methodology and derivation of the Whittle input parameters and assumptions.

15.2.2 Cost inputs

Cost input parameters were based on the Twangiza mine historic operating cost as consolidated in the 2015 production year Budget. For the purpose of the Whittle optimization, capital costs, depreciation, amortisation and other interest / finance charges have been excluded.

Mining costs

For the pit optimization analysis, a base unit mining cost of USD 3.28 /t mined for ore and USD 3.51/t mined for waste has been assumed. A USD 0.30 /t mined adjustment for weathered material has been applied to take into account the reduced drilling and blasting required in the upper formations. The mining cost has also been adjusted for the additional pumping cost of USD0.0005/tonne mined for blocks below the valley floor.

Mining unit costs are outlined in Table 15-1.

Table 15-1: Mining Cost Summary

Category	Ore Unit Mining Cost (USD/t)	Waste Unit Mining Cost (USD/t)
Load & Haul (Free Dig)	2.61	2.84
Drill & Blast	0.30	0.30
Grade Control	0.36	0.36
Total Mining	**3.28**	**3.51**

Processing cost

Process costs have been based on the 2015 Budget which is derived from a zero based approach with reference to the 2014 actual costs and adjusted to account for planned cost saving measures, the increased proportion of transition and fresh ore over the LOM and the increase to a 1.7Mtpa throughput rate.

The total process operating cost encompasses metallurgy and process plant maintenance. The average direct process cost was estimated at USD 18.59 /t processed.

General Administration cost

The General and Administration (G&A) cost was also based on 2014 historical estimates as provided under the 2015 Budget adjusted to reflect planned cost savings and increase in production. The average G&A cost was estimated at USD 12.29/t processed.

Revenue and Selling costs

A gold price of USD1,200 per ounce was used for the purpose of reserves estimation and life of mine planning. It was judged to be consistent with prevailing industry estimates. A royalty calculated at the rate of 1% of gross revenue is payable to the Government of DRC.

Selling costs cover all direct and indirect operating costs incurred in carrying out the gold sales and off-site activities. They include Refinery & Shipment, Government Royalties, Head Office Management in the DRC and in Toronto. The average selling cost is USD 63.19 /ounce.

A summary of the Government royalties, refining and selling costs is provided in Table 15-2.

Table 15-2: Government Royalties, Refining and Selling Costs Summary

	Costs (USD/ounce produced)
Refinery and Shipment	15.20
Government Royalties	12.50
Head office cost (Toronto)	12.93
Head office cost (DRC)	21.55
Banro Foundation	1.00
Total Royalties, Refining & Selling	63.19

A breakdown of the costs and parameters used in the Whittle optimization runs are shown in the Table 15-4.

15.2.3 Mining factors

For the pit optimisation, the mining grade dilution factor was set at 5% (at zero grade) and mining recovery at 100%.

15.2.4 Cut-off grade

A recoverable gold cut-off Grade, (“COG”) has been calculated by dividing the overall marginal operating cost of one tonne of ore by the recovered value of the gold contained therein. This is applied to the recoverable gold grade variable housed in the resource block model. This approach simplified the reporting process by incorporating variable recovery by rock type within the recoverable grade value of the resource model. For the various deposits the COG was determined as shown in Table 15-3

Table 15-3: Recoverable Gold Cut-Off Grade by Deposit

	Unit	Total Mine	Main	Central	North	East	West
Cut-Off grade	g/t	0.84	0.84	0.84	0.86	0.84	0.85

15.2.5 Whittle results

A range of optimizations was prepared with the results for a gold price of $1,200 per ounce and a mining cost of USD3.51/t as shown in Figure 15-1, Figure 15-3, Figure 15-5 and Figure 15-7 for each of the deposits.

The updated Whittle shells compare well with the 2014 pit designs so these pit designs have been used for reporting of the end December 2014 Mineral Reserves. Figure 15-2, Figure 15-4, Figure 15-6 and Figure 15-8 below show the block model and the practical pit design.

Table 15-4: Whittle Parameters for Open Pit Optimization

	Unit	Total Mine	Main	Central	North	West
Plant Capacity	Processed tpa	**1,700,000**				
Mining Costs						
Ore	USD/tonne mined	**2.92**	2.92	2.92	3.22	3.20
Waste	USD/tonne mined	**3.15**	3.15	3.15	3.15	3.15
Grade control	USD/tonne mined	**0.36**	0.36	0.36	0.36	0.36
COSTM	**USD/tonne mined**	**3.51**	**3.51**	**3.51**	**3.51**	**3.51**
Rehabilitation Costs						
Waste dumps	USD/waste tonnes mined	**0.10**	0.10	0.10	0.10	0.10
Post Mining Costs						
Process Plant Costs - Carbon In Leach	USD/tonne treated	**7.73**	7.73	7.73	7.73	7.73
Assay	USD/tonne treated	**0.71**	0.71	0.71	0.71	0.71
Power	USD/tonne treated	**7.35**	7.35	7.35	7.35	7.35
Engineering (Maintenance) Costs	USD/tonne treated	**3.21**	3.21	3.21	3.21	3.21
Additional ore cost	USD/tonne treated	**(0.41)**	(0.41)	(0.41)	0.12	0.09
Processing Costs	**USD/tonne treated**	**18.59**	**18.59**	**18.59**	**19.11**	**19.08**
Infrastructure, Overheads and Sundries (G&A)	USD/tonne treated	**10.49**	10.49	10.49	10.49	10.49
Sustaining Capital (Tailings Dam Lifts, Pad Expansions)	USD/tonne treated	**1.80**	1.80	1.80	1.80	1.80
G&A costs	**USD/tonne treated**	**12.29**	**12.29**	**12.29**	**12.29**	**12.29**
COSTP	**USD/tonne treated**	**30.87**	**30.87**	**30.87**	**31.40**	**31.37**
Selling Costs						
Refinery and Shipment	USD/Ounce Produced	**15.20**	15.20	15.20	15.20	15.20
Government Royalty	USD/Ounce Produced	**12.50**	12.50	12.50	12.50	12.50
H/O Management Fee (Toronto)	USD/Ounce Produced	**12.93**	12.93	12.93	12.93	12.93
H/O Management Fee (Banro Congo Mining)	USD/Ounce Produced	**21.55**	21.55	21.55	21.55	21.55
Banro Foundation	USD/Ounce Produced	**1.00**	1.00	1.00	1.00	1.00
Selling cost / oz	**USD/Ounce Produced**	**63.19**	**63.19**	**63.19**	**63.19**	**63.19**
Rehandle - Owner Mining	**USD/tonne treated**	**0.39**	**0.39**	**0.39**	**0.39**	**0.39**
Gold Price (USD/oz)	**1200**					
Cut-Off grade = (costp - rehab dumps)/((Gold Price/31.103475)*(1-Selling Cost %))						
Cut-Off grade	**g/t**	**0.84**	**0.84**	**0.84**	**0.86**	**0.85**

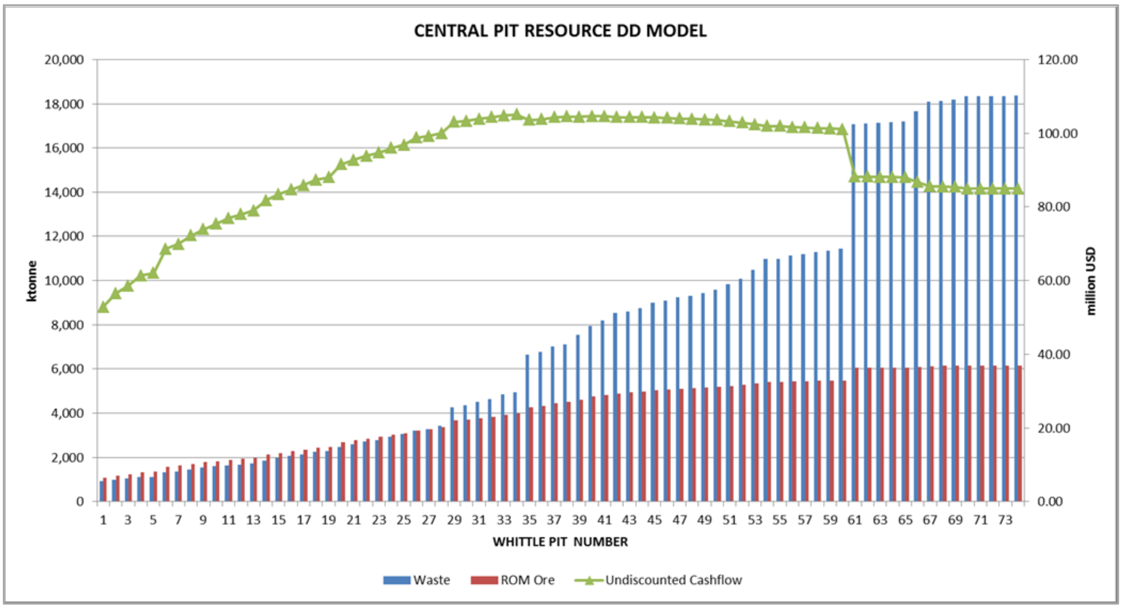


Figure 15-1: Whittle Optimization Results Twangiza Central Deposit December 2014

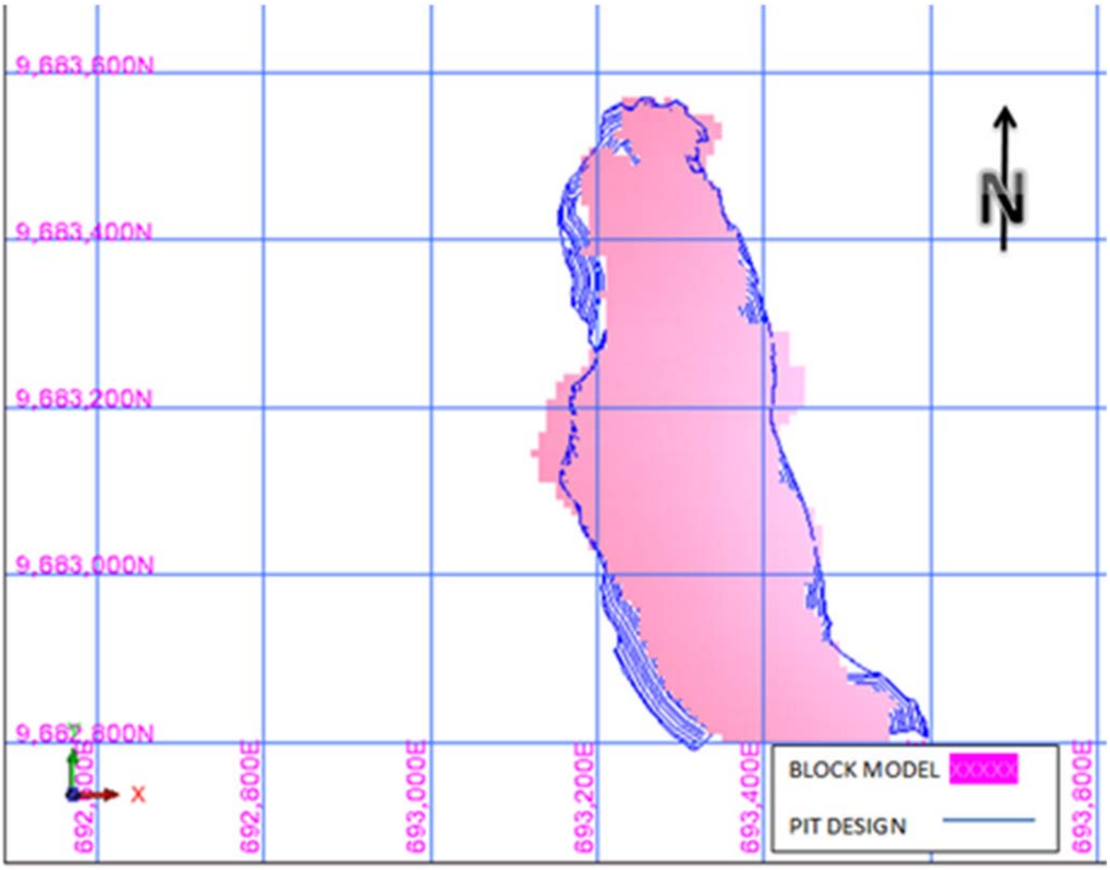


Figure 15-2: Twangiza Central Practical Pit Design Versus Blocks Within Whittle Pit Number 38

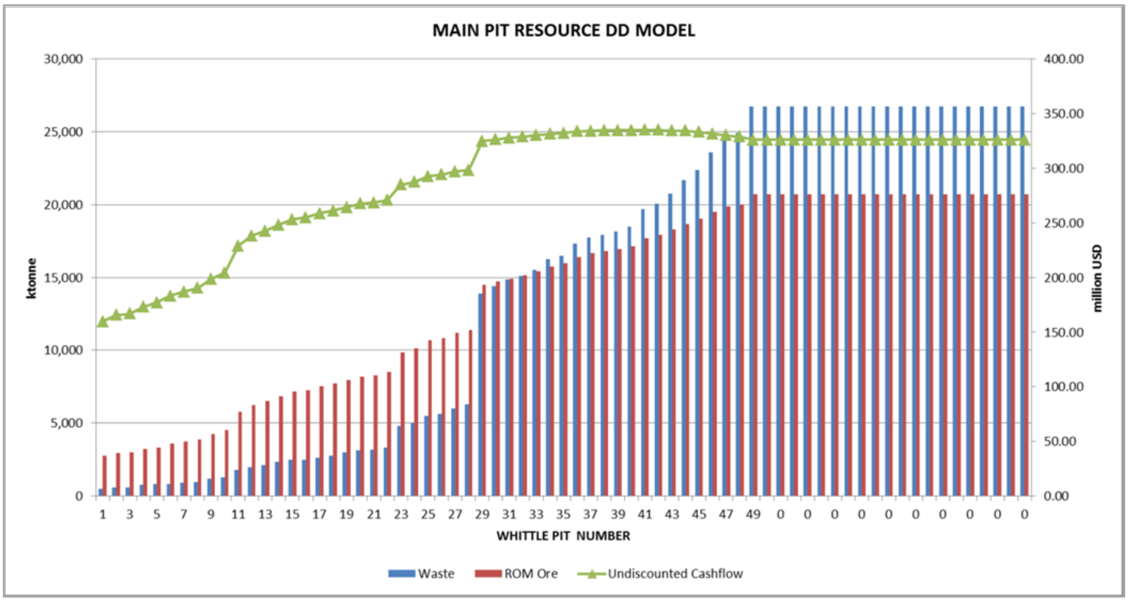


Figure 15-3: Whittle Optimization Results Twangiza Main Deposit December 2014

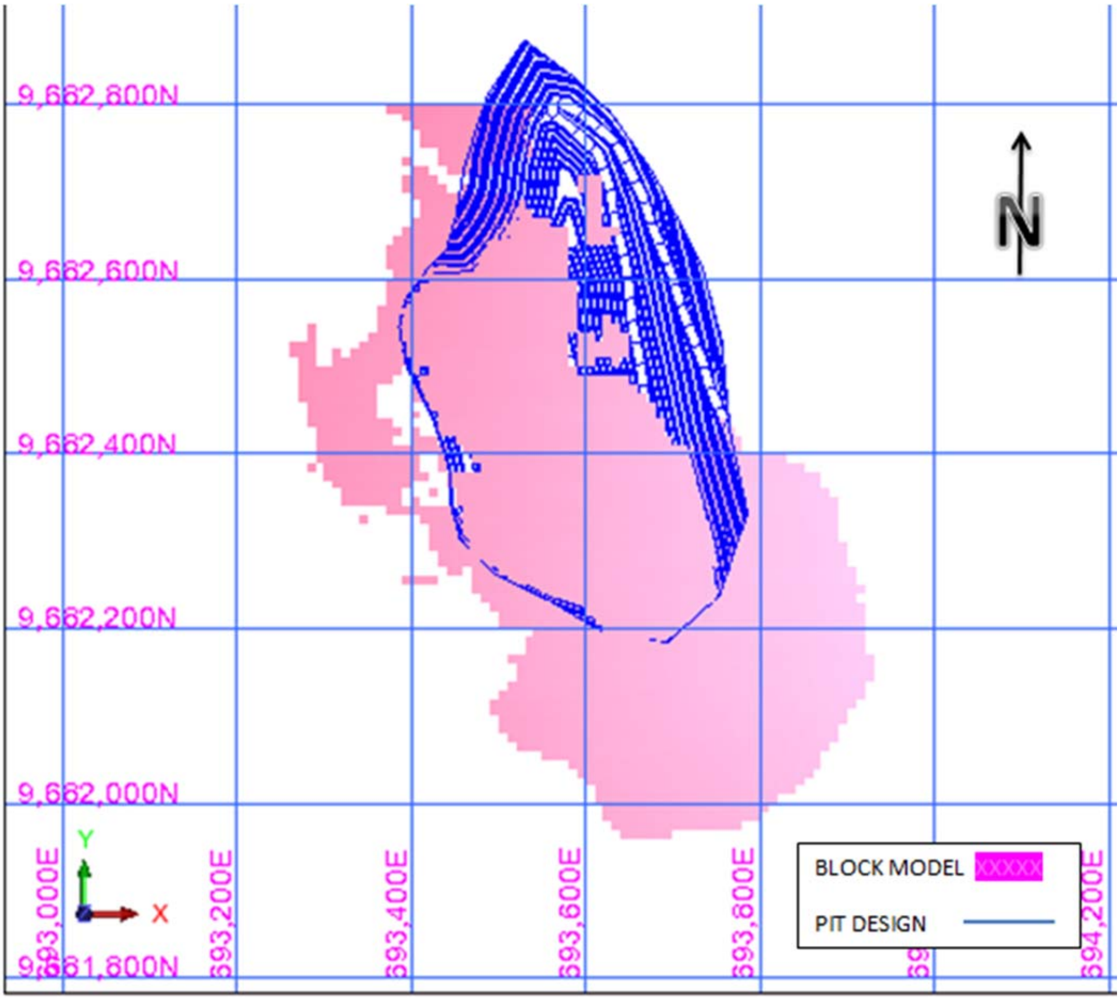


Figure 15-4: Twangiza Main Interim Practical Pit Design Versus Blocks Within Whittle Pit Number 12

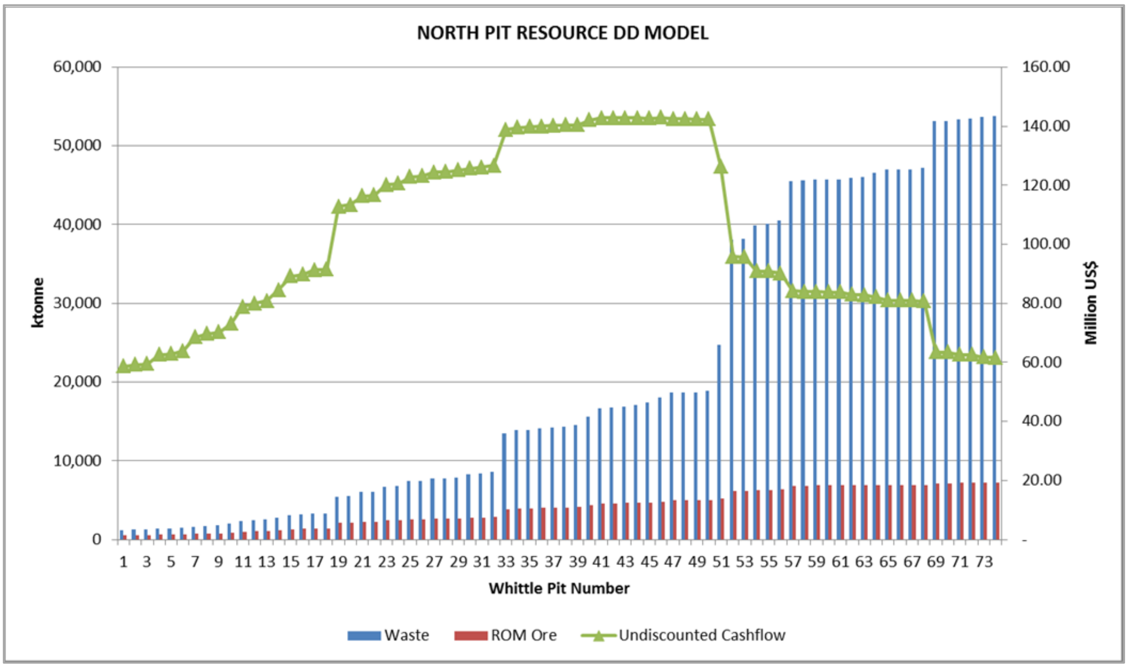


Figure 15-5: Whittle Optimization Results Twangiza North Deposit December 2014

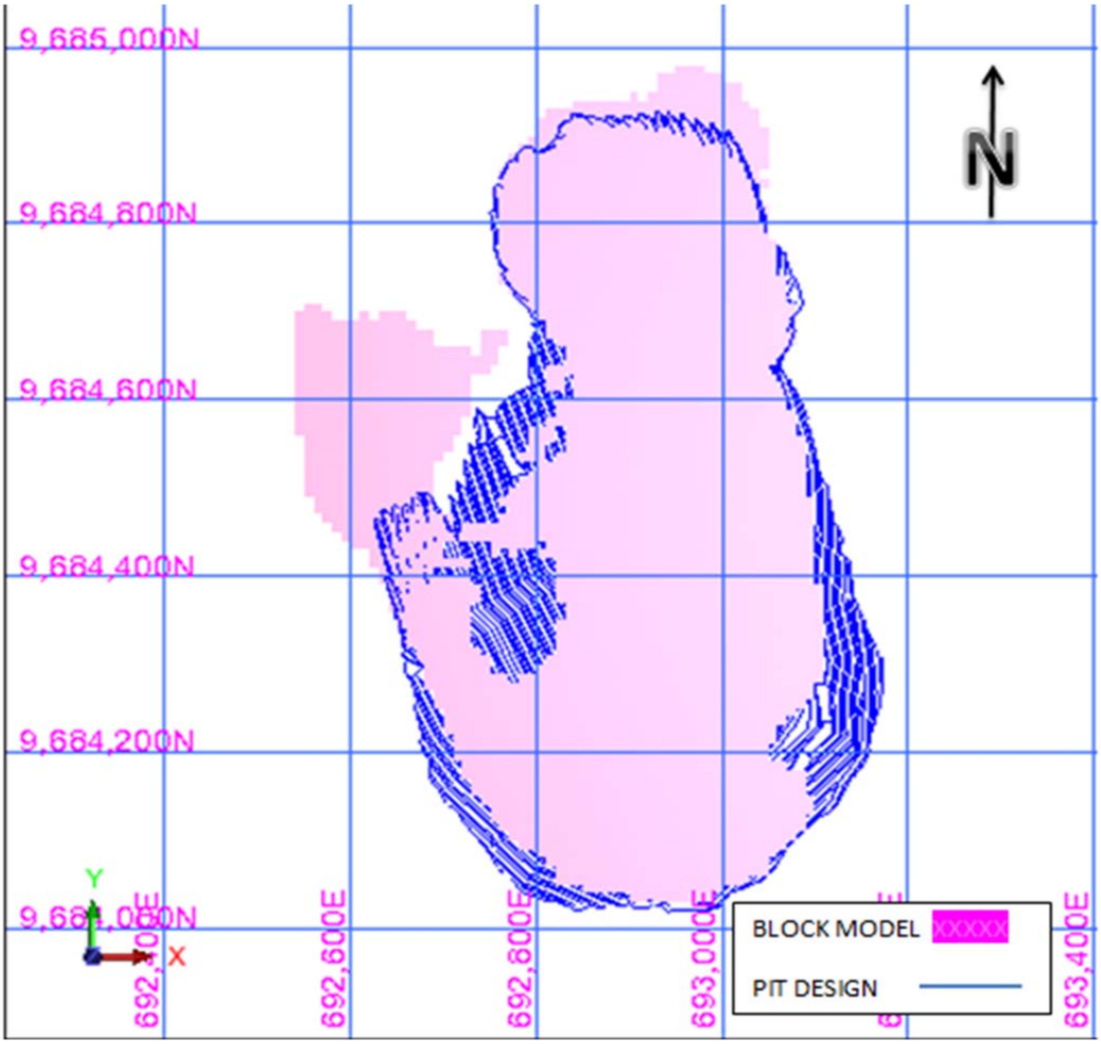


Figure 15-6: Twangiza North Practical Pit Design Versus Blocks Within Whittle Pit Number 29

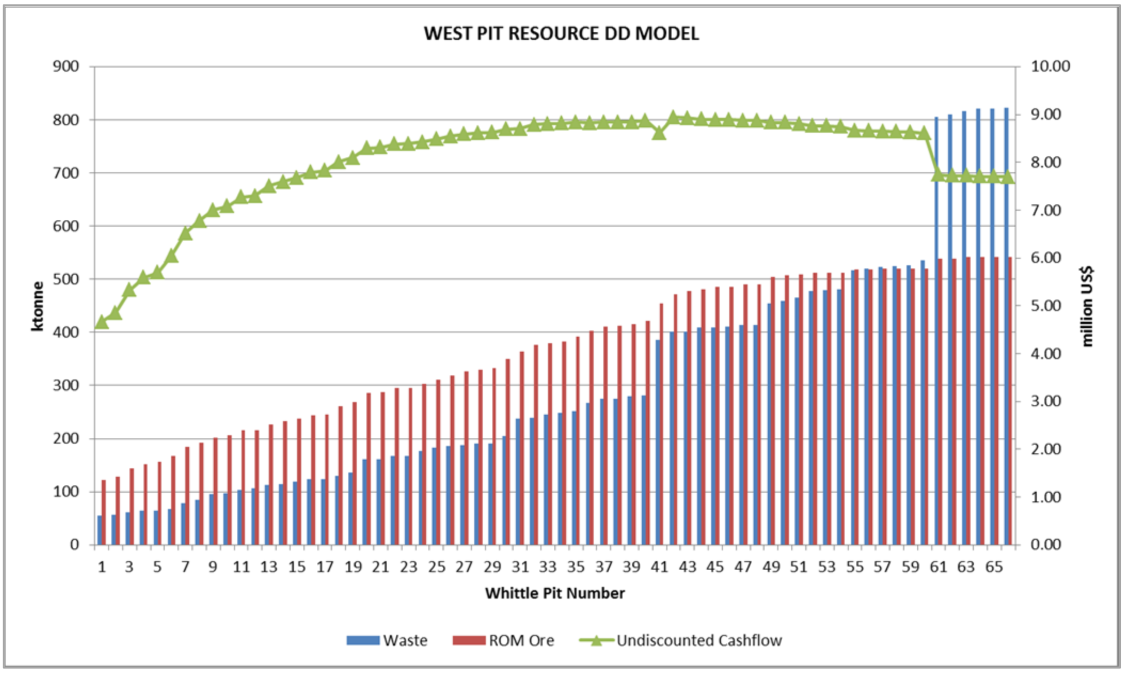


Figure 15-7: Whittle Optimization Results Twangiza West Deposit December 2014

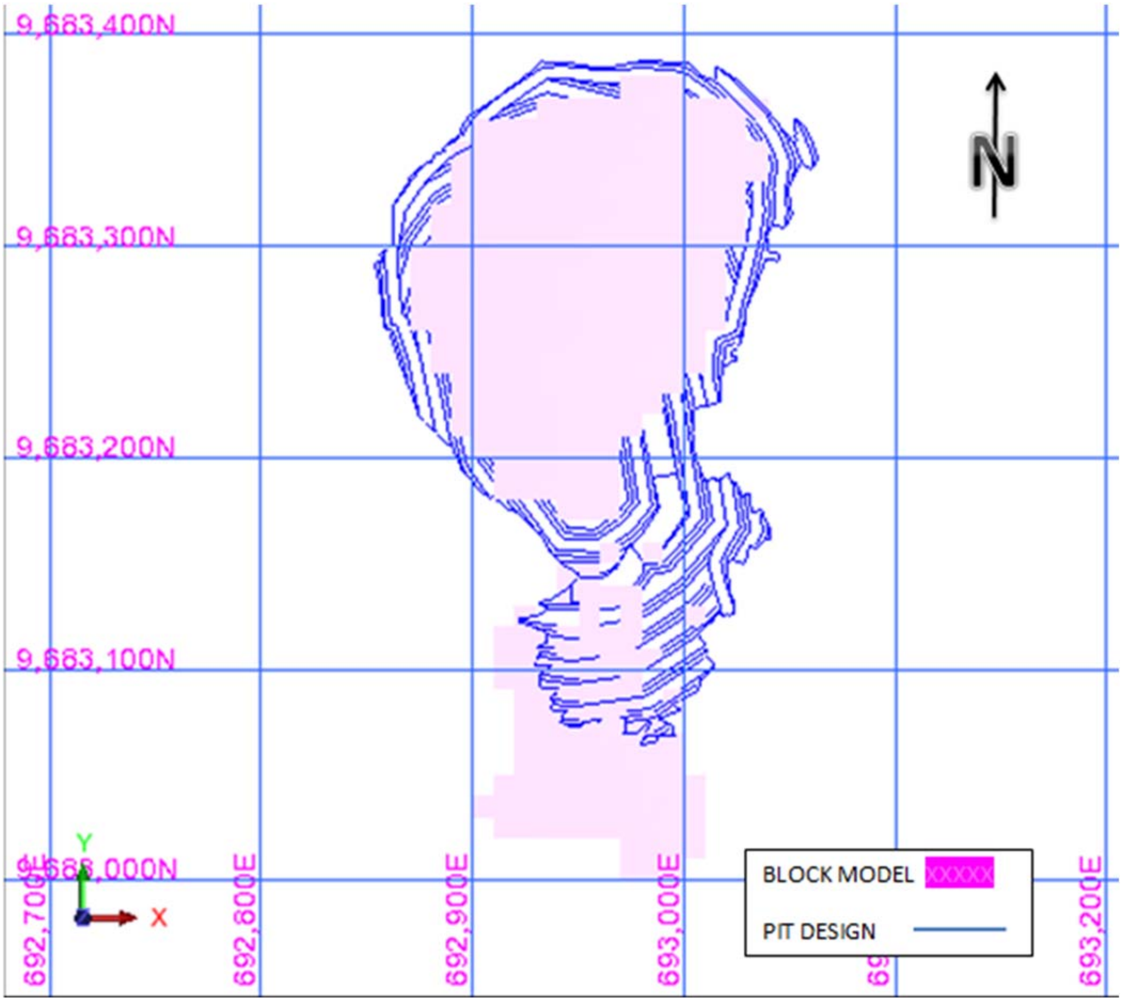


Figure 15-8: Twangiza West Practical Pit Design Versus Blocks Within Whittle Pit Number 41

Table 15-5: Comparison of Inventory Between EOY2014 Optimum Pit and 2014 Practical Pit Design

Item	Description	Unit	Twangiza Main	Twangiza Central	Twangiza North	Twangiza West
WHITTLE INPUTS FOR OPTIMISATION	Gold price	USD/ounce	1200	1200	1200	1200
	Mining costs	USD/tonne mined	3.51	3.51	3.51	3.51
	Processing costs	USD/tonne processed	18.59	18.59	19.11	19.08
	General and administration costs	USD/tonne processed	12.29	12.29	12.29	12.29
	Royalties and selling costs	USD/ounce	63.19	63.19	63.19	63.19
EOY2014 OPTIMUM WHITTLE PIT INVENTORY	Ore mined to process	000 tonnes	17,701	3,974	4,588	472
	Insitu grade	g/t	2.29	2.23	2.84	2.04
	Diluted grade	g/t	2.18	2.12	2.70	1.93
	Recoverable grade	g/t	1.59	1.78	2.24	1.57
	Contained gold	000 ounces	1,238	271	398	29
	Recoverable gold	000 ounces	906	227	330	24
	Waste mined	000 tonnes	19,687	4,959	16,645	401
	Total material	000 tonnes	37,388	8,933	21,233	873
	Strip ratio	t/t	1.11	1.25	3.63	0.85
	Undiscounted Cashflow	Million USD	352	104	157	9
2014 PRACTICAL PIT DESIGN INVENTORY	Ore mined to process	000 tonnes	13,872	2,826	2,971	304
	Insitu grade	g/t	2.42	2.16	2.99	2.05
	Diluted grade	g/t	2.30	2.05	2.84	1.95
	Recoverable grade	g/t	1.61	1.67	2.34	1.55
	Contained gold	000 ounces	1,026	187	271	19
	Recoverable gold	000 ounces	717	152	224	15
	Waste mined	000 tonnes	28,180	9,418	30,809	1,225
	Total material	000 tonnes	42,053	12,244	33,780	1,529
	Strip ratio	t/t	2.03	3.33	10.37	4.02
	Undiscounted Cashflow	Million USD	240	42	43	2

Selection of optimized pit shell

The full optimization generates nested pit shells using an incremental revenue factor ranging from 0.5 to 1.5 with a step size of 0.0125 (1.25% of revenue) to generate a maximum of 81 pit shells. The optimization algorithm uses a cut-off grade ore selection method. A revenue factor of 1.0 equates to the gold price of USD 1,200/oz adopted in the optimization. As part of the optimization process the block model is flagged with a number (1-81) representing the range of nested pit shells. The flagged model is then exported from Whittle and imported into Surpac for graphical inspection and further evaluation.

Conventionally, the pit number with revenue factor equal to 1 (one) is referred to as the optimal pit. Selection of the optimum pit will also depend on consideration of net cashflow (discounted and undiscounted), average mining cost, marginal mining cost and total ounces. The selection of the optimized final pit shell was based on the maximum undiscounted net cash flow.

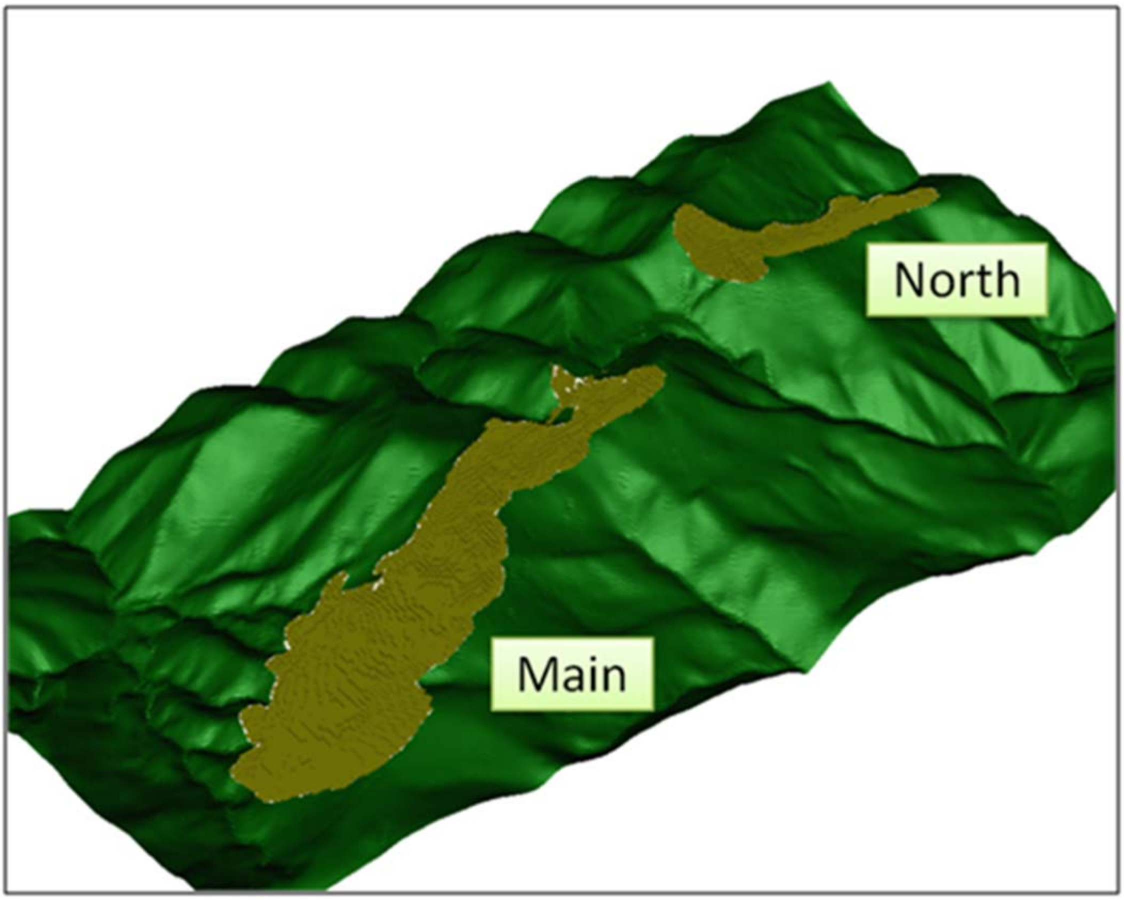


Figure 15-9: Optimized Pit Shells Selected For Practical Design

15.3 Practical pit design

Practical pit designs were prepared using optimized pit shells as templates for all deposits except for the Valley Fill which is already accessible and will be free dig mined to remove sand fill from the river valley.

Surpac software was used to prepare the practical pit, and to incorporate the haul roads, ramps and berms together with appropriate allowance for inter-ramp slope angles. The final economic shell chosen was used as a guide to select pushbacks within the ultimate pit in order to schedule the mining of the pit in a continuously profitable sequence. The pushbacks and ultimate pit were then processed using a minimum mining width algorithm in Whittle in order to apply appropriate practical mining constraints to generate a pit shell for use in pit engineering design work.

Detailed and practical pit designs were produced from the chosen pushbacks to confirm the practicability of mining the deposits using the optimal pit shells as a guide. Care was taken to keep designed strip ratios within the optimal shell strip ratios and a series of options for optimal positioning of ramps was reviewed in order to avoid losses and excessive waste introduction to the final engineered pit design. In-pit and ex-pit haul roads were designed based on these criteria with a continuous gradient of -/+10% at a width of 20 m to provide sufficient room for two way traffic flow. Engineering pit design parameters were as follows;

- Nominal Bench height 10m
- Berm width 6m

- Ramp gradient 10%
- Ramp width (with safety berms) 20m
- Bench face angle (Upper Oxides) 32°
- Bench face angle (Lower Oxides) 38°
- Bench face angle (Transition) 60°
- Bench face angle (Fresh) 70°

A total of five pits have been designed as follows:

- Main Intermediate 1 – is an interim pit within the Twangiza ultimate pit exploiting the oxide zone using Whittle pit number 12 as a template. This pit has been designed to minimize the initial scheduled stripping ratio. A plan view is shown in Figure 15-10
- Main Intermediate 2 – is a second interim pit also within the main Twangiza ultimate shell targeted to extract all the Oxide material to blend with the Fresh and Transition material contained within the optimized shell which forms part of this reserve. This is referred to as cut 2 using Whittle pit number 23 as a template as shown in Figure 15-11.
- Main Final – the design is based on Whittle pit number 36 as a template. This pit forms part of the phase two programme for the Twangiza operations as discussed in the 2009 Feasibility Study.
- Twangiza North – this pit exploits the North ore body using Whittle pit 29 as a template, as shown in Figure 15-12.
- Twangiza Central – this pit exploits the southern portion of the North ore body as shown in Figure 15-13. It uses Whittle pit 38 as a template.
- Twangiza West – this pit lies to the North-West of the Twangiza Main Pit and is shown in Figure 15-14.
- Twangiza Valley Fill – this pit lies to the North-West of the Twangiza Main Pit and is shown in Figure 15-15

Principal haul roads have been designed to connect the working areas to the primary crusher and the waste dumps. The final layout of pit designs is shown in Figure 15-16.

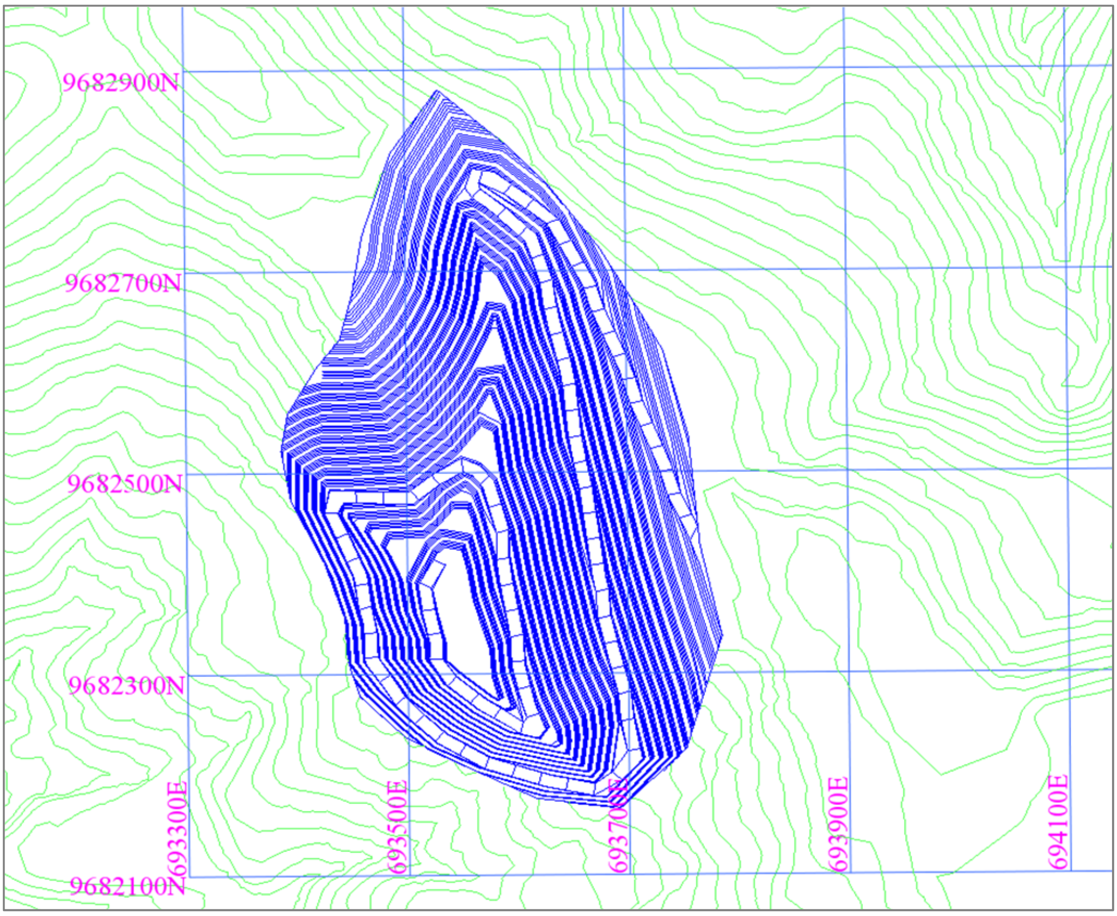


Figure 15-10: Twangiza Main Intermediate Pit (Cut 1) – Plan View

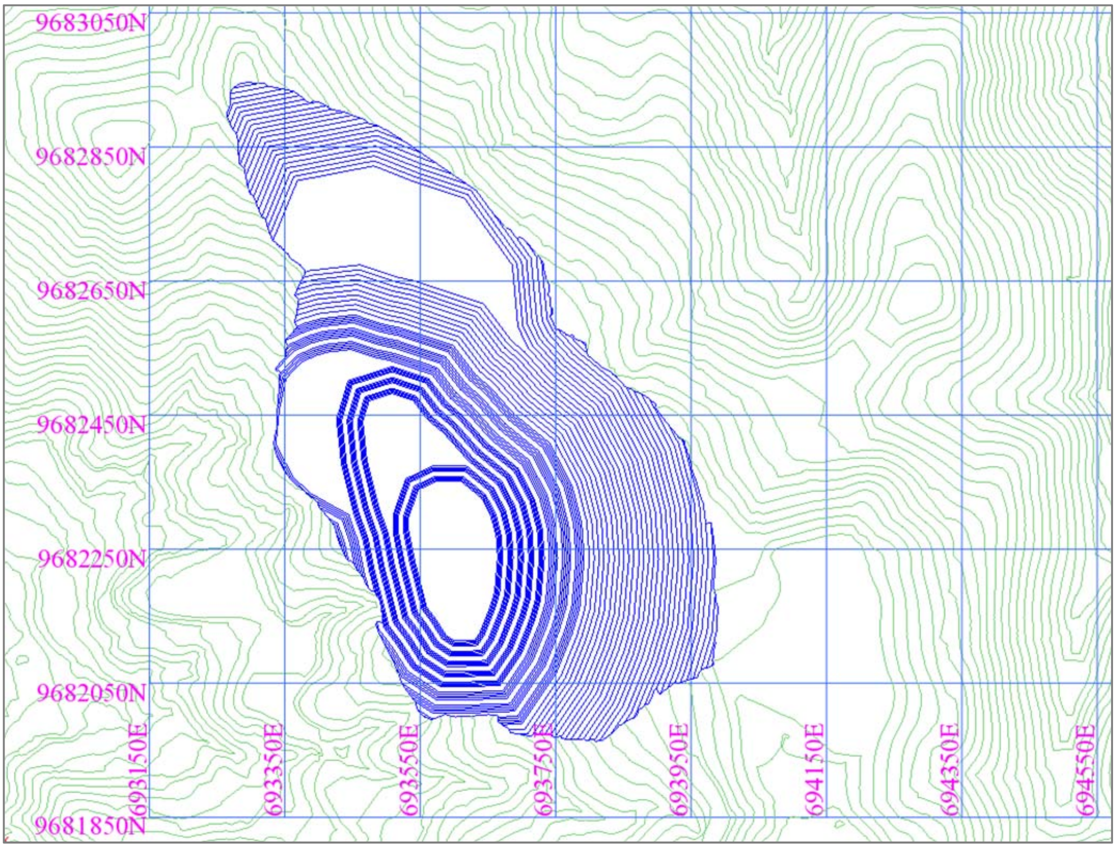


Figure 15-11: Twangiza Main Intermediate Pit (Cut 2) – Plan View

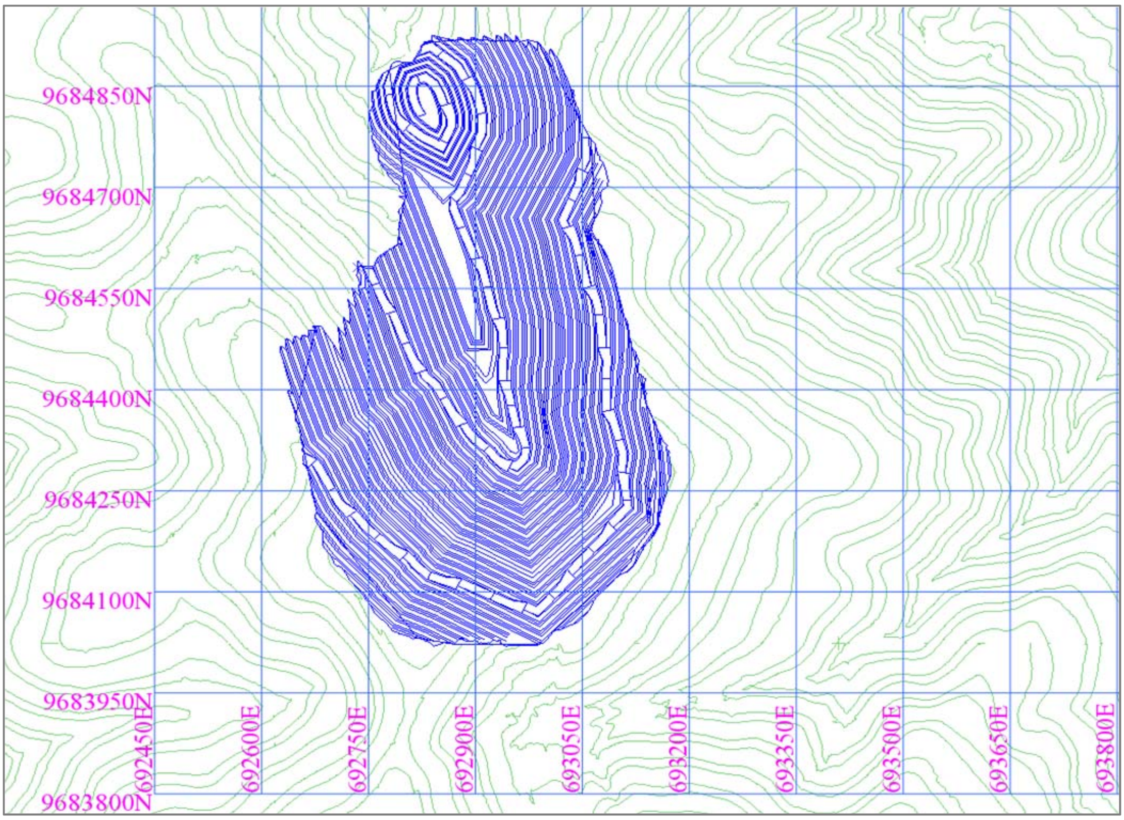


Figure 15-12: Twangiza North Pit – Plan View

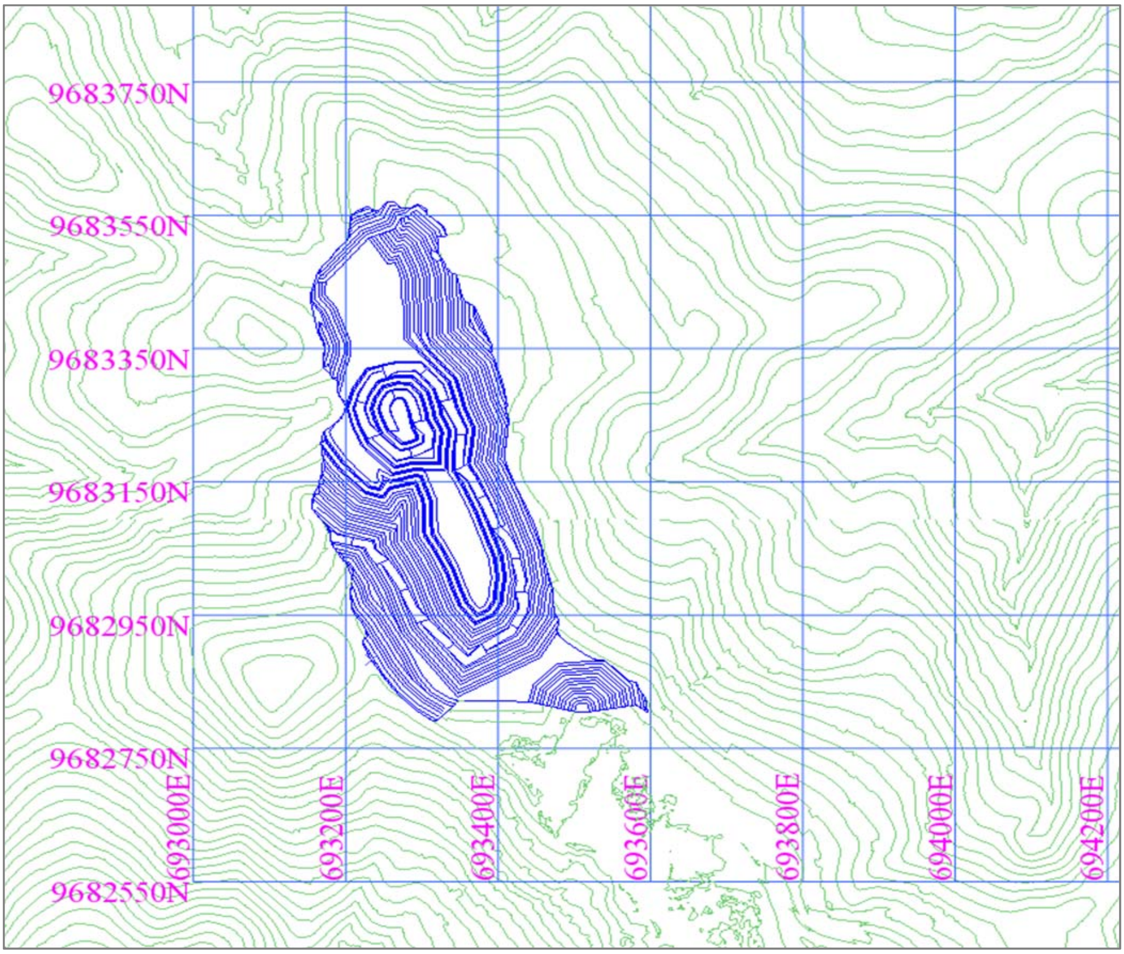


Figure 15-13: Twangiza Central Pit – Plan View

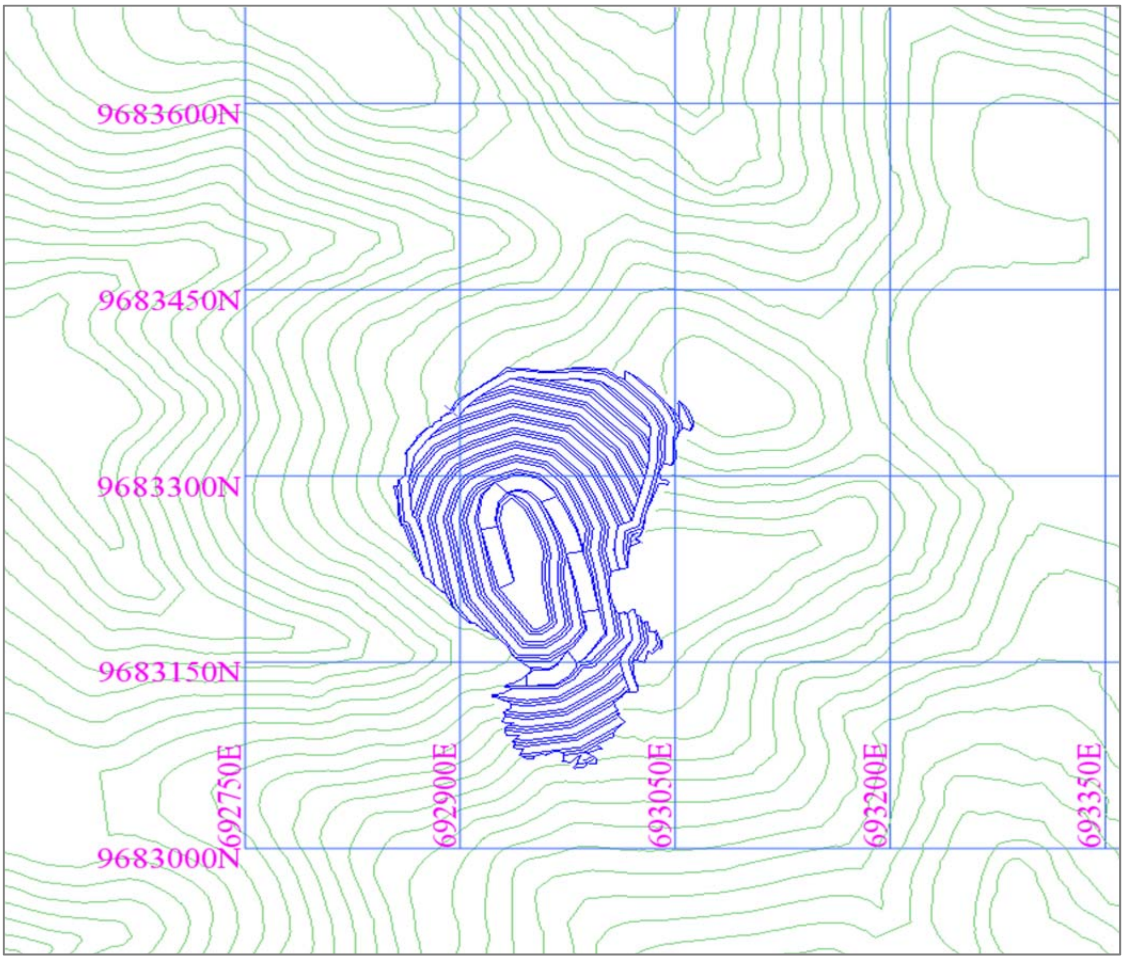


Figure 15-14: Twangiza West Final Pit – Plan View

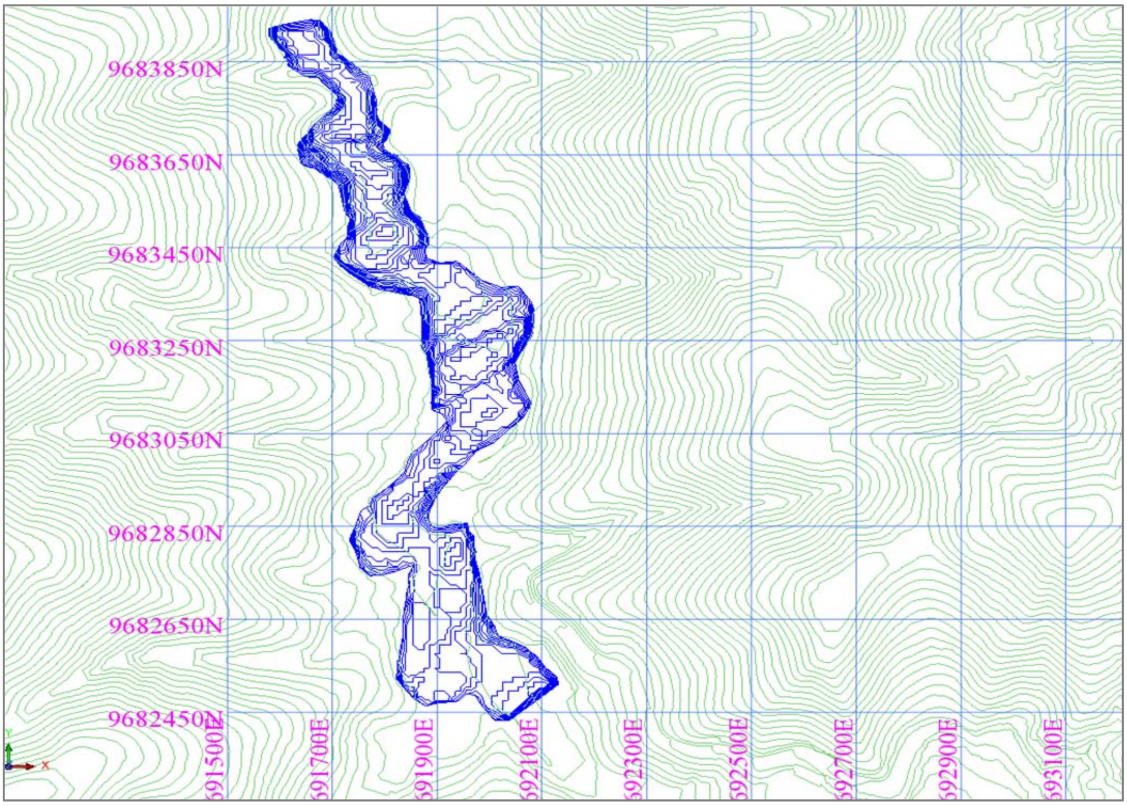


Figure 15-15: Twangiza Valley-Fill Final Pit – Plan View

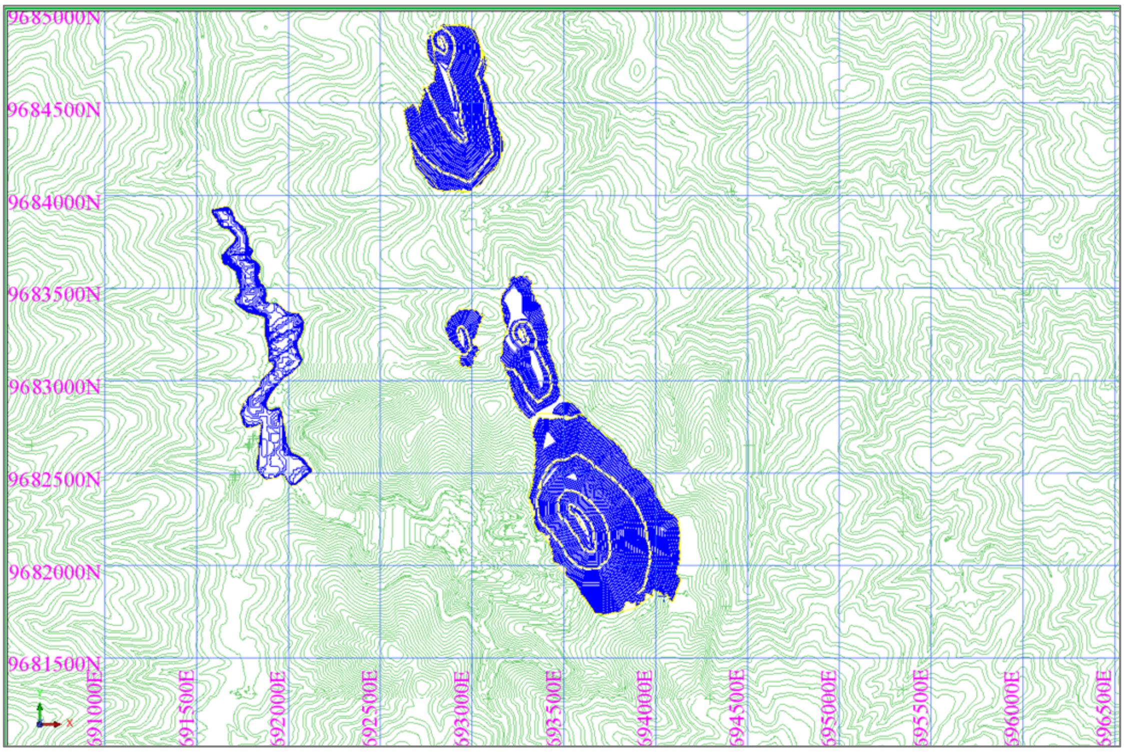


Figure 15-16: Twangiza Final Open Pits – Plan View

15.4 Mine production schedule

The scheduling process consisted of developing a mine plan that is economically optimum using the inventory included in the practical pits. The schedule methodology adopted a simple bench by bench approach with selective mining of ore from waste.

The material mined is classified into three categories namely: High Grade (HG), Low Grade (LG) and Waste (Wst). This is accomplished by applying cut-off grades to the material in the pit.

The schedule by material type is given in Table 15-7 and the overall mine production schedule is given in Table 15-8.

An ore reserve COG of 0.84g/t, based on the recoverable gold grade, is used to differentiate waste from Run-Of-Mine ("RoM") ore. Ore is further categorised into HG and LG by applying an HG COG of 1.0g/t recoverable. Both HG and LG comprise RoM material that can be blended to meet the total plant throughput rate of 1.7 Mtpa.

Grade control mark-up in the pit is undertaken using coloured flagging tapes defining the various categories of material on each mining flitch or bench. The current practice is for the HG, LG and Wst material to be flagged with Red, Yellow and Blue tapes respectively.

All HG material is sent to either the RoM stockpile or direct tipped into the ROM bin whereas all LG material is sent to the LG stockpile.

The material movement is focused on maintaining a smooth total mined profile whilst achieving the required 1.7 Mtpa throughput target.

During the first five years a blend of the oxide and transition material is delivered to the plant to target a process recovery above 82%. Any excess of transition and fresh material is stockpiled to be processed when all the oxide RoM is depleted.

The current LG stockpile is predominantly oxide material which has been accumulated since the commencement of operations. The stockpile closing balance at End-of-Year 2014 is provided in Table 15-6.

Table 15-6: Stockpile Closing Balance as at December 31, 2014

Stockpiles	Tonnes	Gold Grade (g/t)
RoM Stockpiles	69,905	2.44
Low Grade Stockpiles	501,050	0.88
Total Stockpile Balance	570,955	1.07

Table 15-7: Annual Mine Production Schedule by Material Type

Material Type	Units	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	Total
Oxide Quantities														
Waste Volume	kbcm	115	1,658	4,775	2,682	3,758	1,905	1,602	2,473	269	29			19,268
Waste Tonnage	kt	188	3,313	9,537	5,497	7,683	3,983	2,500	4,185	452	56			37,393
LG Volume	kbcm	4	52	11	9	20	7	5	52	9	0			171
LG Tonnage	kt	7	103	23	19	42	14	9	97	17	1			332
LG In Situ Grade	g/t	1.15	1.26	1.11	1.19	1.21	1.1	1.13	1.11	1.13	1.06			1.17
LG Diluted Grade	g/t	1.09	1.19	1.05	1.13	1.15	1.04	1.07	1.05	1.07	1.01			1.11
Ore Recoverable Grade	g/t	0.96	1.05	0.93	0.93	1.02	0.93	0.94	0.92	0.94	0.89			0.98
LG Mined Ounces	koz	0	4	1	1	2	0	0	3	1	0			12
LG Recoverable Ounces	koz	0	3	1	1	1	0	0	3	1	0			10
HG Volume	kbcm	639	583	653	283	258	168	86	577	446	32			3,724
HG Tonnage	kt	1,017	1,048	1,324	591	484	350	159	985	765	61			6,784
HG In Situ Grade	g/t	2.9	2.42	2.7	2.59	2.36	3.64	1.82	2.26	2.19	1.65			2.55
HG Diluted Grade	g/t	2.76	2.3	2.56	2.46	2.24	3.46	1.73	2.14	2.08	1.57			2.42
HG Recoverable Grade	g/t	2.35	1.98	2.24	2.16	1.92	3.03	1.47	1.86	1.74	1.31			2.09
HG Mined Ounces	koz	90	77	109	47	35	39	9	68	51	3			528
HG Recoverable Ounces	koz	77	67	95	41	30	34	8	59	43	3			456
Total Oxide Volume	kbcm	759	2,292	5,439	2,975	4,036	2,080	1,694	3,102	725	62			23,164
Total Oxide Tonnage	kt	1,212	4,464	10,884	6,107	8,209	4,347	2,668	5,267	1,234	118			44,509
Transition Quantities														
Waste Volume	kbcm	383	299	346	1,107	757	1,362	232	951	456	354	44		6,291
Waste Tonnage	kt	843	705	841	2,758	1,798	3,533	599	2,149	1,060	906	115		15,308
LG Volume	kbcm	6	11	2	2	6	6	4	23	10	2			72
LG Tonnage	kt	14	24	4	5	13	16	8	54	24	6			167
LG In Situ Grade	g/t	1.89	1.72	1.19	1.61	1.58	1.32	1.31	1.26	1.4	1.45			1.44
LG Diluted Grade	g/t	1.8	1.63	1.13	1.53	1.5	1.25	1.24	1.19	1.33	1.37			1.37
Ore Recoverable Grade	g/t	0.95	0.93	0.92	0.98	1.05	0.93	0.98	0.93	0.95	0.96			0.95
LG Mined Ounces	koz	1	1	0	0	1	1	0	2	1	0			7,357

Material Type	Units	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	Total
LG Recoverable Ounces	koz	0	1	0	0	0	0	0	2	1	0			5,097
HG Volume	kbcm	390	318	112	448	292	341	79	245	366	369	195		3,156
HG Tonnage	kt	762	696	271	1,144	684	891	208	550	862	953	512		7,534
HG In Situ Grade	g/t	2.86	3.26	2.39	2.68	2.46	2.6	2.56	2.54	2.36	2.45	2.7		2.63
HG Diluted Grade	g/t	2.72	3.1	2.27	2.54	2.33	2.47	2.44	2.41	2.24	2.33	2.56		2.5
HG Recoverable Grade	g/t	1.78	1.98	1.76	1.99	1.59	1.72	1.95	1.63	1.38	1.4	1.5		1.69
HG Mined Ounces	koz	67	69	20	93	51	71	16	43	62	71	42		606
HG Recoverable Ounces	koz	44	44	15	73	35	49	13	29	38	43	25		408
Total Transition Volume	kbcm	780	628	459	1,557	1,054	1,709	315	1,219	832	725	239		9,518
Total Transition Tonnage	kt	1,619	1,425	1,116	3,907	2,495	4,439	815	2,753	1,946	1,865	628		23,009
Fresh Quantities														
Waste Volume	kbcm	44	190	0	159	435	848	1,278	333	358	833	1,429	331	6,239
Waste Tonnage	kt	114	515	0	446	1,174	2,337	3,516	879	944	2,227	3,874	907	16,931
LG Volume	kbcm	7	1	0	2	10	8	34	2	12	24	50	17	166
LG Tonnage	kt	19	3	0	6	27	20	93	5	32	64	139	45	453
LG In Situ Grade	g/t	1.27	1.89	0	1.98	1.76	1.44	1.52	1.65	1.63	1.57	1.42	1.62	1.53
LG Diluted Grade	g/t	1.2	1.79	0	1.88	1.67	1.37	1.44	1.56	1.55	1.49	1.35	1.54	1.45
Ore Recoverable Grade	g/t	0.88	1.02	0	0.92	0.97	0.93	0.94	0.96	0.94	0.94	0.94	0.94	0.94
LG Mined Ounces	koz	1	0	0	0	1	1	4	0	2	3	6	2	21
LG Recoverable Ounces	koz	1	0	0	0	1	1	3	0	1	2	4	1	14
HG Volume	kbcm	10	35	0	83	128	141	462	36	27	252	432	108	1,715
HG Tonnage	kt	21	93	0	234	338	386	1,288	96	74	685	1,188	301	4,703
HG In Situ Grade	g/t	3.35	2.32	0	2.96	2.71	2.2	2.21	2.19	2.3	2.41	2.16	1.97	2.29
HG Diluted Grade	g/t	3.18	2.21	0	2.81	2.57	2.09	2.1	2.08	2.19	2.29	2.05	1.87	2.18
HG Recoverable Grade	g/t	1.42	1.38	0	2.17	1.6	1.39	1.4	1.34	1.33	1.38	1.33	1.3	1.42
HG Mined Ounces	koz	2	7	0	21	28	26	87	6	5	50	78	18	329
HG Recoverable Ounces	koz	1	4	0	16	17	17	58	4	3	30	51	13	215
Total Fresh Volume	kbcm	61	227	0	244	573	996	1,774	371	397	1,109	1,912	455	8,120
Total Fresh Tonnage	kt	153	611	0	686	1,539	2,743	4,897	980	1,049	2,976	5,200	1,253	22,088

Material Type	Units	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	Total
Mwana Valley Fill Deposit														
HG Volume	kbcm		197		484	447								1,128
HG Tonnage	kt		326		799	737								1,862
HG In Situ Grade	g/t		2.21		2.21	2.21								2.21
HG Diluted Grade	g/t		2.1		2.1	2.1								2.1
HG Recoverable Grade	g/t		1.85		1.85	1.85								1.85
HG Mined Ounces	oz		21,964		53,855	49,688								125,507
HG Recoverable Ounces	oz		19,328		47,393	43,726								110,447
Ore proportion by degree of Oxidation (Weathering)														
Oxides	%	55	58	83	29	33	22	10	61	44	4			36
Transition	%	43	37	17	60	44	54	12	34	50	54	28		39
Fresh	%	2	5	0	11	23	24	78	6	6	42	72	100	26
Total	%	100	100	100	100	100	100	100	100	100	100	100	100	100

Table 15-8: Mine Production Schedule Summary

Material Type	Units	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	Total
Mining Schedule																
ROM Mined	kt	1,800	2,164	1,595	2,768	2,243	1,626	1,655	1,631	1,700	1,700	1,700	301			20,883
In Situ Grade	g/t	2.89	2.65	2.65	2.55	2.39	2.73	2.22	2.35	2.28	2.4	2.32	1.97			2.49
Diluted Grade	g/t	2.74	2.52	2.51	2.42	2.27	2.59	2.11	2.23	2.16	2.28	2.2	1.87			2.37
Recoverable Grade	g/t	2.1	1.94	2.16	2	1.74	1.92	1.48	1.75	1.54	1.39	1.38	1.3			1.77
Low Grade Mined	kt	39	130	27	30	82	51	110	156	72	71	139	45			953
In Situ Grade	g/t	1.46	1.36	1.12	1.42	1.45	1.31	1.47	1.18	1.44	1.55	1.42	1.62			1.39
Diluted Grade	g/t	1.39	1.29	1.06	1.35	1.37	1.24	1.4	1.12	1.37	1.47	1.35	1.54			1.32
Recoverable Grade	g/t	0.92	1.03	0.93	0.94	1.01	0.93	0.94	0.93	0.94	0.94	0.94	0.94			0.95
Total Ore mined	kt	1,839	2,293	1,622	2,798	2,326	1,677	1,765	1,787	1,772	1,771	1,839	346			21,835
In Situ Grade	g/t	2.86	2.58	2.62	2.53	2.36	2.69	2.17	2.25	2.24	2.37	2.25	1.92			2.44
Diluted Grade	g/t	2.72	2.45	2.49	2.41	2.24	2.55	2.06	2.13	2.13	2.25	2.14	1.83			2.32
Recoverable Grade	g/t	2.07	1.88	2.14	1.99	1.72	1.89	1.44	1.68	1.52	1.37	1.35	1.25			1.74
Opex Waste Mined	kt	1,144	4,533	10,378	8,701	10,656	9,852	6,615	7,213	2,457	3,189	3,989	907			69,633
Total Waste Mined	kt	1,144	4,533	10,378	8,701	10,656	9,852	6,615	7,213	2,457	3,189	3,989	907			69,633
Total Tonnes Mined	kt	2,984	6,826	12,000	11,499	12,981	11,529	8,380	9,000	4,229	4,960	5,828	1,253			91,468
Strip Ratio	w:o	0.62	1.98	6.4	3.11	4.58	5.87	3.75	4.04	1.39	1.8	2.17	2.63			3.19
Process Schedule																
Total Tonnes Processed	kt	1,600	1,700	1,700	1,700	1,700	1,700	1,700	1,700	1,700	1,700	1,700	1,700	1,700	403	22,406
Head Grade	g/t	2.68	2.39	2.52	2.32	2.13	2.71	2.48	2.21	2.16	1.84	2.15	2.17	2.22	1.45	2.29
Contained Gold	kg	4,291	4,067	4,279	3,938	3,618	4,610	4,212	3,750	3,680	3,121	3,649	3,695	3,767	583	51,270
Contained Gold	koz	138	131	138	127	116	148	135	121	118	100	117	119	121	19	1,648
CIL Plant Recovery	%	84	81	85	86	83	75	75	78	71	66	63	66	66	65	75
Gold Recovered	koz	116	106	117	109	96	112	102	94	84	66	74	79	79	12	1,246

15.5 Modifying Factors

As part of the Mineral Resource and Reserve review process, a detailed analysis of the historical mine reconciliation from the commencement of mining in October 2011 through to December 2014 was completed by Twangiza Mining with assistance from SRK (UK). This work was prompted by identification of a significant negative tonnage reconciliation issue, of the order of -42% when compared with the previous resource model.

The analysis resulted in identification of key adjustments to be applied both to the previous resource block model (as detailed in Section 14.3.4) and additional modifying factors to be applied to the adjusted resource block model to estimate Mineral Reserves which are given in Table 15-9.

Table 15-9: Resource to Reserve Modifying Factors

Factor	Tonnage	Grade	Metal
Adjusted resource to actual Grade Control model	0.86	1.03	0.88
Survey pick-up error	0.97	1.00	0.97
Topo effect 2007 and 2010	0.99	1.00	0.99
Mining losses and Dilution	1.00	0.98	0.98
Combined Factors to convert Adjusted Resource to Reserve	**0.83**	**1.00**	**0.83**

Key sources of variance were identified as:

- Density and underground artisanal mining adjustments in the near-surface oxide as detailed in Section 14.3.4.
- Grade control model variance: -12% tonnage, +3% grade, -9% metal due to slight shape change and higher cut-off grade
- Minor losses due to survey markup, topographical variance, mining losses and dilution and stockpile reconciliation accounted for a further -7% of tonnage. This was offset by 4% addition of material from valley fill.
- A relatively low level of mining dilution (2.5%) and ore loss (98%) has been used based on wide average ore widths, good ore continuity, low dig line length to areas ratio and a low strip ratio.

The potential for an underestimation of tonnage arising from a mill weightometer error was examined but remains unconfirmed. This may have occurred where short term throughput rates exceeded the 200t/hour weightometer limit. This requires further assessment as it impacts historical (and therefore forecast) plant recovery estimates.

A detailed series of recommendations were developed to monitor the mine reconciliation and management system.

15.6 Mineral Reserve Estimate

The Mineral Reserve Statement uses the definitions and guidelines given in CIM Definition Standards on Mineral Resources and Mineral Reserves and is reported in accordance with NI 43-101 requirements.

Following a technical review of operating performance in 2014, Twangiza Mining has included transitional and fresh ore types in the updated estimate based on the proven ability of the current plant to economically process non-oxide materials contained within the reserve pit shell. An update of the Mineral Reserves was prepared as at the end of year 2014. The new Reserve incorporates

- Addition of non-oxide material;
- Mining depletion from 1 October, 2011 to 31 December, 2014;
- Revised density as described in Section 14.3.4;
- Revised modifying factors as described in 15.5;
- Additional resource that has been added from Twangiza West; and
- Changes in economic assumptions.

The Mineral Reserve is reported using a breakeven cut-off grade of 0.84g/t Au applied to the recoverable gold grade and a gold price of USD1,200 per ounce. The Mineral Resource was modified using the factors provided in Table 15-9; the Mineral Resource is inclusive of the Mineral Reserve.

The table below shows the updated Mineral Reserves estimated to be contained within the Twangiza practical pit design and associated production schedule.

Table 15-10: Summary of Twangiza Mineral Reserves as at December 31, 2014

Category	Deposit	Tonnes (Mt)	Grade (g/t Au)	Ounces (Moz Au)
Proven	Twangiza Main + North + Central + West	7.47	2.41	0.58
Probable	Twangiza Main + North + Central + West	14.91	2.22	1.06
Proven + Probable		**22.38**	**2.28**	**1.64**

A breakdown of the Mineral Reserves by deposit and stockpile is provided in Table 15-11.

Table 15-11: Mineral Reserve by Pit as at December 31, 2014

Deposit/Pit	Material	PROVEN			PROBABLE		
		Tonnes (Mt)	Gold Grade (g/t)	Gold (Moz)	Tonnes (Mt)	Gold Grade (g/t)	Gold (Moz)
North	Oxide	0.00	0.00	0.00	1.17	3.25	0.12
	Transition	0.00	0.00	0.00	1.38	2.57	0.11
	Fresh	0.00	0.00	0.00	0.43	2.60	0.04
	Subtotal	0.00	0.00	0.00	2.97	2.84	0.27
Main Extension (Central)	Oxide	0.01	1.24	0.00	1.97	1.97	0.12
	Transition	0.00	0.00	0.00	0.75	2.29	0.05
	Fresh	0.00	0.00	0.00	0.10	2.10	0.01
	Subtotal	0.01	1.24	0.00	2.82	2.06	0.19
Main	Oxide Stockpile	0.57	1.07	0.02	0.00	0.00	0.00
	Oxide	4.56	2.36	0.35	1.07	1.51	0.05
	Transition	2.25	2.83	0.20	3.73	2.30	0.28
	Fresh	0.08	2.77	0.01	4.02	2.00	0.26
	Subtotal	7.46	2.41	0.58	8.82	2.07	0.59
West	Oxide	0.00	0.00	0.00	0.18	1.94	0.01
	Transition	0.00	0.00	0.00	0.00	1.21	0.00
	Fresh	0.00	0.00	0.00	0.12	1.97	0.01
	Subtotal	0.00	0.00	0.00	0.30	1.95	0.02
Totals	**Oxide Stockpile**	**0.57**	**1.07**	**0.02**	**0.00**	**0.00**	**0.00**
	Oxide	**4.57**	**2.36**	**0.35**	**4.39**	**2.20**	**0.31**
	Transition	**2.25**	**2.83**	**0.20**	**5.85**	**2.36**	**0.44**
	Fresh	**0.08**	**2.77**	**0.01**	**4.68**	**2.05**	**0.31**
	Subtotal	**7.47**	**2.41**	**0.58**	**14.91**	**2.22**	**1.06**

16 MINING METHODS

16.1 Mining Method and Site Layout

Mining operations are based on conventional open cast techniques. Excavation of the ore and waste rock on 2.5m high mining benches will be performed by hydraulic excavators in backhoe configuration loading out to 40 tonne nominal capacity articulated dump trucks (Bell B40D or Cat equiv.). Mining is free dig in the oxide zone with any harder transition and fresh materials drilled and blasted on 5.0 m benches.

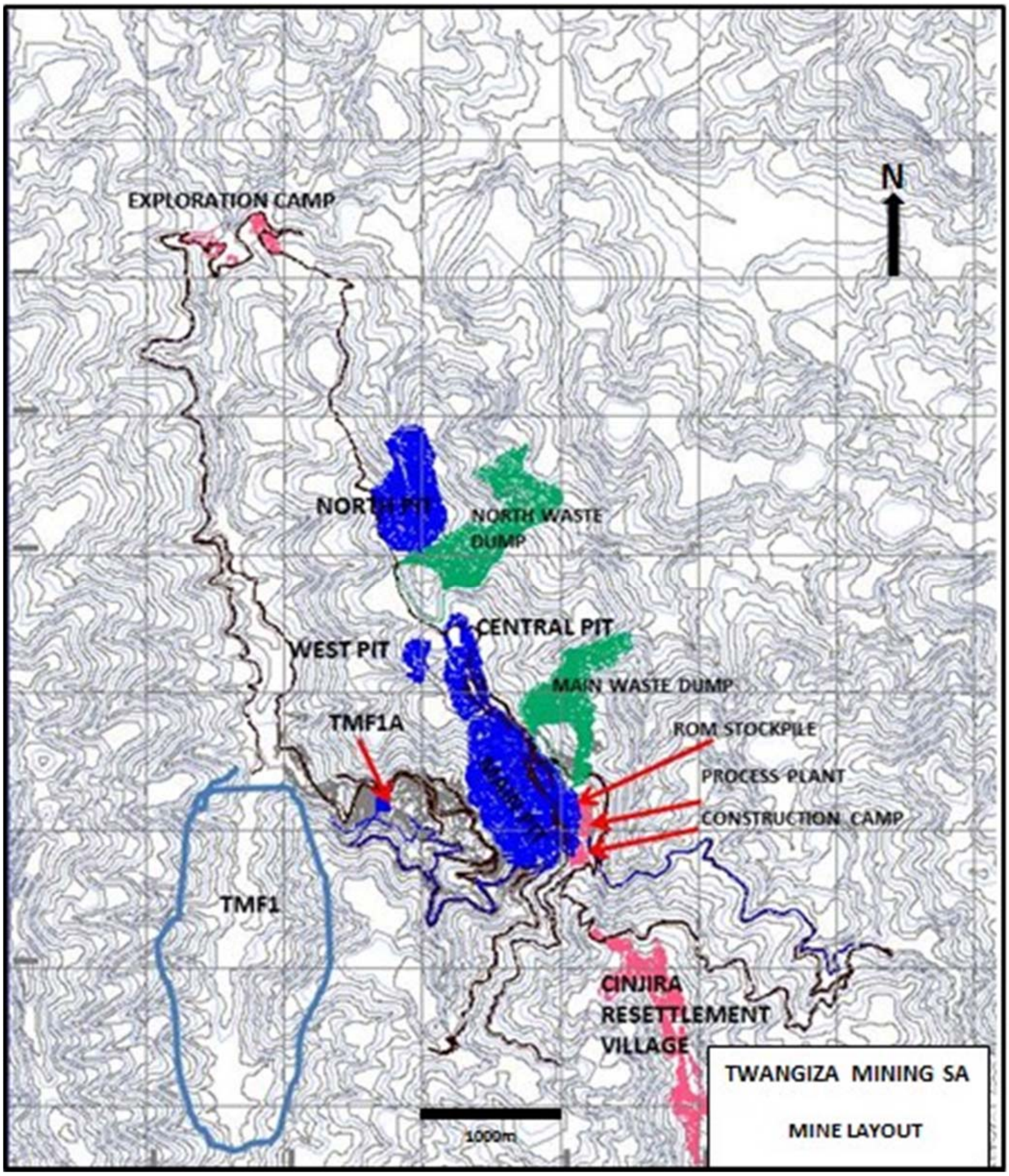


Figure 16-1: Site Layout Plan

RoM will be direct fed to the mill with any excess RoM stockpiled on the RoM pad and low grade ore stockpiles located within 500 m of the pit. The stockpiles will be rehandled using a front-end-loader (Cat 966) on a just-in-time basis to meet the plant throughput requirements during the active mining period. Any remaining stockpile will be treated after mining comes to an end for the Phase 1. Waste material mined is hauled to the TMF 1 wall or a waste dump within a 500 m range from the pit exit. A site layout plan is provided below in Figure 16-1and shows the layout of pits, haul roads, access roads, stockpiles, waste dumps, process plant, tailings dam facility (“TMF”) and site infrastructure.

16.2 Site Preparation

Site preparation requires construction of access roads, vegetation clearing, tree grubbing and topsoil stripping. The main haul road is advanced to allow access between pits and then areas are prepared for stockpiles, waste dumps and field workshops as required for each separate mining area. The haul roads have a 10% gradient and 20 m width considering the type of equipment used in the mining operations.

Vegetation clearing, tree grubbing and top soil stripping are done in conformity to the direction of mining activities and in consonance to the established DRC and/or Internationally accepted mining and environmental regulations. The disturbance of vegetation and top soil stripping are also done progressively according to the mining schedule of activities. Top soil stripping is done to a depth of 200 mm and stockpiled at designated dump locations close to major disturbed areas for future rehabilitation works.

16.3 Drill and Blast

In general the Phase 1 of Twangiza has not required drilling and blasting as mining has been concentrated in the friable and oxidized material. On advancing mining activities into the more competent transition material, minor drilling and blasting will be required where the surface oxidation no longer permits free digging and loading. Drill and blast activities have already commenced at Twangiza although the highest proportion of mill feed ore is still derived from the oxide and upper transition material, reducing the drill and blast activities significantly. In 2015, transition and fresh material requiring drilling and blasting will comprise approximately 59% of total material, reducing thereafter (with increased production from shallower oxide pits and Valley Fill material) to average about 30% over the five year period.

Drill and blast design parameters and input costs have been based on actual operating estimates for the Namoya Mining SA operation in the DRC. Based on this analysis a drill & blast unit cost of USD 0.30 /t has been estimated. A Sandvik Pantera DP 1500i rig will be used to drill the blast holes. Operations technical expertise and training will be sourced from in-house and the Namoya operation.

16.4 Loading and Hauling

Loading and hauling activities will be conducted with two hydraulic excavators in backhoe configuration equipped with 3.7 m^3 nominal capacity buckets. Haulage will be carried out with articulated dump trucks with a rated payload of 40 tonnes.

Excavating and loading of the 5 m mining benches will be undertaken in two flitches, of 2.5 m depth. Mining geologists will closely supervise and monitor the selective mining of ore to minimize dilution and ore loss. They will also supervise and control dispatch of haulage trucks to appropriate destinations by material type - either to RoM pad, low grade stockpile or waste dump in accordance with grade control ore mark-ups based on cut–off grades provided by the Mine Planning Team. The services of other technical support sections including the Survey, Geotechnical and Dewatering teams will also be utilized for such routine mining activities as survey of pit floors, and setting outs, mapping, pit wall monitoring and for water control.

Apart from the main mining fleet, ancillary equipment will be used to support the mining operations. Ancillary equipment encompasses mine units which are not directly responsible for drilling and blasting, loading and haulage of high grade ore, low grade ore and waste materials, but are used to support the major production units and provide safe and clean working areas. Such activities include dozing, batter trimming, road maintenance, floor level clean-up and levelling, safety berm and bund installation and dewatering activities. Pit supervisors are engaged to ensure that the mining personal and fleets are effectively managed and utilized for optimal production.

Some of the waste material from the Main and Main Extended (Central) pit is scheduled to be used for the construction of the TMF Phase 1 wall while the remainder, including the waste mined from other mine pits will be dumped at designated waste dumps within the catchment of the TMF for eventual re-handling and construction of the tailings dam walls, if wet weather conditions do not permit direct tipping onto the pad.

The waste dump development will be based on a geotechnical design which conforms to the mountainous terrain. Parameters used in the design of the waste dumps are described below:

- Dump face angle - 35°
- Batter angle (rehabilitation) - 25°
- Berm width - 21 m
- Lift Height – 15 m
- Average Dump Height – 60 m
- Average Standoff Distance from Pit – 350 m
- Mining Work Schedule

Mining activities will be scheduled according to a continuous 12 hour shift roster – 7 days per week, 2 shifts daily for 365 days of operation. Ore will be preferentially mined on day shift due to higher visibility and to allow for greater supervision. In order to achieve this, a two crew system will be adopted for all direct operations personnel, thus, a shift on day-shift, another on night-shift.

The operating time per shift will be the actual time during the shift that the equipment is productively working and this is equal to the total mechanically available time less all scheduled and unscheduled delays.

The effect of weather on mining operations has been factored into the determination of effective working time and equipment productivity. This means effective working hours have been reduced to reflect rainfall and other weather delays. Estimation of rainfall delays was based on 5-year rainfall average values recorded for the Twangiza Area.

Key elements of the mining work schedule are presented in Table 16-1.

Table 16-1: Mining Work Schedule

Activity	Scheduled Time	Units
Shift Change	90	Minutes per day
Chop time/Break	90	Minutes per day
Working Period	21	Hours per day
Shift Duration	10.5	Hours per Shift
Number of Shifts	730	Shifts per year
Working Time	7665	Hours per year
Weather Delays	669	Hours per year
Effective Working Period	6996	Hours per year

16.5 Manpower

Operations personnel include all paid staff working with Twangiza Mining. It includes labour requirements for the various departments comprising Process Metallurgy, Finance & Administration, Process Engineering, Mining Operations, Mineral Resources Management, Community Relations, Human Resources, Health, Safety & Environment, General Management, Civil Engineering, and Mobile Fleet Maintenance. Additional services required on short term or temporal basis may be sourced from contractors or local hire companies.

Both introductory and extensive training will be required for the national labour due to a lack of local operational and technical skills.

A number of experienced expatriate staff, comprising first level supervisory staff to senior management, will be engaged for both operations and training. Other experts will also be engaged to train the local workforce in technical, operational and maintenance skills on an as-required basis.

Table 16-2 details the annual labour required per department over life of the mine.

Table 16-2: Labour Schedule

Department	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028
PROCESS METALLURGY	122	122	122	122	122	122	122	122	122	122	122	122	122	29
FINANCE & ADMINISTRATION	69	69	69	69	69	69	69	69	69	69	69	69	69	16
PROCESS ENGINEERING	99	99	99	99	99	99	99	99	99	99	99	99	99	23
MINING OPERATIONS	87	114	186	186	177	174	159	150	102	102	102	48		
MINERAL RESOURCES MANAGEMENT	77	77	77	77	77	77	77	77	77	77	77	77		
COMMUNITY RELATIONS	30	30	30	30	30	30	30	30	30	30	30	30	30	7
HUMAN RESOURCES	28	28	28	28	28	28	28	28	28	28	28	28	28	7
HEALTH, SAFETY & ENVIRONMENT	14	14	14	14	14	14	14	14	14	14	14	14	14	3
GENERAL MANAGEMENT	5	5	5	5	5	5	5	5	5	5	5	5	5	1
CIVIL ENGINEERING	117	117	117	117	117	117	117	117	117	117	117	117	117	28
MOBILE FLEET MAINTENANCE	49	49	49	49	49	49	49	49	49	49	49	49		
TOTAL LABOUR	697	724	796	796	787	784	769	760	712	712	712	658	484	115

16.6 Mining Operations

The mine equipment has been selected based upon the annual mine production schedule and equipment productivity estimates. The size and type of mining equipment is consistent with the project scale with peak annual material movements averaging 12 Mt from Production Year 2017 to Year 2020. An asset table and pricing is provided in Table 16-3 while the fleet requirements and a replacement schedule are provided in Table 16-4 and Table 16-5 respectively.

Table 16-3: Mining Equipment Asset Table

Asset Table CY2015	No. of Units	Price per Unit (USD)
Major Mining Fleet		
Hydraulic Excavator Hitachi ZX670	2	643,894
Truck Cat ADT745C	4	504,347
Drill Rig Sandvik Pantera 1500 DR	1	629,640
Track Dozer CatD8R	1	629,640
Grader CAT14M	2	486,216
Minor Mining Fleet		
Hydraulic Excavator Kato 1430	2	298,768
FEL CAT966H	2	346,680
Service Truck Bell B20D	1	271,562
Diesel Truck Bell B20D	1	271,562
Water Truck Bell B20D	1	271,562
Lighting Plants	10	18,000
Light Vehicle	10	53,500
Water Truck Actros MERC	1	197,649

The replacement equipment schedule provided in Table 16-5 takes into account the useful life of each item of plant as scheduled using Xeras software. No replacement after 2023 reflects both a reduction in material movements and therefore fleet numbers combined with parking up of equipment as the remaining useful life of various items of plant is expended.

Table 16-4: Mining Equipment Schedule Matched to Production Requirements

Description		2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027
Hydraulic Excavator Hitachi ZX670		2	2	5	5	5	4	3	3	2	2	2		
Truck Cat ADT745C		4	9	17	17	17	17	13	13	8	8	8	2	
Track Dozer Cat D8R		2	4	7	7	7	7	7	5	3	3	3	1	
Water Truck Bell B20D		1	1	3	3	2	2	2	2	1	1	1		
Service Truck Bell B20D		1	1	2	2	2	2	2	2	1	1	1		
Diesel Truck Bell B20D		1	1	2	2	2	2	2	2	1	1	1		
Lighting Plants		10	10	14	14	14	13	10	10	7	7	7	1	
Light Vehicle		10	10	13	13	13	12	10	10	6	6	6	1	
Hydraulic Excavator Kato 1430	2	2	2	2	2	2	2	2		1	1	1		
Water Truck Actros Mercedes	1	1		3	3	2	2	2	2	1	1	1		
FEL Cat 966H	2	2	2	2	2	2	2	2	2	2	2	2	2	
Sanvik Panterra 1500 DR	1	1	2	2	2	2	2	2	1	1	1			
Grader Cat 14M	2		4	7	7	6	6	6	5	3	3	3	1	

Table 16-5: Replacement Mining Equipment Schedule

Replacement Units	2015	2016	2017	2018	2019	2020	2021	2022	2023
Hydraulic Excavator Hitachi ZX670		1	1						
Water Truck Bell B20D		1					2		
Service Truck Bell B20D				1					
Diesel Truck Bell B20D				1					
Hydraulic Excavator Kato 1430				1		1			
Water Truck Actros Mercedes									1
FEL Cat 966H								1	1

17 RECOVERY METHODS

17.1 Process Flow Sheet

The process plant at Twangiza was originally designed to process oxides ores at a nominal throughput of 1.3 Mtpa. The flowsheet incorporated primary crushing of oxide ore using a mineral sizer, wet scrubbing to remove fine clays, conventional secondary and tertiary crushing to nominally -10 mm, two stages of grinding to 80% passing 75 µm, gravity concentration to recover free gold and intensive cyanidation of the gravity concentrates, cyanide leaching and carbon adsorption of gold, acid washing of loaded carbon, elution and electrowinning of gold and smelting. Cyanide tailings are detoxified using sodium metabisulphite and copper sulphate and the detoxified tailings gravitate to the tailings dam.

Evaluations of the circuit indicated that the plant could be modified to process an increased throughput of 1.7 Mtpa and following treatment of a blend of oxide and transition ore, in a ratio of 3:1, a number of changes were identified and implemented. During this period the grind was slightly coarser and the gold recovery was acceptable at approximately 82%.

17.2 Process Plant Design

Overview of modifications completed to increase throughput from 1.3 Mtpa to 1.7 Mtpa

The initial evaluation of the processing capacity of the existing plant indicated that an annual tonnage of 1.3 Mtpa was possible. Subsequent in-depth investigations by a specialist company established that the processing plant could, with modifications, be modified to increase the annual throughput to 1.7 Mtpa. In addition the modifications took account of the likely increase in the proportion of harder transition ore in the plant feed.

The aim of the process design component of the 2011 economic assessment was to complete a detailed investigation into modifications targeted for upgrading the plant to 1.7 Mtpa and to establish a capital cost associated with these modifications.

Priority was given to the minimizing of production downtime during equipment installation and the construction philosophy.

The installation of new plant equipment was planned with the majority of the installation work taking place during the ramp-up phase of the processing plant towards achieving nameplate capacity of 1.3 Mtpa. The steel structures and pipe work in the areas requiring more extensive modifications would be pre-erected where practical and installed during shutdowns specifically planned for these events, or during periods of planned maintenance.

In order to achieve the increased throughput target each section of the process plant was reviewed and upgrades and modifications identified. The modifications identified were:

- Install a secondary feed system for the addition of ore crushed in the open pit or reclaimed from stockpile.
- Install a cover over the ROM stockpile to reduce the impact of rain on the materials handling characteristics of high clay content ores.
- Install a Rock Breaker at the primary crusher dump hopper.

- Install additional Power Pack on existing Apron Feeder Drive to increase the speed of the unit.
- Install a larger capacity Mineral Sizer - ALP 650 Series with 2410mm wide throat.
- Increase the speed of various conveyors in the crushing circuit.
- Install a more robust Secondary Crusher.
- Install a more robust Tertiary Crusher.
- Modify the system for transferring the scrubber fines between the scrubber and the mil by upgrading the transfer pumps and rerouting the piping to eliminate the numerous bends which restrict the flow of pulp.
- Upgrade the Mill Cyclone Feed pumps, Hydrocyclone Cluster and all associated piping, including the cyclone overflow to CIL.
- Add/ Upgrade Process Water Pumps.
- Install an additional Linear Trash screen.
- Install 4 new CIL Leach Tanks complete with agitators, mechanical screens and carbon transfer pumps to increase the leach residence time.
- Change static and air powered equipment in existing CIL (old CIP) tanks to electrically powered units - mechanical screens and carbon transfer pumps.
- Install larger capacity Air Blowers /Compressors.
- Install an additional Tower Crane in the CIL circuit to improve maintenance downtime.
- Install an additional Elution Circuit complete with Regeneration Kiln to increase the amount of carbon that can be treated and to improve equipment availability.
- Upgrade Raw Water Supply Pump System and Piping.
- Upgrade Gland Service Water System.
- Upgrade Tailings Return Water System (Pumps & Pipe Line).
- Install a new electrical MCC for the new items of equipment.

17.2.1 Front End / Crushing and Screening

Utilizing the existing equipment as it now stands, throughput tonnages of 300 wet tonnes per hour were considered achievable

The existing primary MMD Mineral Sizer was replaced with a larger, more powerful ALP 625 Series, double drive unit, allowing for an increase in throughput up to 300 wet tonnes per hour.

Operating experience with the new ALP Mineral Sizer quickly highlighted shortcomings in the tooth design and tooth replacement strategy, particularly when processing mudstone ore during the wet season. The tooth change out regime on the new ALP Mineral Sizer was radically changed from the design “even” tooth pattern to an “uneven” one increasing the nip angle between the teeth significantly thereby improving throughput and reducing down time and maintenance costs.

The existing apron feeder drive and primary conveyor feed conveyor [21-CVR-01] drive units were found to be adequate and did not need upgrading as had been planned to achieve the throughput of 1.7 Mtpa. The speed of the apron feeder feeding the ALP Mineral Sizer was capped at 72% to eliminate overloading of the primary feed conveyor. The conveyor belt width was increased from 750 mm to 800 mm and the installation of additional training idlers on the carry and return portions of the belt greatly increased belt operating time and reduced spillage.

The scrubber fines pumps were upgraded with new 75 kW motors and upgraded VSD's (variable speed drives).

A single 315mm OD pipeline with smooth long radius bends was installed to replace the original twin 250mm lines with sharp 90 degree elbows that were prone to sanding out.

The original Zenith HPC220, 220 kW Secondary Cone Crusher and the Zenith HPF220, 220 kW Tertiary cone crusher, were removed and replaced with, more robust Secondary and Tertiary, FLSmidth Raptor XL300 Crushers. Installation was completed in June 2013.

17.2.2 Secondary Plant Feed Point

During the implementation of the expansion project it became evident that overall plant availability and utilization could be enhanced by feeding crushed and screened ore into the plant via a secondary feed point. Originally a small feed hopper was installed over 23-CVR-01 in the crushing area to facilitate the loading of alluvial ore mined from below the TMF into the plant. This allowed ore to be fed directly to the grinding circuit via the mill feed bin, bypassing the front end altogether. This concept was developed further during 2014 through the installation of an additional feed conveyor located adjacent to 23-CVR-08, the mill feed bin conveyor. The use of available mobile crushing and screening equipment to provide -25 mm ore to the grinding circuit has now become permanent operating practice providing supplementary feed on an hourly basis as required to keep the grinding circuit operating effectively while the front end of the plant is shutdown.

17.2.3 Upgrade of feed weightometers

At the higher feed tonnages the scrubber feed weightometer and the Mill no. 1 feed weightometer were out of range which potentially introduced significant metallurgical accounting errors in terms of measurement of the feed tonnage. Both weightometers have been changed to larger units.

17.2.4 Milling

The implementation of the expansion to 1.7 Mtpa was incremental in nature and it was realized that an upgrade to the existing grinding circuit was not required at the time but recognized that it would be needed for any further expansion.

However a number of minor changes were made to the installed equipment in the grinding circuit to enhance circuit performance and to effectively operate at 1.7 Mtpa.

The trommel panels on both No.1 and No.2 ball mill were changed from 10 mm polyurethane panels to 10 mm wire screens to increase the effective open area and screening efficiency. The cyclone feed pipelines have been extensively replaced with durable rubber mining hose with smooth long radius elbows for improved pumping efficiency and longer service life. A fourth cyclone was added to the cyclopack increasing the number available from three to four.

An additional 9m^2 Linear Trash Screen, required to handle the increase in flow rate with the planned tonnage increase was ordered and installed in parallel with the current operating Linear Trash Screen.

The existing Linear Trash Screen will be relocated to feed the new #2 CIL Tank to be constructed, and the CIL will be operated as two parallel trains.

17.2.5 CIL

The construction of four new, 412m^3 CIL Tanks (same size as existing # 1 and 2 CIL Tanks) was completed in September 2013. The new tanks have been constructed at the head of the existing CIL train, in order to increase the residence time at the original tonnage and enhance the residence time with the future increase in tonnage.

Four new Kemix agitators are installed in the new CIL tanks. Ten complete Kemix MPS700(P) mechanically swept interstage screens have also been installed throughout the CIL circuit. All the screens will have 800 µm baskets.

The existing static, air swept, wedge wire screens in CIL tanks #8 thru 13 were prone to choking and have been replaced with mechanically swept Kemix MPS700(P) screens, the same as installed in the first eight CIL tanks. All screens will be fitted with 800 µm screens which will allow the tanks to be operated at higher pulp densities than before, thereby allowing better mixing of the carbon in pulp, improved adsorption efficiencies, reduced dissolved gold loses, savings in reagents, increased residence time and improved performance of the CIL in general.

The airlifts for carbon transfer in CIL tanks #8 through 13 have been replaced with vertical spindle pumps, thereby eliminating the excessive use of blower air, which is better utilized for agitation and leaching in the CIL. A total of ten FLSmidth Krebs 4x4 1.8L vertical carbon transfer pumps have been installed throughout the CIL circuit.

Two used (110 operating hours) high capacity Atlas Copco ZE4 200kW 3.5bar blowers have been installed and tied into the existing CIL air sparging system to enable oxygen levels in the CIL to be increased, thereby improving leach kinetics and overall recoveries.

A second tower crane (used) has been installed for construction and servicing of the new CIL tanks. This crane will be positioned so as to enable it to also be utilised to service the milling and cyclone area as well.

17.2.6 Acid Wash, Elution and Regeneration

The existing acid wash and elution system will remain unchanged.

The original vertical carbon regeneration kiln was removed to make room for the new Kemix 300 kg/hour diesel fired horizontal regeneration kiln. A new (second) elution system, complete with horizontal regeneration kiln was installed and commissioned in August 2014 and now operates in parallel with the existing carbon treatment system. This allows for more efficient and more complete stripping of gold from the carbon, ensuring all stripped carbon is regenerated before it is returned to the CIL circuit, thereby improving recoveries and maximizing gold production.

17.2.7 Electro-winning and Smelting

The new elution system included two complete electro-winning cells, gold sludge recovery system, calcine oven, smelting furnace, gold scale, doré handling tools, a bullion safe, work bench and associated fume hoods and extraction fans.

17.2.8 Tailings

The existing tailings carbon safety (catch) screens capacity will be assessed as tonnage throughput increases, to establish if there is a need for additional equipment to accommodate the increase of the capacity of the system.

The present Detox and tailings delivery system will be evaluated at high tonnage throughput.

17.2.9 Water Supply Systems

The capacity of the raw water and tailings return water systems has proven adequate for the expansion to 1.7 Mtpa.

17.2.10 Summary and Conclusions

Overall the expansion from 1.3 Mtpa to 1.7 Mtpa can be considered successful in its overall scope to increase throughput as designed. However, while the expansion project is essentially complete, a number of processing issues have become apparent and will need to be addressed as part of the on-going optimization program.

The operation of a total of 17 CIL tanks in series has proven very difficult from the carbon management perspective. Transferring carbon sequentially upstream through all tanks has proven difficult to manage and control effectively. Numerous operating strategies for slurry movement through the tanks and carbon transfer regimens have been attempted. At present the slurry flows sequentially through all 17 tanks in series. Attempts to operate two parallel slurry streams proved impractical when one single tank was down for maintenance. As of writing this report the first four tanks in the leach train, the new tanks installed in the expansion, are operated as leach tanks. Loaded carbon is transferred up stream only to the old No.1 CIL tank. Thus the circuit now operates as a CIP plant. Further changes to this arrangement may be adopted in the future to fully optimize gold leaching, carbon adsorption and carbon management and effectively process the different ore types present in the ore body.

Residence time in the CIL circuit has been the subject of much debate and deliberation since start-up and ramp-up to 1.3 Mtpa and through the expansion to 1.7 Mtpa. The determination of the optimum residence time required for the oxide, transition and fresh ore is necessary before any further tankage additions are contemplated.

17.3 General Process Plant Description

17.3.1 Introduction

The following describes the flow sheet of the process plant as of December 2014 treating 1.7 Mtpa of a blend of soft oxide and harder transitional ore. The plant now consists of the following sub-sections:

- Mobile Crushing and Screening
- ROM Pad Storage Area and Primary Feed Point
- In Plant Crushing and Scrubbing
- Milling and Classification
- Gravity and Intensive Cyanidation
- Trash Removal
- Leach and Adsorption Carbon-in-Leach (CIL)/Carbon-in-Pulp (CIP)
- Carbon Safety and Detoxification
- Tailings Dam Storage and Return
- Acid Wash
- Elution
- Electrowinning
- Regeneration
- Gold Room
- Reagents and Consumables
- Air Services
- Water Services

17.3.2 Mobile Crushing and Screening

In order to control the size of ROM fed to the Plant, mobile crushers and screens organized under the outside section or ore re-handling section in the pit are used for the realization of pre-crushing and pre-screening of ROM ore which is then selectively routed either to the ROM pad (-100mm to +25mm) or the secondary feed point (-25 mm) for plant feed.

17.3.3 ROM Pad Storage Area Primary Feed Point

The ROM pad as the primary feed point is covered with a roofing structure since the end of Q2 2014. The completion of the ROM pad roof allows dry storage of up to 25,000 tonnes of a mixture of both ROM ore and pre-crushed and pre-screened ore products from the rehandling section in the pit. This solution was implemented to protect stockpiled ore from the rain and thereby control the moisture content of the ore. Historically, ore processed through the plant exhibits significant changes in overall moisture content from dry to wet season. Wet sticky clay ore is extremely difficult to process in the crushing section of the process plant. Controlling the moisture content of ore, especially ores with high fines or clay content, minimizes the effect of changes in the viscosity of the ore. Drier high clay content ore has better flow and processing characteristics than ore that has been exposed to rainfall.

17.3.4 In-Plant Crushing and Scrubbing

Blended ore from either the ROM storage pad or direct from the pit are fed into the in-plant crushing circuit. The front end of the plant will treat ore at a rate of up to 300 wet t/h to a product of size 100% passing 100 mm, through the ALP mineral sizer.

The blended ore is tipped into the ROM bin using rear dump trucks or fed from the ROM pad stockpile by front end loader. The ROM bin has a capacity of 50m³. A variable speed apron feeder withdraws the ore from the bin at a controlled rate and discharges it into the mineral sizer. A small amount of oversize ore (rejects) from the crusher are collected in a sizer rejects stockpile. The mineral sizer reduces the ore to a product size of 100% passing 100 mm. The crushed product is discharged onto the crusher discharge conveyor which, in turn feeds into a rotary scrubber.

Process water is added to the scrubber to produce discharge slurry with a 50% solids density by weight. The scrubber discharge slurry overflows onto a vibrating double deck scrubber discharge screen. The upper screen deck is set at 50mm x 10 mm slotted. The lower deck is 2 mm.

The secondary crusher feed conveyor transfers the screen top deck oversize to the secondary crusher. The bottom deck oversize is conveyed into the mill feed bin by means of a scrubber screen discharge conveyor, transfer conveyor and crusher product conveyor. The screen bottom deck undersize slurry gravitates into a 12m³ scrubber discharge sump, from where it is pumped to the mill discharge sump.

The secondary crusher reduces the ore size to a product of 100% passing 37 mm. The crusher discharge conveyor transfers the crushed ore to the mill feed bin which has a capacity of 480 m^3.

17.3.5 Secondary Feed Point

The secondary feed point has been developed to provide a means of feeding the process plant when the front end of the plant, the mineral sizer, scrubber, scrubber discharge screen, fines pumps, in plant crushing and conveyor belts, are shut down.

Operation of the plant and gold production can be maintained, albeit at a lower overall tonnage, during such times. On regular operating days the secondary feed point is used to supplement the tonnage processed by the front end of the plant.

The -25mm screened fines from the re-handling section or direct from the pit are stockpiled in the security airlock. These fines are fed to the plant through a “grasshopper” conveyor which passes through the security fence discharging directly onto the mill feed bin feed conveyor.

17.3.6 Milling and Classification

The milling circuit consists of one primary ball mill (Mill 1) and one secondary ball mill (Mill 2), which are capable of treating up to 250 t/h of blended ore.

Crushed ore, from either or both primary and secondary feed points is fed into the No.1 ball mill using a variable speed reclaim conveyor. The dimensions of this ball mill are 3.638mØ (inside liners) x 4.660m EGL and its installed power 1300kW. The ball mill discharge slurry overflows onto No.1 mill trommel screen with a screen size of 10mm. The oversize from the screen is collected in No. 1 ball mill scats bunker. The undersize slurry gravitates into a 17 m³ mill discharge hopper, where it combines with the slurry from the No.2 ball mill. The combined mill discharge slurry, with a solids density of 52% by weight, is pumped by one of the cyclone feed pumps to the mill discharge cyclone cluster.

The cyclone cluster classifies the slurry to produce an overflow at a target of 80% passing 75 µm, with a solids density of 36% by weight. The cyclone overflow gravitates to a linear trash screen.

The cyclone underflow, at 67% solids density by weight, gravitates to the 2m³ cyclone underflow splitter box 1, where there is an off take to the gravity circuit which is currently not used. The cyclone underflow then reports to a 2 m³ cyclone underflow splitter box 2, from where 33% of the stream is sent to the No. 2 ball mill via a velocity break box. The dimensions of the second ball mill are 3.088 m Ø (inside liners) x 3.035 m EGL and its installed power 550 kW. 67% of the stream from the splitter box 2 gravitates back to the no. 1 ball mill.

Milling and cyclone spillage is contained in a bunded area that has a sloping floor to direct spillage to two sumps. One sump is allocated to the mill feed spillage and the other the mill discharge and cyclone spillage. Each sump has a vertical spindle spillage pump. Both spillage pumps discharge into the mill discharge sump. There are two more spillage pumps, one for the No.1 ball mill scats bunker and another for the No.2 mill bunker scats bunker area.

17.3.7 Gravity and Intensive Cyanidation

At the current plant throughput of 1.7 Mtpa it is not possible to operate the gravity circuit effectively while maintaining the plant throughput. Effective operation of the Knelson concentrator requires a specific and large volume of flush water. A high throughput thus upsets the water balance in the ball mill discharge pump box limiting overall throughput.

Although this section of the plant is currently not used the following process description still applies.

A portion of the cyclone underflow can be diverted from splitter box No.1 over the 1.12 mW x 2.40 mL gravity feed screen to remove the +2mm material. Dilution water is added onto the screen to dilute the feed slurry to a solids density of 45% by weight. The screen oversize is recycled back to the no.1 ball mill. The screen undersize is fed to the Knelson concentrator to recover the free gold. The tailings from the Knelson concentrator is returned to the cyclone feed sump.

The concentrates from the Knelson concentrator is periodically discharged into a concentrate cone for dewatering, with excess water overflowing to the floor, from where it is pumped to the mill discharge sump using the gravity spillage pump. Concentrates are stored in the concentrate cone until tabling or the intensive leaching cycle is ready. Presently gravity concentrates are treated by means of a Johnson Table and the gold concentrate collected and smelted directly after calcining.

An Intense Cyanide Leach reactor has been installed and is yet to be properly commissioned once the pregnant solution tanks for the system have been relocated to make space for the second complete elution system.

The Intense Cyanide Leach Reactor (ILR) - Gekko will process the gravity concentrates. At the beginning of each batch leach cycle, the entire contents of the feed cone will be discharged into the ILR drum. Excess water from the drum, during the loading cycle, will overflow into the ILR drum sump and will be pumped by the recirculation/transfer pump to the ILR solution storage tank.

The levels in the solution storage tank will be adjusted through the addition of raw water, caustic and cyanide solutions before the commencement of the leaching step. Leaching will be effected by re-circulating the 2% cyanide solution through the rotating reactor drum. The overflow will gravitate to the sump and will be pumped back to the solution storage tank. Hydrogen peroxide will be added to the ILR sump to provide oxygen for the leaching step.

At the end of the leach cycle, which will range between 14–16 hours, the drum will be stopped and the solution in the drum allowed to drain into the ILR sump and pumped to the solution storage tank where it will be clarified by adding flocculant. The clarified solution will be pumped to the gravity pregnant tanks. Wash water will be added to the drum to wash entrained solution from the solids and allowed to clarify in the solution tank before being pumped to the leach tanks.

The leached and washed solids will be emptied by running the reactor drum in reverse and pumped to the mill discharge hopper.

The pregnant solution stored in the gravity electrowinning tanks will be pumped to a dedicated electrowinning cell, wherein gold will be deposited onto steel cathodes.

The fume hood fan is installed to remove potential poisonous and explosive gases evolving during electrowinning.

Owing to the use of strong caustic and sodium cyanide solution in the ILR, a safety shower is provided in this area. It is activated by a foot pedal and equipped with an eye bath.

17.3.8 Trash Removal

The leach feed slurry is passed over a 9 m^2 linear trash screen to remove any tramp material before it is fed to the leach circuit. The screen undersize is sampled, for metallurgical accounting and control, by the leach feed sampler as it gravitates to the leach and CIL circuit. There is a provision for a second 9 m^2 trash linear screen to feed the leach circuit.

Oversize from the trash screen is collected in a trash basket.

17.3.9 Leach and Adsorption Circuit

The trash screen undersize gravitates into a 4 m^3 leach feed boil box. The boil box channels the slurry into the first leach tank, and a provision is made to feed the second leach tank from the box when leach tank 1 is off-line.

The leach and adsorption circuit has a total of seventeen tanks, with a total residence time of 11.5 hours. The circuit is operating currently as a CIP plant. The CIP configuration is composed of 4 leach tanks and 13 adsorption tanks. The slurry and carbon in the tanks is maintained in suspension by the action of the dual impeller tank agitators, each with 22 kW of installed power.

All leach and adsorption tanks are equipped with Kemix interstage wedge wire screens, which prevent any migration of carbon from one tank to another during slurry inter-tank flow. The flow of slurry from one tank to another is affected by the pumping action of the internal impeller mechanism of the interstage screens through an interconnecting launder system.

Each screen is periodically lifted from the tank onto a wash frame for cleaning. A spare screen is provided to replace other screens during cleaning or maintenance of any screen. A high pressure, low volume wash pump is used to clean blocked screens. A tower crane is utilised to lift screens and for general maintenance purposes.

The cyanide solution is added into the feed boil box from a ring main. A TAC1000 automatic cyanide analyser is used to ensure efficient addition of cyanide solution. A provision is also made to manually add cyanide into the first three CIL tanks in the event that cyanide concentration is too low.

Lime slurry is added into the leach feed boil box for pH adjustment. Blower air, at 250 kPa, is introduced (sparged) into the tanks for oxidation during gold dissolution.

Interstage carbon transfer pumps transfer carbon from the last tank through to the first tank in the adsorption circuit. Loaded carbon is pumped from the first adsorption tank using a loaded carbon recovery pump to a 1.8mL x 0.9mW loaded carbon screen. With the expansion programme an extra Elution circuit is installed for more flexibility in gold stripping from loaded carbon. There is therefore provision to route the loaded carbon either to the old elution circuit or to the new one.

Eluted and regenerated carbon is added to the last tank or the second last tank if the last tank is off-line.

Spillage in the Leach and Adsorption circuit area is contained in a bund and pumped by two dissolution area sump pumps.

Three safety showers are provided and each is activated by a foot pedal and equipped with an eye bath.

17.3.10 Carbon Safety and Detoxification

The tailings slurry from the CIL gravitates to the first of 2 cyanide destruction tanks in series. A bypass facility is provided to direct the tailings slurry to the second tank whenever tank 1 is off-line.

Sodium metabisulphite, copper sulphate and blower air (at 250 kPa) is added to the circuit to provide sulphur dioxide, copper catalyst and oxygen, respectively. The free cyanide and/or weakly bound metal cyanide complexes present in the tailings slurry oxidizes to the less toxic cyanate (OCN^-) according to the reaction:

$$CN^- + SO_2 + O_2 + H_2O = OCN^- + H_2SO_4$$

Lime slurry is added into the circuit to neutralise the sulphuric acid that is generated during the reaction, and thus maintain the pH within a range of 8 – 10.

The slurry exiting the second cyanide destruction tank gravitates into a 2 m³ safety screen feed splitter, where it is split and fed to two 4.8 mL x 1.5 mW carbon safety screens. Any carbon that escapes from the adsorption section, as a result of damaged interstage screens, is recovered as screen oversize and collected in a safety screen oversize basket. The screen undersize slurry gravitates into a 20 m³ guard screen undersize hopper. The final tails slurry gravitates from the hopper to the tailings dam. A sample is taken by a cross-cut tails slurry sampler.

Spillage in the carbon safety and detoxification area is contained in a bund and pumped to the feed splitter using a spillage pump.

Two safety showers are provided in this area. Each shower is activated by a foot pedal and equipped with an eye bath.

17.3.11 Tailings Dam Storage and Return

Process water from the tailings dam is pumped by pontoon pumps to a 200 m³ tank. Return water pumps transfer water from this tank to another 200 m³ tank, from where it is pumped to a 1,000 m³ tank plant process water tank.

17.3.12 Acid Wash

Loaded carbon from the first CIL tank is pumped to a 1.8 mL x 0.9 mW loaded carbon screen. Spray water is added onto the screen to wash the slurry off the loaded carbon. The slurry (screen undersize) is returned to the first adsorption tank. The washed loaded carbon gravitates into a 7.6 m³ acid wash hopper, and then to the acid wash column. The column accommodates a loaded carbon batch of 3t.

During acid washing the acid wash pump circulates dilute hydrochloric acid solution (~3% HCl) from a 10 m³ acid wash tank through the acid wash column at a rate of 2 bed volumes per hour for a period of 1 hour. The acid exiting the column is return to the acid wash tank via internal strainers that prevents any carryover of carbon.

At the end of one hour of acid washing, the acid wash pump is stopped. The content of the acid wash column is rinsed with a volume of raw water equivalent to 2 bed volumes. The rinse effluent exiting the column is directed to the carbon safety and detoxification circuit. Rinsed carbon is hydraulically transferred to the elution column.

Periodically, typically after every 4 acid washes, the dilute acid in the acid wash tank is too contaminated for further use. The solution is pumped to the carbon safety screen. Fresh acid wash solution is prepared by filling the acid wash tank with raw water to a pre-determined level and then pumping approximately 909 litres of 33% HCl in order to make up a 3% HCl solution strength.

Spillage in the acid make-up area is contained in a bund that has a spillage pump which pumps the spillage to the carbon safety screen.

A safety shower is provided as a result of the use of acid solution in this area. It is activated by a foot pedal and is equipped with an eye bath.

17.3.13 Elution

Two elution circuits (old and new elution circuit) are installed to process loaded carbon at an acceptable rate to recover the gold. The two circuits are independently connected to two electrowinning circuits to deposit the gold, followed by two independent regeneration kilns for carbon re-activation.

Gold elution from the loaded carbon is carried out using the Zadra method. The elution liquor (12.5 - 2.9 % caustic solution) is pumped from the eluate tank, via secondary and primary heat exchangers into the elution column under elevated temperature and pressure. This promotes a chemical reaction where gold adsorbed onto the carbon is removed by washing with a number of bed volumes of hot water. The gold is thus stripped from the carbon into the eluate solution.

The elution column is operated under a pressure of typically 300 to 350 kPa for the old elution and 400 to 450 kPa for the new elution. Heat required for the elution cycle is provided by diesel fired elution heaters. The heater burners heat up thermal oil, which is used to transfer heat to the eluant solution entering the primary heat exchanger en route to the elution column. The pregnant electrolyte leaving the column is cooled by the fresh eluant solution in the secondary heat exchanger. The primary and reclaim heat exchangers are plate and frame type.

The pregnant electrolyte flows out of the elution column to the electrowinning cells. In the cells gold is deposited on the stainless steel wool cathodes. The electrolyte gravitates from the cells back to the eluent solution tank from where it continues to be cycled through the elution column until the elution is completed. The barren eluent is utilized for a number of elutions before it is discarded to the Leach circuit and a fresh batch of strip solution made up.

Once gold has been stripped from a batch of loaded carbon, the elution cycle is complete. The eluted carbon is hydraulically transferred from the elution column to the regeneration circuit or to the last tank.

To reduce scale build up in the heat exchangers a sulphamic acid solution is periodically circulated by the sulphamic acid dosing pump through the heat exchangers. Spillage in the elution area is contained in a bund that has a spillage pump which pumps spillage to leach.

A safety shower is provided in this area. It is activated by a foot pedal and equipped with an eye bath.

17.3.14 Electrowinning

The pregnant solution (eluate) from the elution column is circulated through the electrowinning circuit until acceptably low gold values are achieved on both the eluted carbon and the electrolyte. Gold is electrowon from the eluate in cells using stainless steel wool cathodes. It is deposited onto the cathodes as loosely adhering sludge. The circulation of the projected 63.5 m^3 of eluate through the elution column and electrowinning cells continues for about 16 hours during which time the pregnant solution grade decreases from a target of 120 to 5 ppm gold.

Loaded cathodes from electrowinning are periodically remove from the cells to the cathode wash table in the gold room, where the gold sludge is washed off by means of a high pressure washer. Gold sludge from the cells is collected in a concentrate cone and drained into buckets, from where it is transferred to the filter press in the gold room.

A fume hood fan is installed to extract potentially poisonous and explosive gases that evolve during electrowinning from the cells and discharge outside the gold room.

Owing to the use of caustic in this area, two safety showers are provided. Each safety shower is activated by a foot pedal and equipped with an eye bath.

Spillage in the barren solution tank area is contained in a bund and pumped to the Leaching circuit. Spillage generated in the electrowinning cell area is also pumped to head of the CIL.

17.3.15 Regeneration

At the end of an elution cycle, eluted carbon is hydraulically transferred from the elution column to the barren carbon storage tank for regeneration. A provision is made for eluted carbon to bypass the regeneration. The eluted carbon gravitates from the barren carbon storage tank to the carbon regeneration kiln feed box which is fitted with a small dewatering screen. Excess water is drained off as screen undersize, and the oversize (carbon) is fed by spiral screw feeder to the 300 kg/hour. Horizontal Diesel Fired Regeneration Kiln.

The kiln, operating at 700 °C, will treat the entire carbon batch in a period of 10 hours.

The regenerated carbon is quenched with water, before it can react with atmospheric oxygen, in a 7.2 m^3 barren carbon transfer tank, from where it will be hydraulically transferred to a vibrating 1.2 mL x 0.6 mW re-activated carbon fines screen. The screen undersize (carbon fines) will gravitate into the guard screen undersize hopper. The screen oversize (activated carbon) will gravitate into the last adsorption tank.

17.3.16 Gold Room

The filtered concentrate (gold sludge) from the vacuum pan filter is placed in stainless steel calcining trays, up to 6 trays at a time and the trays are loaded into the calcining furnace operated at 800°C for drying. The calcining trays are then removed from the furnace and placed on a cooling table and allowed to cool down.

When the dried product from the calcining furnace has cooled down, it is weighed, mixed with fluxes (stored in the flux storage box) in determined proportions. The gold sludge/flux mixture is transferred to a smelting crucible in a diesel-fired smelting furnace operated at 1,200 to 1400□C.

During smelting metal oxides form slag and at the end of smelting the furnace crucible contents are poured into cascading moulds mounted on a cascade trolley. The bullion collects in the first mould with any excess collected in the second mould while slag flows and is collected in a slag collection crucible. A steel slab is included to protect the concrete floor.

Once cooled, the gold bars are cleaned, sampled, stamped and stored in the safe prior to dispatch.

Two safety showers are installed in the electrowinning and gold room area. They are activated by a foot pedal and equipped with an eye bath.

17.4 Consumables

17.4.1 Mill Balls

Grinding media, 75 and 50mm Ø balls are used in No.1 ball mill and 50 and 25mm Ø balls are used in No. 2 ball mill. For ease of transportation, grinding media is delivered in 200 litre drums. Grinding media to mill 1 and ball mill 2 are charged using a bottom discharge skip and a hoist.

17.4.2 Cyanide

Sodium cyanide is delivered to site in 1t bulk bags and packed into wooden crates, to limit the danger of spillages during transportation. The bulk bags are transported using a fork lift from the cyanide storage area to the cyanide mixing area, located in the reagent make-up area.

Prior to the addition of cyanide briquettes into the cyanide tank, the pH of the water in this tank is adjusted to about 10 using caustic soda solution in order to prevent any formation of hydrogen cyanide at low pH values.

When the pH has been adjusted, the hoist lifts the cyanide bags to the bag breaker from where they are discharged into a 21 m^3 covered cyanide mix tank. The tank agitator ensures that cyanide briquettes are completely dissolved during the make-up process to form a 25% cyanide solution by weight.

The cyanide solution is transferred from the make-up tank to a 65 m^3 cyanide solution storage tank using the transfer pump.

One of the cyanide solution feed pumps transfers cyanide solution into the cyanide ring main from which cyanide is tapped off into the leach feed boil box or CIL tanks. A safety shower is installed in the cyanide make-up area. It is activated by a foot pedal and equipped with an eye bath.

Spillage in the cyanide make-up area is contained in a bund and a spillage pump is used to pump cyanide into the leach feed boil box.

17.4.3 Caustic

Caustic is delivered to site in 25 kg bags packed onto wooden pallets. The 25 kg bags on pallets are transported using a fork lift from the caustic storage area to the caustic tank for make-up area. The dissolution of caustic takes place in a 5m^3 covered caustic tank. Once delivered to the caustic area, the reagent hoist is used to lift the caustic pallet to a platform on top of the caustic tank from where an operator manually lift one 25 kg bag at a time onto the bag breaker. The caustic pearls discharged into the caustic tank. The caustic tank agitator ensures that the caustic pearls are completely dissolved during the make-up process to form a 20 % caustic solution by weight.

The caustic dosing pumps only runs for the time required to deliver the various quantities of the reagent to various distribution points, including the cyanide make-up, acid wash, and barren eluent tank (elution).

A safety shower is installed in the caustic make-up area. It is activated by a foot pedal and equipped with an eye bath.

Spillage in the caustic make-up area is contained in a bund and a spillage pump is used to pump it into the leach feed boil box.

17.4.4 Lime

Un-slaked lime (quick lime) is delivered to site in 1t bags and kept in a lime store. The bags are hoisted and transferred to the lime loading hopper by overhead crane, where the bags are cut open. The hopper will be equipped with a dust filter that will help keep the operation dust free.

Lime powder is transferred from the hopper to a 45 m³ silo by means of a rotary feeder and a transfer blower. The lime silo is also equipped with a dust collector. A rotary feeder and an inclined screw feeder transfer lime from the silo, at a rate of 4 t/h, to the lime slaker during the slaking period. An electric silo vibrator is fitted at the bottom of the transfer hopper to enhance discharge of lime from the silo.

The lime slaker continuously convert calcium oxide (quicklime) into calcium hydroxide or slaked lime in the form of slurry of controlled consistency, and dilutes it to the required density of 15% by weight.

Water and lime is added to the first compartment of the slaker in measured proportions and vigorously agitated by slaker mixers.

The shape of the vessel promotes efficient slaking and the overflow into the adjacent compartment ensures the necessary retention time, thus preventing the discharge of un-slaked material. The temperature is between 70° and 75°C.

Slaked lime overflows from the second compartment of the lime slaker onto a vibrating grit classifier for the removal of grit. The oversize grit is stockpiled and removed by a front end loader, while the screen undersize gravitate into the slaked lime sump fitted with a mixer to keep the lime particles in suspension. A lime transfer pump transfers slaked lime to a 140 m³ lime storage and dosing tank, where it will be kept in suspension by a mixer.

Lime slurry is pumped to the ring main by the lime dosing pumps, one on-line and another on standby, to the milling, CIL and cyanide detoxification circuits. Unused lime slurry in the ring main is returned to the dosing tank.

The spillage in the lime make-up area is contained in a bund and transferred by a spillage pump to the grit classifier.

Owing to the presence of hot lime slurry in the make-up and dosing areas, a safety shower is provided. It is activated by a foot pedal and equipped with an eye bath.

17.4.5 Sodium Metabisulphite

Presently the sodium metabisulphite mixing and dosing system is out of order and the reagent lifted by tower crane and is manually added to the detoxification tanks.

Under normal operating conditions Sodium metabisulphite is delivered to site in 1 tonne bulk bags. The bulk bags are transported using a fork lift from the storage area to the sodium metabisulphite make-up area.

The detoxification reagents hoist is used to lift the bags to a bag breaker, from where they are discharged into a 19 m^3 covered sodium metabisulphite mixing tank. The tank is equipped with a mixer that ensures that the powder is dissolved completely during the make-up process to form a 25 % sodium bisulphite solution by weight.

Sodium metabisulphite solution is transferred from the make-up tank to a 21 m^3 dosing tank using a transfer pump.

Duty and standby variable speed hose dosing pumps are used to pump sodium metabisulphite solution to the carbon safety and detoxification circuit at a controlled rate.

A mixing tank fan and a dosing tank fan are used to extract any sulphur dioxide that is generated during the make-up of sodium metabisulphite solution.

17.4.6 Copper Sulphate

Presently the Copper Sulphate mixing and dosing system is out of order and the reagent lifted by tower crane and is manually added to the detoxification tanks.

Copper sulphate is delivered to site in 25 kg bags on 1 tonne pallets. The pallets are transported using a fork lift from the storage area to the copper sulphate mixing area.

Once delivered to the makeup area, a hoist is used to lift the pallet onto the platform where an operator loads the bags onto a bag breaker, from where they are discharged into a 3 m^3 covered copper sulphate mixing tank equipped with a mixer which ensures that the crystals are dissolved completely during the make-up process to form a 15 % copper sulphate solution by weight.

The 15 % copper sulphate solution is transferred from a make-up tank to a 7 m^3 dosing tank using the transfer pump.

Duty and standby variable speed hose pumps transfer the copper sulphate solution to the carbon safety and detoxification circuit at a controlled rate.

A safety shower is installed in the sodium metabisulphite and copper sulphate make-up area. It is activated by a foot pedal and equipped with an eye bath.

Spillage is contained in a bunded area and spillage pumps transfer the spillage to the safety screen feed splitter in the carbon safety and detoxification area.

17.4.7 Plant Diesel

Plant diesel is transferred from the main diesel storage facility to a 7 m^3 diesel storage tank from where it is pumped by the diesel pump to a 2.2 m^3 header tank. Diesel gravitates from the header tank to the elution heater, regeneration kiln and smelting furnace. Diesel for use in the laboratory is pumped from the main diesel storage facility to a separate 2.2 m^3 header tank.

Excess diesel from the elution heater and regeneration kiln is return to the storage tank.

17.4.8 Carbon

Activated carbon is delivered to site in 500 kg bulk bags. The bulk bags are transported using a fork lift from the storage area to the Adsorption circuit. When required, the carbon is added into the last adsorption tank.

17.4.9 Hydrochloric Acid

Hydrochloric acid is delivered in 290 kg (210 litres) plastic drums at 33% strength by weight. The drums are shrink-wrapped and palletized for safety reasons and easy storage. When required, the palletized hydrochloric acid drums are transported using a forklift from the storage area to the acid wash area. Acid is pumped from the drum using a drum pump.

17.4.10 Air Services

Compressed Air

Two air compressors, one working and one on standby, deliver 180 Nm^3/h of compressed air, at a pressure of 750 kPa, to the plant air receiver via one of the air filters. The compressed air is distributed to the lime make-up area, workshop and various points in the plant for general usage.

Instrument air is supplied by a dedicated compressor. An instrument air drier and filters are provided in order to ensure that instrument air is moisture-free and of good quality prior to storage in an air receiver.

A provision is made to supply air from the discharge of the plant air compressor to the instrument air filters and dryer, if the instrument air compressor is off-line.

All three compressors are housed in a compressor shed.

Blower Air

There are three aeration blowers, two on-line and one common standby. The first blower provides oxidation air (at 250 kPa) to the first ten CIL tanks and the three cyanide destruction tanks. The second blower provides air to the airlifts in the last six CIL tanks for carbon transfer.

Leach aeration blowers, one on-line and another on standby, deliver 1981 m^3/h of blower air (at 350 kPa), which is distributed to the leach and CIP and carbon safety and detoxification circuits.

17.4.11 Water services

Water Abstraction and Storage Dam

Raw water from a storage dam is drawn by two centrifugal pumps (both on-line) and transferred to a raw water tank and a process water tank in the plant. A third pump is provided as a standby pump.

Process Water

Return water from the tailings storage facility and raw water top-up from the raw water storage tank constitutes process water. This water is directed to a 1,000m^3 process water tank.

Duty and standby low pressure high volume process water pumps transfers process water to the scrubber, ball mills and the gravity scalping screen.

High pressure spray water pumps, one on-line and another on standby, distribute spray water to several screens throughout the plant.

A high pressure hosing water pump is used to supply hosing water to the plant.

Raw Water

Raw water from a storage dam is routed to a 1,000m³ raw water tank located close to the processing plant and alongside the process water tank

Raw water pumps are used to distribute water to the crushing and milling areas (for dust suppression), intensive cyanidation, reagents make-up, potable water supply, carbon transfer, process water top-up, gland water tank top-up, final tails sump flushing water and high pressure wash pumps.

Gravity concentrator fluidising water requirements are provided through the use of dedicated pumps.

Potable Water

A raw water break tank receives raw water from the raw water pumps. The raw water undergoes purification in the water treatment plant. Purified (potable) water is pumped to a 100 m³ potable water storage tank. The backwash/rinse water from the water treatment plant is directed to the storm water drain.

Two potable water distribution pumps, one on-line and one on standby, supply water to the safety shower and potable water headers via a hydrosphere, which is used to maintain pressure in the potable water system.

Gland Water and Safety Shower Water Distribution

Gland water pumps, one on-line and another on standby, will transfer gland water from a 10 m³ tank to the glands of all slurry pumps that require gland water.

Potable water from the hydrosphere is distributed to various safety showers throughout the plant.

18 PROJECT INFRASTRUCTURE

18.1 Infrastructure

18.1.1 Roads

Road access to the Twangiza Project had been maintained along an existing 31 km section of access track linking the N2 national route from Bukavu, to the existing Twangiza exploration camp on the Twangiza property.

This section of road which has undergone extensive upgrade through widening of the road surface and adjustment of the alignment to insure minimum turning radii and maximum gradients is currently under constant maintenance to allow for the transport of all materials and equipment required for the successful continuous operation of the mine. The maintenance work of this portion includes the widening of some portions of the road surface, gravelling, and repairs of bridges. Suitable layer works are being done to ensure the strength of the road and to provide an all-weather surface. Adequate drainage has been provided to ensure free flow of rain water. This portion of road is made up of 3 bridges, namely bridges 4-6 which are currently in good condition.

A further 5 km of road linking the exploration camp and the process plant has been constructed to the same specification as the access road.

To allow for the construction of the tailings management facility, a road approximately 4.5 km long has also been constructed linking the mine haul roads to the embankment location of this facility. As this road is being used for the hauling of materials for embankment construction by the mine ADT fleet, a road width of approximately 20 m has been put in place.

18.1.2 Process Plant Buildings

Buildings including the mine offices, mine laundry and canteen, first-aid/consultation room, assay laboratory and security gatehouses are in place within the process plant site. Some of these buildings are of concrete and block work construction and local building materials. Local Bukavu based building contractors were used to carry out these works.

18.1.3 Process Plant Warehousing and Workshop

Warehouses and workshops are of steel framed, steel-sheet cladding construction founding on concrete floor slabs and bases.

The warehouses areas have been calculated based on the consumption of the relevant process plant consumables, as well as reasonable storage durations based on origin of supply and transport risk. Warehouse offices are within the general warehouse building.

As per Item 18.1.6 a separate contained area inside the general warehouse have been dedicated (for storage) of special lubricants and greases, for the mining operation. In addition, a designated caged area is in place for flammable products such as solvents and paints.

18.1.4 Process Plant Ancillary Infrastructure

Additional ancillary infrastructure such as compressor house, a helipad and sewage and water treatment plants with associated reticulation are in place for successful mining operations.

18.2 Accommodation

Five camps are currently in operation at Twangiza; these are in the following distribution:

Construction camp:	130 rooms with 227 residents;
Training Centre Accommodation:	26 rooms with 28 residents;
Dispatch Camp:	12 rooms with 17 residents;
Operators Camp:	110 rooms with 110 residents; and
Exploration Camp:	81 rooms and 39 tents with 114 residents.

These housing units are in 3 varying styles dependant on employment status, namely Senior Management, Supervisor and Junior staff.

Three out of the five camps include kitchen and dining facilities, a laundry, offices, an infirmary, a security/gatehouse and recreational facilities.

Most of the buildings are made up of imported prefabricated panel construction in order to minimise construction time and ensure early availability of accommodation and office requirements for mine construction. The 50mm thick panels are made of pre-painted chromadeck exterior with polyurethane insulated infill. Some however, are made of locally manufactured hydro-form bricks provided by local contractors to generate employment for the local community.

Services including electric, water and sewage reticulation as well as on-site water and sewage treatment plants have been provided.

18.3 Security

Various levels of security have been established within the Process Plant area.

The entire mineral processing plant from primary crushing through to the gold-smelting is completely enclosed to create a high-security area. The 6m high fence comprises galvanised, anti-cut, anti-climb panels including a barbed wire coil on the top. A no-go zone has been created by the re-location of existing diamond mesh /barbed wire fencing to the outside of the primary fence. Ultimately this fence will be replaced by a fence of similar specification to the primary. The electrowinning and gold room area are further enclosed within the high-security area. Storm water and Sewage systems within this area exit the boundary fence via gold-trap structures.

A medium security area adjacent to the process plant has been established containing the reagent storage and make-up areas, the fuel farm and power plant. The external perimeter to this area has been established utilising the diamond mesh/barbed wire fencing currently used for construction purposes. Ultimately this fence will be replaced by a fence of a higher specification. Within this medium security area a secondary fence is to be established around the plant stores area.

The administration area comprising offices, training centre and clinic is contained by a fence similar to that used at the process plant and plant stores area, but of a lower specification.

A plant wide CCTV system has been established, together with a complimentary lighting system, in order to provide for adequate perimeter monitoring. The security control room is equipped with recording and storage equipment and remote monitoring of the CCTV system.

A single access point to the process plant area is the primary access control system comprising vehicular boom gates, manned 24 hours per day. Secondary access control is provided to the administration area, the plant store and the high-security areas. Tertiary access control has been established for the gold-room

The increase of throughput from 1.3 Mtpa to 1.7 Mtpa did not impact the security requirements of the process plant.

18.4 Power requirements

The power generating plant at the Twangiza operation is rented from Aggreko, and Aggreko is also operating the plant. A total of 11 containerised, 1250 KVA generator modules, are installed to provide the power requirements for the increased throughput to 1.7 Mtpa. The modules have been de-rated by 17% due to the altitude of the site.

The existing electrical infrastructure is sufficient for the upgrade to 1.7Mtpa. This includes medium voltage switchgear, transformers and cabling. The exception was the 400 volt motor control centres which had to be extended. The size of the mobile switch rooms, housing these motor control centres, was adequate to include all changes.

The power cost from Year 4 to the life of the mine is expected to be reduced by 30% from USD 7.35/tonne processed to USD 5.15/tonne processed, following the planned supply of hydroelectric power from the Ulindi 1 river. The Hydro power plant is planned to be installed and run by a third party to supply power to Twangiza Mining on a usage billing agreement.

Studies by Knight Piesold show the economic gain in switching from diesel-generated power to hydroelectric power. The hydroelectric project will require an estimated capital of USD 40 million to install a capacity of 6.5 MW plant.

18.5 Communications

External telephone communications are currently provided by two cell phone operators and their services are satisfactory.

Internet connectivity is provided by two satellite systems, one at the plant site and the other at the exploration camp. The plant, administration area and stores area are linked via a fibre optic network. A microwave link between the two satellite systems at the exploration camp and plant site provides a failover as a backup should one system breakdown.

18.6 Sewage collection and treatment

Various ablution block/s have been provided in the relevant areas within the process plant and associated infrastructure, to which potable water from a water treatment plant is provided. All sewage is piped to septic tank and French drain systems outside the boundaries of the existing terraces. Gold-traps are installed on the lines where sewage pipes cross the high security fencing of the process plant area.

The blend throughput rate of 1.7 Mtpa is not expected to impact the sewage requirements of the process plant.

18.7 Fuel and lubricant storage and distribution

Fuel is delivered to the mine from Mombasa or Dar-es-Salaam by road tankers. The current fuel facility at Twangiza consists of three 80,000 litre tanks adjacent to the bonded lay down area and a further two 80,000 litre tanks south of the plant site in the proposed mine workshop area.

The main fuel farm is operational with two 1 million litre vertical tanks for storage purposes and two 80,000 litre day tanks providing fuel to the plant and generators.

A complete fuel management system will be installed at a later date.

18.8 Tailings Management Facility

18.8.1 Introduction

As part of the economic assessment, an evaluation of tailings storage capacity has been undertaken. A revised strategy for tailings management has been adopted by Twangiza Mining, in that the construction of the current Tailings Management Facility (TMF), namely TMF 1A, will be halted at an elevation of 2060.9 m (AMSL) and a new facility, TMF1, will be constructed in an adjacent valley.

The benefit will be an improved tailings to fill ratio of 5.7:1 (LOM) compared to TMF 1A, which has a planned final tailings to fill ratio of 1.135:1. The final volume of TMF1A will be 7.37 Mm^3, or a storage capacity of some 8.18 Mt, which is based on an average tailings dry density of 1.109 tonnes/m^3, the density which has been achieved to date.

18.8.2 Rainfall

Design values used by AMEC based on the assumption that a maximum recorded 24 hour rainfall event of 120 mm at Bukavu is equivalent to a 1 in 10 year event. Precipitation at the Twangiza site is generally higher than at Bukavu so this value was increased to 155 mm. AMEC produced rainfall estimates based on these assumptions.

However, SLR reviewed actual rainfall data and revised the estimations upwards as part of their design report for TMF1A. Both sets of estimates, for a 24 hour storm period, are summarised in Table 18-1.

Table 18-1: 24 Hour Design Storm Event Estimates

	Return Period	Intensity (mm)
AMEC	1:10	155
	1:100	180
	1:200	185
	1:1,000	210
	1:10,000	240
SLR	1:50	235
	1:100	289
	1:200	356

18.8.3 Production rates and design life

Based on the current mine plan, the required storage capacity for tailings is 23Mt oxide and non-oxides ore at a rate of1.7 Mtpa over a 14 year mine life. Two TMF's are planned, the current TMF1A and TMF1, which will be constructed in an adjacent valley.

TMF1A was originally designed to store 14.3Mtonnes (13.62 m^3) at a nominal dry density of 1.05 tonnes per cubic metre by Metago PTY of South Africa (now SLR Pty). The tailings dry density has currently exceeded this target having achieved 1.109 tonnes per cubic metre. However, due to unfavourable tailings to fill ratio of 1.135:1 and the inability to store revised LOM reserves, a revision to the tailings containment plan has been carried out by Twangiza Mining and a decision to construct an additional tailings management facility, TMF1, has been taken. At the same time it was decided to halt the construction of TMF1A in mid-2016 when it will have reached a level of 2060.9 m and have a capacity of some 8.18 Mt (7.37 Mm^3) at 1.109 Mg/m^3.

TMF1 will be constructed over the life of the mine and will have a capacity of 22.80 Mt (20.56 Mm^3) at a dam height of 70 m. Notwithstanding the above, there is an option to raise the dam height to 110 m at a later date should this be required to store potential additional reserves. This would provide a tailings capacity of some 65 Mt for TMF1.

18.8.4 Particle size distribution and specific gravity

The tailings particle size distribution, adopted from the 2011, 1.3 Mtpa study, was summarised as follows:

- 100% passing 2mm (by mass).
- 96% passing 0.85mm (by mass).
- 69% passing 0.075mm (by mass).
- 55% passing 0.020mm (by mass).

More recent gradings, carried out at Twangiza in August 2014 are shown in Table 18-2.

Table 18-2: Particle Size Distribution and Sieve Analysis

		Sieve Size							
	Passing%	25µ	38µ	45µ	53µ	63µ	75µ	106µ	150µ
Jun-14	Finest	73	75	78	82	84	88	93	97
	Mean	62	66	68	71	73	76	83	89
	Coarsest	43	48	51	54	58	60	75	84
Aug-14	Finest	74	77	78	80	82	83	89	94
	Mean	64	67	69	72	74	77	84	89
	Coarsest	51	57	59	64	67	70	77	84

Furthermore, hydrometer testing carried out in house recorded values of between 7% and 21% of material less than 2 µm.

Based on the above grading analyses and previous laboratory test work, the material classifies as a "CL" material according to the Unified Spoil Classification System i.e. a low plasticity clay although it borders on being a "ML" (silt).

18.8.5 Slurry characteristics

The tailings slurry, adopted from the 1.3 Mtpa study, was taken as 37% solids by mass (provided by SENET) which at a particle specific gravity of 2.84 t/m^3 yields a slurry bulk density of 1.32 t/m^3.

Initial work carried out by Patterson and Cooke found the specific gravity of samples to be as follows:

- Main Pit: Main Oxide: 2.80 t/m^3, Main Transition: 3.04 t/m^3, Main Fresh: 3.33 t/m^3.
- North Pit: North Oxide: 2.74 t/m^3, North Transition: 2.79 t/m^3, North Fresh: 2.99 t/m^3.

Additional work soil testing work carried out for other purposes has indicated specific gravities in the range 2.82 to 2.85 t/m^3. Based on these an average specific gravity of 2.83 t/m^3 has been adopted for calculation purposes. Considering the 37% solids suggested in the 1.3 Mtpa study, this yields an average slurry density of 1.315 t/m^3.

However, tailings solids are currently being discharged at between 30% and 33% by mass, which at a particle specific gravity of 2.83 t/m^3 yields a slurry bulk density in the range 1.24 to 1.27 t/m^3. With more transition material being processed, average slurry bulk density is likely to increase marginally to between 1.25 and 1.28 t/m^3.

Based on existing data from TMF1A and considering the high rates of rise, a minimum average dry density of 1.09 t/m^3 is expected in TMF1, for a pool size of 30%. A lower starting value of 0.75 t/m^3, rising to 1.109 t/m^3 over the first two years of operation, has been used for the rate-of-rise assessment.

Table 18-3, based on 1.7 Mtpa, presents the expected geotechnical parameters that the tailings will exhibit through its life.

Table 18-3: Tailings Parameters

Source	Unit	Value
Tailings Slurry		
Solids by mass	% bulk density (t/m^3)	30-33
Particle Specific Gravity	t/m^3	2.83
Slurry Bulk Density	t/m^3	1.24-1.27
Dry density	t/m^3	0.486
Moisture Content	%	203-233
Slurry Void Ratio		5.75-6.6
Deposited Tailings		
Average Bulk Density (Saturated)	t/m^3	1.485
Average Dry Density	t/m^3	0.75
Moisture Content	%	98.12
In situ void ratio		2.77
Deposited Freshly Consolidated Tailings		
Average Bulk Density (Saturated)	t/m^3	1.717
Average Dry Density	t/m^3	1.109
Moisture Content	%	54.96
In situ void ratio		1.57

18.8.6 Survey information

Survey information used in the study has been provided by Twangiza Mining's in-house survey department as point data and as an electronic contour map. Aerial photography of the area concerned has also been provided. The survey is based on a combination of the aerial survey completed in 2008 and a more recent ground survey.

18.8.7 Climate data summary

Details on the climate are provided in the report "Surface Water Investigations for the Twangiza Gold Project" by Metago dated April 2010. In summary the area has relatively high rainfall of about 1.35 m/year, evaporation of 1.2m/year, with the rainfall season lasting from mid-October to April, while the 4.5 month dry spell is from May to mid-September.

The design storm events calculated by SLR are as follows:

- 1:50 year 24 hour storm event is 235mm.
- 1:100 year 24 hour storm event is 289mm.
- 1:200 year 24 hour storm event is 356mm.

18.8.8 Rate of rise of tailings

The study requires the development of stage capacity relationships for the tailings dam in order to define the relationships between volume, area, height, rate of rise and production rate of the facility at any point in time during the design life. The stage capacity relationships are used to monitor the rate of rise of the tailings dams and to determine the required height of the start embankment of the tailings facility.

The stage capacity curves shown in Figure 18-1 indicates that a starter wall (TMF1) with a crest elevation of 1977 m AMSL is required for a 22.5 month capacity or 3.14Mt. TMF1A, completed to Stage 5, will contain 8.18Mt.

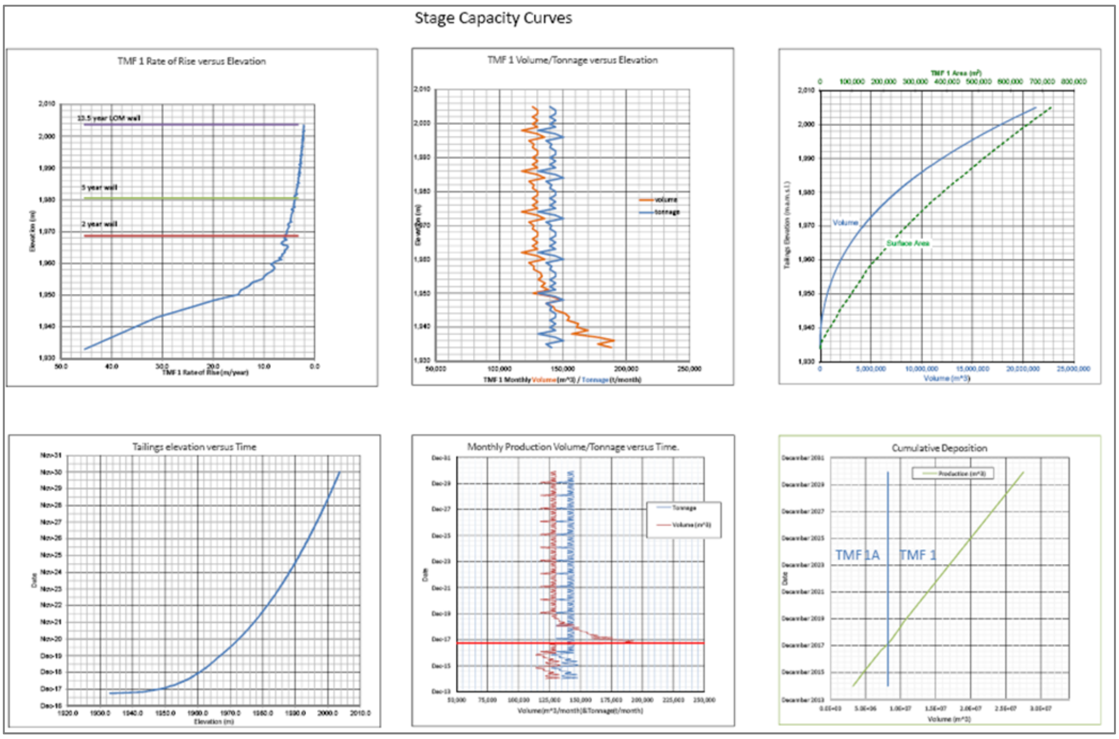


Figure 18-1: TMF Stage Capacity Curves

18.8.9 TMF construction works

A schedule of quantities has been drafted for the construction works for both TMF1A and TMF1, as the stream diversions required remains the same as for the 1.3 Mtpa study. Construction rates have been based on rates calculated in-house. However, these rates are generally in accordance with estimates provided by external contractors. The assumption has been made that an average of 10 m of material must be removed under the foundation of the wall which must be replaced with suitable compacted fill. Additional assumptions include:

- Crest width of 40m. Appendix XIV of the DRC Mining Regulations requires a crest width of greater than H/5 + 3m. This translates into a crest width of approximately 11.0 m for the 2 year starter wall, increasing to 17.0m for the LOM. However, this would prove problematic to construct without specialist plant, therefore a 30 m wide pad plus sand drain and impermeable clay curtain has been added resulting in a provisional width of 40 m. This has significant impact on the wall volumes and a review of construction techniques is required as significant savings are possible.
- Wall fill will predominantly be won from mine waste mixed with borrow material from within a 1.5 km radius to ensure fill moisture content is suitable for compaction.

The TMF costing excludes:

- Any lining of the TMF basin or any other ARD mitigation measure. Water quality is monitored on a daily basis and TMF1A is not seen to be impacting local ground water. The presence of a significant artesian/sub-artesian head underlying the TMF basin provides a degree of protection towards groundwater quality.
- Detoxification plant; To date, on TMF1A, detoxification carried out ex-process has proved adequate to control TMF water quality.
- The tailings delivery pipeline from the plant to the basin of the TMF1.

- The tailings delivery and return water pumping system (floating buildings, pumps, motors, and all associated mechanicals and return water pipeline. However, it is anticipated that the infrastructure already in place for TMF1A will remain in operation and return water will only require pumping from TMF 1 into TMF1A, minimising additional works.
- A grout curtain being required under the base of the wall. A grout curtain was not necessary for TMF1A and it is considered unlikely to be required for TMF1.
- Access Roads. Almost all necessary access roads are already in place. Maintenance is expected to be the primary cost.

18.8.10 Conclusions

The main conclusions in the TMF study are summarised as follows:

- The Construction of TMF1A will cease at a height of 2060.9 mASL resulting in a volumetric capacity of 7.37 Mm3 or 8.18Mt. This will provide tailings storage until the end of October 2017.
- A starter wall for the new tailings management facility, TMF1, with a crest elevation of 1977 m AMSL (40 m crest height) is required to store 22.5 months of tailings, 3.14 Mt.
- A production rate of 1.7 Mtpa is anticipated, giving a further 13.4 year LOM life for TMF1 following the filling of TMF1A (predicted October 2017), brings the total tailings capacity to 15.4 years.
- TMF1 wall will be raised during the dry seasons of 2018 and 2019 to a height of 1987 mAMSL (7.9 Mt capacity) with a further rise to 1995 mAMSL during the dry seasons of 2020 and 2021, which will result in storage for 13.0 Mt. The final rise, carried out during the dry seasons of 2023 and 2024 to an elevation of 2005 mAMSL providing a final volume for tailings in TMF1 of 20.56 Mm^3 or 22.80 Mt.
- The capital cost estimate for the TMF1 starter wall is approximately USD 19.7M.
- The total capital cost required for the construction of the remaining TMF1A and TMF1 is approximately USD 76.6M.

It must be stressed that a starter wall for a start-up basin capacity/life of 22.5 months as opposed to the recommended minimum of 2 years increases the risk of potentially not being able to deposit tailings at the end of the 22.5 months if the wall raise is not in place at that time. Typical factors that need to be considered in assessing this increased risk include:

- The vast majority of each wall raise can only be undertaken the 5 driest months of the year, although during the earlier part of the wet season some daytime working is generally possible. Nevertheless, two dry seasons will be available for construction during the 22.5 month period for the first wall raise.
- Sufficient plant and equipment must be provided based on the application of an efficiency factor due to maintenance, break downs, rain related delays, and where the actual borrow areas will be located along with their haul road development etc.
- The footprint or size of each wall raise is limited even though the volume placed is large, i.e. there is limited “working” area or space to move plant (place, spread, compact, test etc.) and this needs to be evaluated in detail. Generally, a 30 m wide pad width is necessary to maximise efficiency unless specialised plant is used.

19 MARKET STUDIES AND CONTRACTS

19.1 Gold Price Trend

The gold price has shown a downward trend since the Twangiza Mine started operations as can be seen in Figure 19-1, which illustrates the gold price's performance over a fourteen year period.

SRK subscribes to a market consensus forecast which is updated quarterly; the 5 year (long term) gold price forecast has changed from 1,200 USD/oz to 1,150 USD/oz in the six months preceding the date of this report.

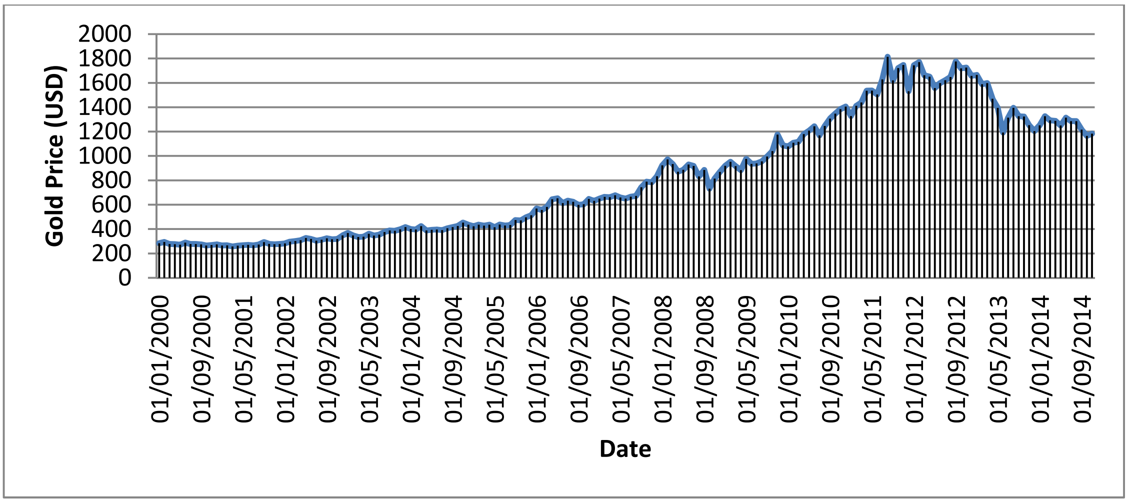


Figure 19-1: Gold Price In USD Trend 2000 – November 2014

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental and Social Impact Assessment status

SRK (SA) conducted an Environmental and Social Impact Assessment as part of the 2009 Feasibility Study. Neither SRK (SA) nor SRK (UK) have done any further Environmental Investigations or additional Social Impact Assessments since the report was filed.

The ESIA was written to meet potential lenders' Equator Principles standards, which cover DRC requirements. Since some specific elements were to be formulated slightly differently in order to meet DRC requirements for reporting, an update ESIA was compiled in French and submitted to the DRC Ministry of Mines.

Twangiza Mining has in the interim undertaken a relocation program as part of the mine construction and has also considered an alternative site for the Tailings Storage Facility.

In November of 2009, Twangiza Mining appointed Metago Environmental Engineers (Pty) Ltd (now part of the SLR Group), an independent environmental consultancy based in South Africa, to compile an interim ESIA report focussing specifically on the 1.3 Mt per annum "Phase I" of the Twangiza Gold Project based on the information and specialist studies available at that time,

Metago identified scope of work in their March 2010 interim ESIA report which were completed by Twangiza Mining, enabling SLR Consulting Ltd to prepare an updated ESIA report for Phase I of the Twangiza Gold Project which put forward a conceptual environmental and social management plan. The updated ESIA report drew significantly upon the content of the following documents:

- the interim environmental impact assessment (EIA) and associated specialist studies and stakeholder involvement process conducted by SRK Consulting SA (Pty) Ltd (SRK) for the full extent of the 5.0 million tonne per annum operation (SRK Consulting 2009);
- additional specialist investigations as amended in 2010 to suit the Phase I scope, and
- the social impact assessment and resettlement action plan reports compiled by Hilde Van Vlaenderen for the Twangiza project (H. Van Vlaenderen 2010a and 2010b, respectively).

Table 20-1: Summary of Key Legislation and Relevant Compliance

LEGISLATION	COMPLIANCE
EXPLOITATION (MINING) PERMIT	
In terms of Title V of Decree no 038/2003, the company must apply for an Exploitation (mining) Permit. In order to apply for such a permit the company must be the title holder of a valid Exploration Permit(s)	Twangiza Mining holds exploitation permits
ENVIRONMENTAL IMPACT STUDY (EIS) AND ENVIRONMENTAL MANAGEMENT PLAN (EMP)	
The environmental obligations are set out in Title XVIII of decree no 038/2003. With the exception of temporary quarrying, any mining operation requires an approved Environmental Impact Study (EIS*) and an Environmental Management Plan for the Project (EMPP*). Schedule IX (Contents of EIS and EMPP) sets out the contents of the EIS and the EMPP and provides detail regarding specific management measures and standards that are required. * EIS = ESIA; EMPP = ESMP, ESIA and ESMP being IFC/WBG/international terminology	An Environmental and Social Impact Assessment (ESIA)* and Environmental and Social Management Plan (ESMP*) have been completed and an updated French version submitted to DPEM (Ministry of Mines) in 2013. The following management plans have been prepared as part of the Environmental and Social Management plan (ESMP – equivalent to EMPP): Resettlement Action Plan Stakeholder Engagement Plan Community Development Plan Conceptual Rehabilitation and Closure Plan

20.2 Key risks to the project

20.2.1 Social conflict arising from resettlement

Risks associated with social conflict due to involuntary resettlement have been managed to date through implementation of a resettlement action plan (RAP) based on data from the social survey and formal and effective consultation with the community. Based on the positive results of the implemented program, Twangiza intends to continue carrying out similar programs to address required resettlement based on the footprint of the operations. These required activities will continue to be carried out through programs that rely on ongoing community forums and constant involvement of local leadership and state officials. In addition, employment opportunities and sustainable development efforts by Twangiza are carried out in a manner to target and prioritize affected communities where possible.

20.2.2 Twangiza North Resettlement

A survey of households carried out in early 2015 has indicated that the area identified to be the affected area from the development of the North Pit has approximately 763 households. This reflects an influx to prior evaluations as a result of the operations of Twangiza and the opportunity for economic benefit.

A significant proportion of the 1,276 artisanal miners who ceased mining on the Twangiza footprint in mid-2010, are engaged in artisanal mining activities at the Twangiza North Pit, and around the mine periphery on the alluvial deposits in the river systems. These original miners have been joined by an influx of transient artisanal miners and current estimates are that there are between 2,000 and 2,500 artisanal miners who will require to be relocated from the North Pit by Quarter 2 of 2016.

Twangiza Mining has identified some minor deposits on its concessions which could be made available to artisanal miners and has established with the national and provincial mining authorities that it may be possible to take legal steps to make the deposits concerned accessible to registered artisanal mining cooperatives.

Based on the mine profile, there is a need to resettle approximately 763 households on a phased programme between Quarter 4 of 2015 and Quarter 1 of 2017. A new Luhwindja Community Forum, similar to previous resettlement activities, has been established to facilitate the ongoing consultative engagements with the community. The community social surveys have been updated for 142 households closest to Twangiza Main Pit, and it is planned to update the social data for some 621 households in the areas close to Twangiza North Pit by the end of Quarter 3 2015.

The households will not be resettled at the Cinjira Resettlement Village, but at alternative land blocks identified within Luhwindja. The options have been narrowed down to 11 sites. The costs associated with acquiring and developing the land are not yet established.

20.2.3 New TMF Resettlement

An additional resettlement project will be required at Twangiza in order to construct a new tailings dam, because of the need to replace the current TMF1A. Resettlement activities for the new TMF1 will be carried out in a manner consistent with past and present resettlements including a focus on community forums and working closely with local and regional leadership. The new TMF1 site community has not yet been surveyed.

20.2.4 General Resettlement Considerations

The total resettlement requirement for the project, including the new TMF, is estimated by Twangiza Mining to cover 830 households with appropriate allowance made in capital cost estimates. The cost estimate may increase to overcome additional challenges which may arise when executing the resettlement process with the various stakeholders involved, including those who have not been a party to historical resettlement activities.

20.3 Local political instability

There is a longstanding local political conflict in the Twangiza area relating to the traditional chieftainship succession, which has affected Twangiza activities from inception in the exploration phase and has periodically continued to negatively affect community stability.

20.4 Biophysical aspects

There are potential liabilities associated with the residual impacts of the extensive artisanal mining that has taken place in the project area. Metal and metalloid contamination (arsenic, chromium, lead, iron, manganese) is severe at many of the sampling points downstream of artisanal mining areas, although no mercury has been detected. Mercury may be adsorbed onto riverbed sediments, but these have not been tested.

20.5 Environmental monitoring

Routine monitoring is being conducted in the general Twangiza catchment area to assess pollution from both artisanal and commercial mining activities, in addition to any other source of pollution that may arise in the region. Surface and ground water quality are being assessed on a monthly basis for the full spectrum of analysis including the ICP-MS Scan and a few parameters like pH, conductivity, free CN are being assessed on a daily basis at some key points.

See Figure 20-1 for the Twangiza Water Monitoring Point.

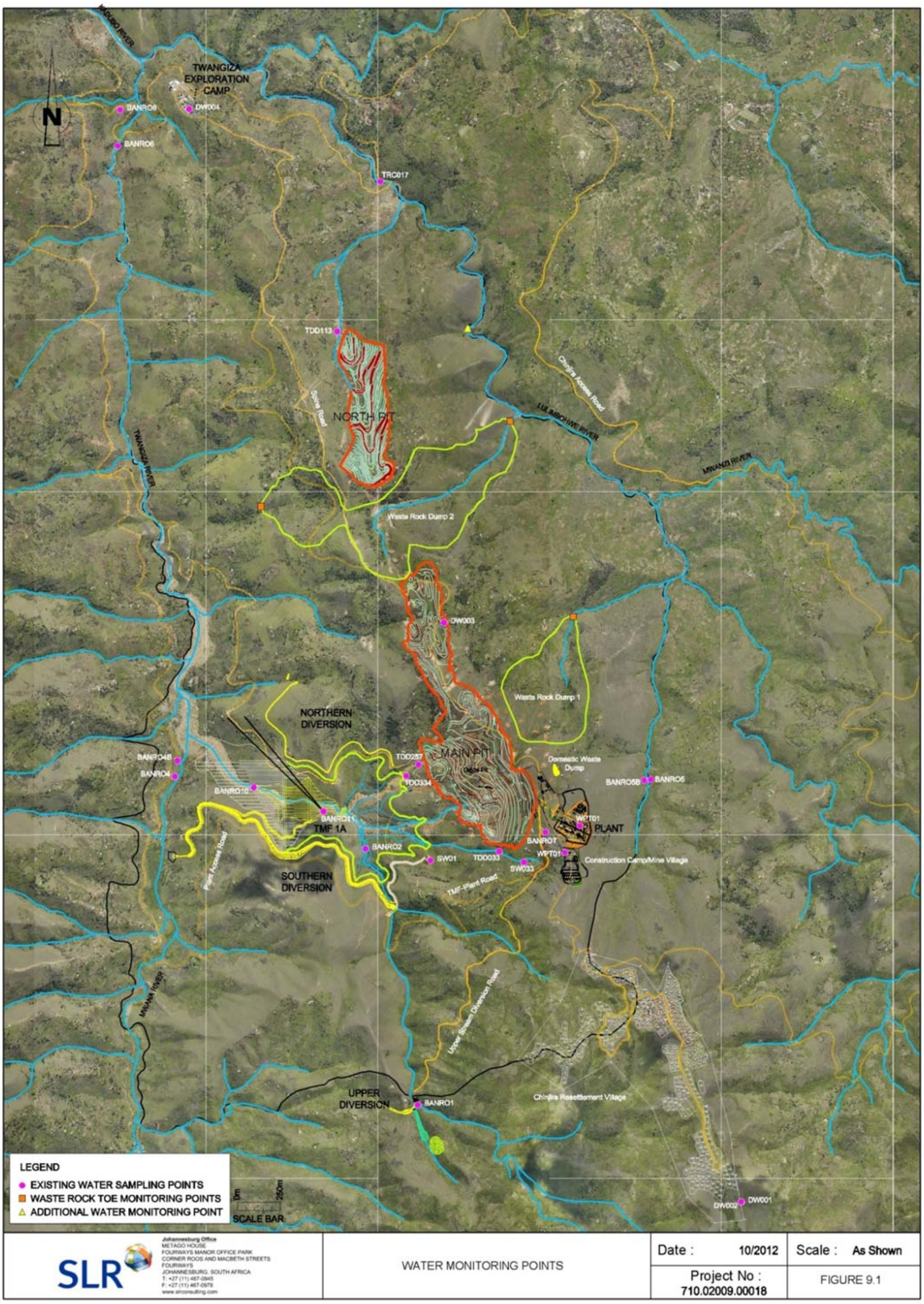


Figure 20-1: Plan Showing Twangiza Water Monitoring Positions

21 CAPITAL AND OPERATING COST

21.1 Operating Cost Estimate

21.1.1 Summary

Costs have been estimated based on a “zero-based” (first principles) cost analysis following review of 2014 historical costs, with appropriate allowance for the future operating environment, productivity and cost saving initiatives.

A number of changes are reflected in the forecast costs, notably there are savings expected from:

- lower oil price
- transition to hydro power source
- restructured staffing,
- renegotiated contracts, and
- more direct supply chain.

The unit savings per processed tonne are enhanced by the higher plant production rate, however the unit savings per ounce produced is tempered by the lower gold recoveries in later years.

A summary of the unit operating costs actually incurred in 2014 and forecast operating costs in the mine plan are given in Table 21-1.

21.1.2 Diesel and Power

During 2014, there was a significant price decline in the global oil market which subsequently reduced the cost of diesel at Twangiza based on the nature of supply contracts. During 2014 the average diesel price delivered to site was USD 1.52/litre which has been renegotiated by Twangiza Mining to USD 0.84/litre for 2015. In addition, mining diesel costs are expected to benefit from new mining fleet additions with improved fuel efficiency.

From 2018 onwards, power is planned to be supplied to Twangiza from a planned hydroelectric plant at Ulindi 1 River. This is projected to reduce the total power cost from USD 7.35/ tonne processed to USD 5.15/tonne processed; or 30% lower.

A preliminary engineering report and Solicitation of Interest (SOI) document has been finalised and will be distributed to potential financing and developing partners soon. The power supply agreement will be a usage billing agreement between Twangiza Mining and the successful power supply company; the initial capital requirement is planned to be repaid over 10 years which forms part of the power cost of USD 5.15/tonne processed.

21.1.3 Sundry and Expenses

During 2014, Twangiza initiated a number of in-house modification and improvement projects associated with ongoing process optimization as well as the plant expansion which started in 2013. As a result of these activities, additional resources were required by Twangiza Mining which resulted in increased employee related costs, such as legal and compliance, to be incurred during the period.

During 2014, the Namoya Mining SA gold project, another Banro subsidiary, progressed forward with gold production levels which allowed for the renegotiation of bullion transportation agreements to allow for certain costs to be shared due to the relative proximity of the two operations.

In addition to the above, certain non-recurring costs were incurred by Twangiza Mining in 2014 including fees associated with building certifications at the Cinjira resettlement and consumable storage costs that resulted from a previously employed procurement methodology.

21.1.4 Payroll

During the early stages of operations, as is typical with new operations in regions without a history of industrial mining, the Twangiza Mining management structure included a large number of expatriates. During 2014, an organizational delayering process was executed which ultimately led to a reduction of expatriate personnel along with the delayering of the organogram to create a flatter structure which has been operational in 2015. In addition to this delayering process, a review of all contractor labour was undertaken to eliminate non-essential staff and costs that existed from the capital projects and a reduction in contract labour to reflect the standards of an operating mine.

21.1.5 Processing Materials and Contracts

Previously, most chemicals were purchased from intermediate handlers who had mark-ups on product costs and fees. New arrangements are now in place to purchase chemicals directly from the manufacturers. This process was assisted by the increased scale of operations. Furthermore, management decided to contract a transporter for in-country haulage which offers cheaper transport costs and no management and personnel fees. In addition to the review on the in-country haulage, all the external haulage which was originally coming through Mombasa port was rerouted through Dar es Salaam port and this reduced the haulage cost significantly.

Even though this process started in late 2013, there were stocks of consumables which were bought at the higher pricing. Those stocks were used during the 2014 production year and from 2015 onwards the benefits of the newly negotiated contracts are being seen.

21.1.6 Mining Materials and Contracts

Due to limited funds, the stock of spares to maintain the mining fleet was limited which necessitated the rental of certain equipment from contractors. In addition to the higher cost of contractor equipment, the cost of spares to fix the owner operated fleet increased due to working capital limitations.

Furthermore, Twangiza had a haulage contract involving six Bell B40 ADT trucks, the terms of which have been renegotiated with better pricing.

Table 21-1: Summary of Unit Operating Costs

Item	Actual H2 2014 (Annualised)			Forecast in Mine Plan		
	USD million / annum	USD / t processed	USD / oz poured	USD million / annum	USD / t processed	USD / oz poured
Mining						
Payroll	5.40	3.53	48	3.99	2.50	45
Materials	3.71	2.43	33	2.60	1.62	29
Contractors & Consultants	3.13	2.05	28	0.90	0.56	10
Diesel & Power	3.42	2.23	30	1.89	1.18	21
Sundry Expenses	0.39	0.25	3	0.15	0.09	2
Mining Total	16.06	10.49	142	9.53	5.95	107
Processing						
Payroll	6.15	4.02	54	5.08	3.17	57
Materials	9.11	5.95	80	11.76	7.35	132
Contractors & Consultants	1.96	1.28	17	1.28	0.80	14
Diesel & Power	15.16	9.91	134	11.80	7.37	133
Sundry Expenses	3.61	2.36	32	2.17	1.35	24
Processing Total	36.00	23.52	318	32.09	20.04	360
G&A						
Payroll	5.39	3.52	48	5.68	3.55	64
Materials	2.07	1.35	18	0.73	0.46	8
Contractors & Consultants	6.78	4.43	60	6.40	4.00	72
Diesel & Power	0.74	0.49	7	0.69	0.43	8
Sundry Expenses	6.63	4.33	59	7.12	4.45	80
G&A Total	21.61	14.12	191	20.62	12.89	232
Total Operating Costs	73.67	48.12	651	62.24	38.88	699

21.1.7 Basis of Estimates

Mining general

The mining operating costs were based on an owner operated system as opposed a contractor operated system. Unit costs were obtained from an in-house zero based budget prepared for the production year 2015. The estimation of mining operating costs by period was based on projected equipment performance and unit operating costs.

Plant operating labour

Derived from first principles using assumptions and costs estimated in the in-house zero based budgets prepared for the production year 2015.

Plant maintenance labour

This was derived from first principles using assumptions and costs estimated in the in-house zero based budgets prepared for the production year 2015.

Plant consumable materials

This was derived through a combination of projected reagent consumption and the in-house zero based budgets prepared for the production year 2015.

Power

The average continuous monthly power consumption for process plant was determined by taking into account the power rating for each piece of major equipment and the projected running times as outlined in the design criteria. The power costs were then determined by taking into account the operating costs and the calculated monthly electrical consumption in kWh.

Plant maintenance costs and supplies

This cost has been obtained from the in-house zero based budgets prepared for the production year 2015.

General and Administration costs (including assay costs)

This caters for administration labour; which has been derived from first principles and a range of other costs associated with administration such as camp costs, office supplies, telephones, computers, safety supplies, clinic supplies, vehicles, insurance and head office expenses as captured in the in-house zero based budget prepared for the production year 2015.

Administration tax

An administration tax of 5% to cover the cost of importation of plant, machinery and consumables has been included in the projected capital and operating costs.

21.2 Capital Cost Estimate

21.3 Introduction

As part of the economic assessment, capital costs were estimated for the upgrade of the Twangiza process plant to facilitate processing of a blend of oxides and non-oxide material types to meet the projected plant throughput rate of 1.7 Mtpa. Capital costs were also estimated to purchase additional/replacement mining fleet to meet the material movement requirements over the Life of Mine. Due to the increase in the ore to be processed due to the addition of the non-oxides, the tailings capacity needs to be increased to accommodate the increase in projected tailings that will be generated over the life of mine. Additional capital provisions have been made to cover the cost of constructing an alternative TMF, which is expected to be ready by mid-2017. This approach is designed to significantly reduce the cost of the alternative option to raise the wall of the existing tailings dam (TMF1A).

Table 21-2: Twangiza Capital Cost Summary

Item	Cost (USD million)	Comment on Estimate Accuracy
Capitalised Expenditure		
Mining – Sustaining Capital	26	(+/-5% accuracy) Based on quotations received and used in 2015 zero based budget
Processing – Sustaining Capital	6	(+/-5% accuracy) Based on quotations received and used in 2015 zero based budget
TMF Construction Capital	46	(+/-10% accuracy) Based on Design BOQs and costs inputs from 2015 zero based budget
Tailings – Sustaining Capital	30	(+/-10% accuracy) Based on Design BOQs and costs inputs from 2015 zero based budget
General & Administration - Sustaining Capital	21	(+/-5% accuracy) Based on quotations and reviewed compensation rates and used in 2015 zero based budget
Banro Foundation	1	Statutory rate of US$1.00/ounce
Total – Capitalised Expenditure	**130**	

21.3.1 Basis of Estimates

The accuracy of the capital estimate is considered by Twangiza Mining to be within ±10%. Most items are based on historic data which have been generated since the beginning of operations, some are based on quotations received from vendors. The TMF is based on Twangiza Mining's estimate which is factored from previous dedicated technical studies.

Mining – Sustaining Capital

The additional initial and replacement mining fleet requirements have been estimated based a multi-pit operation approach sufficient to achieve peak total material movements requirements sustain a suitable ore blend to feed the plant at a 1.7 Mtpa throughput target. New mining fleet capital amounts to USD19 million, fleet replacement capital is estimated at USD5 million with a further USD1.5 million allocated for grade control, heavy duty workshop construction and tooling.

Process – Sustaining Capital

Due to the increased hardness of the plant feed, measures have been implemented to replace all liners and wear plates to economically and safely handle the projected feed rate. A full time primary crushing unit has been designed to be installed to crush and screen RoM to the required -10mm before it is fed through a mineral sizer and auxiliary feed point.

TMF Construction Capital

Total capital of USD 76 million will be required to construct TMF facilities, split in two portions. The first portion of USD 46 million will be required for TMF construction capital and will be used to procure material, geotechnical studies, land compensation and relocation and starter wall construction. The second portion totalling USD 30.5 million will be expended in Years 2015 – 2025 to cover the cost of continuous rising of the wall.

General and Administration – Sustaining Capital

Provision has been made on the capital estimates for North Pit artisanal miners and Kantambwe compensation and relocation.

21.4 Escalation

No escalation has been allowed for in the estimate. Rates are based on historical cost from the Twangiza operations and the zero-based budget for the 2015 production year.

21.5 Taxes

In early 1997, Banro, SOMINKI and the government of the DRC ratified a new 30 year mining convention that provided for SOMINKI to transfer its gold assets to a newly created company, Société Aurifère du Kivu et du Maniema, SARL (“SAKIMA”). In addition to this asset transfer, the new mining convention included a ten year tax moratorium from the start of commercial production, the ability to export all gold production, the ability to operate in US currency, the elimination of import duties and title confirmation for all of the concessions.

After the 10 year tax holiday, as described above, the project is subject to corporate taxation at the rate of 30% on profit.

Royalty payments, as agreed with the Government are paid at the rate of 1% of gross revenue.

22 ECONOMIC ANALYSIS

22.1 Economic Analysis

The base case was developed using a long-term gold price of USD1,200 per ounce and a 5% discount rate. The financial model also reflects the favourable fiscal aspects of the mining convention governing the Twangiza project, which includes 100% equity interest and a 10 year tax holiday from the start of production in September 2012. An administrative tax of 5% for the importation of plant, machinery and consumables has been included in the projected capital and operating costs. Calculated sensitivities show the significant upside leverage to gold prices and the robust nature of the projected economics to operating assumptions.

Table 22-1: Financial Analysis Summary

ITEM	UNIT	AMOUNT
LIFE OF MINE GOLD PRODUCTION	koz	1,246
PRODUCTION PERIOD	years	14
ANNUAL GOLD PRODUCTION FOR FIRST 5 YEARS	koz	109
TOTAL CAPITAL COSTS	USD/oz	104
ALL IN COSTS	USD/oz	888
POST-TAX NET PRESENT VALUE	**USD million**	**285**
NET CASHFLOW AFTER TAX AND CAPEX	**USD million**	**395**

22.2 Model Assumptions

The assumptions used in the financial analysis are given in the table below.

Table 22-2: Financial Model Assumptions

Item	Unit	Value
Revenue		
Plant Throughput	'000tpa	1,700
Gold Price (Lower Limit)	USD/oz	1,000
Gold Price (Base Case)	USD/oz	1,200
Gold Price (Upper Limit)	USD/oz	1,600
Discount Rate	%	5%
Fuel Price		
Diesel	USD/litre	0.84
Fiscal		
Tax Free Holiday	years	10
Tax Rate (Year 1 – 10)	%	
Tax Year (Beyond Year 10)	%	
Government Royalty	%	1.00
Depreciation	%	
Conversion Factors		
Kilograms To Ounces	kg/ troy ounce	32.1505
Diesel Fuel Density	t/m^3	0.85
Exchange Rate	ZAR : USD	7.496
Other		
Refining Charges, Dore Transport and Insurance	USD/oz	15.20
Percent Of Capital Expenditure (Year 2015)	%	35%

22.3 Sensitivity Analysis

A sensitivity analysis was performed on the after tax profits by varying the gold price between USD1,000 and USD1,600 per ounce. The results are summarized below.

Table 22-3: Sensitivity Analysis on Gold Price

GOLD PRICE (USD/oz)	NET PRESENT VALUE (USD million)						
	1600	1500	1400	1300	1200	1100	1000
NPV 0.0%	847	734	621	508	395	282	169
NPV 5.0%	612	530	449	367	285	203	122
NPV 8.0%	516	447	378	310	241	172	103
NPV 9.5%	476	413	350	286	223	159	96
NPV 10.0%	464	402	341	279	217	156	94
NPV 12.5%	411	356	302	247	193	139	84
NPV 15.0%	366	318	270	222	173	125	77

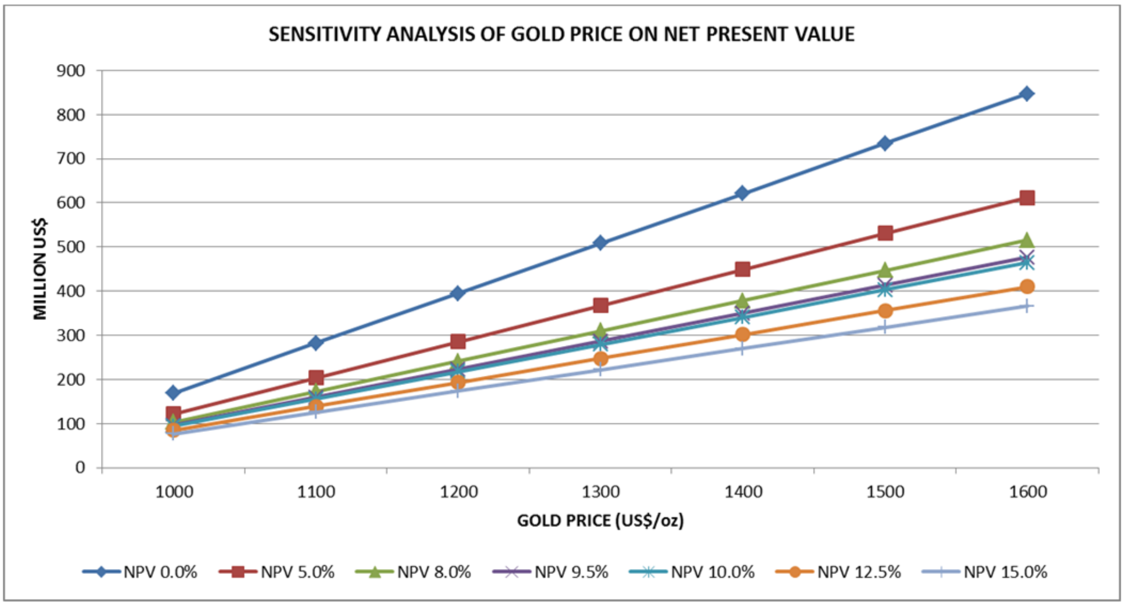


Figure 22-1: Sensitivity Analysis of Gold Price Versus Net Present Value

A sensitivity analysis of NPV (5% discount rate) to total operating and capital costs +30% has also been prepared as provided in Figure 22-2. Included in the sensitivity analysis are comparative estimates for project NPV calculated using historical costs based on full calendar year 2014 ($62M) and H2 2014 ($177M). It is evident that the changes implemented during 2014 generally indicate an improving trend for unit cost reduction and productivity improvement.

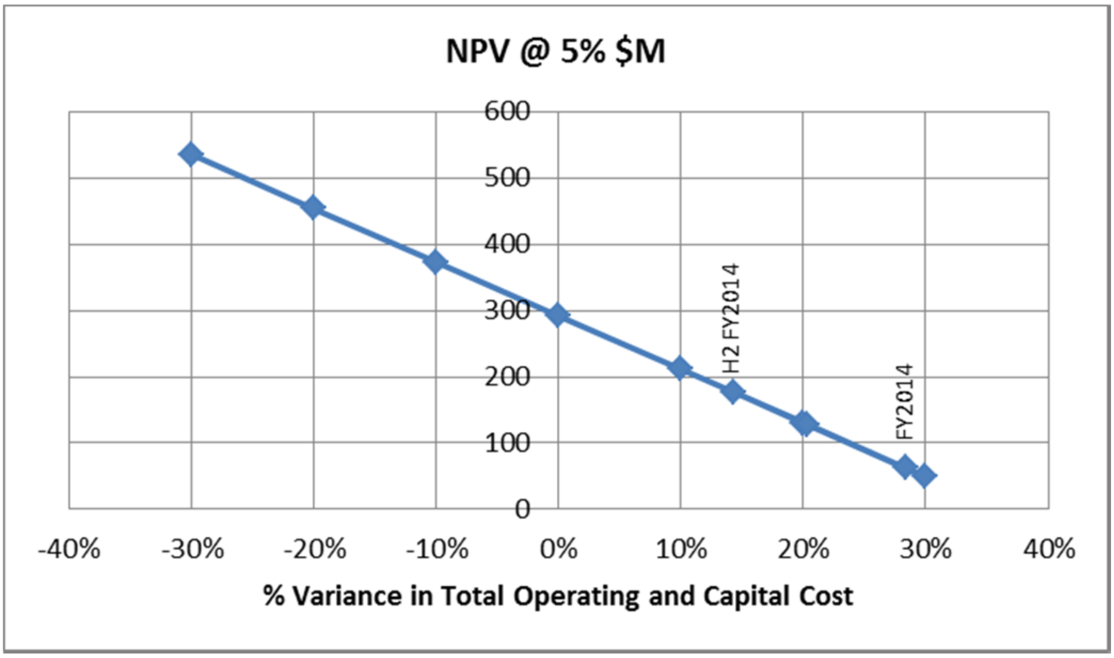


Figure 22-2: Sensitivity Analysis of Total Operating and Capital Cost Versus Net Present Value

22.3.1 Project life cash-flow

Table 22-4: Cash Flow Summary

Revenues		Year_2015	Year_2016	Year_2017	Year_2018	Year_2019	Year_2020	Year_2021	Year_2022	Year_2023	Year_2024	Year_2025	Year_2026	Year_2027	Year_2028	Total
Gold Produced	oz	115,897	105,752	117,121	108,591	96,305	111,790	102,160	94,396	84,289	65,832	73,675	78,647	79,463	12,149	1,246,311
Gold Price	USD/oz	1,250	1,200	1,200	1,200	1,200	1,200	1,200	1,200	1,200	1,200	1,200	1,200	1,200	1,200	1,202
Revenue	**USD 000**	**144,871**	**126,903**	**140,545**	**130,309**	**115,566**	**134,148**	**122,592**	**113,275**	**101,147**	**78,999**	**88,409**	**94,376**	**95,355**	**14,578**	**1,501,368**
Operating Costs																
Mining cost	USD 000	9,436	21,497	37,793	36,215	40,884	36,311	26,392	28,345	13,319	15,620	18,354	3,945	0	0	288,108
Grade Control cost	USD 000	1,091	2,352	4,341	3,871	4,430	4,171	3,032	3,256	1,530	1,794	2,108	453	0	0	32,429
Process Plant Costs - Carbon In Leach	USD 000	12,364	13,498	13,498	13,498	13,498	13,498	13,498	13,498	13,498	13,498	13,498	13,498	13,498	3,196	177,569
Assay	USD 000	1,142	1,246	1,246	1,246	1,246	1,246	1,246	1,246	1,246	1,246	1,246	1,246	1,246	295	16,387
Power	USD 000	11,780	12,499	12,499	8,749	8,749	8,749	8,749	8,749	8,749	8,749	8,749	8,749	8,749	2,072	126,366
Engineering (Maintenance) Costs	USD 000	5,128	5,601	5,601	5,601	5,601	5,601	5,601	5,601	5,601	5,601	5,601	5,601	5,601	1,326	73,676
Rehab Provision (Pits & Dumps, TMF, Ponds, Demobilization)	USD 000	64	1,086	1,070	1,202	320	1,369	1,292	981	1,039	582	652	729	433	346	11,249
Infrastructure, Overheads and Sundries (G&A)	USD 000	19,949	12,478	12,478	12,478	12,478	12,478	12,478	12,478	12,478	12,478	12,478	12,478	12,478	2,955	172,665
Refinery and Shipment cost	USD 000	1,854	1,608	1,781	1,651	1,464	1,700	1,553	1,435	1,282	1,001	1,120	1,196	1,208	185	19,041
Royalty & Government Charges	USD 000	1,449	1,322	1,464	1,357	1,204	1,397	1,277	1,180	1,054	823	921	983	993	152	15,579
H/O Management Fee (Toronto)	USD 000	1,577	1,368	1,515	1,404	1,246	1,446	1,321	1,221	1,090	851	953	1,017	1,028	157	16,197
H/O Management Fee (Banro Congo Mining)	USD 000	2,629	2,279	2,525	2,341	2,076	2,410	2,202	2,035	1,817	1,419	1,588	1,695	1,713	262	26,994
TOTAL OPERATING COST	**USD 000**	**68,462**	**76,833**	**95,810**	**89,612**	**93,194**	**90,375**	**78,640**	**80,024**	**62,701**	**63,662**	**67,268**	**51,590**	**46,947**	**10,946**	**976,260**
Total Operating Cost per ounce	USD/oz	591	727	818	825	968	808	770	848	744	967	913	656	591	901	783
EBITDA	**USD 000**	**76,409**	**50,070**	**44,735**	**40,697**	**22,372**	**43,773**	**43,952**	**33,251**	**38,446**	**15,337**	**21,141**	**42,786**	**48,408**	**3,632**	**525,108**
Capital Expenditure																
Project Capital - TMF Construction	USD 000	17,950	5,000	0	8,000	0	0	5,000	5,000	5,000	0	0	0	0	0	45,950

Mining Capital (New & Replacement Fleet and Others)	USD 000	4,694	5,397	4,801	4,728	3,886	299	543	347	544	0	0	0	0	0	25,239
Process Capital (Plant Modification and Others)	USD 000	941	2,000	3,500	0	0	0	0	0	0	0	0	0	0	0	6,441
Grade Control Capital	USD 000	271	0	0	0	0	0	0	0	0	0	0	0	0	0	271
General & Admin Capital	USD 000	10,562	10,000	0	0	0	0	0	0	0	0	0	0	0	0	20,562
Sustaining Capital (Tailings Wall Lifts)	USD 000	2,884	3,064	3,064	3,064	3,064	3,064	3,064	3,064	3,064	3,064	0	0	0	0	30,467
Banro Foundation	USD 000	116	106	117	109	96	112	102	94	84	66	74	79	79	12	1,246
TOTAL CAPITAL TAXES AND LEVY	**USD 000**	**37,418**	**25,567**	**11,483**	**15,900**	**7,046**	**3,475**	**8,709**	**8,505**	**8,693**	**3,130**	**74**	**79**	**79**	**12**	**130,177**
Total Financing Cost per ounce	USD/oz	323	242	98	146	73	31	85	90	103	48	1	1	1	1	104
ALL-IN COST (Cash plus Capital)	**USD 000**	**105,880**	**102,400**	**107,292**	**105,512**	**100,240**	**93,849**	**87,350**	**88,529**	**71,393**	**66,792**	**67,342**	**51,669**	**47,026**	**10,958**	**1,106,437**
All-In Cost per ounce	USD/oz	914	968	916	972	1,041	840	855	938	847	1,015	914	657	592	902	888
Project Cashflow																
Net Cashflow	**USD 000**	**38,991**	**24,502**	**33,253**	**24,797**	**15,326**	**40,298**	**35,242**	**24,746**	**29,753**	**12,207**	**21,068**	**42,707**	**48,329**	**3,620**	**394,931**
NPV Sensitivities																
NPV 0.0%	$ m	**395**														
NPV 5.0%	$ m	**285**														
NPV 8.0%	$ m	**241**														
NPV 9.5%	$ m	**223**														
NPV 10.0%	$ m	**217**														
NPV 12.5%	$ m	**193**														
NPV 15.0%	$ m	**173**														

23 ADJACENT PROPERTIES

23.1 Adjacent Properties

There are no adjacent properties

24 OTHER RELEVANT DATA AND INFORMATION

No other information is considered necessary.

25 INTERPRETATION AND CONCLUSIONS

25.1 Geology

SRK (UK) considers the geology at Twangiza Main and North to be well understood. The folded geometry of the stratigraphy and sills has been modelled in detail allowing a confident interpretation of the lithological distinction of ore types. The weathering that has affected the deposit is also modelled in a series of continuous layers which differentiated ore types of differing metallurgical behaviour and densities in sufficient detail for the resource model.

25.2 Resource Model

The zone of mineralisation within the fold hinge is reasonably continuous and well sampled by adits and resource drilling. Data quality and quantity have been reviewed and are sufficient in SRK (UK)'s opinion to support the level of confidence in the Mineral Resource.

Density data from drilling samples has been factored to bring the block model tonnage estimate in line with actual tonnages recorded in historical production. The adjustment was supported by a recent small programme of in pit density check samples. This programme should be continued to consolidate new views on the ore density.

The model has been appropriately depleted to the end of December 2014 pit survey.

Continue to monitor dry density and moisture content of ore by sampling daily and analysing on a monthly basis.

25.3 Metallurgy and Processing

25.3.1 Historical Plant Performance

While the plant has been running for approximately three years it has not been operated at the design conditions due to issues relating to shortage of funds. Oxide ore handling has been problematical especially in the wet season; a roof has only recently been installed over the ROM crusher and re-handling area. A lack of spare parts resulted in:

- low availability of primary, secondary and tertiary crushing
- ball mills where new liners have not been available, and
- some CIL tanks were off-line due to non-availability of spares for the agitators.

The supply of reagents and other plant consumables has been limited resulting in:

- a lack of steel balls for the mills which has impacted throughput, grinding circuit product size and an excessive amount of mill oversize;
- cyanide addition has been limited which has resulted in poor leaching conditions; and
- the lack of supply of activated carbon has reduced the adsorption efficiency in the CIL circuit and increased the soluble gold losses in the tailings.

Overall these issues have resulted in lower than expected gold recovery, 82.5% gold recovery compared to 88 to 90% in the feasibility study.

25.3.2 Recent Plant Modifications

The plant has been modified to process 1.7 Mtpa ore. In the latter half of 2014 the crushing circuit, grinding circuit and the downstream cyanidation and adsorption circuit all operated at the increased rate, so from a simple capacity perspective, as opposed to circuit performance, they can handle the increased tonnages.

The installation of the covered RoM stockpile has improved ore handleability. The modification to the layout of the primary crusher has resulted in more consistent feeding of the crusher and has improved access for maintenance. In addition a revised replacement strategy for the sizer teeth has reduced equipment downtime and teeth replacement operating costs.

In addition to these modifications, the expected increase in the proportion of more competent non-oxide ore in the feed blend in the next few years should also improve ore handleability issues. SRK (UK) is of the opinion that these changes will increase the circuit utilisation with resultant increases in plant throughput going forward.

SRK (UK) is of the opinion that once the equipment is operating according to design and with sufficient reagents and consumables the overall gold recovery from oxide ores will increase from the historically achieved figures.

25.4 Mine Plan

Pit optimisations have been conducted based upon resource models revised for historical reconciliation to December 2014 and as such, are understood to reflect the current understanding of the resource model and its exploitation performance on mining.

Optimisations have also been based largely on costs generated from first principles, a 'zero-based' approach, following a review of historical costs. The cost structure is estimated to be considerably lower than it has been historically and is dependent on the achievement of key productivity and performance targets and operating strategies (hydroelectric supply, process throughput and recovery of harder ores, drill and blast performance, mining fleet operating strategies).

As a consequence of the lower forecast costs, the cut-off grades are lower and optimisation and design inventories are higher. This is mitigated to a degree by the resource grade profile, low stripping ratios and relatively flat optimisation cashflow profiles.

25.5 Operating & Capital Costs

Costs have largely been generated from first principles, a 'zero-based' approach, following a review of historical costs. There are a number of key reasons for this approach:

- Diesel fuel price five year forecasts are significantly lower than recent 2014 performance at USD1.52/l, representing a reduction in a significant input cost;
- The operation proposes to source future power from a hydroelectric supply at an estimated 30% lower price than currently obtained from diesel sources;

- The mining fleet basis has been upgraded to meet the future increase in stripping ratio from less than 1.0 historically to an average 3.2 in the future on a waste tonnage to ore tonnage basis, an escalation to an average 12 Mtpa over the five year period. As such this represents a higher productivity, economy of scale and potentially lower cost structure going forward;

- Procurement and some contracts for supply of materials and services have been rationalised to provide a potentially lower cost structure going forward;

- Increased working capital will remove restrictions, additional costs and operating inefficiencies;

- Historical reconciliation of both ore and waste movements indicates that while ore reconciliation has been negative, strong positive reconciliation of waste movement has occurred. While the strongly positive waste movement reconciliation likely includes rehandle and borrow pit movements, a downwards correction of truck factors for historical movements of the order of 20-30% will yield a corresponding escalation in historical unit waste mining costs; and,

- An increased proportion of transition and fresh material types will incur additional mining costs (drill and blast, excavation costs, secondary breakage) and process costs (throughput, wear and recovery).

An analysis of historical costs for 2014 indicates that the proposed zero-based cost structure approach represents a significant cost reduction of up to 30% on certain unit cost estimates going forward. This is a risk to the project if the proposed future operating strategy, cost reductions, operating efficiencies and mining fleet operating efficiencies are not achieved. This will require monitoring of unit costs for mining, processing and G&A. It will also require further technical investigation and management to ensure that:

- Additional tailings capacity is achieved at projected capital construction and operating costs. Approvals and contracts will need to be negotiated and agreed;

- Hydroelectric power supply is sourced at projected costs in accordance with the projected timeline. Approvals and contracts will need to be established in a timely fashion;

- Process throughput rates and recoveries are achieved and/or plant modifications implemented in order to meet targets; and,

- Drill and blast production rates and costs are achieved based on proposed patterns. This will depend on operating performance. The supply and cost of explosives and accessories will need to be managed in order to ensure that the cost and timely supply is achieved on a regular basis.

During 2014, the nature and scale of the Twangiza operation changed significantly with the completion and commissioning of the process plant expansion project. Following commissioning in mid-2014, operating levels increased to reach the current 1.7Mtpa design capacity by the fourth quarter. Recognizing management's priority during H2 2014 was to maintain targeted LOM throughput, many cost savings in the LOM base case included in this report had yet to be implemented and realized as at December 31, 2014; SRK (UK) requested a sensitivity on the financial model to understand the consequence of using various historical operating cost profiles on a going forward basis. While this cost profile is not assumed to be indicative of future operations, it quantifies the implications if the forecasted cost profile is not realized. This sensitivity indicates that the project would return a considerably lower NPV over the LOM when considering solely the historical costs. The achievements made by the Twangiza Mine while advancing from a 1.3Mtpa throughput to the 1.7Mtpa LOM throughput lends support to the operation's ability to reduce future costs and improve productivities, but further delivery of anticipated cost savings must be realized to achieve the forecast. This highlights the importance of achieving the cost, productivity and performance programmes and planning timelines to achieve the base case NPV in this report. Operating costs and mine reconciliation will need to be monitored stringently to ensure that unit cost performance targets are being achieved.

Capital costs for key initiatives in respect of the tailings dam and hydroelectric supply will need to be confirmed by firm quotation and tendering.

25.6 Infrastructure

There is no engineered construction schedule for TMF1 at the moment, so there is a risk of interruption to the production schedule if TMF1 construction takes longer than currently expected.

25.7 Environmental

Whilst Twangiza Mining has a good track record of relocating families and artisanal miners, significant challenges remain before mining in Twangiza North and TMF1 construction can begin given that there will need to be dealings with a different community group from before.

The costs associated with the relocation are significant, to some extent these costs can be based on similar historical activity and previously established community relationships; but there are also new challenges which may require greater effort. The project can afford the estimated cost; however there is clearly a need to properly plan and execute the relocation exercise to minimise any delay to the project or additional costs.

25.8 Financials

A discount rate of 5% has been applied for calculation of project NPV over the LOM. SRK (UK) regards this rate as slightly low given the operating environment and country risk rating. Generally the discount rate should incorporate allowance for the risk-free interest rate, operational and country risk premiums. A discount rate of 8-10% is regarded as more appropriate.

26 RECOMMENDATIONS

26.1 Geology

Further work will eventually be required to continue exploration at previously identified targets on the Twangiza concessions, should the need arise to find more oxide ore at any time.

26.2 Resource model

26.2.1 Density

The resource model has been modified to reflect new ideas concerning ore density which have been mainly based on historical tonnage reconciliation and also on a few in-pit check density samples. SRK (UK) recommends continuing with in pit density sampling using the same method, seeking to build a geo-referenced database of samples on each flitch taken on a square grid spacing of approximately 20m.

The team on site needs to revise and unify its understanding of wet density, moisture content and dry density as applied to ore material. Further, the distinction between density and specific gravity values used by the plant for slurry calculations also needs to be clearly understood.

26.2.2 Reconciliation

It will be important to continue monitoring production grades, gold recoveries and densities on a monthly basis going forward. Periodically, (at least monthly or quarterly), a reconciliation exercise should be completed to assess whether further modifications need to be made to the resource block-model.

26.2.3 Grade Control Modelling

The creation of wireframes based on grade control sampling and mapping is an area that could be improved in terms of geological accuracy, 3D continuity and speed with which wireframes can be created. The grade control block models should incorporate newly captured in-pit dry density data. New and improved ways of interfacing with mine survey data should be considered. Better block model version control should assist with historical production analysis in the future.

26.3 Metallurgy and Processing

26.3.1 Measuring and Reporting Production

- Instigate a more accurate "measurement" of solids (scats) removed from the circuit returned to the stockpile.
- Determine the true density of the oxide and transition ore types for use in the mass flow loop calculation. The density of feed solids should be checked at least every 6 months as the ore type change over time.
- The feed grades to the plant are variable – include the modelled ore hardness in the mine-plan to try and achieve a more consistent feed blend to the plant.

- The new feed weightometers will improve the accuracy of the measurement of the feed tonnage to an acceptable level. Review of the plant operating statistics indicated that the measurement of the scrubber underflow slurry feed reporting to mill no.1 is potentially erroneous. The slurry flow rate and the slurry density are measured and computed using a standard mass flow loop to give a percentage solids and a solids tonnage. This calculation uses the true solids density. It is apparent that the density used by operations is too high and does not represent the true density of oxide and/or transition ore. This can potentially introduce a 10% error in to the mass flow calculation. Further true solids density measurements should be made to correct this situation. SRK (UK) recommends that this is checked at least every six months to reflect the changing ore blend with time.
- The grinding circuit can theoretically process 1.7 Mtpa of mixed ore to the required grind size provided sufficient grinding media is available. However, as the proportion of harder non-oxide ore increases it is likely that the circuit product size will coarsen and this may impact the leaching efficiency slightly. SRK (UK) recommends that the effect of the changes in ore blend on grind size, including an assessment of the effect on leaching performance, is closely monitored in the future.

26.3.2 Technical Improvements

- Review the reasons for the low percent solids in the leach feed. This reduces the residence time in the leach/CIP circuit and will be affecting overall gold extraction in the circuit. If the clay content is causing viscosity issues in the pulp necessitating lower percent solids – consider use of viscosity modifiers.
- Target lower gold in solution losses from CIP by adjustment of carbon inventory and carbon movement.
- The crushing circuit has historically been operated to produce a coarse product as feed to the mill. At times the top size has been 30mm which is too coarse for a ball mill and has a detrimental effect on mill performance. With adequate supply of crusher wear parts the crushing circuit should be operated to produce the design product, grinding circuit feed, of nominally -10 mm.
- The CIL circuit volumetric capacity has been increased by the addition of tanks in order to maintain the CIL residence time residence time at approximately 15 hours at the revised throughput of 1.7 Mtpa. The testwork performed indicates that 15 hours residence time is at the lower end of what is required especially for some of the transition and fresh ore types, therefore further increases in residence time should be researched and implemented.
- Operating data indicates that the percentage of solids in the leach feed is variable. A low percent solids in the grinding circuit product from the hydrocyclone overflow or the leach feed increases volumetric flow and reduces leaching residence time. This may be due to the feed characteristics, especially the clay content of the ore, but may also have been influenced by the less than optimal operation of the two mills. If clay is a problem this should either be addressed as part of the ore blending strategy to maintain a maximum level of clay in the feed, or through the addition of viscosity modifiers to the circuit. The latter would incur additional operating costs.

The gravity and intensive cyanidation circuit has not been operated continuously to date. Testwork indicated that GRG was present in oxide and transition ores and thus this circuit should be recommissioned and brought in to operation as soon as possible. If the circuit is operated without the gravity circuit there is a risk that coarser gold particles may not be fully leached within the leaching circuit and there may be losses to tailings. In addition it is noted that the adsorption circuit is designed to process ore post gravity and thus operation of the gravity circuit will reduce the gold load on the carbon adsorption system and prevent potential losses to tailings.

26.3.3 Fund Fully to Maximise Performance

- Avoid the spares issue with the crushing and grinding circuit as it results in very inefficient operation of the circuit (excessive sized solids in mill feed).
- Maintain the correct levels of steel balls, cyanide and carbon in the circuit.
- The grinding mills have been operating with reduced steel loads. Reduced steel load in the grinding mills affects mill power draw and thus circuit performance. With adequate supply of steel grinding media the correct design steel loadings in the two grinding mills should be maintained which in turn will allow the grinding circuit product to achieve the design 80% -75 microns.
- Twangiza Mining has operated the CIL circuit with starvation amounts of cyanide. Once sufficient cyanide is available the performance of the leach should be evaluated to determine the optimum cyanide concentration and thereby optimum leaching performance.

Carbon inventory in the CIP circuit affects soluble gold losses to tailings. Once adequate supplies of activated carbon are available Twangiza should maintain the design carbon concentrations in the circuit. Once stable operation is achieved the soluble gold losses should be assessed and optimised.

26.3.4 Additional Testwork

- Review historical metallurgical testwork and identify and perform additional testwork on potential feed solids that will be processed in the next few years. This should include:
    a. testing and studies to further evaluate the grinding requirements of future ore types, work index and power draw variations, grinding equipment size and throughput of the mills, grind size achievable, etc.
    b. the leaching requirements of future ore types to especially the residence times required for satisfactory leaching. This may result in additional leaching tanks being required.
    c. Further testwork is required to identify an effective treatment route for CMS material.
- Further comminution and leaching tests should be performed on samples of fresh (sulphide) ore in the next two years in order to assess the impact on circuit performance and identify modifications required for successful treatment of ore blends containing an increased proportion of harder fresh ore.

- Further testwork should be performed on samples of transition and fresh ores from all zones in order to be able to optimise the leaching circuit for future ore blends. This may necessitate the addition of new leaching tank capacity but this should be assessed from both a metallurgical and an economic perspective.

26.3.5 Ore Type Definition and Management

- Metallurgical testwork performed in the study phase of the project indicated that CMS ore is refractory and does not exhibit acceptable leaching characteristics. This ore should not be added to the blended feed and if mined should be stockpiled separately.
- The feed grade to the plant has been variable. Better communication between geology-mining-plant should be instigated to improve ore blending and feed consistency.

26.4 Mine Plan

SRK (UK) recommends further pit optimisation, design and sensitivity analysis to better assess Mineral Reserve and project cashflow sensitivity to operating cost escalation.

Review of current optimisation shells compared to the December 2013 pit designs used for the December 2014 Mineral Reserve shows that there are some differences that should be better accounted for by redesign of the pits based on the revised resource models and optimisation sensitivity analyses.

In particular, SRK (UK) recommends creating or revising pit designs for:

- Twangiza Main Intermediate Pit (Cut 2) to provide for ramp access in the design;
- Valley Fill pit design; and,
- Waste dump designs over the LOM.

The current schedule has different proportions of hard and soft ore types and in particular first / transitional / oxide ore type each year.

SRK (UK) is of the opinion that there may be a risk associated with this variability and that the risk can be mitigated to some extent by rescheduling to even out the ore types in the blend each month.

No designs for waste dumps have been presented to SRK (UK) for review, it is recommended that designs are completed to assist with planning going forward.

26.5 Operating & Capital Costs

Continue monitoring operating costs on a monthly basis and periodically reviewing these at least annually.

Some of the larger items in the capital costs going forward, such as the TMF, require additional design and engineering work to increase confidence in the cost estimate.

26.6 Infrastructure

TMF1 requires additional design work and there should be a dedicated engineering and construction schedule to increase confidence in successful delivery of expanded tailings storage capacity in time for the mine plan; without this there is a risk that production will be interrupted.

26.7 Environmental

SRK (UK) recommends that the on-going relocation process be given a high priority; otherwise parts of the mine plan will be at risk or delayed.

27 REFERENCES

In addition to references in the table below, a number of references to metallurgical testwork are given throughout Section 13.

SRK (UK) has accumulated numerous spreadsheet and monthly report documents from the mining operation; these are not individually detailed here.

AUTHOR	DATE	TITLE	SOURCE
SENET	13th September, 2007	Preliminary Assessment NI 43-101 Technical Report, Twangiza Gold Project, South Kivu Province, Democratic Republic of Congo	Banro
SENET	17TH July 2009	UPDATED FEASIBILITY STUDY NI 43-101 TECHNICAL REPORT, TWANGIZA GOLD PROJECT, South Kivu Province, Democratic Republic of Congo	www.sedar.com
SENET	24th March 2011	ECONOMIC ASSESSMENT NI 43-101 TECHNICAL REPORT, TWANGIZA PHASE 1 GOLD PROJECT, South Kivu Province, Democratic Republic of the Congo	www.sedar.com
Venmyn Deloitte (Ref:D1417R)	12th May 2014	Independent National Instrument 43-101 Technical Report on the Namoya Gold Project, Maniema Province, Democratic Republic of the Congo Prepared for Namoya Mining SARL (a subsidiary of Banro Corporation)	www.sedar.com

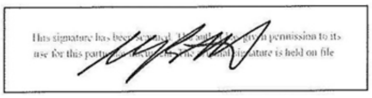
28 DATE AND SIGNATURE PAGE

The independent Qualified Persons (within the meaning of NI 43-101) for the purposes of this report are Martin Pittuck and David Pattinson. Further review and authoring has been undertaken by Allan Blair; these authors represent SRK Consulting (UK) Limited.

The SRK authors have undertaken an extensive review of Twangiza Mining's technical and economic data and have reviewed Twangiza Mining's contributions to this report.

Daniel Bansah, Head of Projects and Operations of Banro Corporation has provided a QP certificate; he has supervised the compilation of material provided by Twangiza Mining and has overall responsibility for certain sections in this report.

Signed, this 29th day of July, 2015.



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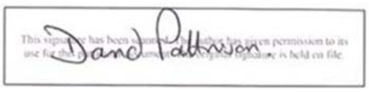

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Corporate Consultant (Minerals Processing & Metallurgy)
SRK Consulting (UK) Limited

29 CERTIFICATES OF QUALIFIED PERSONS

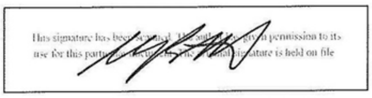
CERTIFICATE OF QUALIFIED PERSON

Martin Pittuck

I, Martin Frank Pittuck, MSc., C.Eng, MIMMM do hereby certify that:

1. I am Director and Corporate Consultant (Mining Geology) of SRK Consulting (UK) Limited with an office at 5th Floor, Churchill House, Churchill Way, Cardiff CF10 2HH, United Kingdom.
2. This certificate applies to the technical report with an effective date of July 29, 2015 and titled "NI 43-101 Technical Report, Mineral Resource and Reserve Update, December 31 2014, Twangiza Gold Mine, Democratic Republic of the Congo" (the "Technical Report").
3. I am a graduate with a Master of Science in Mineral Resources gained from Cardiff College, University of Wales in 1996 and I have practised my profession continuously since that time. Since graduating I have worked as a consultant at SRK on a wide range of mineral projects, specializing in precious and rare metals. I have undertaken many geological investigations, resource estimations, mine evaluation technical studies and due diligence reports. I am a member of the Institute of Materials, Minerals and Mining (Membership Number 49186) and I am a Chartered Engineer.
4. I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "Qualified Person" for purposes of NI 43-101.
5. I visited the Twangiza property between 12th and 15th March, 2015.
6. I am responsible for Sections 2 to 16, 20 and 22 to 29 of the Technical Report.
7. I am independent of the issuer as described in section 1.5 of NI 43-101.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101 and the sections of the Technical Report I am responsible for have been prepared in compliance with NI43-101.
10. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed the 29th day of July, 2015.



Martin Frank Pittuck, MSc. C.Eng, MIMMM

Director and Corporate Consultant (Mining Geology)

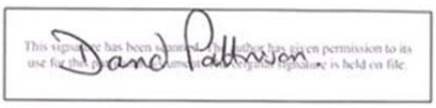
CERTIFICATE OF QUALIFIED PERSON

David Pattinson

I, David Pattinson, PhD., BSc., C.Eng, MIMMM do hereby certify that:

1. I am a Corporate Consultant (Minerals Processing) of SRK Consulting (UK)) Limited with an office at 5th Floor, Churchill House, Churchill Way, Cardiff CF10 2HH, United Kingdom.
2. This certificate applies to the technical report with an effective date of July 29, 2015 and titled "NI 43-101 Technical Report, Mineral Resource and Reserve Update, December 31 2014, Twangiza Gold Mine, Democratic Republic of the Congo" (the "Technical Report").
3. I am a graduate with a Doctor of Philosophy degree in Minerals Engineering gained from Birmingham University, UK in 1982 and I have practised my profession continuously since that time. Since graduating I have worked for an international engineering company for 23 years and then as a consultant at SRK working on a wide range of mineral projects including design and commissioning activities, technical studies and numerous due diligence reports in gold and base metal plants. I am a member of the Institute of Materials, Minerals and Mining (Membership Number 46888) and I am a Chartered Engineer.
4. I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "Qualified Person" for purposes of NI 43-101.
5. I visited the Twangiza property between 12th and 15th March, 2015.
6. I am responsible for sections 17, 18 and 19.1 to 18.7 of the Technical Report.
7. I am independent of the issuer as described in section 1.5 of NI 43-101.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101 and the sections of the Technical Report I am responsible for have been prepared in compliance with NI43-101.
10. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed the 29th day of July, 2015.



David Pattinson, BSc, PhD, C.Eng, MIMMM

Corporate Consultant (Minerals Processing)

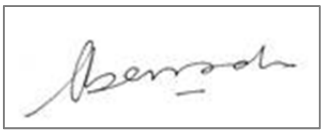
CERTIFICATE OF QUALIFIED PERSON

Daniel Bansah

I, Daniel Kenneth Bansah, MSc.(MinEx), MAusIMM(CP), do hereby certify that:

1. I am the Head of Projects and Operations of Banro Corporation, 1 First Canadian Place, Suite 7070, 100 King Street West, Toronto, Ontario, M5X 1E3, Canada.
2. This certificate applies to the technical report with an effective date of July 29, 2015 and titled "NI 43-101 Technical Report, Mineral Resource and Reserve Update, December 31 2014, Twangiza Gold Mine, Democratic Republic of the Congo" (the "Technical Report").
3. I am a graduate of University of Science and Technology, School of Mines, Ghana with a degree in Geological Engineering (1988). I also have a Master of Science in Mineral Exploration with Distinction gained from Leicester University, United Kingdom, and I have over 26 years of continuous professional experience in the gold mining industry. I was Banro Corporation's Vice President of Exploration from 2007 to 2013. Prior to joining Banro in 2004, I was the Group Mineral Resource Manager with Ashanti Goldfields, with responsibilities for the coordination, auditing and compilation of Ashanti's Mineral Resources and Ore Reserves in Africa. I am a Member and a Chartered Professional of the Australasian Institute of Mining and Metallurgy.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "qualified person" for purposes of NI 43-101.
5. I have had prior involvement with the property that is the subject of the Technical Report, in that I have worked on the Twangiza property as an employee since September 2005 to the date of this certificate. I have personally visited the property on many occasions with the last visit on July 8, 2015.
6. I am responsible for items 18.8 and 20 of the Technical Report.
7. I am not independent of the issuer as described in section 1.5 of NI 43-101 by virtue of being Head of Projects and Operations of Banro Corporation and a director of Twangiza Mining SA.
8. I have read NI 43-101 and the sections of the Technical Report I am responsible for have been prepared in compliance with NI 43-101.
9. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed the 29th day of July, 2015.



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