# FIRESTONE DIAMONDS plc

## LIQHOBONG MINE EXPANSION PROJECT

# DIAMOND RESOURCE AND RESERVE REPORT

**October 2015**

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## EXECUTIVE SUMMARY

The purpose of this internal technical report is to document the work activities conducted by independent consultants leading to the publication of the latest SAMREC compliant Diamond Resource and Reserve statements as at 30 September 2015 reflected in Firestone Diamond's 2015 Annual Report and as announced on 6 October 2015. The technical report layout is based on the SAMREC (2009 version) guideline but, because it is internal, does not constitute a competent person report (CPR) as defined in the AIM Rules.

Since being discovered in the 1950's, the Liqhobong Main Pipe has undergone a number of work studies under various owners culminating in a Definitive Feasibility study published by Firestone Diamonds at the end of 2012 and followed by an update a year later in November 2013. This was followed by the successful raising of the required capital to fund the Project and commencement of construction during mid-2014.

The 2015 SAMREC compliant Diamond Resource was based on a detailed re-logging of the main pipe borehole core during 2014 leading to the construction of a new 3D geological solid model. As part of the re-logging exercise, new density measurements were collected down all holes where competent core existed which allowed a local block estimate of density to be conducted for the first time. The latest grade estimate was again based on the wide diameter holes drilled during 2008/2009. The revenue estimate was based on size frequency distributions attained from surface bulk samples collected during 2008 and a re-pricing of the diamond assortment as at August 2014. The 2015 Diamond Resource stated at a 1.25mm bottom cut-off contains an Indicated Resource to a depth of some 183m below surface, estimated to comprise 9.5 million carats in 35 million tonnes of kimberlite at an average grade of 27 cpht. Below 183m an Inferred Diamond Resource containing some 13.5 million carats in 48 million tonnes at an average grade of 28 cpht has been estimated.

The derivation of a new Diamond Resource and block model afforded the opportunity to compile a new life of mine plan. After signing off of the relevant operating and economic assumptions and modifying factors, a Whittle pit optimisation study was conducted by independent Competent Persons for both concentric and split shell mine designs. A split shell design was selected as the most optimal mine plan delivering a 2015 SAMREC compliant Probable Reserve of some 9.3 million carats in 35.3 million tonnes at an average recovered grade of 26.4 cpht over 10 years of a 15 year life of mine. Over and above the Indicated Resource, the 2015 split shell mine plan also assumes the mining of a portion of the Inferred Resource totalling some 16.7 million tonnes.

The Liqhobong Project is currently half way through the construction phase with first diamond production expected towards the end of 2016. The project still faces a number of risks of which the most pronounced at the moment is the political instability in Lesotho and extreme weather conditions.

# 1. INTRODUCTION

## 1.1 Purpose of Report

This technical report was compiled to summarise the work activities conducted by independent consultants that produced the latest SAMREC compliant Diamond Resource and Reserve statements as at 30 September 2015 that were released to the market as part of Firestone Diamonds' Annual Report to Shareholders and as announced on 6 October 2015. The technical report layout is based on the SAMREC (2009 version) guideline but, because it is internal, does not constitute a competent person report (CPR) as defined in the AIM Rules for companies and AIM note for Mining and Oil & Gas Companies.

The latest Diamond Resource and Reserve figures are an update to the JORC and SAMREC compliant figures that were compiled in a Competent Person Report by A.C.A. Howe International (Leroux, 2010) which were used as input to a Definitive Feasibility Study (DFS) published by Firestone at the end of 2012 and updated at the end of 2013.

## 1.2 Project Outline

Firestone Diamonds plc (Firestone), an AIM quoted diamond development company, acquired the Liqhobong Diamond Mine (Liqhobong or the Project) situated in the northern Lesotho Highlands in an all share transaction from Kopane Diamond Developments Plc (Kopane) in September 2010. The mine is owned by the Liqhobong Development Company (LMDC) which in turn is owned 75% by Firestone and 25% by the Government of Lesotho. Firestone completed a DFS on the project in 2012 which entailed an expansion of the existing mine to treat 3.6 million tonnes of kimberlite from the Main pit. This was followed by an updated DFS released to the market during November 2013 which contained the following key operating and financial parameters:

| | Unit | November 2013 DFS |
|---|---:|---:|
| Ore mined | Mt | 53.7 |
| Average strip ratio | Waste/ ore | 2.28 |
| Plant capacity | Mtpa | 3.6 |
| In situ grade | Cpht | 32.07 |
| Average annual production | Mcts pa | 1.15 |
| Mining cost | ZAR/ t mined | 21.5 |
| Processing cost | ZAR/ t processed | 57.8 |
| Site SG&A | ZAR/ t processed | 12.5 |
| Steady state site operating expenses | US$/ ct | 43.93 |
| Royalty | % | 8% |
| Initial capital expenditure | ZAR million | 1,854 |
| Initial capital expenditure | US$ million | 185 |

*Table 1. 2013 DFS key parameters*

During January 2014 the company announced that it was fully funded to execute the project as it was successful in raising a total of US$222.4 million which was a combination of debt (US$82.4 million) and equity (US$140 million). The project officially commenced construction work on 1 July 2014 with a planned completion date of Q4 2016.

### 1.2.1 Regional Geology

The following information was obtained from Leroux (2010). Lesotho is entirely underlain by flat-lying Paleozoic rocks of the Karoo Supergroup. While Permian Dwyka-Ecca Group shales and marine sediments form the base of the succession, they are not exposed in Lesotho. The Dwyka-Ecca Group rocks are overlain by arenaceous continental sediments of the Beaufort and Stormberg Groups. The sedimentary rocks are capped by an accumulation of Cretaceous age amygdaloidal basalt flows up to 1,700 metres thick belonging to the Drakensburg Group. Feeder dykes and sills of basalt are common within the underlying 1,000 metres of sediments.

Kimberlites represent the final phase of igneous activity in Lesotho, and were emplaced during the Cretaceous in WNW-ESE trending structures. Out of the 400 known occurrences of kimberlite in Lesotho, 39 are pipes, 23 are blows and 343 are dykes. All but one pipe, and virtually all of the diamondiferous dykes are found within the WNW trending Lemphane-Robert Kimberlite Belt in northern Lesotho (see Figure 1). The Liqhobong kimberlites are located in the northern portion of this 100 kilometre long and 20 kilometre wide zone of intrusives.

The present topography is the product of three episodes of glaciation during the Pleistocene. The lowlands are underlain by sediments, whereas the Drakensburg basalt forms the highlands.

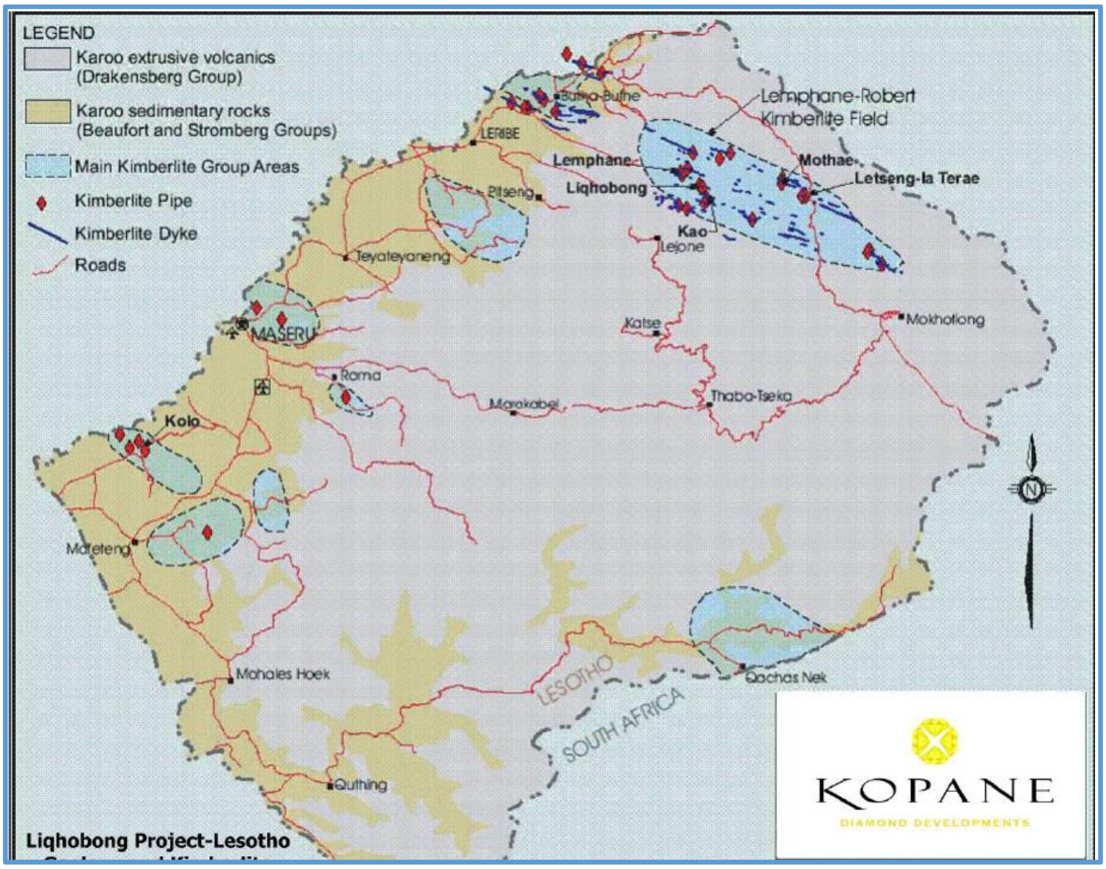



*Figure 1. Geology map with kimberlite occurrences (Leroux, 2010)*

### 1.2.2 Local Geology

The Liqhobong Mine lease area contains a cluster of at least five diamondiferous kimberlite bodies; namely the Main Pipe, Satellite Pipe, Discovery Blow, the Blow and a NW-SE striking dyke that is traceable from the perimeters of both the Main and Satellite Pipes (Figure 2). The surface areas of these pipes and blows are ~8.5 hectares, 1.6 hectares, 0.15 hectares and 0.1m hectares respectively. All the currently known five diamondiferous kimberlites of the Liqhobong cluster are within a strike length of at least 2.5 km (Leroux, 2010). Both blows and the Satellite Pipe are generously endowed with kimberlite indicator minerals (KIMs) occurring as xenocrysts. The main pipe is comparatively less enriched in the kimberlite indicator minerals but has abundant olivine macrocrysts and mantle xenoliths, the latter which commonly bear the KIMs.

The focus of this report is the 2015 Diamond Resource and Reserve update of the Main Pipe.

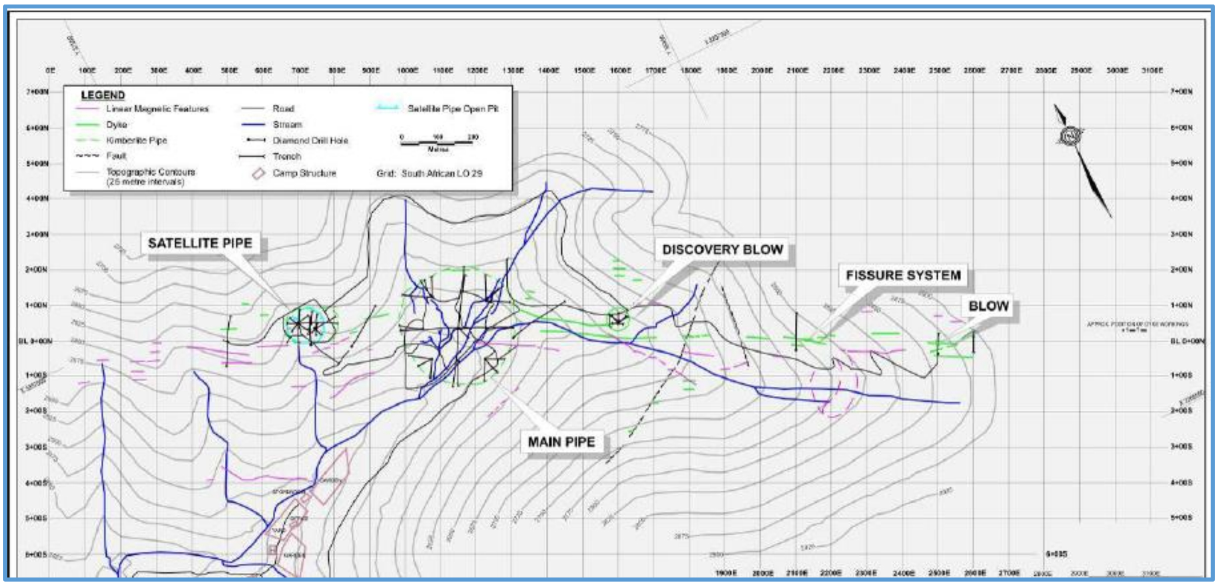



*Figure 2. Location of Liqhobong kimberlite bodies (Leroux, 2010)*

## 1.3 Project Location

The Liqhobong diamond mine lease covers an area of just over 7.6 km$^2$ and is located at the head of the Liqhobong valley in the Maluti Mountains of northern Lesotho in the Butha-Buthe district approximately 140 kilometres ENE of the capital Maseru (Figure 3). The nearest town is Ha Lejone located 18 km southwest and is accessible by 35km of unpaved road. Liqhobong is accessible from South Africa via the R26/R70 official border posts at the Maputsoe Bridge (Ficksburg) and from the R26/R711 via Caledonspoort. From the border post at Maputsoe, the A25 route via Ha Lejone is the most direct route to site, via a tarred dual lane road which changes to a poorly maintained dirt road from Ha Lejone. An alternative route exists, via the Moteng Pass leading to the turn off at Mothae Junction where the road continues to Kao and Liqhobong Mines. This road on this route is in a good condition and used to transport all large equipment to the mine. LMDC recently completed a new 5.8km access road to the Liqhobong mine.

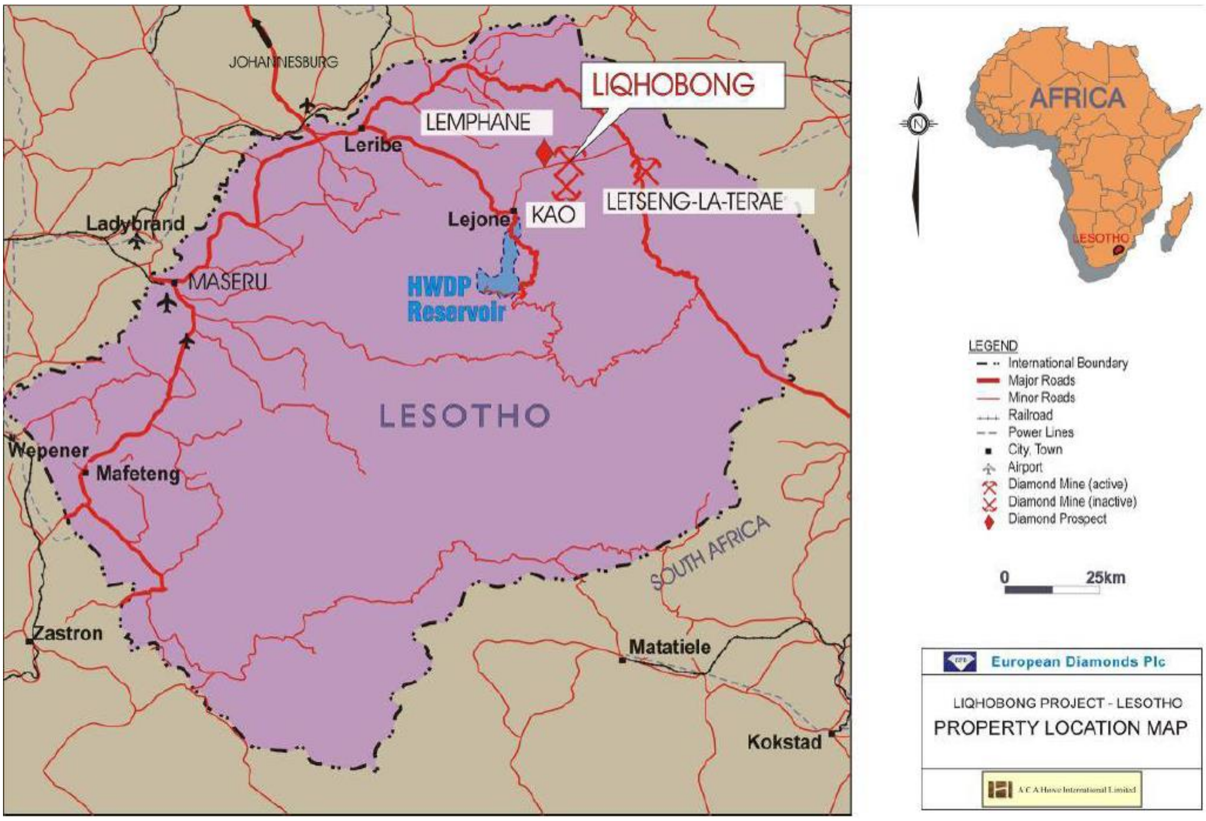



*Figure 3. Location map (Leroux, 2010)*

The extended mining lease no. 001-14/15 was issued in terms of section 33 of the Mines and Minerals Act to LMDC on 16 May 2014. The mining lease boundaries are confined within the following coordinates shown in Table 2 and Figure 4 below.

| UTM 35 S WGS 84 | | |
|---|---|---|
| **Point** | **Easting** | **Northing** |
| SP01 | 656299 | 6792400.650 |
| SP02 | 657243.520 | 6793773.230 |
| SP03 | 659734.300 | 6792058.680 |
| SP04 | 659251.890 | 6791369.609 |
| SP05 | 658021.500 | 6792226.400 |
| SP06 | 657544.270 | 6791534.670 |
| A | 658784.470 | 6791695.640 |
| B | 657655.220 | 6791695.640 |
| C | 657446.570 | 6791393.220 |
| D | 657216.970 | 6791552.770 |
| E | 656203.150 | 6791552.770 |
| F | 656058.891 | 6792180.405 |
| G | 655809.525 | 6792437.843 |
| H | 655630.439 | 6792524.211 |
| I | 655410.709 | 6792473.407 |
| J | 654816.294 | 6792848.726 |
| K | 654593.820 | 6793130.140 |
| L | 654593.820 | 6793872.720 |
| M | 655515.320 | 6793872.720 |
| N | 656932.190 | 6793320.400 |

*Table 2. Mining Lease coordinates*

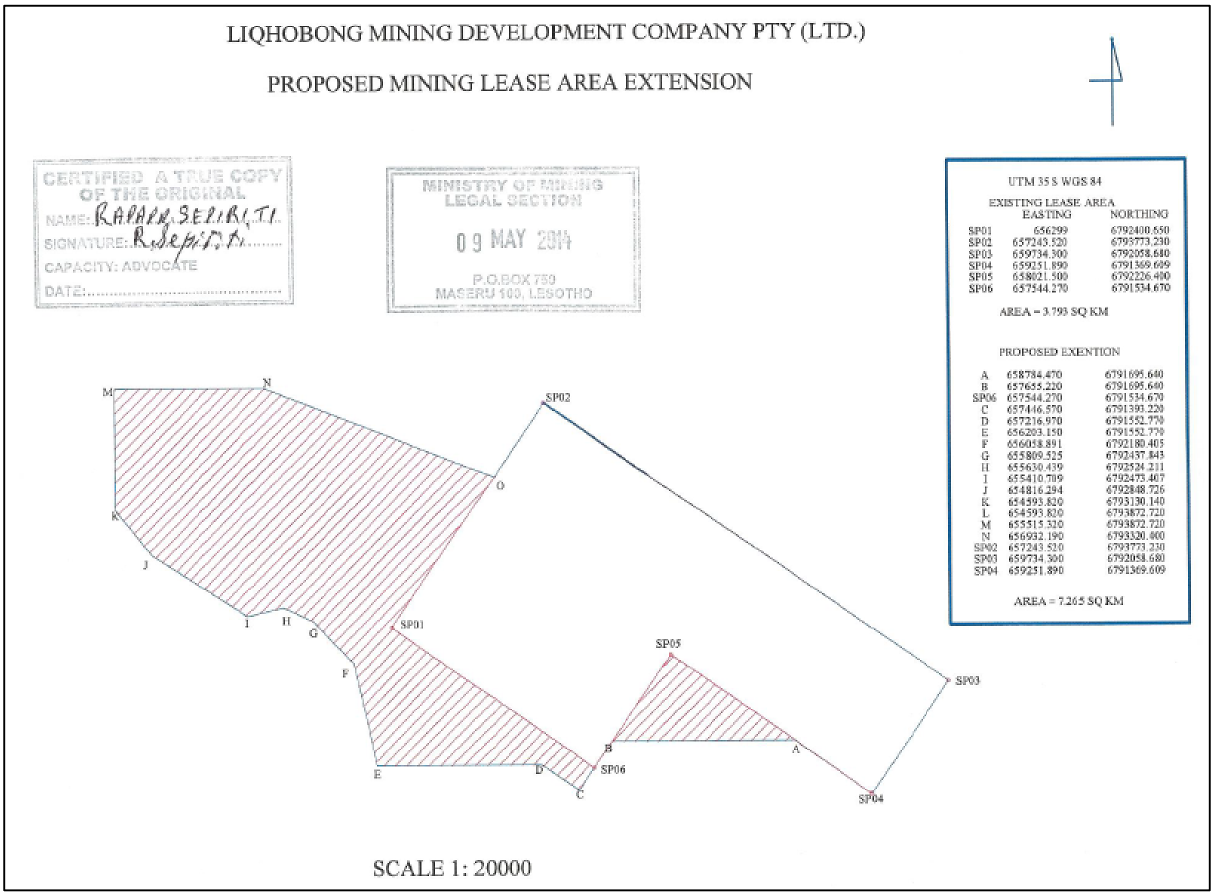



*Figure 4. Mining Lease area map*

## 1.4 Topography and Climate

Liqhobong lies at an elevation of 2,560 metres above mean sea level (masl). The largest body of water is the Motete River, located approximately 2km from the mine downstream of the Liqhobong stream. The Motete River is a major tributary of the Malibamatso River which flows into the Katse Dam. Due to the steep valley flanks the drainage pattern in the area is highly incised. Liqhobong lies in a region of high altitude and experiences a dry, temperate climate with warm and rainy summers and cold, dry winters. Extremes in weather conditions are experienced due to the altitude. Snow is common in the highlands between May and September. Most of the rainfall takes place during the summer months (see Table 3 below). June and July are normally the driest months. The severe winter climate at times restricts on-site operations whereas other delays may occur as a result of severe rainstorms in the summer.

| Month | Average rainfall (mm) | Average rainfall days |
|---|---|---|
| October | 88.8 | 16.0 |
| November | 117.7 | 18.7 |
| December | 131.5 | 21.2 |
| January | 149.5 | 22.5 |
| February | 151.6 | 20.0 |
| March | 117.7 | 19.3 |
| April | 59.0 | 12.4 |
| May | 26.3 | 6.5 |
| June | 11.7 | 3.7 |
| July | 11.2 | 3.3 |
| August | 25.7 | 5.3 |
| September | 43.9 | 7.9 |
| **Total rain days** | | **156.8** |
| **Mean annual precipitation** | **934.5 mm*** | |

** Note that the mean annual precipitation will not necessarily equal the sum of the monthly averages.*

*Table 3. Summary of average rainfall data for Liqhobong mine*

## 1.5 Legal Tenure

On 28 April 2014 Firestone announced that LMDC has entered into a Revised Mining Lease Agreement with the Lesotho Government for an initial period ending on 24 April 2024 that can be renewed in accordance with the Lesotho Minerals and Mining Act, 2005. In compensation for the requirement to pay withholding tax, the Lesotho Government has agreed that the royalty rate on diamond sales will be reduced from 8% to 4% from first production, until such time as LMDC has obtained benefit to the value of US$20.0 million. Thereafter, the royalty rate will revert to 8% of diamond sales.

The 7.6 km$^2$ mining lease no. 001-14/15 was issued in terms of section 33 of the Mines and Minerals Act to LMDC on 16 May 2014 as mentioned in section 1.3.

LMDC completed a comprehensive Environmental and Social Impact Assessment (ESIA) during 2012. This was reviewed by the Lesotho Department of Environment (DoE) and subsequently on 13 August 2012 the DoE issued an environmental clearance certificate for the Liqhobong expansion project.

## 1.6 History

The following information was obtained from Leroux (2010). The kimberlites were "discovered" as part of the Lesotho-wide program of Colonel Jack Scott in the late 1950's. The kimberlites were then subsequently evaluated by Scott's company; Basutoland Diamonds Limited (Basutoland). Basutoland had participation rights reserved for De Beers in return for funding, and made use of geologists from Anglo-American Corporation. From the late 1950's to 1996, the Property had been the subject of three exploration and mining programmes which were carried out by various groups:

a) Basotoland Diamonds Limited (circa 1950's)
b) Ministry of Overseas Development (1963 – 1966)
c) Liqhobong Cooperative (1979 – 1994)

From 1996 to 2003, MineGem Inc. carried out several systematic exploration programmes on the Liqhobong site which comprised of:

- Winterized camp construction;
- Survey grid establishment;
- Ground magnetic geophysical surveys;
- Petrographic, Kimberlite Indicator Mineral Chemistry and Microdiamond studies;
- Core drilling and sampling programmes on the Satellite Pipe, Main Pipe and the Blow;
- Surface bulk sampling programme of both the Satellite Pipe and Main Pipe (1998);
- Resource Estimation Study of the Satellite Pipe (1997); and,
- Feasibility Study completed by Bateman Engineering Limited of the Satellite Pipe (2001).

From 2003 to 2009, Kopane's exploration efforts on the Liqhobong site have consisted of the following work programmes and engineering studies:

***2003-2005***

- Re-logging of the MineGem drill core;
- Geotechnical study on the Satellite Pipe;
- Reverse circulation pipe geometry drilling on the Main Pipe
- Microdiamond analysis work on the Main Pipe; and,
- Mine development and construction of a diamond process plant based on the results of the Bateman Engineering Feasibility Study for the Satellite Pipe.

***2006-2007***

- Main Pipe internal pre-feasibility study (2007)

Based on the favourable results obtained from the 2007 internal pre-feasibility study, Kopane commissioned a Definitive Feasibility Study on the Main Pipe. The various work programmes consisted of the following:

***Main Pipe DFS Geological Work Programme (2008-2009)***

- Pipe geometry core drilling programme;
- Geotechnical core drilling programme
- Wide diameter reverse circulation drilling and sample processing programme; and,
- Large-scale surface bulk sampling and processing programme.

***Engineering Studies (2008 - 2010)***

- Metallurgical and Process Plant studies;
- Tailings Disposal and Environmental studies;

- Power line study; and,
- Infrastructure studies.

The following is a summary of the 2008-2009 DFS Work Programmes.

***DFS Wide Diameter Drilling (WDD) Programme***

From February 2008 to November 2009, a total of twenty eight 0.455 metre diameter drill holes totalling 4,413.70m was completed on the Main Pipe. Bauer Technologies South Africa (Pty) Ltd. (Bauer) were contracted to carry out the WDD mini-bulk sampling drilling programmes. One Prakla RB-40 RC drill rig and associated Bauer de-sanding / washing plant were utilized to carry out the WDD drill programme. The WDD drill holes were carried out in order to obtain geological and diamond grade information of the various kimberlite facies previously identified from the surface core drilling programmes. The WDD holes were drilled vertically on a nominal 50m by 50m grid pattern across the kimberlite footprint of the Main Pipe (Figure 6).

***Production and Sampling Results – DFS WDD Mini-Bulk Sampling Programme***

A total of 480.20 carats (or 6,645 stones) of commercial sized diamonds >0.85mm were recovered from a total of 1,142 wet tonnes of +1.0mm kimberlite material (i.e. recovered kimberlite chips from the WDD drill rig desanding plant). The kimberlite chips were processed through LMDC's batch sampling dense media separation (DMS) process plant. Due to the small amount of WDD chip samples recovered from the WDD sample lift intervals, the clay-rich upper intervals were grouped together for processing purposes.

***Production and Sampling Results –DFS Large-Scale Bulk Sampling Programme***

A total of 8 individual surface bulk sample pits were excavated whereby a total of 12,721.90 carats of commercial sized diamonds >0.85mm were recovered from a total of 33,921 dry tonnes of kimberlite which were processed through LMDC's Satellite plant facility (Figure 5).

***DFS Core Drilling***

From May 2009 to September 2009, a total of nine HQ/NQ diameter (four Main-series and five SRK geotechnical) core holes totalling 2,952m were completed on the Main Pipe. Geomechanics of Johannesburg, South Africa were contracted to carry out the core drilling programme using an Atlas Copco drill rig.

The goal of the Main-series core drilling programme was to obtain additional 3D geological information (i.e. kimberlite country rock contact points) of the Main Pipe at depth as well as to obtain core for metallurgical test work. All of the Main-series core holes intersected the kimberlite - country rock contact. The SRK-series core holes were drilled to obtain additional geotechnical information of the basalt country rock and kimberlite contact.

***Main Pipe DFS 2009 Mineral Resource Estimate***

The Mineral Resource estimate presented in the ACA Howe CP report (Leroux, 2010) was reported to have been estimated in conformity with accepted guidelines as per the JORC and SAMREC codes. Table 4 below summarises the Main Pipe Diamond Resource as at December 2009 at a bottom cut off of 1mm.

| Elevation | Volume | Density | Tonnes | Grade | | Carats |
|---|---|---|---|---|---|---|
| masl | $m^3$ | $T/m^3$ | T | $c/m^3$ | cpht | c |
| 2600 | 1,100 | 2.64 | 2,900 | 0.82 | 31 | 900 |
| 2580 | 218,200 | 2.60 | 566,700 | 0.70 | 27 | 152,500 |
| 2560 | 961,800 | 2.59 | 2,495,600 | 0.72 | 28 | 690,800 |
| 2540 | 1,571,500 | 2.60 | 4,083,900 | 0.78 | 30 | 1,219,800 |
| 2520 | 1,584,300 | 2.60 | 4,119,100 | 0.79 | 30 | 1,252,500 |
| 2500 | 1,554,300 | 2.60 | 4,042,800 | 0.81 | 31 | 1,260,700 |
| 2480 | 1,540,400 | 2.60 | 4,011,400 | 0.84 | 32 | 1,298,100 |
| 2460 | 1,523,600 | 2.61 | 3,972,200 | 0.87 | 33 | 1,325,700 |
| 2440 | 1,507,900 | 2.61 | 3,934,300 | 0.87 | 33 | 1,304,600 |
| 2420 | 1,485,200 | 2.61 | 3,874,900 | 0.84 | 32 | 1,243,900 |
| 2400 | 1,454,300 | 2.61 | 3,792,800 | 0.82 | 31 | 1,188,300 |
| 2380 | 1,404,500 | 2.61 | 3,662,300 | 0.79 | 30 | 1,104,700 |
| **Total** | **14,806,000** | **2.60** | **38,556,000** | **0.81** | **31** | **12,041,600** |
| 2360 | 1,358,500 | 2.61 | 3,541,000 | 0.86 | 33 | 1,165,400 |
| 2340 | 1,323,700 | 2.61 | 3,451,500 | 0.87 | 33 | 1,155,500 |
| 2320 | 1,304,900 | 2.61 | 3,403,700 | 0.89 | 34 | 1,161,000 |
| 2300 | 1,278,400 | 2.61 | 3,334,800 | 0.90 | 34 | 1,148,400 |
| 2280 | 1,240,600 | 2.61 | 3,235,700 | 0.90 | 35 | 1,116,600 |
| 2260 | 1,200,000 | 2.61 | 3,128,900 | 0.90 | 35 | 1,080,100 |
| 2240 | 1,164,200 | 2.61 | 3,034,700 | 0.90 | 34 | 1,046,600 |
| 2220 | 1,146,000 | 2.61 | 2,986,900 | 0.90 | 35 | 1,030,800 |
| 2200 | 1,144,500 | 2.61 | 2,982,800 | 0.90 | 35 | 1,029,500 |
| 2180 | 1,144,500 | 2.61 | 2,982,800 | 0.90 | 35 | 1,029,500 |
| 2160 | 1,144,500 | 2.61 | 2,982,800 | 0.90 | 35 | 1,029,500 |
| 2140 | 1,144,500 | 2.61 | 2,982,800 | 0.90 | 35 | 1,029,500 |
| 2120 | 1,144,500 | 2.61 | 2,982,800 | 0.90 | 35 | 1,029,500 |
| 2100 | 1,144,500 | 2.61 | 2,982,800 | 0.90 | 35 | 1,029,500 |
| 2080 | 1,144,500 | 2.61 | 2,982,800 | 0.90 | 35 | 1,029,500 |
| 2060 | 1,144,500 | 2.61 | 2,982,800 | 0.90 | 35 | 1,029,500 |
| 2040 | 572,300 | 2.61 | 1,491,400 | 0.90 | 35 | 514,800 |
| **Total** | **19,744,600** | **2.61** | **51,471,000** | **0.89** | **34** | **17,655,200** |
| **Grand Total** | **34,550,600** | **2.61** | **90,027,000** | **0.86** | **33** | **29,696,800** |

*Table 4. 2009 Main Pipe Mineral Resource estimate at 1mm BCO (Bush and Grills, 2009)*

The Diamond Resource stated in the ACA Howe report included in Firestone's admission document relating to the acquisition of Liqhobong in 2010 has a slightly higher grade (34.3 cpht) and total carats (31.14 Mct) than reflected in Table 4. This is due to the application of different modifying factors than those reflected in Bush and Grills (2009). This Technical Report will reconcile back to the 2009 Diamond Resource reflected in Table 4.

***Main Pipe Diamond Valuations (2008 and 2010)***

From August to October 2008, Kopane commissioned a diamond valuation of the 2008 Main Pipe DFS large-scale bulk sample diamond parcel which consisted of 12,721.9 carats. A value of US$86 per carat was obtained from an independent valuation of this parcel in November 2008.

***LMDC Pilot Plant Operation (February 2011 to October 2013)***

The Pilot Plant at Liqhobong was constructed initially to treat the harder ore of the smaller Satellite Pipe. Upon acquisition of the Project in September 2010 by Firestone Diamonds, the Satellite Pipe was nearing depletion and all mining activity was focused on the 8.5 hectare Main Pipe. In order to treat the much softer Main Pipe ore via the Pilot Plant acquired from Kopane at 100 to 120 tonnes per hour, a number of modifications were undertaken.

*Figure 5. Pilot Plant at Liqhobong Mine.*

The Pilot Plant commenced production during February 2011 at a throughput capacity of approximately 0.4 Mtpa. After modifications and upgrades the throughput rate was increased to 0.65 Mtpa by October 2011. The Pilot Plant was closed in October 2013 in anticipation of securing funding for the Expansion Project. The table below summarises the Pilot Plant production statistics as derived from Firestone Diamonds Annual Reports according to financial years (www.firestonediamonds.com).

| **Production** | | **FY2014** | **FY2013** | **FY2012** | **FY2011** |
|---|---|---|---|---|---|
| Ore treated | tonnes '000 | 199 | 623 | 488 | 72 |
| Recovered grade | cpht | 21.6 | 25.1 | 33.6 | 32.4 |
| Carats recovered | carats | 42 929 | 156 131 | 164 050 | 23 368 |
| **Revenue** | | | | | |
| Gross diamond sales | US$ '000 | 3 954 | 15 516 | 8 221 | |
| Carats sold | carats | 58 086 | 166 712 | 139 556 | |
| Price achieved | $/ct | 68 | 93 | 59 | |

*Table 5. Summary of Pilot Plant production data*

***Firestone DFS Updates (2012 to 2013)***

Firestone Diamonds published the results of its Definitive Feasibility Study (DFS) for the Liqhobong Main pit in October 2012. The DFS was based on a Diamond Resource estimate prepared by Z Star Mineral Resource Consultants (Z Star) dated 31 August 2009 (Bush and Grills, 2009). Z Star was subcontracted by A.C.A. Howe to conduct the grade estimate that formed part of the Competent Person Report (CPR) compiled on behalf of Kopane (Leroux, 2010). The 2012 DFS contained the following highlights:

- Pre-tax Net Present Value of US$441 million applying an 8% discount rate and 44% internal rate of return (IRR) with 28 month payback period
- Post tax IRR of 40% and NPV (applying an 8% discount rate) of US$335million
- Average annual production of 1.2 million carats commencing 2015
- 15 year life of open pit mine
- Processing 3.6 million run of mine tonnes per annum
- Total capital expenditure for the plant and associated infrastructure of US$167 million
- Average diamond price of US$100/ct, escalated at 3% real per annum, excluding full potential from recovery of large and special stones

A year later in November 2013, Firestone released an update to the 2012 DFS. The updated 2013 DFS contained the following changes compared to the 2012 DFS:

- An updated diamond price assumptions (Bush, 2013), which assumed a base case average expected price of US$107 per carat and an upside price of US$156 per carat. These figures were derived by including the large stone potential of the Liqhobong main pit by extrapolating both the SFD and assortment models from +10.8 carats to +100 carat size.
- The 2013 DFS exchange rate assumption was a flat ZAR10:US$ over the life of the Project compared to an average of ZAR9.76:US$1 in the October 2012 DFS
- Construction project capital cost increased from US$167 million to US$185.4 million when compared to the October 2012 DFS.

The 2013 updated DFS showed the following economics:

| | US$ per carat | Project Post-tax $NPV_{8\%}$ (US$M) | Project Post-tax IRR (%) |
|---|---:|---:|---:|
| **Updated DFS** | | | |
| Base Case | 107 | 379 | 30 |
| Upside Case | 156 | 728 | 45 |
| | | | |
| *Previous DFS (October 2012)* | *100* | *335* | *40* |
| Note: at 3% diamond price growth, all costs are flat real in the model only diamond revenue has been escalated. | | | |

*Table 6. 2013 Updated DFS project economics*

# 2. PROJECT DATA

## 2.1 Primary Data Elements

The following chapter comments on the status of the primary data elements used to derive the 2014 Diamond Resource estimate for the Liqhobong Main Pipe. All the pre-2010 work descriptions are derived from the A.C.A. Howe CPR (Leroux, 2010). Prior to the systematic exploration and evaluation work carried out by Kopane/Firestone and predecessor MineGem Inc, the Liqhobong kimberlites had not been subjected to modern exploration or evaluation techniques. Even the most basic information such as surface limits of the pipes had not been established.

### 2.1.1 Delineation drilling - Minegem

MineGem embarked on a major exploration program in 1996 which was designed to delineate the geometry of the Satellite Pipe, Main Pipe and the Blow as well as confirm the presence of kimberlitic dykes as identified from geophysical surveys. The Main Pipe was tested to a depth of approximately 150m with 25 drill holes named Main 1 to Main 25.

### 2.1.2 Delineation drilling - Kopane

During 2003, Kopane embarked on a delineation drilling program targeting greater depth in the Satellite Pipe and to further define the geometry of the Main Pipe. A total of 8 HQ core holes were drilled into the Main Pipe named Main 26 to Main 33.

Between September 2006 and November 2006, Kopane drilled a further 15 drill holes (HQ collars to 25m and NQ tails) to understand the continuity, shape and thickness of the various Main Pipe kimberlite facies to a depth of approximately 160m from surface so that a 3D geological model and resource estimate could be compiled. These holes were named Main 34 to 47.

From May 2009 to September 2009, a total of 9 HQ/NQ (four Main-series 48 to 51 and five SRK geotechnical) core holes totalling 2,952m were completed on the Main Pipe. The goal of the Main-series core drilling programme was to obtain additional 3D geological information (i.e. kimberlite country rock contact points) of the Main Pipe at depth as well as to obtain core for metallurgical test work. All of the Main-series core holes intersected the kimberlite - country rock contact. The SRK-series core holes were drilled to obtain additional geotechnical information of the basalt country rock and kimberlite contact drilling towards the Main Pipe.

### 2.1.3 Wide Diameter Drilling and Sampling

From 29 February 2008 to 27 November 2009, a total of twenty eight 0.455 metre diameter drill holes totalling 4,413.70m were completed on the Main Pipe. The holes were named WDD 1 to 27 with WDD 16 drilled twice (WDD-16 and WDD-16 TWIN). The WDD drill holes were carried out in order to obtain geological and diamond grade information of the various kimberlite facies previously obtained from the surface core drilling programmes. The WDD holes were drilled at –90° on a nominal 50m by 50m grid pattern across the kimberlite footprint of the Main Pipe (Figures 6 and 7). A total of 480.20 carats of commercial sized diamonds >0.85mm (or 6,645 stones) were recovered from a total of 1,142 wet tonnes of +1.0mm kimberlite material which was processed through LMDC's batch sampling DMS process plant.

### 2.1.4 Bulk Sampling

During 2008, a total of 8 individual surface bulk sample pits were excavated from the four known kimberlite facies in order to obtain a macro diamond parcel for diamond grade and valuation purposes (Figure 6). Each sample pit perimeter measured a nominal 20m by 20m with variable depths (from 3m to 5m). A total of 12,721.90 carats of commercial sized diamonds >0.85mm were recovered from a total of 33,921 dry tonnes of kimberlite processed through LMDC's Satellite Plant (Table 7).

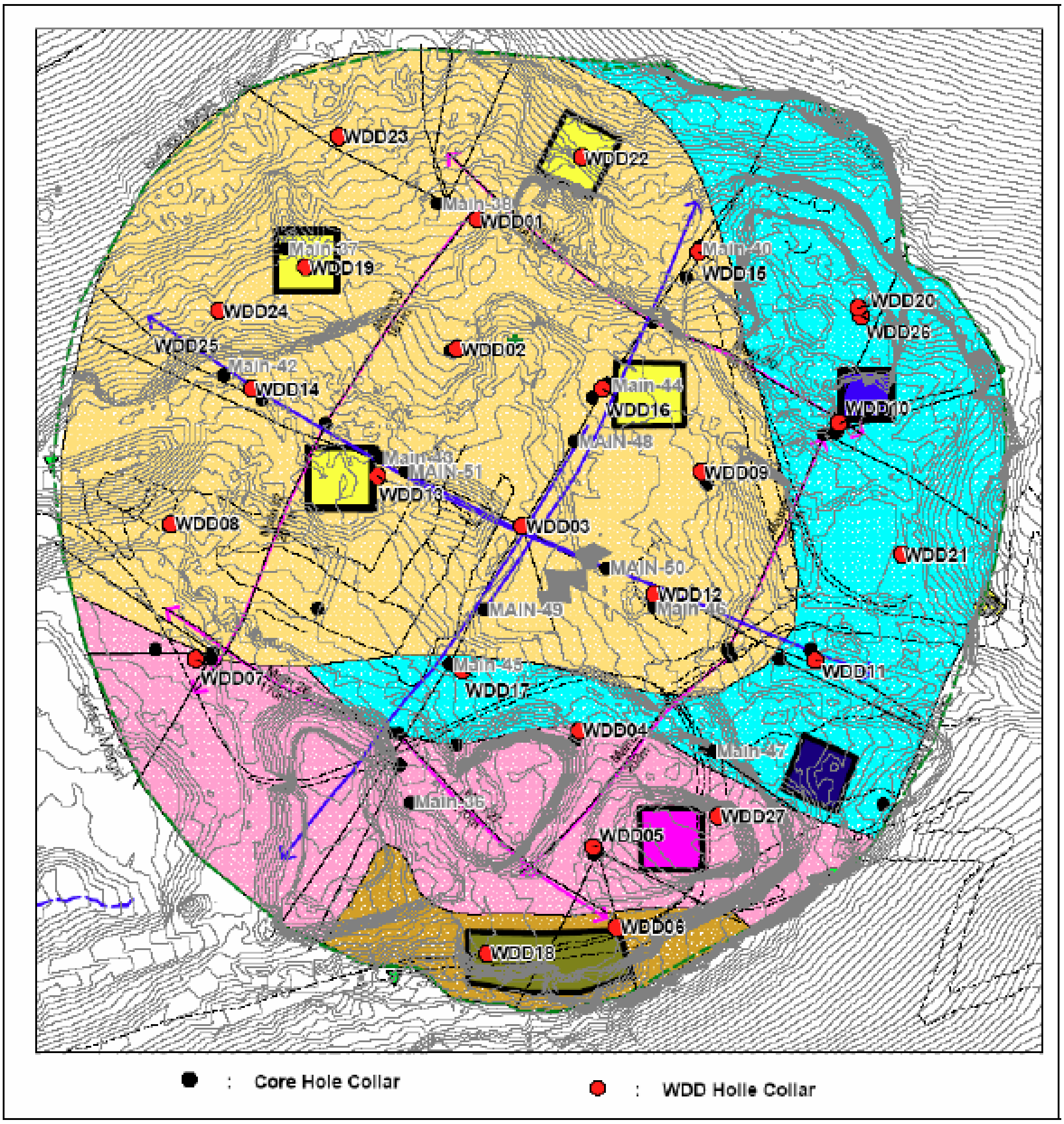



*Figure 6. Bulk sampling and WDD locations*

*(K2 = Yellow; K4 = Blue; K5 = Red; K6 = Brown. Squares are bulk samples)*

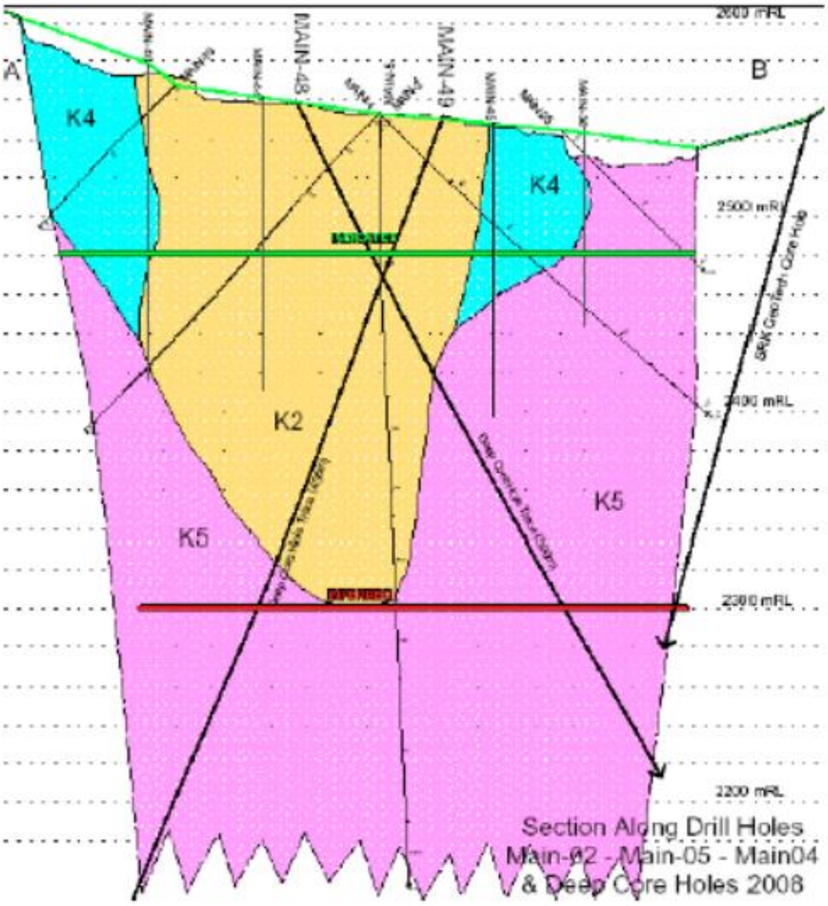



*Figure 7. Section through Main Pipe showing a number of delineation and WDD holes*

| Bulk Sample Number | Dry Tonnes | Carats | Grade (cpht) | Carats recovered |
|---|---|---|---|---|
| MAIN - K5-A | 3813.45 | 2858.20 | 74.95 | 2858.20 |
| MAIN - K6-A | 4943.13 | 1982.64 | 40.11 | 1982.64 |
| MAIN - K2-A | 4583.12 | 1617.90 | 35.30 | 1617.90 |
| MAIN - K2-C | 3670.87 | 926.94 | 25.25 | 926.94 |
| MAIN - K2-B | 4854.09 | 1360.09 | 28.02 | 1360.09 |
| MAIN - K2-D | 3843.49 | 1368.07 | 35.59 | 1368.07 |
| MAIN - K4-A | 4488.50 | 1335.77 | 29.76 | 1335.77 |
| MAIN - K4-B | 3724.65 | 1272.29 | 34.16 | 1272.29 |
| **TOTAL** | **33921.31** | **12721.90** | **37.50** | **12721.90** |

*Table 7. 2008 Bulk Sampling Program*

The high sampling grade (75cpht) of the K5 unit (Table 7) as well as the high boart content (38.5%), prompted the acquisition of a focused mining sample from the K5 unit. Focused mining is in effect production but limited to a single (focused) geological unit. A total of 19 899 carats from 38 688 tonnes for an average grade of 51cpht were recovered from the K5 facies between April and May 2008. This K5 focused mining sample included 13.7% boart (Bush, 2012). The bulk samples were used to derive size frequency distribution models (Bush, 2012) and, in the case of K5, was derived from focused mining which applied to the diamond assortment data provided a revenue estimate.

#### 2.1.5 Core logging

Commencing during Q3 2013, LMDC personnel undertook an exercise to re-log the Liqhobong Main pit core inventory. The focus of the core logging was to 1) identify and confirm the various facies and contacts logged by previous workers and 2) for the first time conduct a lithic count along 1m scanlines focussed on identifying mantle derived clasts and dilution. The newly derived core logs were used as an input to build the 2014 geological solid model.

#### 2.1.6 Density Measurements

During 2014, LMDC personnel undertook an exercise to measure density data from available drill-hole core to replace the sparse dataset used for the 2009 estimate. Kimberlite core samples were collected, sawed, weighed and measured with a calliper every 20m down all holes where competent material exists (Bosma, 2014). The dataset comprised sampling from some 40 drill holes with a total of 338 density samples. This data was imported into Datamine Studio 3™ and coded based on the new geological solids model. The data was then utilised for local block estimation of density (Lohrentz and Bush, 2014).

#### 2.1.7 Production data

Diamond assortment (US$ / carat / sieve size) data from previous recoveries by the LMDC Pilot Plant (December 2010 to November 2013) is available for analysis. The production assortment categories were repriced by an independent diamond valuator based on August 2014 diamond values (Erikson, 2014) and used as part of the 2015 Liqhobong Diamond Resource and Reserve update.

### 2.2 Spatial Data Accuracy

#### 2.2.1 Surface Core Drill Collar Surveying

The following comes from the A.C.A. Howe CPR report (Leroux, 2010).

Upon the completion of each surface core hole (Main-series and SRK-series), each drill collar was re-surveyed by the mining contractor’s mine surveyor with the use of an optical theodolite. By convention, each measurement was taken on the west side of the drill collar. MMIC’s mine surveyor would record the X, Y, and Z (RL) coordinates digitally for each drill hole into an Excel spreadsheet.

Subsequently in 2008, a topographical survey was carried out using an aircraft mounted LIDAR system that scanned the ground surface with a 50kHz laser producing a dense Digital Terrain Model (DTM). As a necessary part of the project, a GPS survey was undertaken to produce check points to check the laser survey and it was noted that the local heights of the mine beacons MB1, 2 and 3 had a datum shift of -96.86m (Southern Mapping Company, 2008).This was confirmed by the main project contractor, DRA Mineral Projects in 2012 (DRA, 2012). The 2014 final block model produced by Z Star was moved upwards by 96.86m to compensate for this (Lohrentz and Bush, 2014).

#### 2.2.2 Core Drill Downhole Surveying

Down-hole surveying was completed using two primary methods: a multi-shot surveying tool and a gyroscopic tool. The multi-shot surveying tool was utilized below the kimberlite (in non-magnetic sediments) and throughout the borehole. Due to moderate to strong magnetism of some of the kimberlite units (primarily caused by magnetite replacement of serpentine), down hole surveys of the core holes were carried out by Borehole Surveys South Africa (BSSA). Each core hole was surveyed with BSSA's self-orienting downhole survey system. The self-orienting downhole survey system has an azimuth and an inclination accuracy of 0.1 degree. Down-hole measurement readings were taken at 6 metre intervals. All of the down-hole survey data was digitally acquired and recorded as Excel files by BSSA and sent to Kopane by e-mail. The down-hole survey data was reviewed by Kopane and Howe personnel before being incorporated into Kopane's database.

#### 2.2.3 WDD Down-hole Caliper Surveying

Upon the completion of each WDD hole, a down-hole caliper survey was carried out by Howe personnel. The goal of the WDD caliper survey was to record and estimate the volume (in $m^3$) of material drilled out along the length of the WDD hole for diamond grade estimation purposes. The caliper survey consists of lowering a mechanical 3-arm caliper system with the use of a winch and cable system. Each arm can extend up to a maximum distance of 1.5 metres in length. The survey methodology consists of lowering the caliper to the bottom of the WDD hole, extending the arms until they contact the WDD hole wall and then raising the instrument at a constant rate so that the calliper arms can measure the WDD hole profile in real-time. The information was recorded on a laptop and then processed by Howe personnel for processing and interpretation. The data was presented as a graphic 2D downhole Excel spreadsheet.

### 2.3 Data Management

The Liqhobong Expansion Project is currently still in the construction phase and no IT infrastructure is in place. As such LMDC has not yet invested in an industry standard database. This is planned for 2016 after completion of the office buildings and installation of IT infrastructure at the Liqhobong mine site in Lesotho. In the meantime the data described in 2.1 and used to compile the 2014 Diamond Resource is stored on Excel spreadsheets and backed up monthly.

### 2.4 Data Verification

All the recently acquired datasets including the core logging and density measurements can be verified by comparing the content of Excel spreadsheet to original logs. This unfortunately is not the case for some of the older datasets. However, the older datasets were reviewed and verified by A.C.A. Howe (Leroux, 2010). Quality assurance and quality control programmes were in place during the Kopane WDD drilling and bulk sampling programmes. This QA-QC covered database management, adherence to geological procedures (sampling, logging, data entry) and data validation. Any identified errors were cross checked with hard copies and corrections were made where necessary. ACA Howe provided third party supervision and monitoring services to LMDC between February 2008 and July 2009.

## 2.5 Geological data

As mentioned previously, during Q3 2013, LMDC personnel undertook an exercise to re-log the Liqhobong main pit core inventory (Rapopo, 2015). The results were used to derive an updated geological and block model during 2014. Fifty nine drill holes with a total of 10,143 metres drilled across the Main Pipe were logged. Four main kimberlite facies (K2, K4, K5 and K6) had previously been identified in the Main Pipe by previous workers. Subtle variations exist such that each of the K2 and K5 facies are divisible into two sub-facies; namely TKB1 and TKB2 for K2 and TKB3 and TK1 for K5. The major differences between the K2 and K5 facies in general are the prominence of basement clasts, occurrence of lapilli and carbonate segregationary features as well as comparatively more calcite veining in K2 and their virtual absence in K5. All these features in K2 can be attributed to the comparatively more hydrous nature of the K2 kimberlite magma.

The recent core logging was conducted in two stages. The first exercise involved subdivision of each drill hole into a kimberlite facies/sub-facies (either TKB1, TKB2, TKB3, TK1, K4, or K6) based on the compositional and textural variations, noting the sizes of the largest crustal xenoliths, largest mantle xenoliths and visually estimating the olivine (both phenocrysts and macrocrysts) component within each defined unit. The second exercise involved a lithic count down a scan line, taken at 1 metre increments from top to the bottom of each drill hole. Crustal xenoliths were counted and measured if they were larger than 5mm and mantle xenoliths and xenocrysts were measured if larger than 4mm in size. Also, the biggest five olivine macrocrysts, five mantle xenoliths and five lapilli (where present) grains were measured over a meter at every 5m interval; these did not have to lie along a scan line. The general distribution of sub-facies/facies across the Main Pipe is shown in Figure 8, looking W-E from the North.

Rapopo (2015) concluded that basement clasts are generally prominent in the K2 facies, very rare in K4 and K5 and absent in the K6 facies. This occurrence of basement clasts almost exclusive to the K2 facies could imply it being the earliest intrusion or an indication of the crustal level at which the kimberlite magma erupted. An early intrusion would imply that K2 was the first to break through the crustal basement, opened up a fissure and consequently entrained fragments of the basement. However the apparent co-magmatic nature between K2 and K5 and the K2+K5 transitional units do not necessitate that K2 was emplaced first. Likewise, the sharp contact between K2 and K6 and the occurrence of K6 autoliths in K2 rule out K2 having been the first pulse. It thus becomes more plausible to suggest that the inclusion of basement clasts in the K2 facies rather reflect the level at which the magma became violent. Therefore the establishment of the Main Pipe was probably achieved by several pulses of kimberlite melt fractions. In this manner, subsequent pulses of a kimberlite from one source may undergo differential degree of alteration or incorporate different quantities of mantle and crustal clasts.

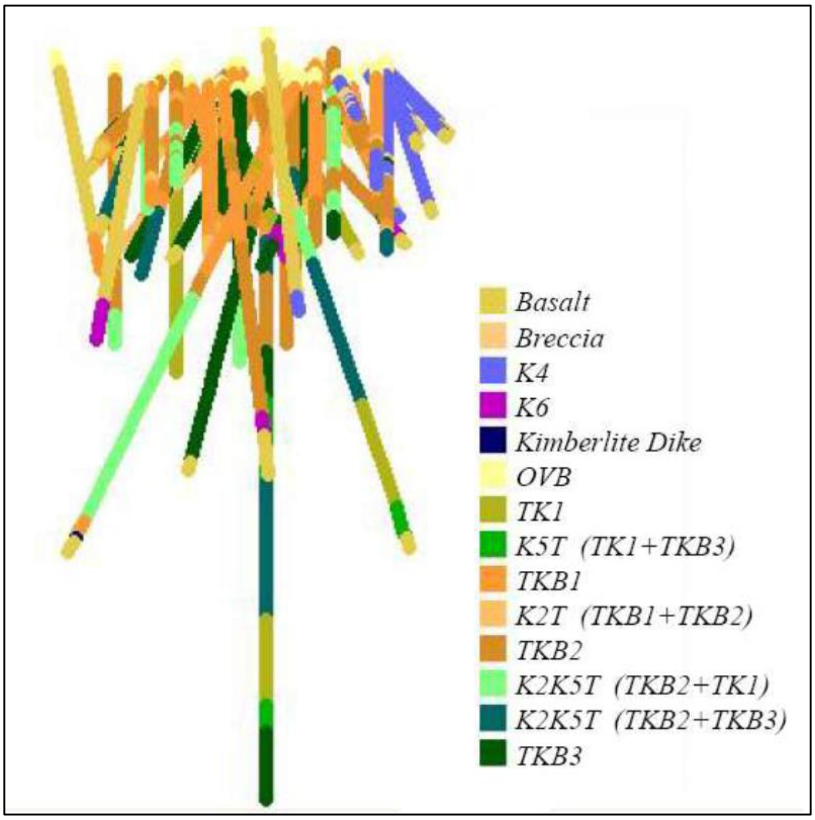



*Figure 8. Down hole facies characteristics of the Main Pipe drill holes (Rapopo, 2015).*

## 2.6 Density data

Density measurements were taken of all the main pit core where possible. A dry competent sample was taken roughly at every 20 metre interval where competent core exists. The procedure for density calculations involved cutting with a core cutter, the ends off the core cylinder to facilitate accurate measuring. The core cylinder is then measured along the long axis. Two measurements were taken and recorded on a log sheet. Three diameter readings, one on each end and the third at the centre of the core were measured with a Vernier calliper. The average of the three readings were taken as the "true" diameter. The cut core was then weighed on a scale that was calibrated every morning. Volume of the core was then calculated using the equation $V = \pi \times (0.5d)\ 2\times l$ (where $\pi$ = pi, d =diameter and l = length of the core). Subsequently, density was calculated according to the formula $\rho$ (density) = m/v (where m = mass and v = volume. The density measurement exercise was audited by Firestone's Mineral Resource Manager (Bosma, 2014).

Table 8 summarises the density values in tonnes/$m^3$ (Lohrentz and Bush, 2014).

| Raw | | | | | | | |
|---|---|---|---|---|---|---|---|
| Facies | # Samples | Min. | Max. | Mean | Vari. | Std. Dev. | $C_V$ |
| None | 8 | 2.61 | 2.72 | 2.65 | 0.00 | 0.04 | 0.01 |
| K2 | 190 | 2.19 | 4.17 | 2.60 | 0.03 | 0.16 | 0.06 |
| K4 | 17 | 2.54 | 2.85 | 2.65 | 0.01 | 0.08 | 0.03 |
| K5 | 115 | 2.25 | 3.77 | 2.65 | 0.02 | 0.15 | 0.06 |
| K6 | 8 | 2.47 | 2.93 | 2.71 | 0.02 | 0.13 | 0.05 |

*Table 8. Summary statistics of the SG data by facies (tonnes/m$^3$)*

# 3. SAMPLING

The focus of this chapter is on the WDD and bulk sampling campaigns of which the outputs were used to inform the latest grade and revenue estimates. These sampling campaigns and their governance was well covered by the A.C.A Howe CPR (Leroux, 2010) and is summarised below.

## 3.1 Sampling Method

### 3.1.1 Wide Diameter Drilling

Bauer Technologies South Africa (Pty) Ltd. (Bauer) was the contractor and operator for the WDD rig. One Prakla RB-40 RC drill rig and associated Bauer de-sanding / washing plant was utilized for the collection and recovery of kimberlite cuttings >0.8mm. The WDD rig is capable of drilling a 17.5 inch diameter hole to a depth of approximately 300m. Bauer's Prakla RB-40 drilling rig is designed to carry-out air assisted fluid flush RC drilling. WDD drilling sampling methods rely on circulating drilling fluid to clear cuttings from the drill bit, stabilize the drill-hole and lift cuttings to surface using a reverse circulation airlift system. The rotating open-centre drill bit liberates the kimberlite cuttings while the introduction of compressed air into the inner air rods airlifts the drilling fluid and sample (cuttings) to surface, where the cuttings sample is recovered over a 0.8mm wedge wire screen and bagged, while the drill fluid re-circulates back down the drill-hole, less the -0.8mm fines that are settled out into a mud tank or pit.

All +0.8mm cuttings returned by the Prakla RB-40 drill rig was processed and sized through the de-sanding plant. Processed material were collected in $1m^3$ single-walled, woven polypropylene bags, where they were labelled and numbered by a pre-determined sample interval and bag number once the bulk bag was full. The bulk bag was then securely tied and tagged with a pre-numbered security strap at the drill rig. Once the bulk bag was securely sealed, it was then loaded onto a crane truck for shipment to a secure storage area located at the project site in order to await processing through LMDC's on-site 5tph DMS plant.

### 3.1.2 2008 Bulk Sampling program

A total of 8 individual surface bulk sample pits were excavated and collected from the four known kimberlite facies in order to obtain a macro-diamond parcel for diamond grade and valuation purposes. Each sample pit perimeter measured a nominal 20m by 20m with variable depths (from 3m to 5m). All the large-scale bulk sample material for a particular sample site were loaded and transported to the Satellite Plant via articulated dump trucks for processing. LMDC and or A.C.A. Howe personnel supervised the removal and transport of the large-scale bulk material so that it was stockpiled according to bulk sample site. Each batch sample was identified with a sign denoting what bulk sample site it was derived from. The kimberlite muck for each large-scale bulk sample was piled on top of a sand-clay rich base.

## 3.2 Sample Preparation and Treatment

### 3.2.1 2008- 2009 Wide Diameter Drilling

LMDC purchased and commissioned a 5 tph DMS batch sampling process plant in order to process and recover diamonds from the WDD mini-bulk samples. The following describes the sample processing procedure for the WDD mini-bulk samples:

- All sub-sample bags for each WDD hole sample interval were transported to the 5 tph DMS plant for processing. LMDC and or ACA Howe personnel supervised the removal and transport of the bulk bags for a particular WDD sample interval from the secure storage facility to the 5 tph plant and reviewed a sample check list for the WDD sample interval to make certain that no sub-sample bags were left behind;
- The +0.8mm recovered wet weight of each sub-sample was obtained (with portable scale) and recorded onto a pre-designed “Mini-Bulk Production Sheet” prior to being emptied into the feed hopper;
- The WDD mini-bulk sample was withdrawn at a steady rate controlled by a profile gate through a primary jaw crusher to reduce all particles to < 25mm;
- The sample was then fed into a scrubber and passed over a feed prep de-watering/de-sliming screen prior to entering the mixing box;
- In the mixing box, the product plus circulating medium (ferrosilicon + water) was mixed and then pumped via a centrifugal pump through a 250mm DMS cyclone for separation;
- Both heavy and light size fractions report to the split sinks-floats screen respectively whereby both products were washed clean of the circulating medium;
- For each sample the sinks DMS concentrate reported to individual canisters stored in a secure cage. The floats product reported to a screen which separates +6mm from -6mm material. The -6mm product was washed to remove FeSi and reported to bulk bags for storage, with the +6mm being conveyed to the re-crush crusher.
- All excess water was drained prior to weighing the DMS concentrate canisters;
- The DMS concentrate canister was labelled with the following DMS sample number format (e.g. WDD-007-2-DMS) onto the canister and was then individually weighed and recorded onto the “Mini-Bulk Production Sheet”. The corresponding pre-numbered security tag was also recorded for that DMS concentrate canister;
- All DMS concentrate canisters were then transported and stored within the Satellite Process Plant’s dedicated secure compound. ACA Howe personnel monitored and escorted the samples to the secure compound;
- Upon receipt of the DMS concentrate canisters, a Chain of Custody form was filled out by LMDC’s on duty process plant security guards and stored for their records.
- From the dedicated area, each sealed DMS concentrate sample canister and a chain of custody sample submission form was delivered to LMDC’s sort house for DMS concentrate processing and diamond sorting.

To prevent contamination of samples by diamonds from previously run samples strict guidelines were followed. Prior to processing a new sample batch, the plant equipment (conveyors, crushers, hoppers, chutes, scrubber, sumps and pump boxes) was thoroughly cleaned inside and out. All screens were scrubbed and flushed. The DMS circuit was run empty until all material trapped in the system was flushed out. The plant floor and base for the

5 tph plant consisted of a concrete base, which was periodically swept. All sumps were cleaned and hosed off prior to processing the next WDD sample interval.

3.2.2 2008 Bulk Sampling program

The LMDC's Satellite process plant was used to process and recover diamonds from the large-scale bulk samples. At the time the LMDC's Satellite process plant consisted of the following circuits:

- A 75 tonne per hour crushing circuit;
- Separate fines (-6mm to +1mm) and coarse (-32mm to +6mm) DMS circuits. The coarse DMS cyclone was a 60 tonne per hour Dense Media Separation (DMS) circuit which consisted of a 510mm diameter separating cyclone; and the fines DMS cyclone was a 360mm diameter separating cyclone,
- A recovery circuit consisting of a concentrate dryer, Rare Earth magnetic separator drums, 2 Debex VE116 X-Ray diamond sorting machines and a grease table (Vipro Vibrating Products (PTY) Limited).

The following describes the sample processing procedure for the large-scale bulk samples:

- Wet tonnages were determined by a load cell attached to the front-end loader and by a weightometer located mid-way along the run-of-mine (ROM) belt. Moisture content samples (2-3kg) were collected off the ROM belt cutter every 2-4 hours during steady state operation for overall bulk sample batch dry tonnage determinations.
- All kimberlite material was fed into the Primary Static screen where the oversize reported to the side and the undersized material reported to a vibrating grizzly feeder (VGF) for additional sizing before the primary jaw crusher. All undersize and crushed VGF oversize material reported to the ROM belt. Collection and re-handling / treatment of +500mm grizzly oversize and spillage were broken up and then hauled with the front-end loader (FEL) on a regular basis back to the primary feed bin.
- All ROM head feed reported to the scrubber for attritioning. The scrubbed material was discharged through a 32mm square aperture trommel screen whereby the +32mm material reported to a secondary cone crusher for size reduction and the -32mm reported directly to a double-deck feed prep screen fitted with a 1mm slotted aperture bottom screen and a 8mm square aperture top screen for coarse and fines DMS feed screening.
- The fines and coarse size fraction kimberlite product reported to dedicated mixing boxes and DMS cyclones for separation. The sinks and floats fractions from both DMS cyclones reported to a sinks screen and floats screen respectively.
- The DMS sinks product then reported to a surge bin for downstream diamond recovery (drying, sizing, magnetic separation of the ferro and para-magnetic minerals, x-ray and grease table).
- The light fraction (floats) reported to a double-deck floats screen where the +6mm oversize reported to a tertiary cone crusher for size reduction in order to liberate any locked diamonds.

- The tertiary crushed product then reported back to the scrubber for attritioning before returning to the DMS circuit.

Contamination of samples by diamonds from previously run samples can adversely affect sample results. Therefore, strict guidelines were followed to prevent batch sample cross-contamination. Prior to processing a new sample batch, the plant equipment (conveyors, crushers, hoppers, chutes, scrubber, sumps, pump boxes) were thoroughly cleaned inside and out. All screens were scrubbed and flushed and de-blinding. The DMS circuit was run empty until all material trapped in the system was flushed out. The plant floor and base for the Satellite plant consisted of a concrete base, which was periodically shovelled and swept. All sumps were cleaned and hosed off prior to processing the next batch sample.

## 3.3 Sampling Recovery

### 3.3.1 Wide Diameter Drilling

The following Standard Operating Procedures were implemented in the LMDC sort house for the WDD DMS concentrate. During the sorting work, an A.C.A. Howe representative was present at all times to monitor the procedures/process, documentation and ensure sample integrity. The process consisted of the following steps (Leroux, 2010):

- Receipt by LMDC sort house manager of secure DMS concentrate samples from the secure storage facility from ACA Howe Representative and LMDC Manager.
- Inventory of the DMS concentrate samples by number and security seal tags by the LMDC sort house manager.
- Reconciliation of sample numbers and security tag numbers to the sample submission form provided by ACA Howe.
- LMDC affixes WDD sample numbers.
- Opening of DMS concentrate canisters by LMDC sort house manager under the supervision of an ACA Howe representative.
- Transfer of DMS concentrate sample to pans for drying along with cursory selection of visible diamonds.
- Weighing and description of +2ct diamonds and documentation of all diamonds from initial cursory sort.
- Drying of DMS concentrate.
- Sieving of dried DMS concentrates with a 12mm, 6mm, 3mm square aperture screens (with catch pan) sieve nest and weighing of size fractions.
- Labelling of all fractions of the WDD DMS sample.
- Escorted transport of labelled size and magnetic fractions to LMDC’s secure sorting room.
- Hand sorting of each sample fraction by 3rd party diamond pickers (MSA Geoservices).
- Each fraction sorted twice by two different sorters.
- Verification of all selected diamonds.
- DTC sieving of all diamonds in sample, and weighing of each sieve fraction.
- Weight and description of each diamond.
- Data recorded and entered into LMDC’s diamond report.

All diamonds were packaged and sealed in double bagged WDD sample interval labelled transparent zip-lock bags and stored in a dedicated diamond safe located within the sorting room. A dual custody system of two security locks placed on the diamond safe were in place whereby the locks could only be opened simultaneously by the LMDC sort house manager and an ACA Howe representative.

### 3.3.2 2008 Bulk Sampling program

Once all of the –12mm DMS concentrate was dried, sized and passed through the magnetic separators for removal of the magnetic fractions, the non-magnetic fraction of the DMS concentrate size fractions (+1 to -3 millimetre, +3 to -6 millimetre and +6 to -12 millimetre) were processed through two Impulelo® X-Ray Diamond Sorter Units (Model number CDX 116 VE). All three individual sized fractions were sized at the concentrate dryer and automatically fed to the Sortex receiving hopper for processing.

The Sortex units were designed on the principle of diamonds fluorescing and-or luminescing when bombarded by X-rays. The diamond bearing concentrates passes photomultiplier tubes that detect fluorescent material (i.e. particles emitting light) which has been irradiated by X-rays. Excitation of the photomultiplier tubes triggers the ejector gate doors to open and force the diamonds (and other fluorescent material plus gangue) into a separate stainless steel canister from the gangue minerals. The Sortex tailings are diverted to the Vipro grease table for reprocessing.

A Vipro® VGT 1220x1625 model grease table was employed to concentrate the +1 to -6 millimetre, and +6 to -12 millimetre X-ray tailings. Most diamonds are hydrophobic (i.e. non-wettable) and thus will adhere to grease specially formulated for diamond recovery.

All recovered diamonds picked from both the Sortex and grease table were placed into the sort-house's glove box for weighing and description before being stored in a safe under the supervision of an LMDC security officer.

## 3.4 Sampling Governance

### 3.4.1 Wide Diameter Drilling

ACA Howe's chain of custody and security protocols at the 5tph DMS plant were designed around a two-lock system such that two individuals must simultaneously carry out the removal, transport and escort of the DMS concentrate from the secure area at all times. There was no video surveillance system set up for the 5tph DMS plant. However, a security officer from LMDC's security service accompanied all transport of DMS concentrate canisters to the Satellite Plant's recovery circuit for processing.

### 3.4.2 2008 Bulk Sampling program

The following chain of custody procedures and protocols were carried out and adhered to by LMDC and its mining contractors during the large-scale bulk sampling programme:

- LMDC and/or ACA Howe geologists verified that all sample material for each bulk sample site was cleanly mucked out by the LMDC's mining contractor;

- When the bulk sample material was transported to the Satellite Plant's ROM stockpile area and in order to avoid sample mix-ups, LMDC and/or ACA Howe geologists verified that the kimberlite material for each batch sample was transported to its specified location on the ROM stockpile area by the mining contractor and that any other kimberlite material in the ROM pad area was clearly demarcated off by fencing pickets and flagging tape;
- In order to avoid sample spillage, all of the mining contractor's excavator, ADT and loader operators were given specific instructions by LMDC and ACA Howe not to overload their buckets when transporting kimberlite from the bulk sample site area to the Satellite ROM stockpile area; and,
- A LMDC security officer was present to observe the mucking of kimberlite from the Satellite ROM pad to the primary feed bin.

# 4. DIAMOND RESOURCE ESTIMATE

This chapter describes the derivation of the 2015 SAMREC compliant Diamond Resource estimate. The work was conducted during 2014, by independent consultants, Z Star Mineral Resource Consultants and signed off by their Principal Mineral Resource Analyst, Mr David Bush (Lohrentz and Bush, 2014). Compared against the previous estimate the following new inputs were considered:

- *Geology* - Commencing during Q3 2013, LMDC re-logged the Liqhobong main pipe core inventory. The focus of the core logging was to 1) identify and confirm the various facies and contacts logged by previous workers and 2) for the first time conduct a lithic count along 1m scanlines focussed on identifying mantle derived clasts and dilution. The newly derived core logs were used as an input to build the 2014 geological solid model. (Rapopo, 2015).
- *Density* - during 2014, it was decided to initiate a density recording campaign to replace the sparse dataset used for the 2009 estimate. Kimberlite core samples were collected, sawed, weighed and measured with a calliper every 20m down all holes where competent material exists (Bosma, 2014).
- *Bottom-cut-off* – the new Treatment Plant will have an effective bottom-cut off of 1.25mm. This is a change compared to the 2009 estimate which was stated at a 1mm bottom cut-off.
- *Boart factor* – the 2015 grade estimate is a gem diamond estimate (all boart was excluded), i.e. no boart factor was applied as was the case in the previous 2009 estimate.
- *Depletions* – the final new in-situ model was depleted against the surface topographic elevation dated April 2014 to compensate for mining conducted up to that date.
- *Revenue* – First Element Diamond Services was requested during August 2014 to revalue the Liqhobong parcels based on current market trends (Erikson, 2014). The $/ct/sieve class information for the 2014 re-pricing was used to update the Liqhobong revenue forecast by Z Star including an adjustment for the bottom cut off and an estimate of large stone potential not recovered by the Pilot Plant.
- *Positioning* – during a 2008 LIDAR survey of the Liqhobong mine site it was discovered that the mine's survey points relative to the beacons used were 96.86m below the true position. The 2014 model was based on the collar positions of the drill-holes and then afterwards lifted by 96.86m to bring it to the correct elevation.

## 4.1 Geological Solids and Block Models

Creation of the 3D solids (wireframe) model was undertaken by orientating sections in plan view while visualising the following in CAE Studio 3 (Lohrentz and Bush, 2014):

- Current drill-hole data and logs (19m clipping either side); and

- Existing 3D solids model outline on that level.

Section clipping was set to 19m and a string interpretation for each of the facies was given on a 20m spacing in the vertical direction. Strings were adjusted to ensure that the resultant solids cut drill-hole data where desired and that adjacent facies did not overlap.

A separate solid for each of the facies (K2, K4, K5 and K6) was created from 2 697 masl down to a depth of 2 137 masl. An overburden surface was then created taking into account the drill-hole data as well as the previous model and the current surface topography so as to ensure correct slope angles on the perimeter of the pipe. The individual facies solids were then clipped to this surface to create the final solids models for each of the facies. Figure 9 shows a plan view of the 2 600 masl elevation and illustrates the change from the 2009 geological solids model (solid pale colours) and the 2014 interpretation (outlines).

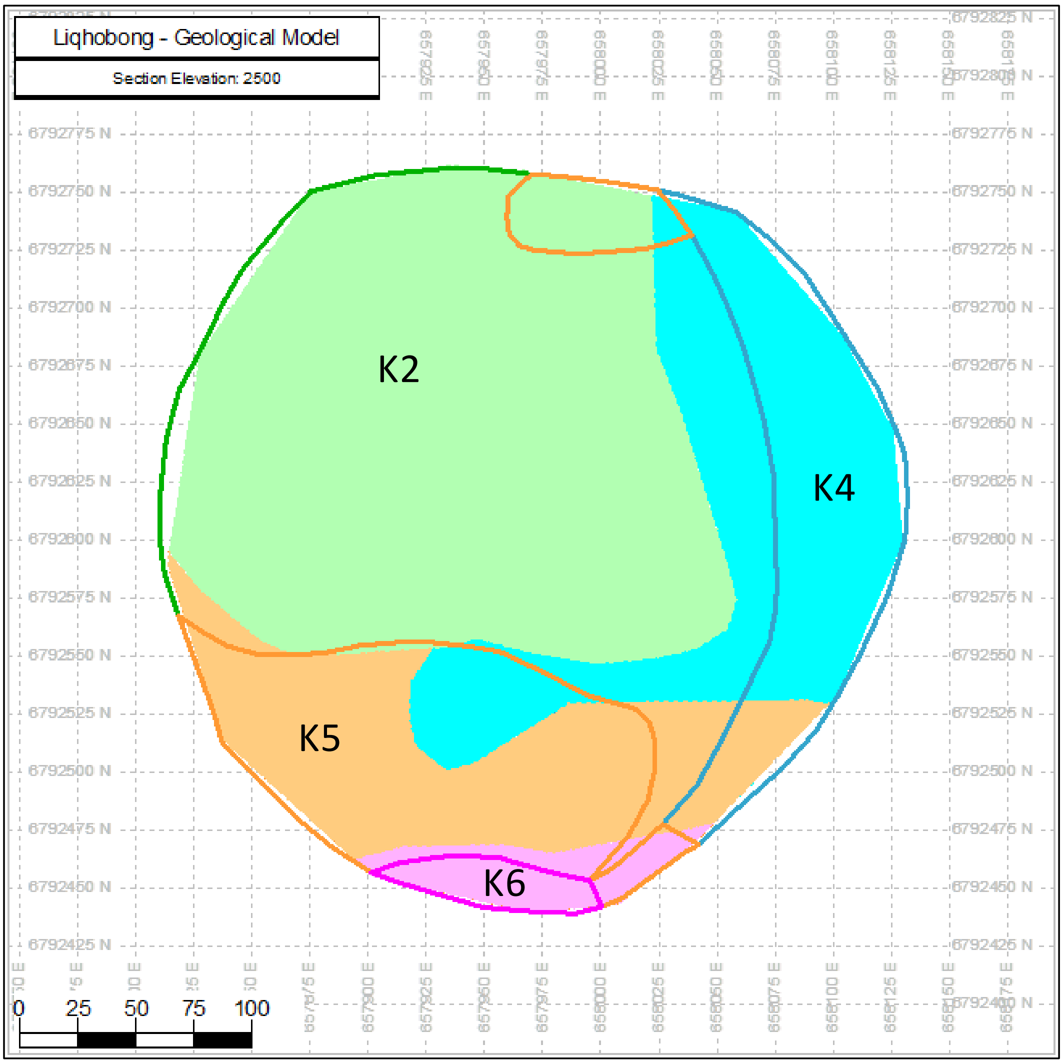



*Figure 9. Plan view of the 2014 (outlines) and 2009 (solid) geological model*

A significant proportion of “transition” material was logged as “K2+K5” towards the southern portion of K2 from approximately 2 480 masl and below. Given the trend in K5, the previous interpretation for K5 and the lack of data for supporting this being a separate facies, these samples were mostly included in the K5 facies. This portion however falls almost entirely in the Inferred portion of the pipe as it is generally of lower confidence both in terms of geology and grade.

The "finger" of K4 in the south of the pipe is no longer present with K2 also moving towards the west somewhat at the expense of K4. Other changes include minor modifications to the perimeter outline of the pipe as well, while the primary change is the reduction in volume of K6 due to a significant change in drill-hole logging interpretation.

The outer perimeter of the pipe has remained largely unchanged from 2009 except that the solids have been tapered from the last reliable information using the general trend observed in the upper portions of the pipe. Figure 10 demonstrates clearly where the previous model (2009 – black) had vertical walls below 2 340masl. The tapering in the 2014 pipe results in less volume at depth than in the 2009 model.

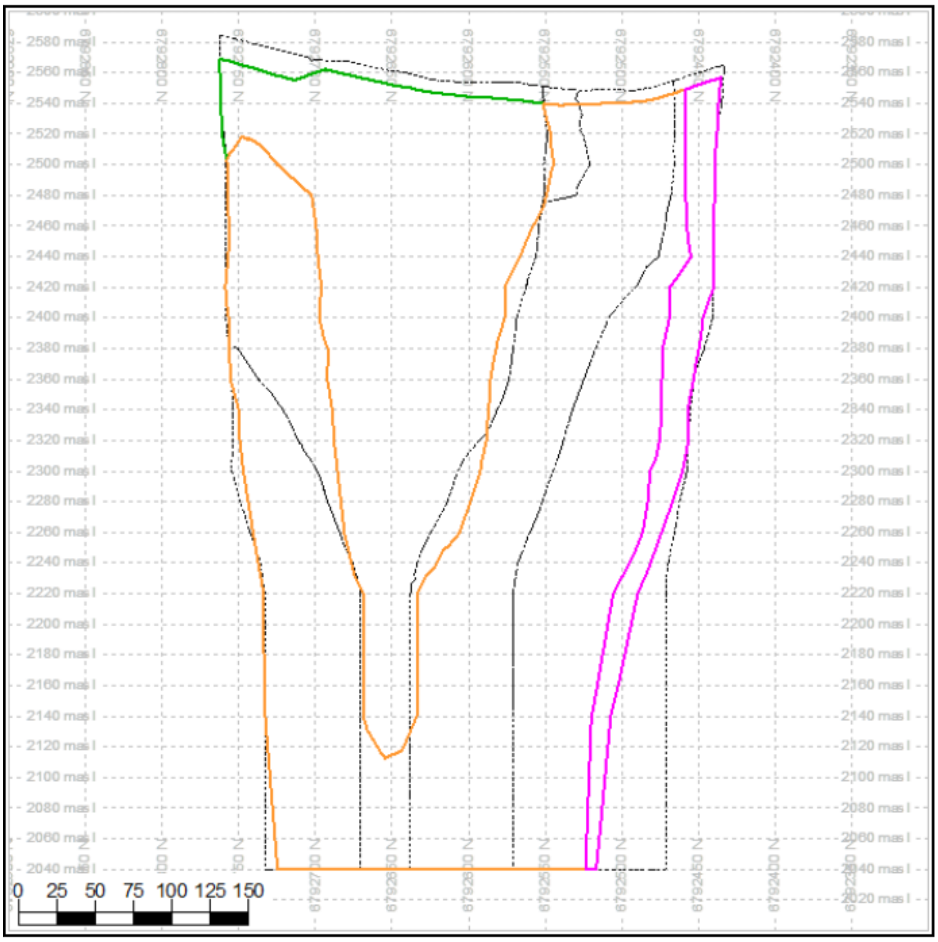



*Figure 10. Section looking east through the 2009 (black) and 2014 geological solids*

As a result of the latest geological logging and subsequent modelling exercise, the volume of K5 increased whilst the volume of K6 and K4 reduced as can be seen from Table 9 below. The overall reduction in volume compared to the 2009 model of 8.3% can be attributed to the introduction of an overburden surface, tapering of the pipe at depth, and depletions as a result of mining.

| | Volume (m³) | | | | |
|---|---|---|---|---|---|
| Facies | Total | Mined | In Situ | 2009 | % Difference |
| K2 | 12 962 100 | 264 300 | 12 697 900 | 12 916 300 | -1.69% |
| K4 | 1 582 100 | 157 800 | 1 424 300 | 2 460 100 | -42.10% |
| K5 | 16 268 500 | 220 900 | 16 047 600 | 11 266 100 | 42.44% |
| K6 | 1 523 900 | 12 300 | 1 511 600 | 7 908 600 | -80.89% |
| Total | 32 336 600 | 655 200 | 31 681 400 | 34 551 100 | -8.31% |

*Table 9. Volume reconciliation between the 2009 and 2014 models*

The solids (wireframe) model was populated with blocks sized 50m x 50m x 20m in the x, y and z dimensions respectively for the purposes of estimation. To ensure that the block model honours the solids model, a minimum sub-cell size of 5 x 5 x 2.5m was used. Table 10 below shows that the block model volume fairly represents the solid model volume.

| | K2 | K4 | K5 | K6 |
|---|---|---|---|---|
| Wireframe | 13 064 274 | 1 582 245 | 16 179 368 | 1 522 640 |
| 5x5x2.5 | 12 962 125 | 1 582 063 | 16 268 500 | 1 523 875 |
| % Difference | 0.79% | 0.01% | -0.55% | -0.08% |

*Table 10. Volume comparison of the geological solids model to the sub-cell block model*

## 4.2 Density Estimate

The availability of an updated density dataset made it possible to estimate density by means of Ordinary Kriging (Lohrentz and Bush, 2014). Variography for all facies proved somewhat poor with only K2 and K5 producing variograms. However, no horizontal spatial structure was observed and the “down the hole” variograms from each facies have been assumed to hold for all directions as an omnidirectional variogram. Neighbourhood analysis was undertaken for density in the same manner as for the grade variable. It should be noted that the vertical search range in all cases has been limited to 50m so as to reduce the effect of smoothing the estimates in the vertical direction.

The bench average estimates were compared back to the sample density data as part of the validation of the estimates. There is generally a good correlation between input sample data and estimated values. It is clear that the 2014 estimates represent a dramatic improvement over zonal averages applied previously which would not have been possible without the density sampling exercise undertaken by Firestone (Lohrentz and Bush, 2014).

Table 11 below shows the difference in density and tonnage per facies between the 2014 and 2009 models.

| 2014 In Situ Resource | | | | 2009 In Situ Resource | | | |
|---|---|---|---|---|---|---|---|
| Facies | Volume ($m^3$) | Tonnes | SG (tonnes/$m^3$) | Facies | Volume ($m^3$) | Tonnes | SG (tonnes/$m^3$) |
| K2 | 12 697 900 | 32 968 400 | 2.60 | K2 | 12 916 300 | 33 815 300 | 2.62 |
| K4 | 1 424 300 | 3 744 100 | 2.63 | K4 | 2 460 100 | 6 345 800 | 2.58 |
| K5 | 16 047 600 | 42 691 300 | 2.66 | K5 | 11 266 100 | 29 654 100 | 2.63 |
| K6 | 1 511 600 | 4 024 300 | 2.66 | K6 | 7 908 600 | 20 849 200 | 2.64 |
| Total | 31 681 400 | 83 428 100 | 2.63 | Total | 34 551 100 | 90 664 400 | 2.62 |

*Table 11. Tonnage reconciliation between the 2009 and 2014 models*

## 4.3 Grade Estimate

No new grade data was available so the same input data was used as in the 2009 estimate which are the sampling results from the 27 WDD holes described in sections 2 and 3 of this report. Grade estimation was undertaken by means of Ordinary Kriging. This was chosen as only a single data source (one sample support size) was available (the WDD samples). Estimation was undertaken in stones/$m^3$ as per Bush (2009). Following is a summary of the grade estimation process as described in Lohrentz and Bush (2014).

### 4.3.1 Compositing

The WDD data were sampled in 20m lifts. However there is some variability in the sample lengths, as the holes were sampled from the collar elevation down not taking cognisance of bench elevations. As such the data have been composited to 20m to coincide with the bench centroids of the block model. Compositing has resulted in alignment of samples with benches in the block model, which is a more ideal orientation for geostatistical estimation. Compositing of the data has had a marginal effect on the classical statistics of the grade variable (stones/$m^3$) as many samples were already 20m in length (Table 12).

| | Raw - 2014 | | | | | | | 20m Bench Composited - 2014 | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Facies | # Samples | Min | Max | Mean | Vari. | Std.Dev. | $C_v$ | # Samples | Min | Max | Mean | Vari. | Std.Dev. | $C_v$ |
| K2 | 123 | 2.76 | 25.37 | 8.74 | 16.00 | 4.00 | 0.46 | 118 | 2.88 | 23.67 | 8.69 | 12.79 | 3.58 | 0.41 |
| K4 | 29 | 5.31 | 17.13 | 9.96 | 8.90 | 2.98 | 0.30 | 29 | 5.31 | 17.13 | 9.86 | 7.75 | 2.78 | 0.28 |
| K5 | 56 | 1.83 | 22.30 | 12.05 | 25.45 | 5.04 | 0.42 | 53 | 1.89 | 20.08 | 12.02 | 23.44 | 4.84 | 0.40 |
| K6 | 6 | 6.58 | 15.72 | 11.99 | 8.89 | 2.98 | 0.25 | 6 | 6.58 | 15.72 | 11.96 | 8.56 | 2.93 | 0.24 |

*Table 12. Statistics comparing the raw and 20m bench composited WDD grade data*

### 4.3.2 Variography

The WDD sampling data was recoded to the latest geological solid model which has a larger proportion of K5 and less K4 and K6. Unlike the 2009 estimate, robust variograms of stones per $m^3$ were obtained for the two major facies K2 and K5 (Figures 11 and 12). Generation of variogram models for the K2 and K5 facies lends these two facies to being estimated with hard boundaries. Limited sampling in the K4 and K6 facies however resulted in no variograms being able to be modelled and as such first pass estimation of these facies was with soft boundaries. The K2 and K5 variograms were used for the K4 and K6 facies by means of scaling them to the variance of the data to be used for the estimation of these domains. Table 13 summarises the fitted variogram models for each of the four facies.

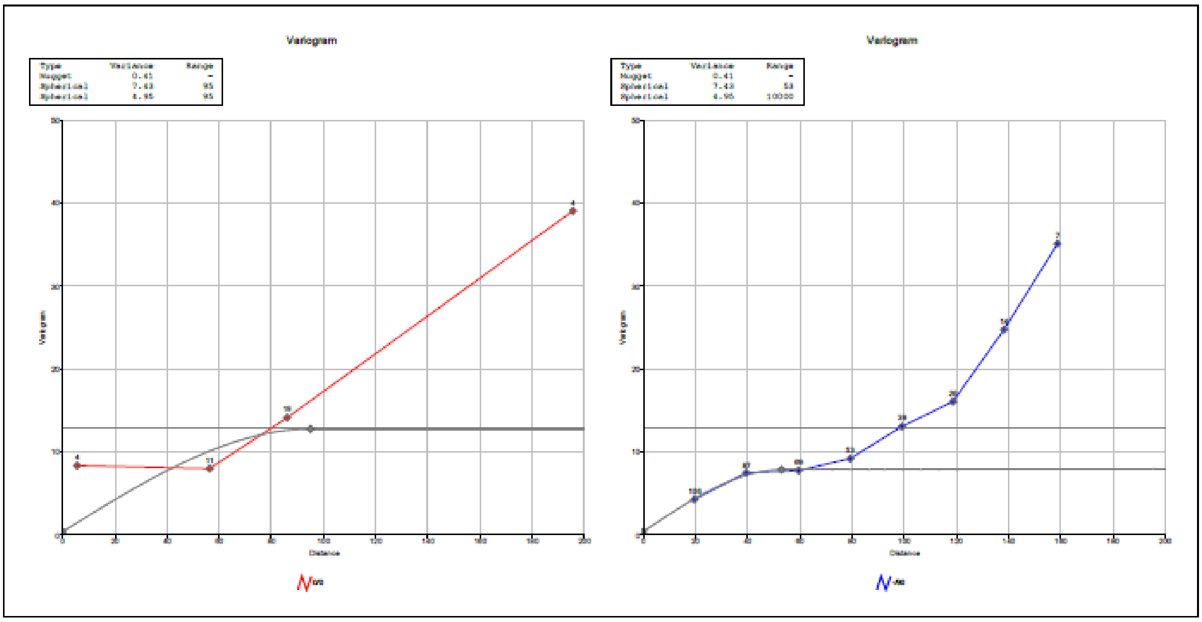



*Figure 11. Fitted variogram model for K2.*

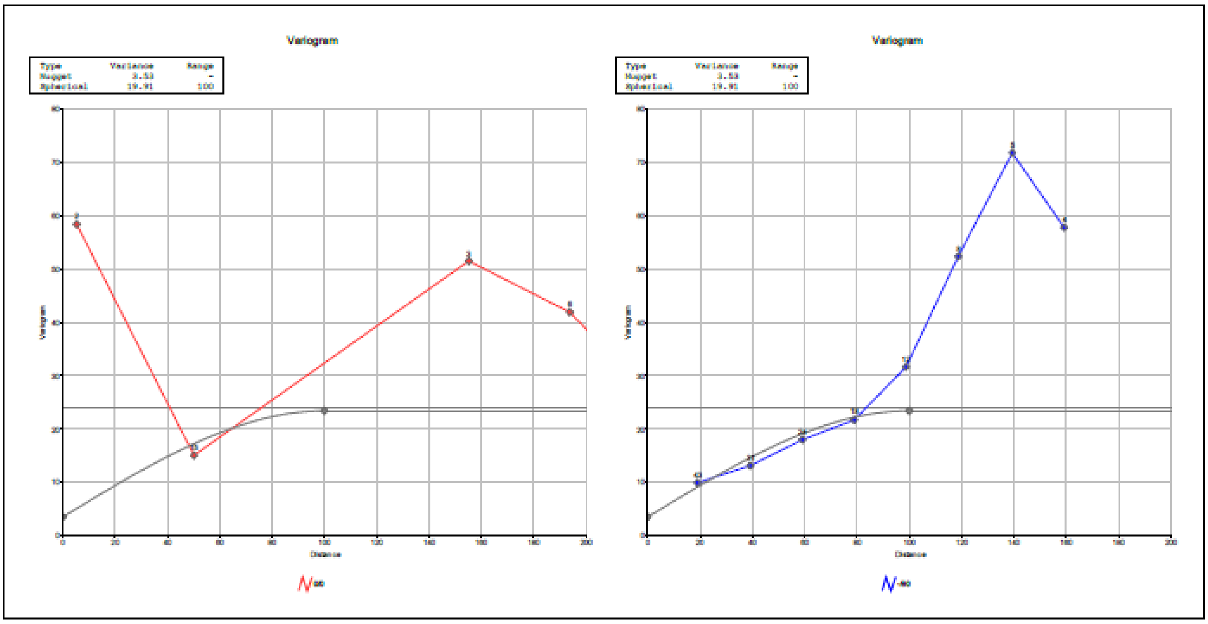



*Figure 12. Fitted variogram model for K5.*

| | | Structure 1 | | | | Structure 2 | | | |
|---|---|---|---|---|---|---|---|---|---|
| Facies | Nugget | $C_1$ | X | Y | Z | $C_2$ | X | Y | Z |
| K2 | 0.41 | 7.43 | 95 | 95 | 53 | 4.95 | 95 | 95 | 10000 |
| K4 | 0.39 | 7.02 | 95 | 95 | 53 | 4.68 | 95 | 95 | 10000 |
| K5 | 3.53 | 19.91 | 100 | 100 | 100 | - | - | - | - |
| K6 | 3.3 | 18.63 | 100 | 100 | 100 | - | - | - | - |

*Table 13. Summary of fitted variogram models for each facies.*

### 4.3.3 Neighbourhood analysis

Neighbourhood analysis was undertaken to ensure that kriged stones/$m^3$ estimates are optimised and unbiased compared to sample data. This is achieved by kriging the domain while varying the neighbourhood parameters and monitoring the various outputs until an optimal neighbourhood is achieved. A summary of the estimation methodology for the optimised estimates is given in Table 14. Neighbourhoods were deliberately truncated in the vertical direction relative to the range of the variograms in this direction so as to ensure that vertical variability as seen in sample bench plots was maintained.

| | | Range | | | # Samples | | |
|---|---|---|---|---|---|---|---|
| Facies | Samples | X | Y | Z | Min. | Max. | Shape |
| K2 | K2 | 76 | 76 | 30 | 3 | 40 | Ellipsoid |
| K4 | K2+K4 | 76 | 76 | 25 | 3 | 40 | |
| K5 | K5 | 100 | 100 | 30 | 3 | 40 | |
| K6 | K5+K6 | 80 | 80 | 30 | 3 | 40 | |

*Table 14. Summary of first pass kriging neighbourhoods per facies*

### 4.3.4 Validation

Kriged estimates of stones/$m^3$ were validated by means of comparing the bench average estimates to the samples as well as histograms of the estimates to that of the samples. Bench plots demonstrate that estimated data provide a good match to sample data from the individual facies and that there is no excessive smoothing of the estimates in the vertical direction relative to the sample data (Figures 13 and 14). The histograms reveal the estimates to be behaving normally with a reduction in variance and good reproduction of the mean grade. A 3D visual inspection of stones/$m^3$ estimates versus samples was also undertaken with no oddities being observed and as such the first pass estimates were considered robust.

### 4.3.5 Population of unestimated blocks

Optimised first pass estimates do not populate the entire block model for each facies. As such a second kriging pass was necessary to populate the remainder of blocks on the well sampled benches. This was accomplished by a combination of both local block estimation and the application of zonal average grades. Table 15 summarises the applied second pass neighbourhoods.

| | X | Y | Z | Min | Max | Shape |
|---|---|---|---|---|---|---|
| K2 | 150 | 150 | 30 | 3 | 8 | Ellipsoid |
| K4 | 150 | 150 | 25 | 3 | 10 | |
| K5 | 150 | 150 | 30 | 3 | 10 | |
| K6 | 150 | 150 | 30 | 3 | 10 | |

*Table 15. Summary of second pass estimation neighbourhoods.*

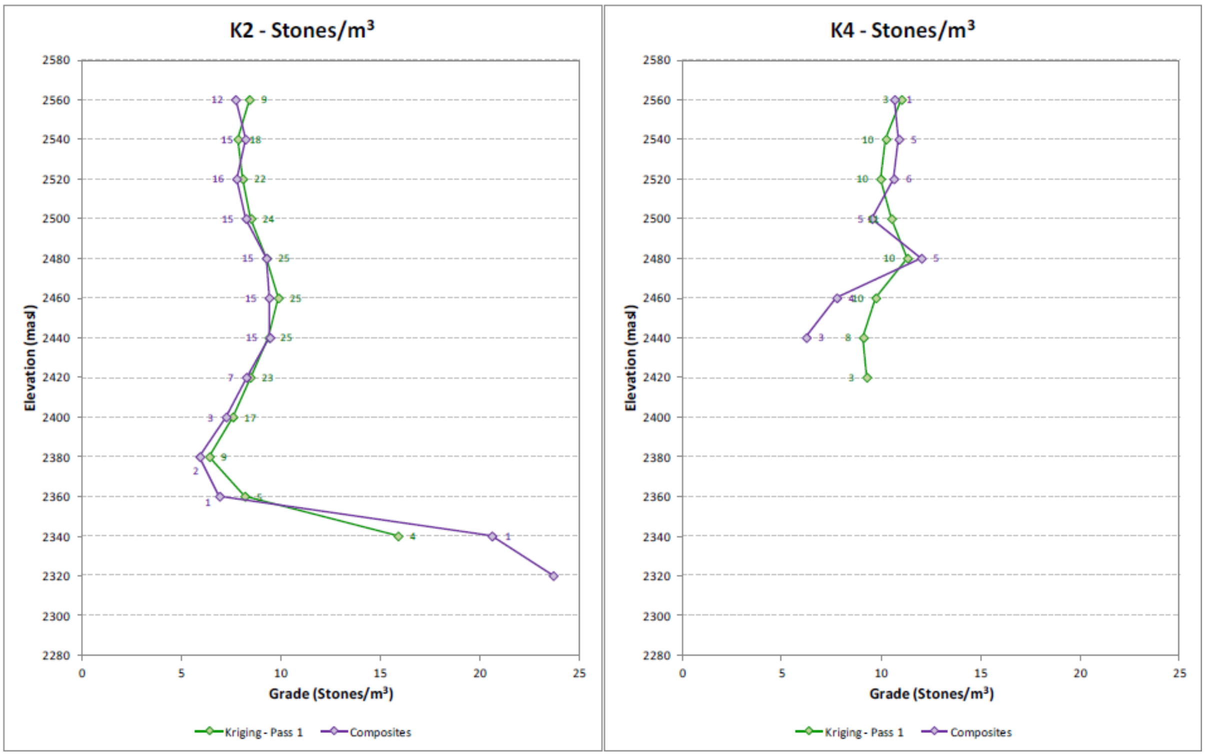



*Figure 13. Bench profile plot of composite sample data compared to optimised first pass kriged estimates for the K2 and K4 facies.*

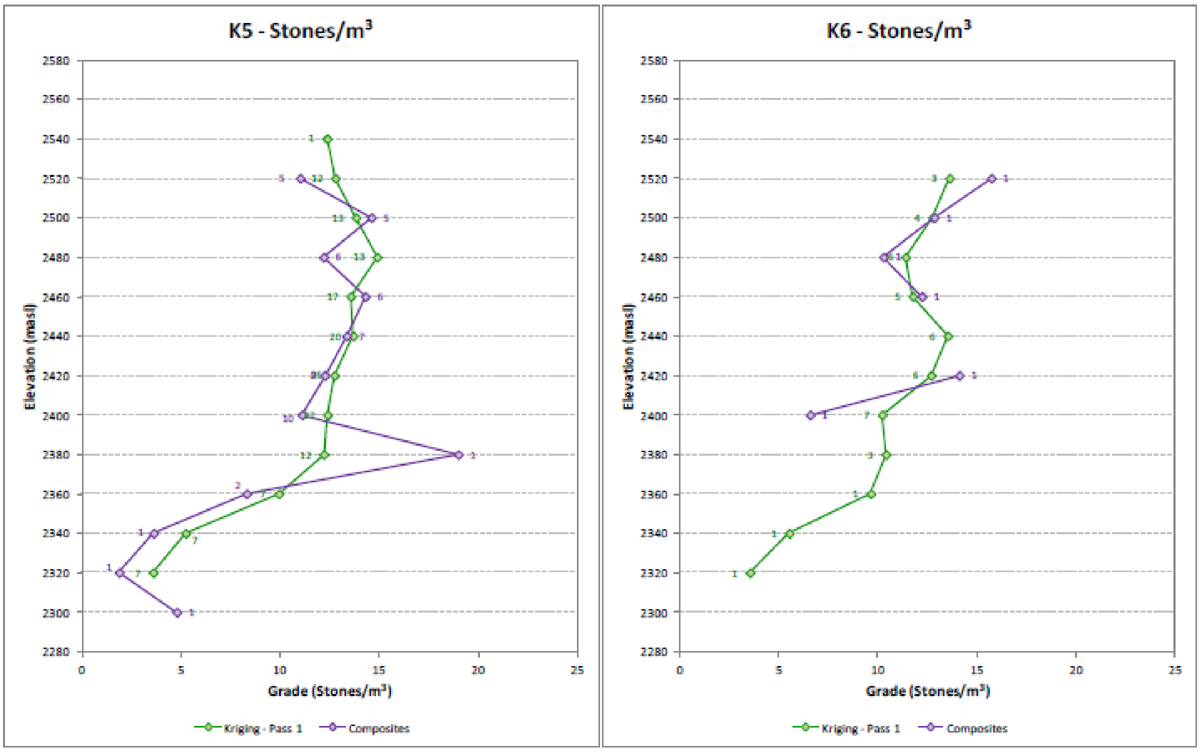



*Figure 14. Bench profile plot of composite sample data compared to optimised first pass kriged estimates for the K5 and K6 facies.*

### 4.3.6 Zonal grade application

Upon analysis of first pass estimated grades, it was concluded that estimates for certain benches could be considered less robust as they were based on fewer sample points. First pass estimates were reviewed and used to calculate a volume weighted mean grade and elevation from which to apply this as a zonal grade. Table 16 describes the benches to which zonal grades were applied, as well as the source benches for the zonal means.

| Facies | Benches Elevation Applied To | Source Bench Elevation (masl) | Zonal Grade (stones/$m^3$) |
|---|---|---|---|
| K2 | 2560 and Above | 2560 + 2540 | 7.96 |
| | 2380 and Below | 2420 + 2400 | 8.06 |
| K4 | 2650 and above - ALL | 2560 + 2540 | 10.35 |
| | 2460 and below | 2480 + 2460 | 10.61 |
| K5 | 2520 and Above | 2520 + 2500 | 13.32 |
| | 2380 and Below | 2400 + 2380 | 12.33 |
| K6 | 2520 and Above leave first pass | 2520 + 2500 | 13.17 |
| | 2400 and below | 2420 + 2400 | 11.47 |

*Table 16. Summary of zonal estimates and sources*

### 4.3.7 Application of average stone size

Carats per $m^3$ per block was calculated by applying the average stone size per facies derived from the WDD data. WDD drill-hole data contained information relating to total carats per sample in addition to the stones data used for the primary estimate. These data were used to calculate an average stone size per facies (Table 17) by summing the carats and stones per facies. Both K4 and K6 contained very few data points and as such the same soft boundaries as applied during estimation were applied here to calculate an average stone size.

| Facies | Source | Average Stone Size (cts/stn) |
|---|---|---|
| K2 | K2 | 0.075 |
| K4 | K2+K4 | 0.074 |
| K5 | K5 | 0.067 |
| K6 | K5+K6 | 0.069 |

*Table 17. Average stone size per facies from the WDD data*

### 4.3.8 Grade adjustment Factors

The new Main Treatment Plant (MTP) is unlikely to replicate the recovery process of the WDD samples and some form of correction to the resource grade estimates was necessary. This was done in two steps:

1. modelling from WDD to Bulk samples to correct for comminution and final recovery; as well as
2. changing the bottom cut off from 1.00mm to 1.25mm to align with the MTP.

Typically the WDD size frequency distribution is finer than the Bulk Sample equivalent as the aggressive comminution process of drilling would liberate the finer sizes while the smaller sample size of the WDD would also preclude the occurrence of larger stones in the sample. The expected “production” size frequency distribution was modelled based on the Bulk Sample distribution with the boart excluded. This is a big change compared to the 2009 estimate which included the boart component and the result is a reduction in grade and carats especially for K5 (Figure 15).

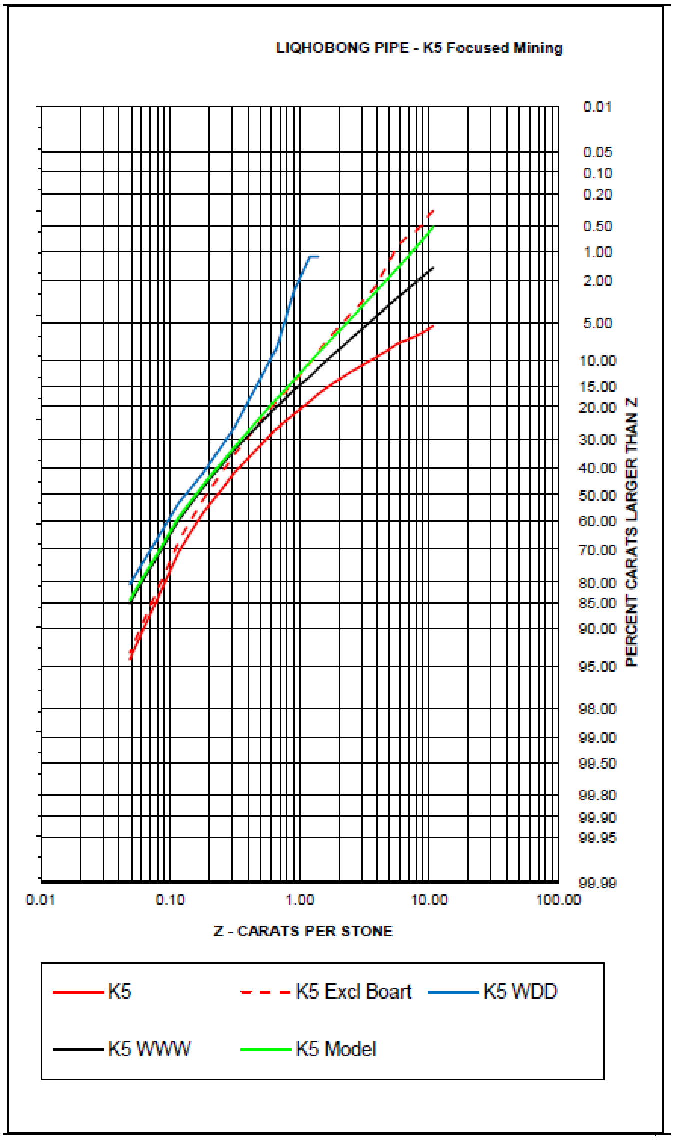



*Figure 15. Size frequency distributions for K5*

The adjustment factor per facies cannot readily be determined from size frequency distributions alone as differences in distribution can be the result of adding or subtracting from the finer or coarser sieve sizes; i.e. a “finer” distribution can result from either more small stones or proportionally less larger stones. It is therefore necessary to consider the differences between distributions at the local sieve size level. This is done using grade – size plots.

It can be seen from Figure 16 below that when the K5 WDD and K5 model grade per sieve size is compared, that the model plot lies above the WDD for all sieve sizes and that the difference between the two sample processes increases as the sieve size increases. The grade gap between the WDD and model is expected as the average grades are 30cpht and 44cpht, respectively. Thus to identify the more subtle differences in each sieve size it is necessary to standardise (i.e. normalise) the two plots to the same grade. It is then possible to identify the differences in stone grade per sieve size and generate factors per size to in effect, place the WDD grade – size plot on the model and mimic the recovery efficiency of the WDD samples as if they were processed as bulk samples.

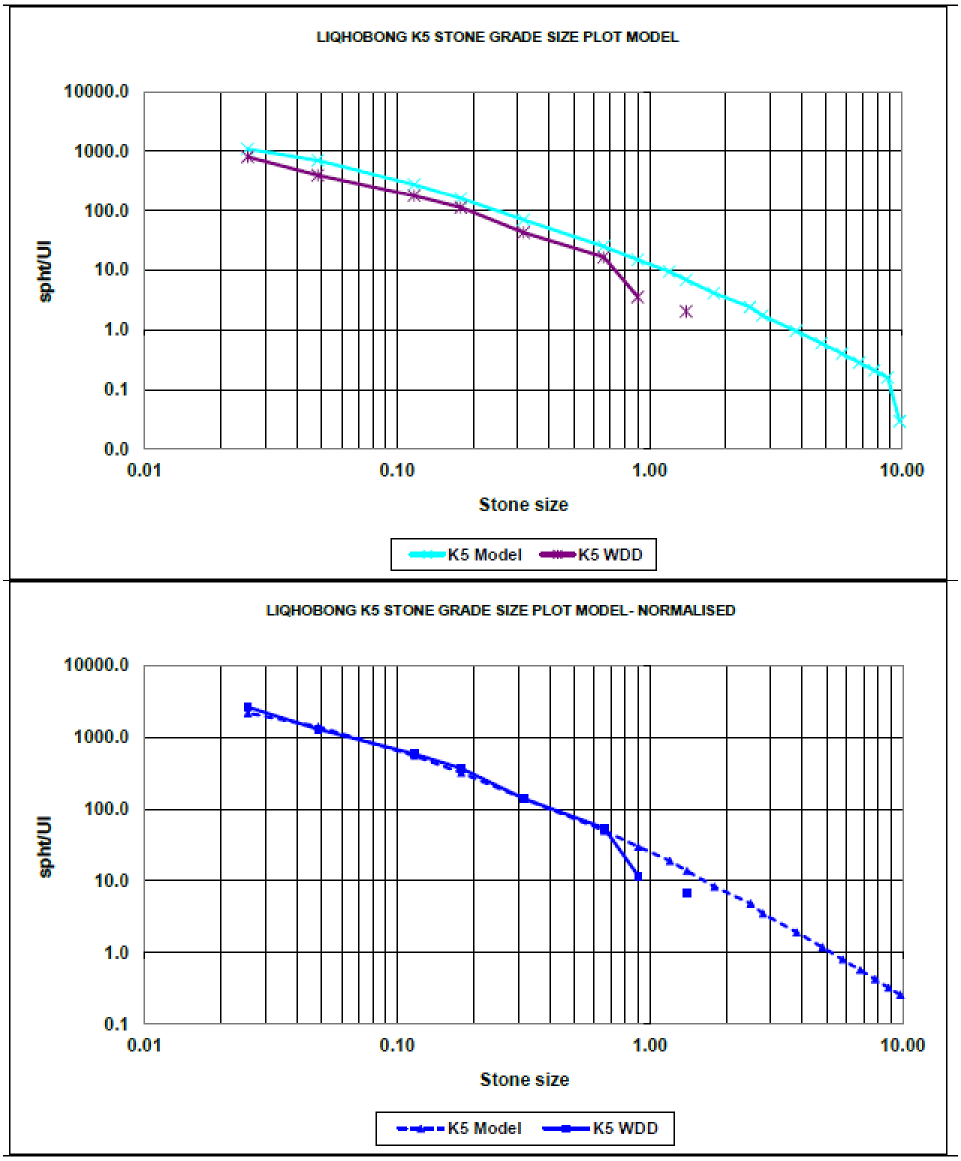



*Figure 16. K5 grade size plots, original and normalised.*

The above was done for all the facies. The factors are lower and less variable than those determined in 2009 and reflect the generally finer size frequency distributions of the models as a result of excluding the boart content (Table 18).

| Geology | Factor | % |
|---|---|---|
| K2 | 1.03 | 2.6 |
| K4 | 1.02 | 2.3 |
| K5 | 1.03 | 3.2 |
| K6 | 1.03 | 3.1 |

*Table 18. WDD to bulk sample factors*

Adjusting for the bottom cut off involves interpolation between sieve sizes which bracket the proposed bottom cut off. A 1.25mm bottom cut off lies within the +3 to -5 sieve size. Interpolation between the two sizes for 1.25mm provides an estimate that 65% of the +3 to -5 sieve size diamonds would remain above 1.25mm with 35% of the +3 to -5 sieve size diamonds passing through a 1.25mm aperture. It is therefore necessary to adjust the +3 sieve size of the model for each geological unit to correct for the increased bottom cut off. The process is shown in Table 19 and results in a modifying factor of 0.95.

| Geology | +3 Sieve % | Adjustment | Adjusted +3 Sieve % | Modifying Factor |
|---|---|---|---|---|
| K2 | 15.57 | 65% | 10.12 | 95% |
| K4 | 15.261 | 65% | 9.92 | 95% |
| K5 | 15.512 | 65% | 10.083 | 95% |
| K6 | 15.629 | 65% | 10.159 | 95% |

*Table 19. Bottom-cut off adjustments*

The two modifying factors (WDD to model and bottom cut off) are cumulative; thus the final modifying factors to estimate grade in the proposed Main Treatment Plant are summarised in Table 20.

| Geology | WDD to Model | Bottom cut off | Combined Modifying factor |
|---|---|---|---|
| K2 | 1.03 | 0.95 | 0.97 |
| K4 | 1.02 | 0.95 | 0.97 |
| K5 | 1.03 | 0.95 | 0.98 |
| K6 | 1.03 | 0.95 | 0.97 |

*Table 20. Combined factor per geological facies*

## 4.4 Diamond Resource Inventory

The in situ 2015 Diamond Resource inventory depleted to the April 2014 mining surface is depicted below (Table 21). To a depth of 2370 masl (2467 masl on the corrected datum) which is the extent of the majority of the WDD holes, the Liqhobong Indicated Resource is estimated to comprise 9.5 Mct in 35 Mt at an average grade of 27 cpht at a bottom cut off of 1.25mm. Below the Indicated Resource down to a depth of approximately 540m below surface an Inferred level of confidence Diamond Resource containing some 13.5 Mct in 48 Mt at an average grade of 28 cpht has been estimated. The total Diamond Resource inventory for the Liqhobong Main Pipe as at September 2015 is estimated to be 83 Mt containing 23 Mct at a grade of 28 cpht.

For comparison to the 2009 Diamond Resource inventory refer to Table 4.

| | | Total Resource | | | | | |
|---|---|---|---|---|---|---|---|
| 20m Bench Centre (New Datum) | 20m Bench Centre (Mine datum) | Volume ($m^3$) | Tonnes | SG (tonnes/$m^3$) | Carats | Grade | |
| | | | | | | Carats/$m^3$ | CPHT |
| 2677 | 2580 | 30 400 | 79 100 | 2,6 | 21 300 | 0,7 | 27 |
| 2657 | 2560 | 311 500 | 805 000 | 2,58 | 196 800 | 0,63 | 24 |
| 2637 | 2540 | 954 100 | 2 457 500 | 2,58 | 597 400 | 0,63 | 24 |
| 2617 | 2520 | 1 501 300 | 3 848 000 | 2,56 | 976 100 | 0,65 | 25 |
| 2597 | 2500 | 1 599 600 | 4 122 100 | 2,58 | 1 100 400 | 0,69 | 27 |
| 2577 | 2480 | 1 576 900 | 4 102 400 | 2,6 | 1 163 100 | 0,74 | 28 |
| 2557 | 2460 | 1 552 500 | 4 083 600 | 2,63 | 1 169 300 | 0,75 | 29 |
| 2537 | 2440 | 1 541 400 | 4 073 000 | 2,64 | 1 177 600 | 0,76 | 29 |
| 2517 | 2420 | 1 529 300 | 4 022 700 | 2,63 | 1 088 500 | 0,71 | 27 |
| 2497 | 2400 | 1 490 100 | 3 917 100 | 2,63 | 1 006 200 | 0,68 | 26 |
| 2477 | 2380 | 1 459 800 | 3 853 800 | 2,64 | 1 036 100 | 0,71 | 27 |
| **Total** | **Total** | 13 546 800 | 35 364 200 | 2,61 | 9 532 800 | 0,7 | 27 |
| 2457 | 2360 | 1 417 600 | 3 725 900 | 2,63 | 1 022 300 | 0,72 | 27 |
| 2437 | 2340 | 1 368 300 | 3 598 400 | 2,63 | 999 200 | 0,73 | 28 |
| 2417 | 2320 | 1 335 600 | 3 530 400 | 2,64 | 981 900 | 0,74 | 28 |
| 2397 | 2300 | 1 286 200 | 3 408 800 | 2,65 | 955 300 | 0,74 | 28 |
| 2377 | 2280 | 1 237 900 | 3 283 800 | 2,65 | 927 300 | 0,75 | 28 |
| 2357 | 2260 | 1 184 600 | 3 138 100 | 2,65 | 892 300 | 0,75 | 28 |
| 2337 | 2240 | 1 132 500 | 2 992 100 | 2,64 | 855 000 | 0,75 | 29 |
| 2317 | 2220 | 1 085 800 | 2 865 800 | 2,64 | 820 600 | 0,76 | 29 |
| 2297 | 2200 | 1 053 600 | 2 799 600 | 2,66 | 795 300 | 0,75 | 28 |
| 2277 | 2180 | 1 028 100 | 2 737 000 | 2,66 | 774 900 | 0,75 | 28 |
| 2257 | 2160 | 999 600 | 2 660 500 | 2,66 | 752 200 | 0,75 | 28 |
| 2237 | 2140 | 972 800 | 2 588 800 | 2,66 | 731 100 | 0,75 | 28 |
| 2217 | 2120 | 944 400 | 2 513 500 | 2,66 | 710 900 | 0,75 | 28 |
| 2197 | 2100 | 914 300 | 2 433 600 | 2,66 | 689 300 | 0,75 | 28 |
| 2177 | 2080 | 890 100 | 2 369 700 | 2,66 | 672 700 | 0,76 | 28 |
| 2157 | 2060 | 861 900 | 2 294 900 | 2,66 | 652 700 | 0,76 | 28 |
| 2137 | 2040 | 421 700 | 1 122 900 | 2,66 | 319 700 | 0,76 | 28 |
| | **Total** | 18 134 600 | 48 063 800 | 2,65 | 13 552 800 | 0,75 | 28 |
| | **Grand Total** | 31 681 400 | 83 428 000 | 2,63 | 23 085 600 | 0,73 | 28 |

*Table 21. Diamond Resource inventory of the Liqhobong Main Pipe.*

*Combined factors of Table 20 applied and depleted to the latest mining surface. BCO = 1.25mm.*

## 4.5 Revenue Estimate

The determination of average price is a combination of two variables namely diamond size, expressed in terms of size frequency distribution (SFD), and diamond assortment, which reflects the contribution of model, colour and quality and is expressed as a US dollar value per size class.

As part of the Diamond Resource update, First Element Diamond Services was requested to re-price the Liqhobong Pilot Plant production parcel (Erikson, 2013). The 2014 revenue estimate was based on a combination of SFD per facies and the assortment ($/carat/sieve size) data received from First Element reflecting the market conditions for the Liqhobong assortment categories during August 2014.

The SFD per facies (excluding boart) was modelled by Bush in 2012 and has not been changed. The same SFD models were used in the factor analysis section (4.3.7) of this report. The modelled size frequency distributions and production data (January 2012 to October 2013) are shown in Figure 17. It can be seen that the adjusted, to a 1.25mm bottom cut off, model size frequency distributions provide a reasonable fit to the historical production data at the larger stone sizes while the models show a greater proportion of smaller stones than production.

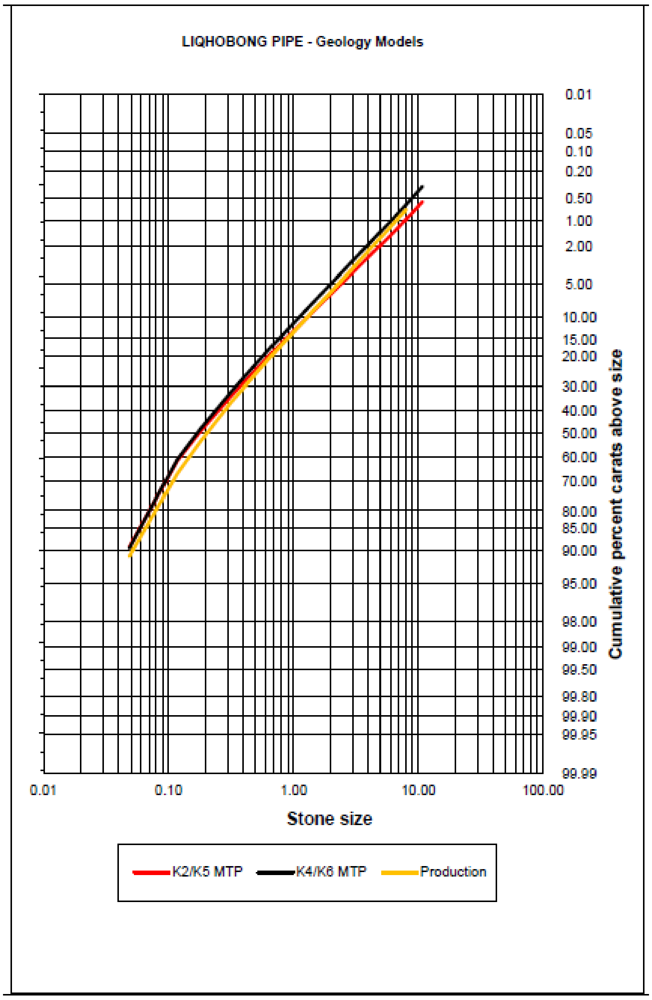



*Figure 17. Comparison of modelled SFD's to Pilot Plant production SFD.*

The August 2014 assortment data was modelled to compensate for the larger sizes that are generally under sampled and not representative of the entire assortment at those sizes. The -3 to +11 DTC sieve sizes have enough stones per size and therefore does not require modelling whereas a linear model was used to estimate assortment at sizes greater than +11 DTC sieve (Figure 18).

By combining the SFD models with the assortment model the modelled average price for each of the Liqhobong geological units can be obtained. The modelled values are further adjusted to be aligned to the new Main Treatment Plant bottom cut off of 1.25mm as reflected below (Table 22).

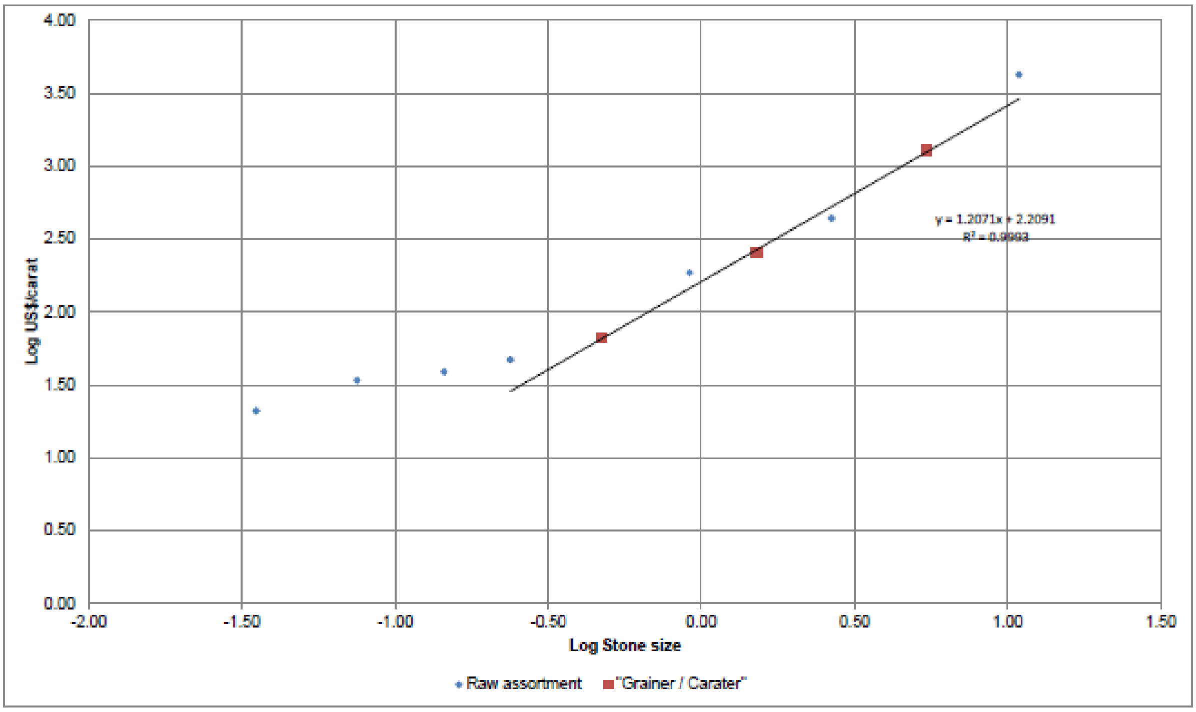



*Figure 18. Assortment modelling.*

| | K2 | K4 | K5 | K6 |
|---|---|---|---|---|
| Average price ($/c) | 124 | 106 | 123 | 106 |

*Table 22. Average $/carat at 1.25mm bottom cut off.*

In addition, similar to the work done in 2013 (Bush, 2013), the revenue estimate was modelled to allow for the extrapolation of values up to 100 carat stones. For both the SFD and assortment models the extrapolation involves applying the log normal relationships to larger stone sizes. Therefore in the case of the size distributions the +10.8 carat percentage is re-allocated between +10.8 carats to +100 carats. A similar process is involved in the assortment model. This resulted in an average price per facies as summarised below in Table 23.

| | K2 | K4 | K5 | K6 |
|---|---|---|---|---|
| Average price ($/c) | 134 | 115 | 133 | 115 |

*Table 23. Average $/carat per facies at 1.25mm BCO including large stone potential*

Applying the latest revenue estimates in the Mine Plan resulted in a weighted un-escalated average diamond price of US$131/ct, which has equated to an average escalated diamond price over the life of mine of US$165/ct, representing an increase of approximately 13% from the 2013 DFS. Given the current market conditions, the 2014 revenue estimate has not been escalated until 2017 in the financial model.

Compared against the previous revenue estimate (Bush, 2013), the average price of the Liqhobong diamond population has increased by some 25% (from $107/carat to $134/carat) for K2 and is a function of increasing the bottom cut off from 1.00mm to 1.25mm (5%) and improvements in the assortment valuation including large stone potential (20%).

Given the current uncertain market conditions, the August 2014 revenue estimate will be used in the financial model but not escalated until 2017. The weighted unescalated average diamond value for the 2015 mine plan and financial model is $131 per carat over the life of mine.

## 4.6 Classification

For the 2009 estimate Z Star regarded only the grade estimate as being at the Indicated level of confidence down to 2467 masl. Due to additional information made available for the 2014 estimate the geology, volume, density and revenue models are now deemed to be at the Indicated level of confidence down to 2467 masl. (Table 24). Similar to the 2009 estimate, all blocks below 2467 masl down to 2137 masl are considered to be at the Inferred level of confidence due to a reduction of sampling points from WDD drilling and hence the application of zonal grades as discussed in section 4.3.6. Numbers of samples per bench drop off sharply below 2467 masl, particularly in the K2 and K5 facies which further supports the Inferred Resource classification.

| Criteria | Classification | Comments |
|---|---|---|
| Geology | Indicated | Detailed re-loggging of the core including clast counts and dilution analysis has resulted in robust facies classification and an improved understanding of the kimberlite emplacement model. |
| Volume | Indicated | Re-logging of core data and re-interpretation of the logs has resulted in marginal overall volume difference but at higher confidence. |
| Density | Indicated | New density measurements lead to the derivation of local block estimates that are at a much higher level of confidence than previous zonal estimates. |
| Grade | Indicated | Robust estimates for the majority of geological facies based on WDD sampling has been achieved. |
| Revenue | Indicated | SFD's from bulk sampling and large production parcel on which to base assortment model provide solid revenue estimates. |

*Table 24. Classification of Liqhobong Main Pipe Diamond Resource.*

Figure 19 shows an East-West cross section through the main pipe with the WDD holes plotted and the Indicated – Inferred boundary.

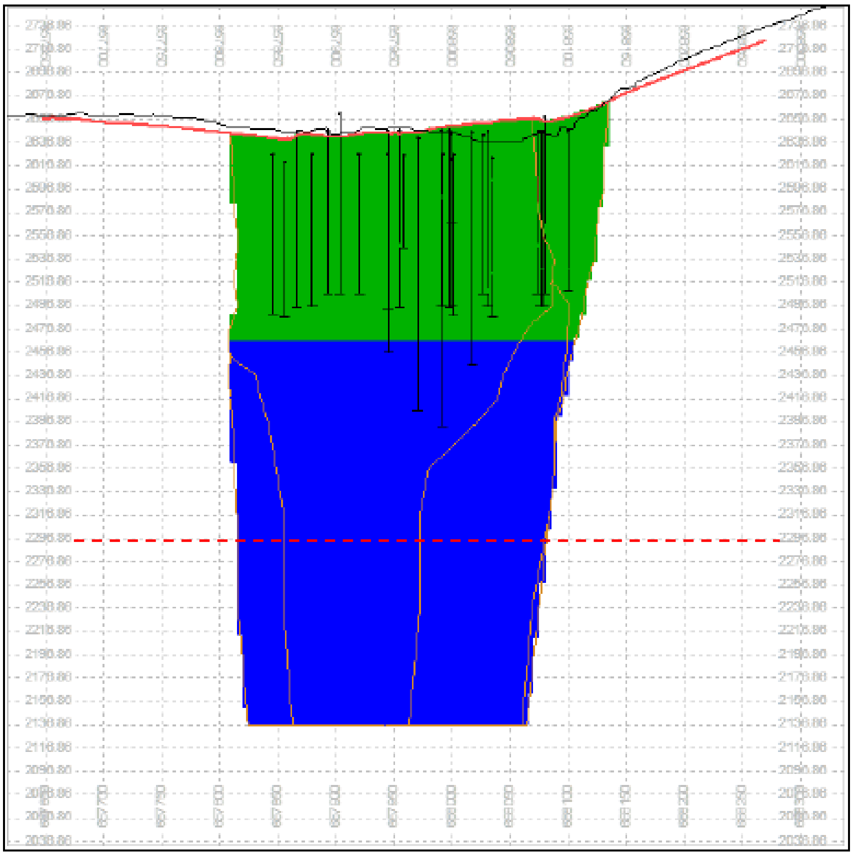


*Figure 19. East-West cross section through the Liqhobong block model.*

*(green = Indicated and blue = Inferred) with WDD holes and April 2014 surface shown.*

## 4.7 Reasonable Prospects for Eventual Economic Extraction

A Mineral Resource should meet the minimum requirement of having reasonable prospects for eventual economic extraction. LMDC's latest mine plan (Section 5) considers a split shell mine design that consists of 3 cuts that extends down to approximately 355m from surface. The red dotted line in Figure 19 illustrates the depth to which the pipe will be mined relative to the Indicated/Inferred boundary. Clearly the Indicated and Inferred Resource above the red dotted line has excellent prospects of being extracted as it is the focus of the Liqhobong Mine Expansion study. The portion of the Inferred Resource below the red dotted line requires some consideration.

The latest mine plan indicates that it will take approximately 15 years to mine down to 355m. The current Inferred boundary extends down to approximately 523m and the deepest vertical core hole terminated in kimberlite at 650m from surface.

The Whittle pit optimisation graph below (Figure 20) shows that the NPV is quite robust even if the amount of waste tonnes are increased by a factor of around 2 to around 220 million tonnes which bodes well for the possibility of another waste cut in the future dependant on the diamond price and cost assumptions at the time.

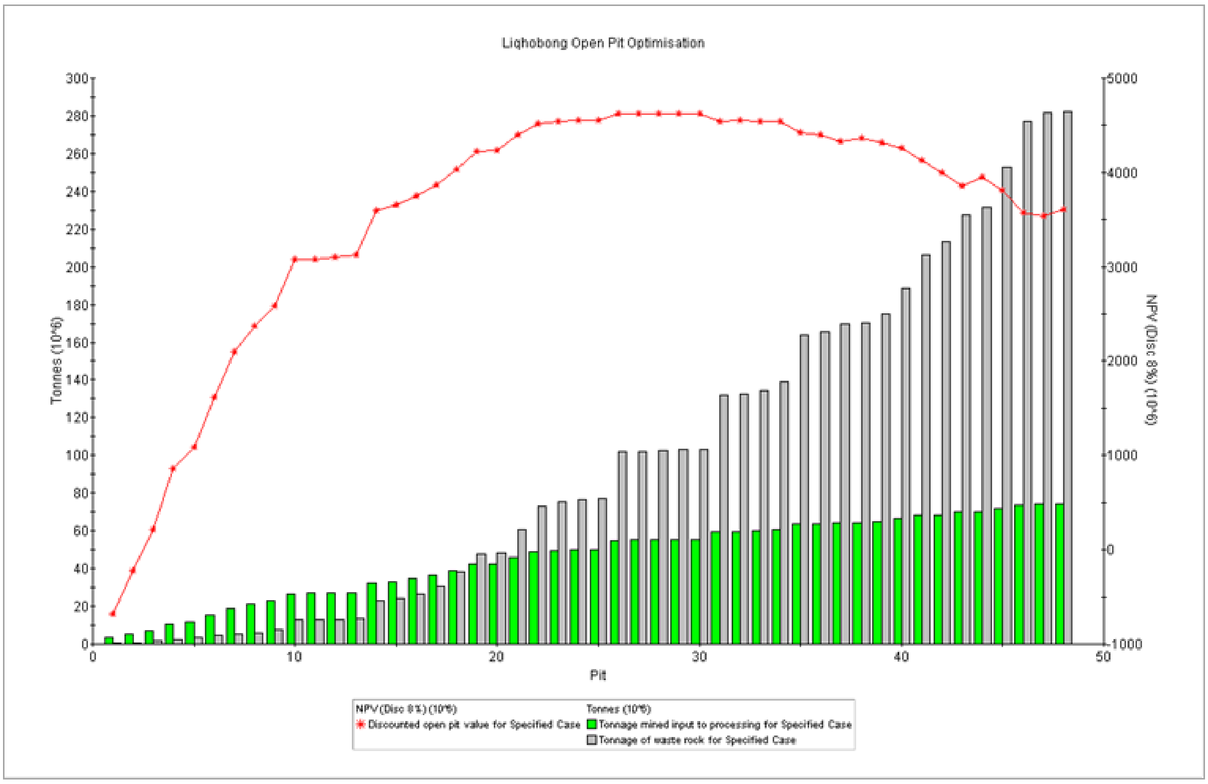



*Figure 20. Whittle pit optimisation plot*

With regard to an underground scenario, an analysis of some South African underground diamond mines (Petra and Lace Mines) show an average cost per tonne of around R200 to R250 per tonne compared to Liqhobong's Rand per tonne in the ground of around R460 assuming the Inferred grade of 28 cpht, revenue of $132 per carat and an exchange rate of R12.50. The difference allows for capex associated with underground development to be repaid. In support of a possible underground scenario it is reasonable to assume that with further drilling one would be able to extend the amount of Inferred Resource below 523m depth.

Therefore given the supply-demand scenario mentioned in Section 10 it is not unreasonable to assume that after 15 years the rest of the existing Inferred Resource down to 523m, which by then would have been converted to Indicated Resource, will be mined either by further waste cuts in an open pit scenario or by underground mining methods. Especially given the fact that there is more K5 at depth relative to K2 and that K5 has a higher $/tonne value, i.e. $41/tonne versus $31/tonne for K2 at current diamond prices. Also given the fact that the existing infrastructure including processing plant would be long paid off by that stage.

## 4.8 Reconciliation against previous Diamond Resource estimate

### 4.8.1 <u>2009 Grade Estimate</u>

As stated before, the 2012 Liqhobong Definitive Feasibility Study as well as the 2013 Updated DFS results were based on a Diamond Resource estimate prepared by Z Star Mineral

Resource Consultants dated 31 August 2009 (Bush and Grills, 2009). Z Star was subcontracted by A.C.A. Howe to conduct the grade estimate that formed part of the Competent Person Report (CPR) compiled on behalf of Kopane (Leroux, 2010).

The 2009 estimate was based on the following inputs:

- Wide diameter drillholes (WDD) – the primary spatial data for grade estimation were 27 x 0.445m diameter WDD holes drilled between February 2008 and November 2009. The WDD drillholes were drilled vertically (-90°) and on a nominal 50m by 50m grid pattern across the entire Main Pipe.
- Bulk samples for each of the 4 kimberlite facies that were collected by Kopane in 2008 and used primarily for revenue and size frequency distribution (SFD) analyses. The bulk samples contained a higher proportion of boart carats than the WDD samples.
- Geology solid and block model supplied by A.C.A. Howe that extended down to a depth of 2040 masl.
- Limited density data was available. A single density figure was used for each of the four facies.

When modelling diamond grade for the different kimberlite facies, Z Star applied modifying factors in the grade estimation process to align the grade and revenue SFD's and compensate for a deficiency of boart diamonds in WDD samples compared to the surface bulk samples. This lead to a substantial increase in grade for some of the facies: K2 (+8%); K4 (+1%); K5 (+19%) and K6 (+9%).

With regards to classification, Z Star considered the grade component of the resource estimate to be Indicated to a depth of 2370 masl (2467 masl according to the corrected survey elevations) and Inferred thereafter (see Table 25 below and Table 4 for the detailed inventory).

| Resource Classification | Elevation masl | Volume M m³ | Density T/m³ | Tonnes | Grade cpht | Carats |
|---|---|---|---|---|---|---|
| Indicated | Surface to 2,370 | 14.806 | 2.6 | 38.556 | 31 | 12.042 |
| Inferred | 2,370 to 2,040 | 19.745 | 2.61 | 51.471 | 34 | 17.655 |
| **Total** | | **34,551** | **2.61** | **90.027** | **33** | **29.697** |

Bottom cut-off 1 mm

Source: Bush and Grills (2009)

*Table 25. 2009 Diamond Resource Estimate*

### 4.8.2 2013 Revenue estimate

The previous revenue estimate was conducted by Z Star in October 2013 (Bush, 2013). The output of this exercise was used to inform the 2013 DFS update which was released on 5 November 2013 which assumed a base case average price of US$107 per carat. The 2013 estimate was based on the following inputs:

- Modelled SFD's generated in the Bush (2012) report derived from the surface bulk samples per facies collected by Kopane in 2008.

- Parcel tender data per sieve class based on Pilot Plant goods sold up to August 2013. The more recent production parcels were used to determine the assortment value per sieve class.

Z Star combined the modelled SFD with the modelled assortment data to derive average prices for each of the geological facies ranging from US$85 per carat for K6 to US$99 per carat for K2. To include the large stone potential of the Liqhobong mine, both the SFD and assortment models were extrapolated from +10.8 carats to +100 carat size. This resulted in a range of possible $/carat values as depicted in the Table 26 below:

| | | Assortment | |
|---|---|---|---|
| | | **Expected** | **Upside** |
| **Size** | **Expected** | 107 | 121 |
| | **Upside** | 125 | 156 |

*Table 26. Large stone potential for the K2 facies at a 1mm BCO.*

The value of $107/ct was used in the Base Case of the November 2013 DFS update.

4.8.3 Reconciliation against the 2015 Diamond Resource estimate

- *Tonnes* – The re-logging exercise has resulted in an increase in K5 mainly at the expense of K6. The graph below (Figure 21) shows an overall decrease in tonnes of -8% which is as a result of:
  - volume changes associated with the new geological model mainly related to tapering of the pipe at depth and the inclusion of an overburden surface which was absent in the 2009 model
  - Depletions as a result of mining

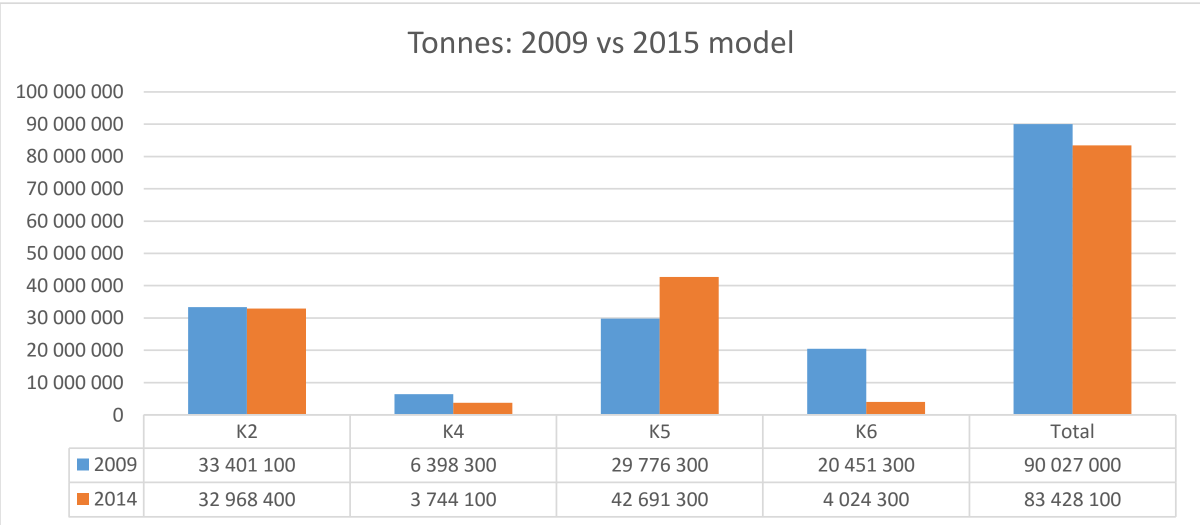
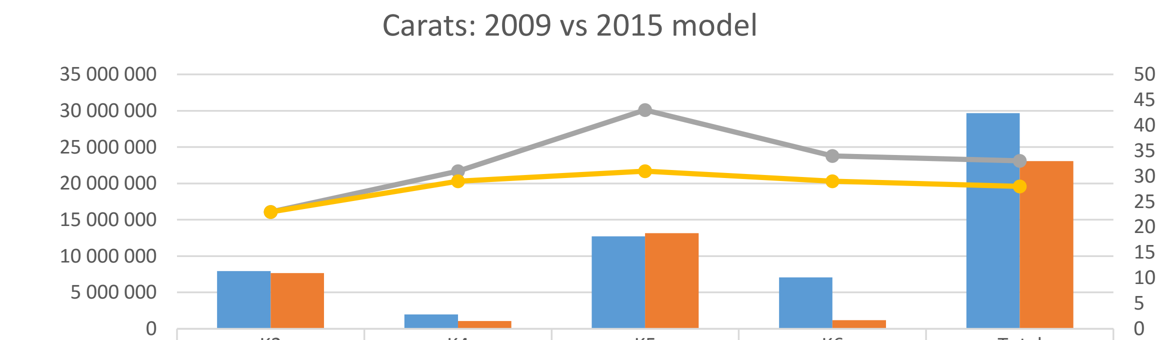



*Figure 21. Tonnage reconciliation*

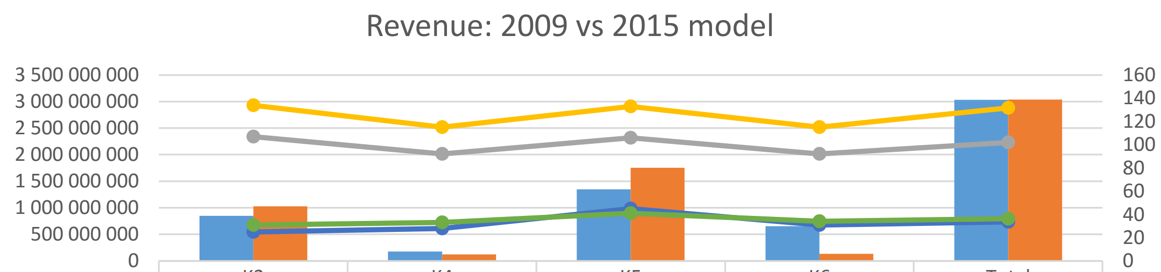
- *Carats* – the graph below (Figure 22) shows an overall decrease in carats of -22% mainly as a result of:
  - Reducing the resource modifying factors including the removal of the 2009 boart factor which led to a reduction in overall grade which is most pronounced for K5
  - decrease in volume of around 8% as discussed above
  - increasing the bottom cut off from 1.00 to 1.25mm




| | K2 | K4 | K5 | K6 | Total |
|---|---|---|---|---|---|
| 2009 | 7 943 600 | 1 953 500 | 12 711 900 | 7 087 800 | 29 696 800 |
| 2014 | 7 648 500 | 1 077 700 | 13 174 700 | 1 184 700 | 23 085 600 |
| 2009 Grade | 23 | 31 | 43 | 34 | 33 |
| 2014 Grade | 23 | 29 | 31 | 29 | 28 |

*Figure 22. Carats reconciliation*

- *Revenue* – The graph below (Figure 23) shows that the overall in-situ value of the 2014 model is slightly higher than that of 2009 due to an increase in $/carat of approximately 29% due to the improved assortment values and increasing the bottom cut off from 1.00 to 1.25mm. The overall $/tonne has increase by 9%.




| | K2 | K4 | K5 | K6 | Total |
|---|---|---|---|---|---|
| 2009 | 849 965 200 | 179 722 000 | 1 347 461 400 | 652 077 600 | 3 029 226 200 |
| 2014 | 1 024 899 000 | 123 935 500 | 1 752 235 100 | 136 240 500 | 3 037 310 100 |
| 2009 $/ct | 107 | 92 | 106 | 92 | 102 |
| 2014 $/ct | 134 | 115 | 133 | 115 | 132 |
| 2009 $/t | 25 | 28 | 45 | 31 | 34 |
| 2014 $/t | 31 | 33 | 41 | 34 | 36 |

*Figure 23. Value reconciliation*

- *Summary* – the relative changes between the 2009 and 2015 Diamond Resource is summarised in Table 27 below:

| | Volume ($m^3$) | Tonnes | SG (tonnes/$m^3$) | Carats | $/tonne |
|---|---|---|---|---|---|
| Grand Total | -8.3% | -8% | 0.4% | -22.3% | 9% |

*Table 27. Reconciliation summary*

The 2015 Liqhobong Diamond Resource estimate update is an improvement to the previous 2009 estimate. After completion and sign-off, the 2015 block model was used to conduct mine design, pit optimisation work and prepare Life of Mine and detailed production schedules which is described in the next section.

# 5. MINE PLAN

## 5.1 Introduction

### 5.1.1 Resource model

The model used was produced by Z-Star Mineral Resource Consultants. A Datamine Studio 3D geological model (lqmoddatum_1105_10m_depleted.dm) was produced with an Indicated Resource to a depth of 183 metres below surface (2650 masl) and an average grade of 27 cpht. An Inferred Resource was estimated from 2467 masl to 2127 masl at an average grade of 28 cpht. A bottom cut-off of 1.25 mm was used for the grade determination. The resource model contained grades and SG per block with block sizes of 50 x 50 x 20 metres (in the x, y and z direction). Sub-cell blocks of 5 x 5 x 2.5 metres are used to cater for boundary delineation between the pipe contact and for internal facies boundaries.

The resource model tapers with depth. At the widest area at the 2520 elevation the model has a volume of 1600$m^3$ tapering to 850$m^3$ at the 2060 metre elevation.

### 5.1.2 Mine Design Criteria

The life of mine (LoM) plan was focussed on an open-pit design as mining of this nature had already taken place during previous phases of the Liqhobong operation. The driving principle was to optimise the economic return of the Diamond Resource based on the prevailing operating, geotechnical and financial inputs, while remaining cognisant of operating constraints. Due to the steep terrain surrounding the Main pit, waste stripping is a key factor in the economics of the project while at the same time posing possible constraints on actual waste stripping capabilities. A Whittle Pit Optimisation process was undertaken to determine the optimal size of LoM pit based on all the relevant inputs. The inputs were agreed and signed off by the LMDC project team for use during the optimisation process. The inputs are tabled below and are described in more detail in subsequent sections of the report. The geo-technical studies carried out to determine the bench designs are described in section 5.3.1.

| Parameters | Unit | Value |
|---|---|---|
| ***1. Resource*** | | |
| Resource block model file | lqmoddatum_1105_10m_depleted.dm | |
| Survey depletions file | Liqobbong Scan April 2014.dxf | |
| ***2. Operating costs*** | | |
| Waste mining cost | ZAR/tonne | 33.50 |
| Ore mining cost | ZAR/tonne | 35.00 |
| Mining cost adjustment factor per tonne (MCAF) | ZAR/bench/tonne | 0.44 |
| Processing cost | ZAR/tonne | 62.00 |
| ***3. Operating efficiencies and factors*** | | |
| Mining Recovery | % | 98% |
| Mining Dilution | % | 2% |
| Processing recovery | % | 100% |
| ***4. Pricing*** | | |
| Diamond Price (Oct 2014) | | |
| K2 | US$/carat | 134 |
| K4 | US$/carat | 115 |
| K5 | US$/carat | 133 |
| K6 | US$/carat | 115 |
| Annual (real) diamond price escalation | % | 3.0% |
| Sales and marketing | % | 1.87% |
| ***5. Royalties and financials*** | | |
| Lesotho Government Royalty (months 1 - 48) | % | 4% |
| Lesotho Government Royalty (months >48) | % | 8% |
| Discount rate | % | 8% |
| USD-Rand FX rate | | 11.00 |
| ***6. Production rates*** | | |
| Milling | mtpa | 3.60 |
| Mining - ore | | |
| Year 1 includes 6 month ramp-up | mtpa | 2.88 |
| Year 2 and to end of LoM | mtpa | 3.60 |
| Steady state monthly rate (ktpm) | ktpm | 300 |
| Mining - waste (maximum allowed) | mtpa | 18.50 |
| ***7. Time Costs*** | | |
| Construction capital costs | | |
| Year 1 (2014) | ZAR m | 637 |
| Year 2 (2015) | ZAR m | 949 |
| Year 3 (2016) | ZAR m | 124 |
| Total | ZAR m | 1 710 |
| SIB capital | | |
| Years 1 - 3 | ZAR m | 12.00 |
| Year 4 onwards | ZAR m | 32.00 |

*Table 28. Pit optimisation inputs*

## 5.2 Mining Method

The mining process at LMDC will be a conventional open-pit operation consisting of drill, blast, load and haul activities which will be carried out by a mining contractor. The pit design has been based on a split shell concept largely to defer waste stripping as much as possible while at the same time providing a double ramp system to mitigate the risk of ramp failure. The pit layout incorporates a concentric cut 1 which has been designed around the existing exposed pit bottom. This cut provides three and a quarter's years of ore supply. Four successive split shell cuts follow, namely Cut 2 South, Cut 2 North, Cut 3 South and Cut 3 North. Each split shell cut will have its own ramp system, however once the north cut meets up with its respective south cut, the ramps join to become a concentric system.

The waste and ore mining fleet is planned to commence with 40 tonne ADT's with a matching excavator. As waste tonnes increase from year five, a CAT 777 (90 tonne) truck or equivalent will be phased in. Haul roads in the pit have been designed at a width of 25m to accommodate the larger trucks and will be at a gradient of 1:10. The ADT ore fleet will continue throughout the LoM and ramp widths in the kimberlite zones will be 17m.

The mining contractor will also provide all the support fleet and will carry out maintenance on the total mining fleet. A small existing workshop will cater for the contractor's initial mobilisation. Following this a new earth-moving workshop and mining office facility will be constructed by the mining contractor for use from the start of year two.

All kimberlite ore will be trucked and tipped into the primary crusher located 560m from the edge of the current pit. The front-end arrangement allows for direct tipping into a crusher tipping bin fitted with an 800mm aperture grizzley and rock-breaker. The crusher bin has a live capacity of 180m$^3$ or approximately 320 tonnes. The ore will then be fed with an apron feeder into the primary crusher and onto the treatment plant. A RoM stockpile area adjacent to the tipping bin will provide 30,000 tonnes of capacity or approximately three days of production. Building a larger stockpile area proved prohibitive due to the steep terrain in the area. The tipping strategy will be to direct tip into the crusher bin during normal production operations. When the mining fleet is not operational due to a shift change or blasting activities, re-handling from the stockpile will take place using a dedicated front end loader. A section through the tipping arrangement is shown below.

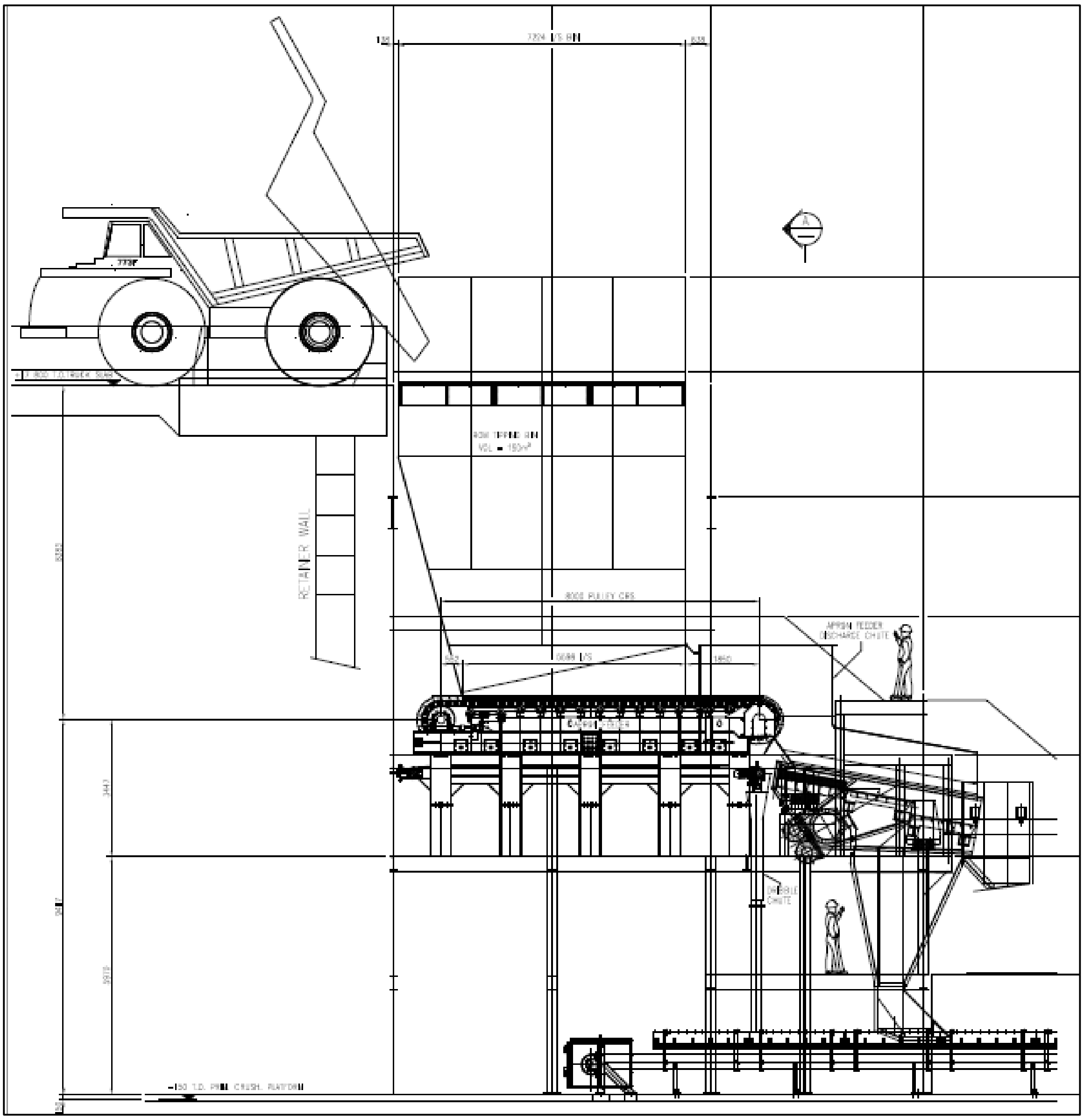


*Figure 24. Section through the ore tipping arrangement*

Waste rock will be trucked and dumped onto the residue storage facility (RSF3) to construct the slimes dam wall with coarse tailings disposal from the treatment plant. Based on the scheduled construction of RSF3 over the LoM, waste rock will be dumped according to various phases. Once the waste rock is dumped on RSF3 in the designated areas a separate contractor responsible for managing the RSF will doze and shape the waste rock as per the construction sequence. The sequence of the waste dumping phases is shown in the following diagram.

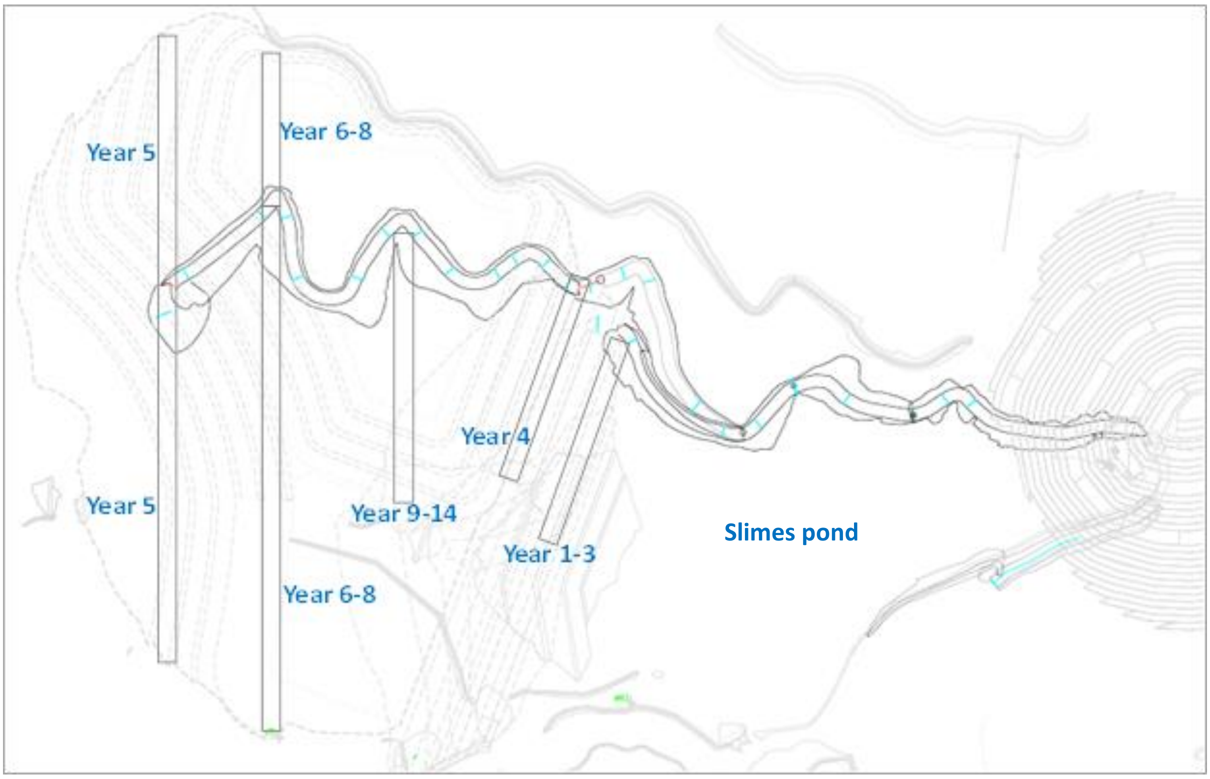



*Figure 25. LoM waste haul roads*

The waste haul road is generally flat to down-grade at 1:10 leaving the pit followed by flat sections on the dam wall. The maximum one way road length from the edge of the pit is in year five at 2,660m.

## 5.3 Modifying factors

In this section of the report the technical modifying factors are discussed. The economic modifying factors are described separately in the Economic Criteria section.

### 5.3.1. Pit slope design

SRK consulting were engaged to carry out slope design studies for the planned open-pit operation. SRK carried out bench, stack and overall pit slope stability analysis in order to produce a recommended slope design. SRK's work initially began in 2008 with the previous owners, Kopane Diamond Development Company (Pty) (Kopane). During this period face mapping data recorded by Kopane along with borehole data logged by SRK was generated. Under Firestone's ownership SRK began further studies in early 2012 making use of a geological block model based on the 2008 data and preliminary pit design shells from Firestone. A slope design report by SRK was then published in June 2012 which was compiled under the Guidelines for Open-pit Design published by the CSIRO (the Commonwealth Scientific and Industrial Research Organisation) (Armstrong 2012). The SRK report produced a bench, slope and structural analysis for the Main pit with design parameters which are shown in Table 29.

| 10 m - 20 m benches | Kimberlite | Basalt | Basalt |
|---|---|---|---|
| **Bench height (m)** | 10 | 20 | 10 |
| **Batter angle (°)** | 90 | 90 | 90 |
| **Berm width (m)** | 8.4 | 12 | 6.7 |
| **Geotechnical berm (m)** | 15 | - | 15 |
| **Stack height (m)** | 60 | 60 | 60 |
| **Stack angle (°) (toe to toe)** | 50 | 59 | 56 |
| **Stack angle (°) (toe to crest)** | 55 | 68 | 61 |
| **Ramp width (m)** | 25 | 25 | 25 |

*Table 29. Pit design parameters for 10 & 20 m benches*

Revisions and reviews by SRK took place during 2014 and 2015 as more detailed pit design work progressed based on the updated Resource Model. The following sections outline the workflow carried out by SRK in arriving at the design recommendations.

*Structural characteristics of the rock units*

Five geotechnical boreholes were drilled during the Kopane drilling campaign, however these were found to have unreliably orientated core. The core from one suitable exploration hole along with face mapping was used to determine the structural environment in the Main pit area. Bedding planes (basalt flows), veins, joints, a dyke and contacts between the kimberlite and basalt were observed. The predominate geological feature is a steeply inclined to vertical joint sets striking NE-SW and NW-SE with a moderate number of joints and bedding dipping shallowly and demonstrating no distinct strike. Veins dip from approximately 15˚ to 60˚ and strike distinctly E-W. These observations indicated that the steep nature of joint sets will limit the occurrence of structurally controlled failures, should they occur.

*Rock mass characterisation*

Nine boreholes from the 2008 drilling program (five geotechnical and 4 exploration) which had been logged by SRK were used as inputs to determine rock mass ratings as per Laubscher's (1990) method. The main observations from the core are listed below;

a) Layered massive and amygdaloidal basalt units alternate throughout the country rock zone and are indicative of successive lava flows, where the amygdaloidal basalt indicates the top of the flows

b) The kimberlite contact against the basalt is generally intact and shows no significant weathering

c) In general, joints are sub-vertical, contain no infill and are rough and undulating or irregular in their small-scale expression.

d) The contact between massive and amygdaloidal basalts is gradational

e) Weathering of basalt was observed down to an average depth of 17 metres while weathering of kimberlite took place down to 53 metres.

f) Field data resulted in intact rock strength measurements of approximately 85 MPa for kimberlite, 95 MPa for amygdaloidal basalt and 105 MPa for massive basalts.

The following picture shows a representative sample of core from the 2008 drilling program. The red pointers indicated contacts between rock units.

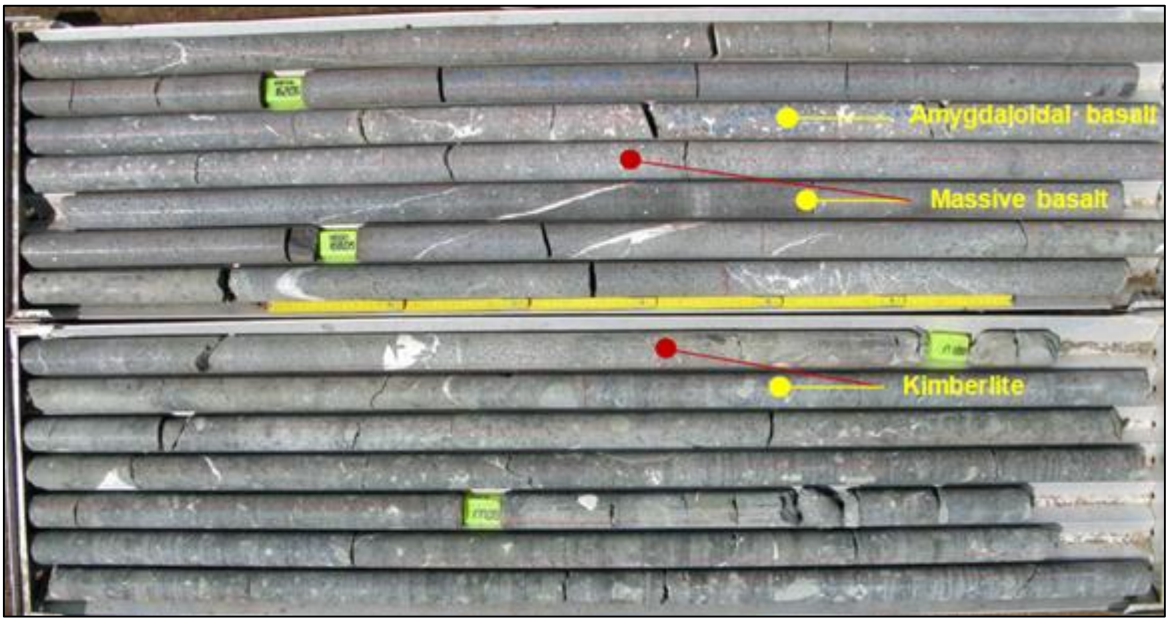



*Figure 26. Core from Borehole SRK04.*

The resulting rock mass ratings (RMRs) calculated for the various rock units are shown below.

| Rock unit | Average RMR |
|---|---|
| Amygdaloidal basalt | 69 |
| Massive basalt | 78 |
| Red amygdaloidal basalt | 85 |
| Kimberlite | 81 |

*Table 30. RMR values*

*Hydrogeology*

No piezometer information was available at the time of conducting the geotechnical investigation, therefore the initial level of the phreatic surface was assumed to be 30 m below the crest of the slope, which is consistent with other Lesotho open pits.

*Test Work*

During the 2012 study, various laboratory tests and evaluations were carried out by RockLab based in Pretoria which consisted of;

a) uniaxial compressive strength tests

b) base friction angle tests

c) tri-axial compressive tests

d) direct tensile strength tests

e) calculations of Young's modulus and Poisson's ratio

The outputs of these tests were statistically described and presented in the following sections.

| Rock unit | | M BAS | A BAS | KIM |
|---|---|---|---|---|
| UCS (Mpa) | Number of tests | 14 | 16 | 6 |
| | Max. | 201 | 233 | 131 |
| | Min. | 76 | 66 | 38 |
| | mean | 113 | 133 | 78 |
| | -stdev | 78 | 90 | 47 |
| | +stdev | 148 | 177 | 108 |
| | CoV | 0.3 | 0.3 | 0.4 |
| Unit weight (KN/m$^3$) | Max. | 2.83 | 2.69 | 2.80 |
| | Min. | 2.64 | 2.46 | 2.46 |
| | mean | 2.73 | 2.60 | 2.68 |
| | -stdev | 2.77 | 2.65 | 2.74 |
| | +stdev | 2.69 | 2.54 | 2.61 |
| | CoV | 0.02 | 0.02 | 0.02 |

*Table 31. UCS and density measurements*

The UCS and density results indicate that the amygdaloidal basalt is the strongest rock with an average UCS value of 133 MPa. The massive basalt is slightly weaker with an average UCS value of 113 MPa, whereas the kimberlite is the weakest rock with an average UCS value of 77 MPa.

In terms of Tri-axial compressive tests, a total of 75 tests were carried out with a confining pressure ranging between 2.5 MPa and 20 MPa and were grouped according to the main rock units. The corresponding plots of minor (σ3) to major (σ1) principal stress is shown below.

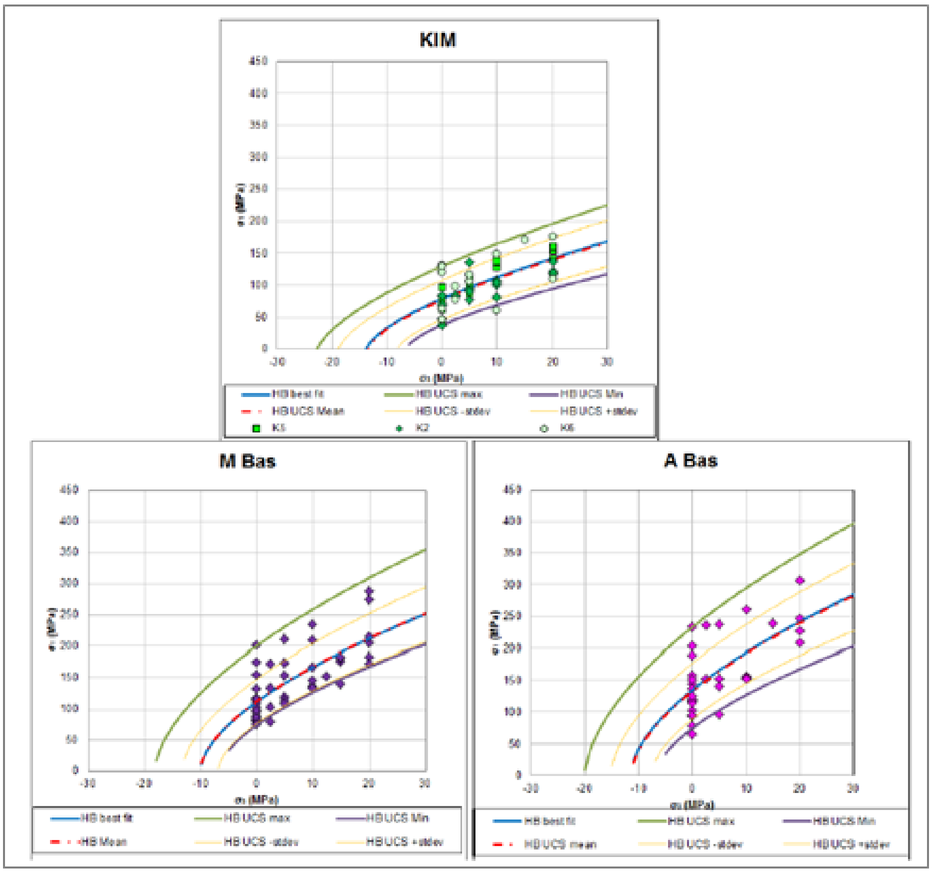



*Figure 27. Tri-axial compressive measurements*

Tests results to determine Young’s Modulus and Poisson’s ratio are shown below.

| Rock unit | | M BAS | A BAS | KIM |
|---|---|---|---|---|
| Young's Modulus (GPa) | Max | 47 | 45 | 45 |
| | Min | 12 | 26 | 14 |
| | Mean | 30 | 34 | 30 |
| | -stdev | 12 | 23 | 19 |
| | +stdev | 48 | 44 | 41 |
| | CoV | 0.6 | 0.3 | 0.4 |
| Poisson's Ratio v | Max | 0.22 | 0.24 | 0.29 |
| | Min | 0.11 | 0.17 | 0.17 |
| | Mean | 0.17 | 0.21 | 0.24 |
| | -stdev | 0.11 | 0.18 | 0.20 |
| | +stdev | 0.24 | 0.25 | 0.27 |
| | CoV | 0.35 | 0.16 | 0.15 |

*Table 32. Young's Modulus and Poisson's ratio measurements*

These results indicate that the average stiffness of the three rock units is similar at around 30 GPa with massive basalt having the greatest variability ranging from 12 to 47 GPa. The average value of 30 GPa indicates a high rock strength. The values of Poisson’s ratio are indicative of the basalt and kimberlite rock types found in this region.

Base friction angle tests were carried out according to ASTM Standard D5607-95. The mean base friction angle for the massive basalt unit is 42˚ for the amygdaloidal basalt unit and 30˚ for the kimberlite unit.

*Rock mass strength properties*

The estimation of rock mass strength properties was based on the borehole data and laboratory test results along with the calculated RMRs. The generalised Hoek-Brown failure criterion was used directly in both Limit Equilibrium analysis (SLIDE) and finite element analysis (PHASE2). The procedure used to estimate the Hoek-Brown material constants is illustrated below.

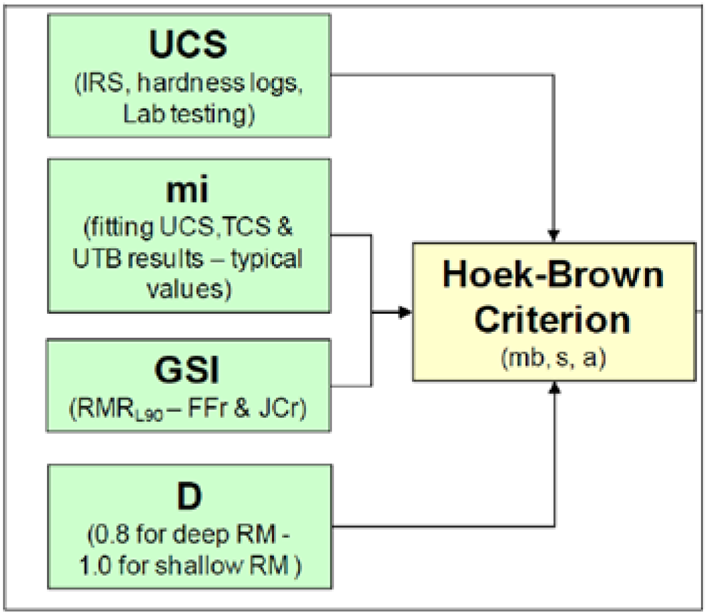



*Figure 28. Hoek Brown methodology*

The resulting rock strength parameters for the three rock units are shown below.

| Shear Strength of Rock Mass | | Hoek Brown Criterion | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|
| Hoek-Brown criterion | | Overall Slope | | | | Stacks | | | |
| | | D = 0.7 | | σ3 max = 2 | | D = 1 | | σ3 max = 0.5 | |
| Geotechnical Unit Description | Code | c (MPa) | Φ (°) | st (kPa) | E (GPa) | c (MPa) | Φ (°) | st (kPa) | E (GPa) |
| M Basalt | M Bas | 10.8 | 55.5 | 5.1 | 16.1 | 9.7 | 56 | 4.8 | 11.5 |
| | | 0.8 | 43.4 | 0.1 | 4.1 | 0.3 | 47.6 | 0.1 | 2.5 |
| | | 2.3 | 52.6 | 0.8 | 10.6 | 1.6 | 55.4 | 0.7 | 6.8 |
| | | 1.1 | 47.2 | 0.3 | 7.3 | 0.6 | 51.6 | 0.2 | 4.5 |
| | | 5.4 | 55.1 | 2.3 | 13.9 | 1.1 | 55.2 | 0.5 | 4.5 |
| | | | | | | | | | |
| A Basalt | A Bas | 9.4 | 56.8 | 4.1 | 16.9 | 8 | 57.3 | 3.7 | 11.7 |
| | | 0.6 | 39.8 | 0.1 | 3.4 | 0.2 | 43.8 | 0 | 2.1 |
| | | 1.7 | 52.3 | 0.5 | 9 | 1.1 | 55.4 | 0.4 | 5.6 |
| | | 1 | 47.7 | 0.2 | 6.1 | 0.5 | 51.7 | 0.2 | 3.7 |
| | | 3.3 | 55.2 | 1.2 | 12.4 | 2.5 | 57.1 | 1.1 | 8 |
| | | | | | | | | | |
| Kimberlite | KIM | 9.2 | 44.4 | 6.6 | 15.9 | 8.1 | 44.5 | 6.1 | 11.2 |
| | | 0.3 | 24.8 | 0 | 1.7 | 0.1 | 27.1 | 0 | 1.2 |
| | | 1.5 | 41.2 | 0.8 | 9.2 | 1.1 | 44 | 0.7 | 5.7 |
| | | 0.6 | 33.7 | 0.2 | 4.9 | 0.3 | 37.9 | -0.1 | 3 |
| | | 4.7 | 44.2 | 3.1 | 13.9 | 3.9 | 45 | 2.8 | 9.4 |
| | | | | | | | | | |

*Table 33. Rock strength parameters*

*Kinematic analysis*

The kinematic analysis conducted, identified low risk with respect to potential toppling, planar sliding and wedge sliding modes of failure respectively. Wedges are steep and stack failures are unlikely and from SRK's experience, the failure of wedges on a bench scale will be restricted to the blasting cycle. Risks from the freeze–thaw cycle must be considered and this aspect of bench stability must be monitored.

*Stability analysis*

Two methods of analysis were carried out, namely;

a) A stack sensitivity analysis based on conceptual slopes of various slope heights and angles using the Limit Equilibrium (LE) method.

b) An overall slope analysis on representative sections of the final pit, using LE analysis and Finite Element Method (FEM).

The stack stability analysis was carried out on homogeneous slopes by varying height, angle and composition of the slope. The ranges of these variables were chosen such that all the possible geometries and geology of stacks at Liqhobong could be accounted for. Therefore, the results of this analysis can be used to assess the stability of designed stacks in the various cut-backs. Three slope heights (100 m, 200 m and 300 m) and four angles (55°, 60°, 65° and 70°) were combined to construct the models. In the models, the slope was in turn composed of kimberlite, massive basalt and amygdaloidal basalt rock unit. The two sets of slope limits were placed in the models to analyse only failures at the scale of the stack excluding skin failures. This result indicates that based on the material properties derived from the previous tests and analysis, all the possible geometries analysed have a FoS higher than 1.5.

The overall slope stability analysis was carried out on two representative cross sections in the proposed final Cut 3 which is the final cut-back. The analysis was carried out using the LE program SLIDE and subsequently using FEM in PHASE2. PHASE2 caters for the contribution of rock deformations in the slope stability analysis and allows for joints to be explicitly included into the models. Therefore the effect of major joint sets on the stability of the slopes could be assessed. In both LE and FEM, a phreatic surface was also included and the stability was re-assessed under saturated conditions.

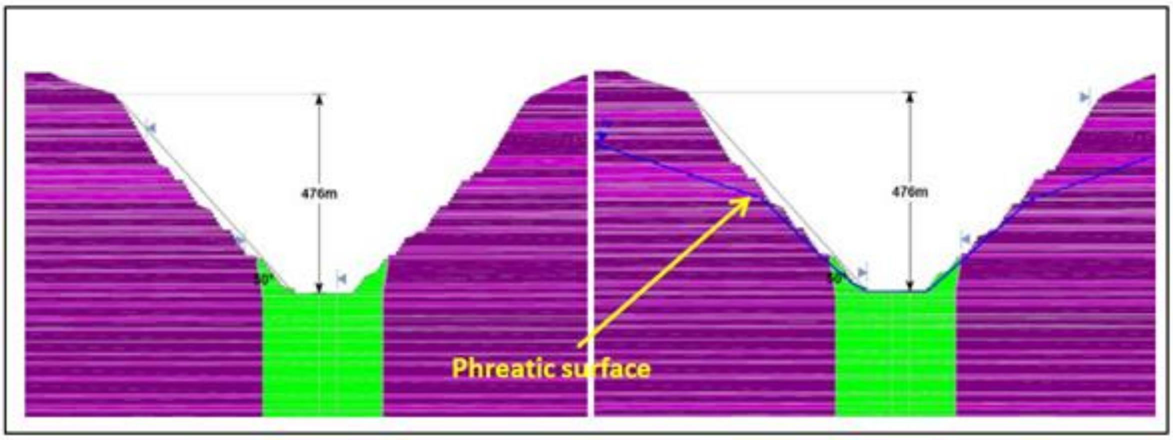



*Figure 29. A representative section through final pit slope*

*Rock-fall analysis*

A rock fall analysis was conducted for basalt stacks to determine the potential impact of various stack heights, bench heights and overall stack angles on rock-fall incidents. The rock-fall analysis was carried out using the Rocscience software package, RocFall 4.0. Combinations of 60 metre, 80 metre and 100 metre stacks with 10 and 20 metre high benches were analysed with 60° and 65° crest to toe slope angles and a 25 metre wide ramp. From this modelling, the likelihood of rock-falls extending beyond the ramp were determined.

| Stack Height (m) | Slope Angle (°) | Bench Height (m) | Number of rocks over 25m Ramp | Probability of Rock Fall Extending Beyond Ramp (%) |
|---|---|---|---|---|
| 60 | 60 | 10 | 97 | 2 |
| 60 | 60 | 20 | 0 | 0 |
| 60 | 60 | 30 | 0 | 0 |
| 80 | 60 | 10 | 96 | 2 |
| 80 | 60 | 20 | 0 | 0 |
| 80 | 60 | 30 | 0 | 0 |
| 100 | 60 | 10 | 136 | 3 |
| 100 | 60 | 20 | 0 | 0 |
| 100 | 60 | 30 | 0 | 0 |
| 60 | 65 | 10 | 804 | 16 |
| 60 | 65 | 20 | 1 | 0 |
| 60 | 65 | 30 | 0 | 0 |
| 80 | 65 | 10 | 1189 | 24 |
| 80 | 65 | 20 | 17 | 0 |
| 80 | 65 | 30 | 0 | 0 |
| 100 | 65 | 10 | 1667 | 33 |
| 100 | 65 | 20 | 32 | 1 |
| 100 | 65 | 30 | 0 | 0 |

*Table 34. Results of ROCFALL analysis*

As expected, steeper angles, higher bench heights and smaller berm widths, lead to greater likelihood of rocks falling into the mining areas.

*Slope design recommendation*

The relative lack of joints in the basalt and kimberlites result in high rock mass strengths and reduce the potential for rock mass failures. This is evidenced from the high FoS's derived from the analysis. The favourable orientation of the geological structures reduce the likelihood of structural failure. Therefore the design of the slopes is driven by the safety of the working area below benches rather than both the safety and the stability of overall slopes. The slope design for Liqhobong is mainly based on bench catchment capacity as bench scale rock falls are the primary risk. It is recommended that the basalt is mined in 20 metre benches, which allows for a crest to toe angle of 68° with 12 metre berms. The 20 metre benches should be pre-split in a single pass to prevent hang-up of loosened blocks that could form a rock fall source. In kimberlite or areas of basalt where a 10 metre bench height is used, a 15 metre wide safety berm is required every 60 metre to catch rocks that do fall beyond the smaller inter-bench berms. It is recommend that the kimberlite slopes be mined at a stack angle of 55° (crest to toe), with a 10 metre high bench. A schematic representation of the design slope angles is shown below.

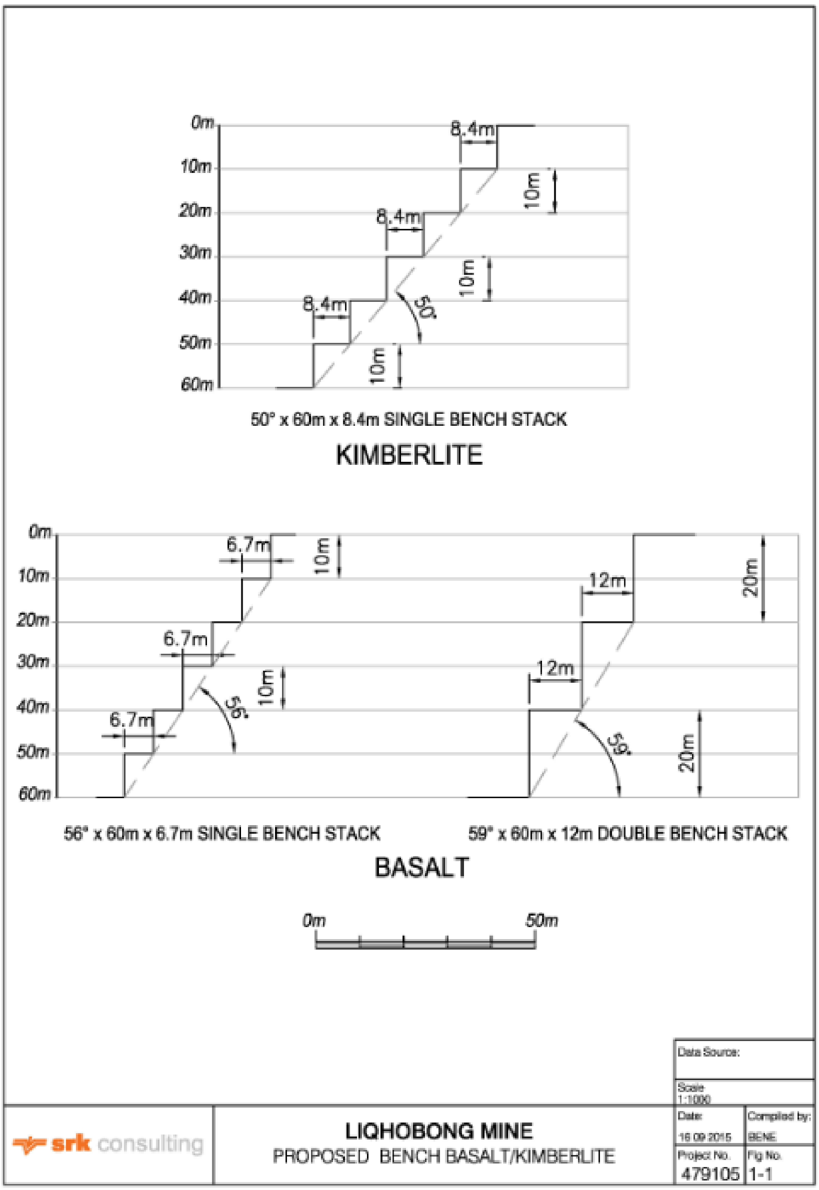



*Figure 30. Slope design parameters for 10m and 20m benches*

These parameters were used in the concentric pit design and the first split shell design as described later in the report. Based on more recent work carried out by SRK, a slope design was produced for a 14 metre and 28 metre double bench design that could be utilised for the Liqhobong pit design. These design parameters were used in a second split shell design only and are shown in the following table.

| 14 m - 28 m benches | Kimberlite | Basalt | Basalt |
|---|---|---|---|
| Bench height (m) | 14 | 28 | 14 |
| Batter angle (°) | 90 | 90 | 90 |
| Berm width (m) | 11.2 | 17.4 | 9.4 |
| Geotechnical berm (m) | 15 | - | 15 |
| Stack height (m) | 84 | 84 | 84 |
| Stack angle (°) (toe to toe) | 51 | 58 | 56 |
| Stack angle (°) (toe to crest) | 56 | 67 | 61 |
| Ramp width (m) | 25 | 25 | 25 |

*Table 35. Pit design parameters for 14 & 28 m benches*

This design allows a steeper overall pit angle, thus reducing waste stripping. A schematic representation of the slope design is shown below.

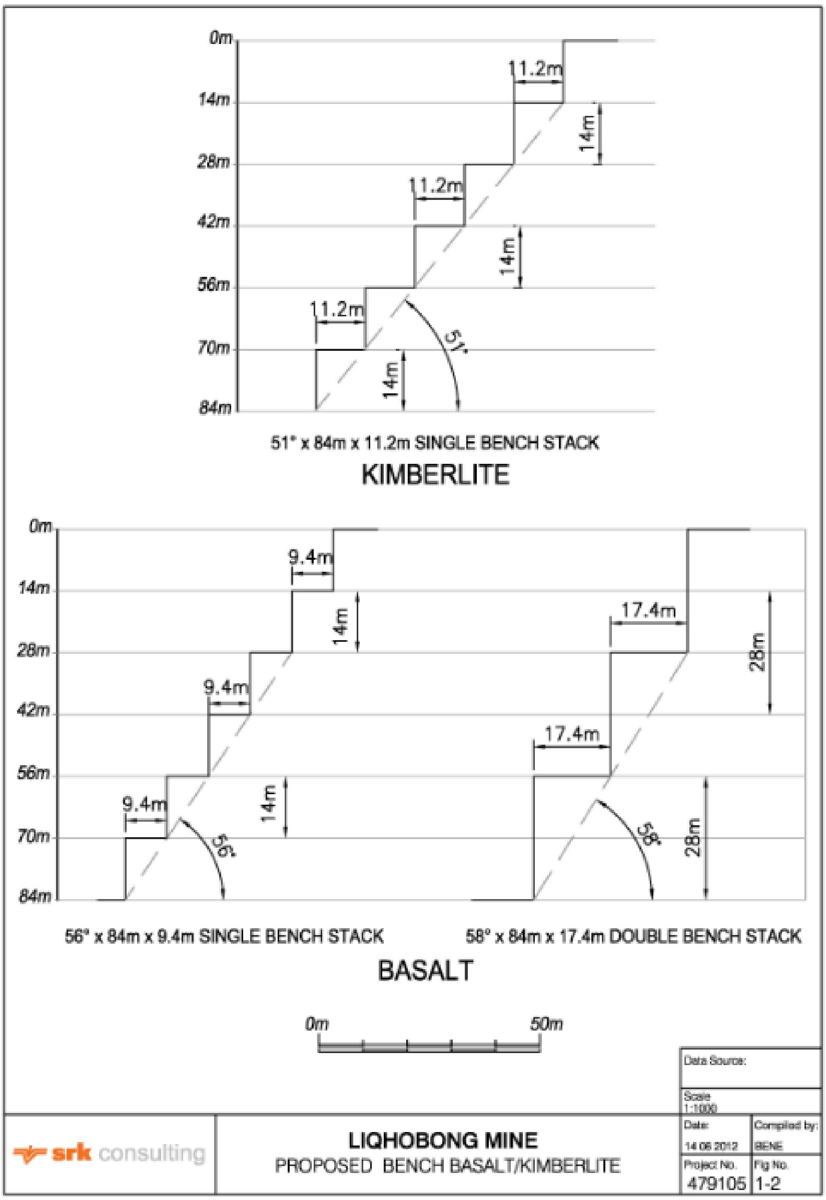



*Figure 31. Slope design parameters for 14m and 28m benches*

### 5.3.2 Dilution

Ore samples treated for grade analysis from the WDD bulk sampling included any internal dilution within the kimberlite ore. Therefore internal dilution is already included in the grade estimates. It is expected that external dilution will enter the ore stream when mining at the basalt contact areas takes place. The initial mining when the pit restarts under the expanded production scenario will be from the exposed ore sections of the pit, post the pre-2013 mining phases. Therefore initially external dilution will be at a minimum. As waste cuts are started and ore mining progresses, dilution will be encountered while ore is mined at basalt interfaces until the end of waste mining at the 2420m elevation. Thereafter there will be approximately 125 vertical metres or the last two years of mining entirely within kimberlite. It was decided that an average 2% dilution factor would account for waste contamination over the LoM.

### 5.3.3 Mining extraction

A mining extraction rate of 98% was used for the Whittle Pit Optimisation and mine planning. This was based on an assumption that a certain amount of ore could possibly end up reporting to waste where the contact between the basalt and kimberlite is of a nature that extracting 100% of the kimberlite in a particular area may result in excessive waste dilution.

### 5.3.4 Mining capacity

For mining productivity, output is based on 350 operating days per year allowing 15 days for weather related down-time. Actual operating time per day has been set at 16.5 hours over a 2 x 12 hour shift arrangement taking into account shift changes, blasting down-time and meal breaks. Ore mining capacity has been set at 3.6 million tonnes per annum or 300,000 tonnes per month to match the treatment plant feed rate. The first year of production will include a six month ramp-up period starting with month one at 20% of planned capacity increasing up to 90% in month six, with full capacity thereafter.

For waste mining capacity calculations the loading rate for the expected size of excavators to be employed was used to determine annual peak tonnage.

| Parameter | CAT 390 exc ADT fleet | CAT 6030 exc 777 fleet |
|---|---|---|
| Operating days per year | 350 | 350 |
| Actual effective loading hrs per day | 16.50 | 16.50 |
| tph loading rate | 580 | 1 600 |
| Tonnes loaded per year | 3 349 500 | 9 240 000 |
| Tonnes loaded per month | 279 125 | 770 000 |
| Loading teams | 2 | 2 |
| Max tonnes per month for scheduling purposes | 558 250 | 1 540 000 |
| Scheduling limit per month | 465 000 | 460 000 |
| Scheduling limit per annum | 5 580 000 | 18 480 000 |

*Table 36. Waste mining schedule rate*

#### 5.3.5 Process recovery factor

A process recovery efficiency of 100% has been determined for the new treatment plant. The design used is considered more advanced in terms of diamond recovery efficiency when compared to the previous sampling and pilot plants at Liqhobong. The latest generation of crushers, scrubbers and screens will be utilised, and advances with DMS and X-Ray process control will be implemented to ensure optimised process efficiency at all times.

Since the current resource estimate is based on actual recovery data from the WDD sample plant and pilot plant, no correction to the current resource grade estimates are considered necessary. Therefore a process recovery factor of 100 percent is to be used. In addition to this, previous MCFs recorded from the Pilot Plant during the last seven months of operation averaged 103.6% indicating that recovered grades were higher than the estimated grades.

### 5.4 Mine Design

#### 5.4.1 Concentric Shell Design

The technical and economic modifying factors were applied to an initial pit design and schedule which was developed on a concentric pit layout as had been used in the 2012 Definitive Feasibility Study.

A series of Whittle pit optimisation runs were produced (Chama 2014) incorporating 10 metre benches and a 20 metre double bench for the final cut with 25m wide haul roads in waste and 17m haul roads in kimberlite. This is to cater for a CAT 777 truck in basalt and 745 ADT's in kimberlite. In conjunction with the SRK bench design inputs described earlier, this resulted in an overall pit slope angle of 53.5 degrees to be used as the Whittle pit slope input. Along with the other relevant modifying factors, the following pit shells were generated with the corresponding values and tonnes.

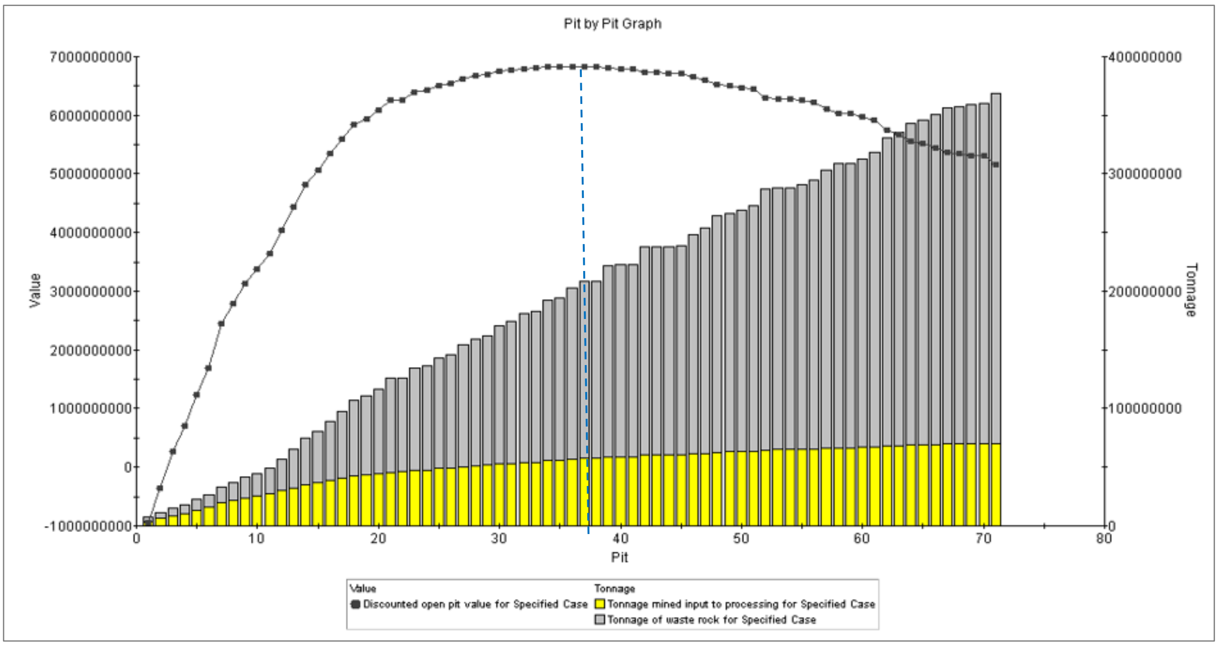



*Figure 32. Whittle optimisation results for concentric design*

The peak NPV values were centred around pit shell 36. A table of the tonnage results for the selected pit shells are shown below.

| 53.5 degrees | | | | | |
|---|---|---|---|---|---|
| Pit shell | Discounted value (R) | Ore tonnes | Waste tonnes | SR | Approx LoM (yrs) |
| 27 | 6 626 072 364 | 50 549 004 | 103 590 067 | 2.05 | 15.2 |
| 28 | 6 669 693 851 | 51 231 472 | 108 178 869 | 2.11 | 15.3 |
| 29 | 6 691 106 893 | 51 671 400 | 110 334 114 | 2.14 | 15.5 |
| 30 | 6 748 215 057 | 52 889 152 | 117 753 348 | 2.23 | 15.8 |
| 31 | 6 767 549 680 | 53 343 988 | 121 046 090 | 2.27 | 15.9 |
| 32 | 6 793 828 066 | 54 213 593 | 126 455 590 | 2.33 | 16.2 |
| 33 | 6 801 729 361 | 54 484 672 | 128 566 969 | 2.36 | 16.2 |
| 34 | 6 826 460 173 | 55 670 873 | 137 186 142 | 2.46 | 16.6 |
| 35 | 6 827 863 658 | 55 806 322 | 138 048 935 | 2.47 | 16.6 |
| **36** | **6 831 379 671** | **56 829 113** | **145 803 975** | **2.57** | **16.9** |
| 37 | 6 829 267 238 | 57 544 979 | 151 326 439 | 2.63 | 17.1 |

*Table 37. Selected pit shells from the Whittle optimisation*

Pit shell 36 was selected for the detailed concentric design and schedule.

The concentric pit design (Chama 2014) incorporated four cuts and a minimum mining width of 60m. A peak waste mining rate of 18.5 million tonnes per annum was set for the scheduling and the ore production of 3.6 million tonnes per annum was used as mentioned previously. A representative section of the resulting pit design is shown below.

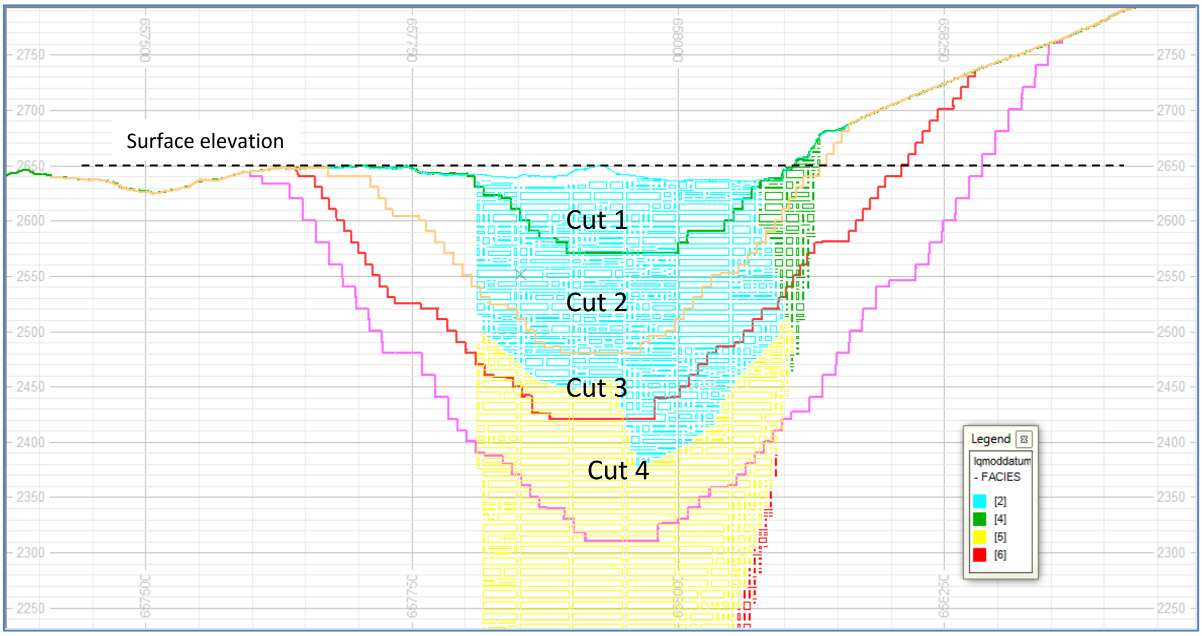



*Figure 33. North-South section through the concentric design*

The final pit depth is 340m below the effective surface elevation of 2650m.

A number of scheduling runs were made and the following LoM production profile was produced which aimed at deferring waste as much as possible while maintaining the ore production rate. A two month pre-stripped reserve was allowed for in the waste scheduling.

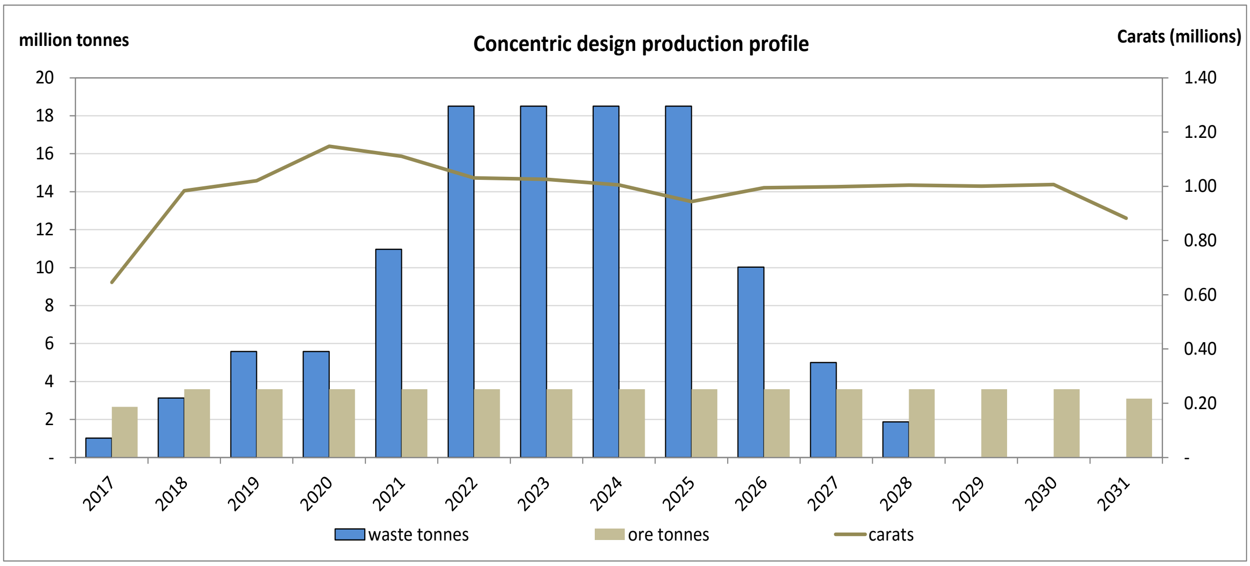



*Figure 34. Production profile for concentric design.*

The plan produces a fairly constant carat profile of around 1 million carats per annum. There is a waste peak of 18.5 mtpa over four years in the middle of the LoM. In order to possibly defer waste further a decision was made to investigate a split shell layout as this design in principle will defer waste stripping into later years of the LoM. This work is described in the following section.

### 5.4.2 Split Shell Design

Two split shell designs were carried out, firstly one with a 10m bench and 20m double bench (Chama 2015) as with the concentric design and a second design with a 14m bench and 28m double bench design (Gallagher 2015). The 28m double bench results in a steeper overall slope angle and reduces waste stripping compared to the 20m double bench. For both designs a 25m wide ramp is used in basalt and a 17m wide ramp in the kimberlite areas. This is to cater for a CAT 777 truck in basalt and 745 ADT's in kimberlite as was the case in the concentric design.

*20 metre double bench design*

One of the key differences between the concentric design and the split shell design is the ramp configuration. The split shell design results in switch-backs as each cut on either side of the pit has its own ramp system. This has an impact on the slope angles as firstly the effective double ramp system results in a flattening of angles in general. Secondly at the locations of the switch-backs there is an effective doubling of the ramp width in that section of the pit which also impacts on the overall slope angle. Switch-back areas are highlighted in the following diagram as an example.

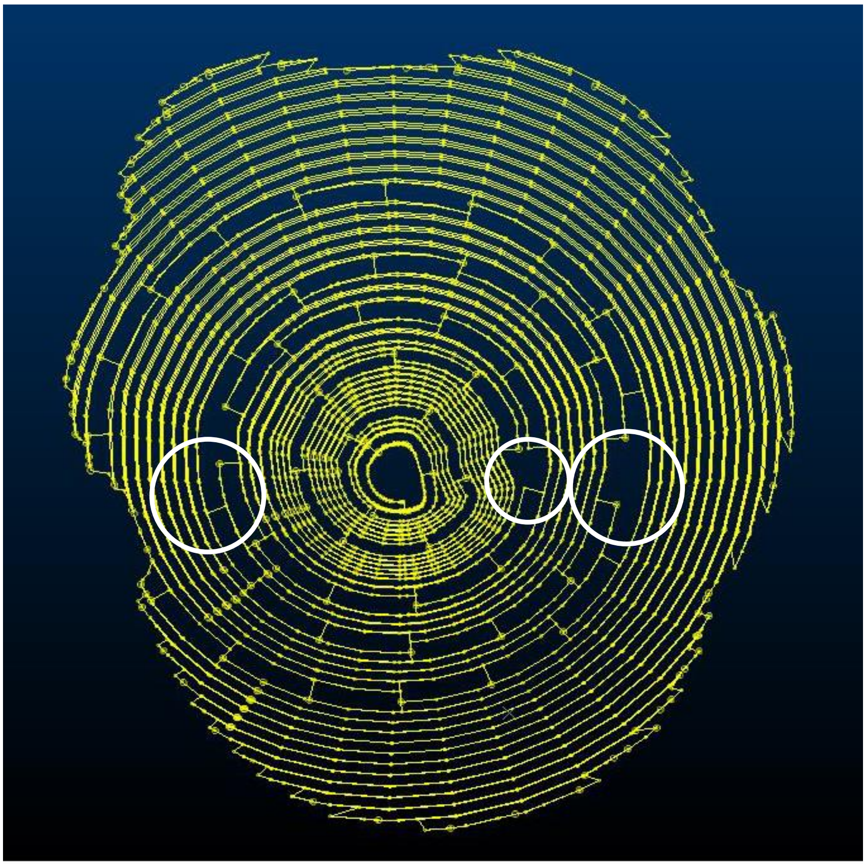


*Figure 35. Plan view of split shell pit design*

The effect of the switch-backs must be taken into account in terms of determining the slope angles for different quadrants of the pit for the Whittle Optimisation slope inputs. For example in kimberlite in the southern section there are three ramp intersections compared to the east section where there were two switch-backs resulting in a five degree slope angle variation. Therefore after carrying out a preliminary pit design, the specific slope angles and slope heights in the different quadrants of the pit are determined to be used in a second Whittle optimisation run to produce a more accurate Whittle shell. An example of the south and east sections are shown in the following diagram where kimberlite slope heights of 100 metres and 110 metres were measured.

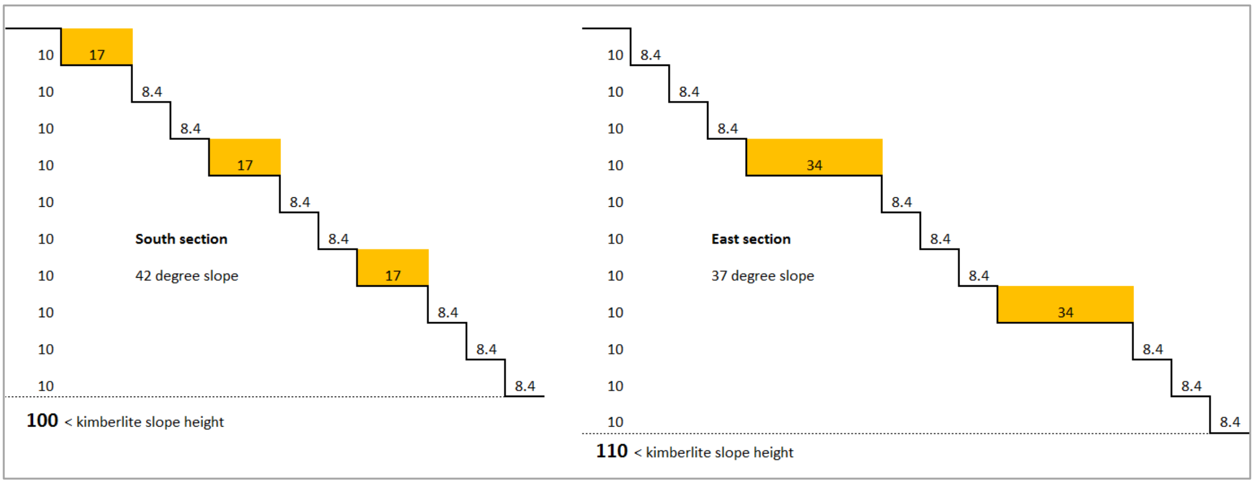



*Figure 36. Effect of switch-backs on slope angles in kimberlite*

The same analysis was carried out for the basalt with similar variances in slope angles in different quadrants of the pit. The slope angles also take into account the slope heights of the basalt in the various quadrants of the pit due to the varying topography surrounding the pipe. The slope inputs for the final Whittle optimisation runs are shown below.

| Whittle slope inputs | Basalt | | | | Kimberlite | | | |
|---|---|---|---|---|---|---|---|---|
| | North | South | East | West | North | South | East | West |
| Bench width | 12.0 | 12.0 | 12.0 | 12.0 | 8.4 | 8.4 | 8.4 | 8.4 |
| Bench height | 20.0 | 20.0 | 20.0 | 20.0 | 10.0 | 10.0 | 10.0 | 10.0 |
| Ramp width | 25.0 | 25.0 | 25.0 | 25.0 | 17.0 | 17.0 | 17.0 | 17.0 |
| Top elevation | 2740 | 2780 | 2760 | 2640 | 2430 | 2430 | 2440 | 2440 |
| Bottom elevation | 2440 | 2440 | 2440 | 2440 | 2330 | 2330 | 2330 | 2330 |
| Slope height | 300 | 340 | 320 | 200 | 100 | 100 | 110 | 110 |
| Slope angle (degrees) | 54 | 54 | 53 | 49 | 42 | 42 | 37 | 41 |
| Azimuth for slope angle (degrees) | 315-45 | 135-225 | 45-135 | 225-315 | 315-45 | 135-225 | 45-135 | 225-315 |

*Table 38. Split shell slope angle summary for 10m and 20m double benches*

The Pit Optimisation results with the pit shells, tonne and values are shown below.

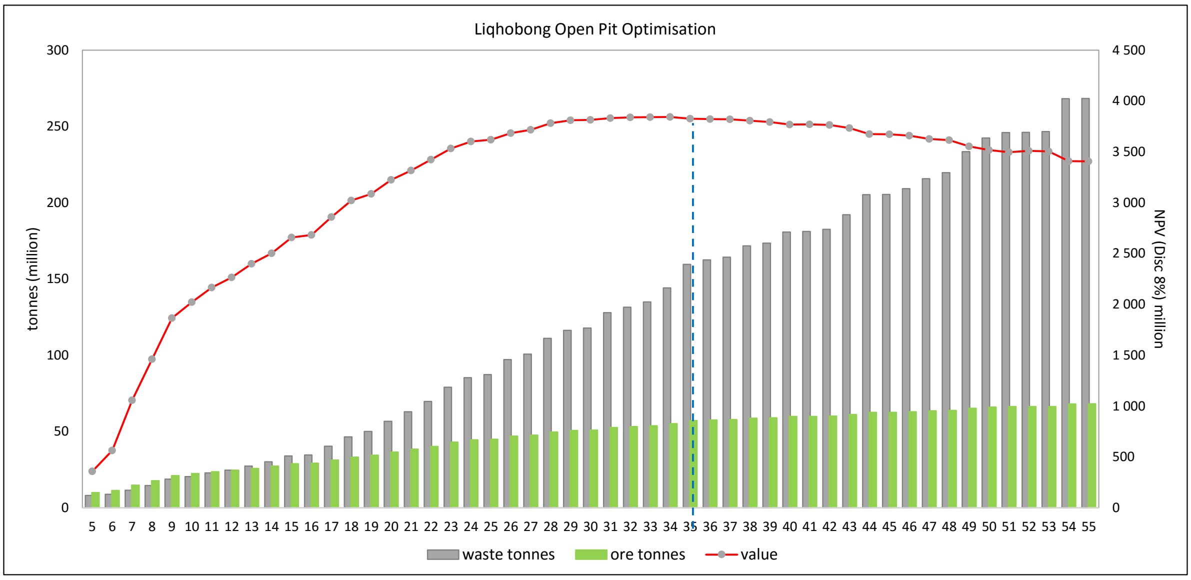



*Figure 37. Whittle optimisation results for 20m double benches*

A selection of the results indicating the optimal Whittle shell are shown below.

| Final pit | Revenue factor | cashflow specified R disc | tonne input specified | Waste specified tonne | Strip ratio specified |
|---|---|---|---|---|---|
| 30 | 0.92 | 3 814 072 078 | 50 955 932 | 117 840 687 | 2.31 |
| 31 | 0.98 | 3 832 310 325 | 52 610 964 | 127 850 359 | 2.43 |
| 32 | 1.00 | 3 839 743 836 | 53 206 652 | 131 475 720 | 2.47 |
| 33 | 1.02 | 3 841 659 591 | 53 713 362 | 134 888 597 | 2.51 |
| 34 | 1.04 | 3 842 980 362 | 55 070 744 | 144 142 894 | 2.62 |
| 35 | 1.06 | 3 826 001 982 | 57 195 207 | 159 612 344 | 2.79 |
| 36 | 1.1 | 3 821 513 638 | 57 606 519 | 162 461 051 | 2.82 |

*Table 39. Split shell 20m double bench optimal pit selection*

The 20 metre double bench split shell resulted in increased waste compared to the concentric design. The optimal pit from the concentric design resulted in 57 million tonnes of ore with 145 million tonnes of waste. The split shell design with an equivalent volume of ore tonnes gives 159 million tonnes of waste and therefore was not considered for further work.

*28 metre double bench design*

In further consultation with SRK and taking into account recent advances in drilling and blasting practices a 14m single bench and 28m double bench design was then used for a second split shell layout. The higher double bench has a wider berm width of 17.4m compared to 12m for the 20m double bench. The impact of using the higher double bench in the basalt results in a steeper slope angle therefore reducing waste.

Using the general slope height for the basalt section of the pit and four 25m wide haul roads as being representative of the planned slope profile, the 28m double bench has a slope angle of 54.6 degrees compared to 53 degrees for the 20m double bench design.

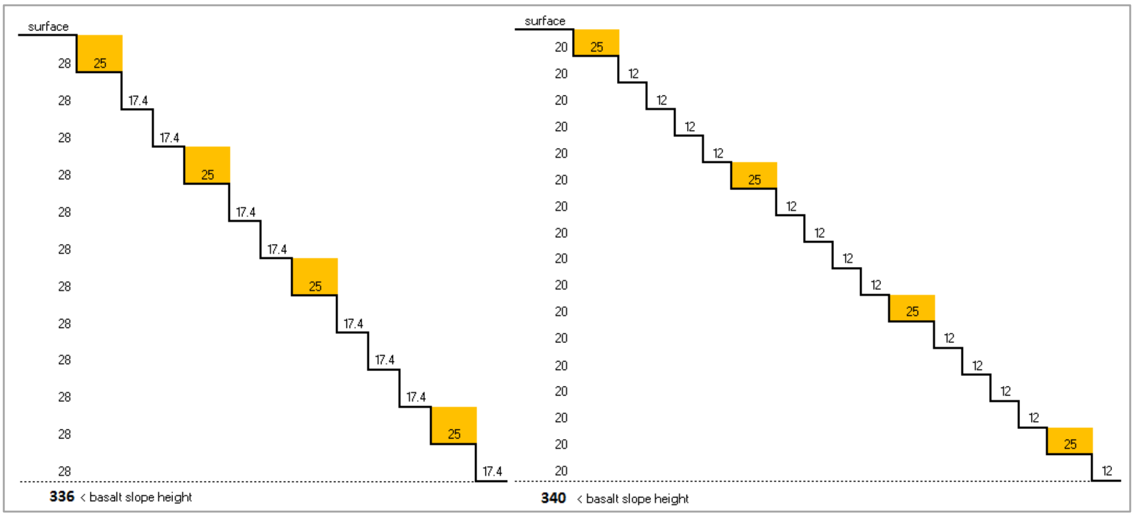



*Figure 38. Slope angle comparisons*

Although the angles are only steeper by 1.6 degrees, this has a favourable impact on the total waste tonnes in the final pit shell. As with the 20m double bench design the various slope angles for different quadrants of the pit were determined from a preliminary pit design and then used for a final Whittle optimisation. The slope angles are shown below.

| **Whittle slope inputs** | **Basalt 25m wide ramp** | | | | **Kimberlite 17m wide ramp** | | | |
|---|---|---|---|---|---|---|---|---|
| | **North** | **East** | **South** | **West** | **North** | **East** | **South** | **West** |
| Bench width | 17.4 | 17.4 | 17.4 | 17.4 | 11.2 | 11.2 | 11.2 | 11.2 |
| Bench height | 28 | 28 | 28 | 28 | 14 | 14 | 14 | 14 |
| Ramp width | 25 | 25 | 25 | 25 | 17 | 17 | 17 | 17 |
| Top elevation | 2 740 | 2 760 | 2 780 | 2 640 | 2 430 | 2 440 | 2 430 | 2 440 |
| Bottom elevation | 2 440 | 2 440 | 2 440 | 2 440 | 2 330 | 2 330 | 2 330 | 2 330 |
| Slope height | 300 | 320 | 340 | 200 | 100 | 110 | 100 | 110 |
| Slope angle (degrees) | 55 | 53 | 55 | 50 | 46 | 40 | 46 | 43 |
| Azimuth for slope angle (degrees) | 315 - 45 | 45 - 135 | 135 - 215 | 215 - 45 | 315 - 45 | 45 - 135 | 135 - 215 | 215 - 45 |

*Table 40. Split shell slope angle summary for 14m and 28m double benches*

The resulting pit shells with their respective tonnes and values are shown in the following graph.

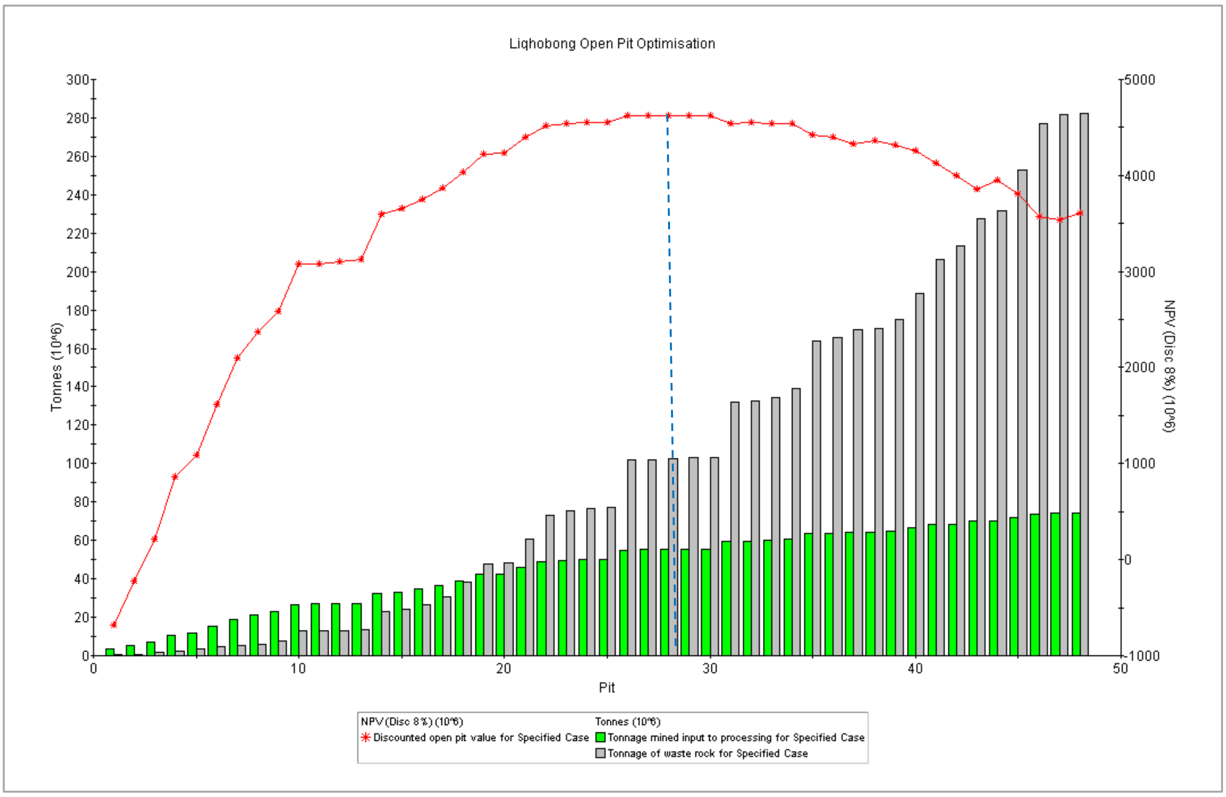



*Figure 39. Whittle optimisation results for 28m double benches*

A selection of the results around the optimal pit shells and the associated valuations and tonnes are shown below.

| Final pit | cashflow specified Rands disc | Tonnes Treated specified | Waste specified tonne | Total Tonnes specified | Stripping ratio |
|---|---|---|---|---|---|
| 25 | 4 555 333 174 | 50 236 775 | 77 021 330 | 127 258 105 | 1.53 |
| 26 | 4 618 202 287 | 55 080 911 | 101 879 540 | 156 960 451 | 1.85 |
| 27 | 4 621 017 084 | 55 130 656 | 102 138 565 | 157 269 221 | 1.85 |
| 28 | 4 621 385 904 | 55 197 517 | 102 560 437 | 157 757 954 | 1.86 |
| 29 | 4 619 696 045 | 55 244 104 | 102 862 375 | 158 106 479 | 1.86 |
| 30 | 4 618 237 534 | 55 326 975 | 103 434 369 | 158 761 344 | 1.87 |
| 31 | 4 540 304 903 | 59 722 481 | 132 282 191 | 192 004 672 | 2.21 |

*Table 41. Whittle pit shell results for the 28m double bench design*

Pit shell 28 was selected to generate a final detailed design. A sensitivity on the optimal pit shell was carried out to test the impact of varying the main inputs of mining costs, processing costs, capital cost and revenue on the NPV. The results are shown in the following graph.

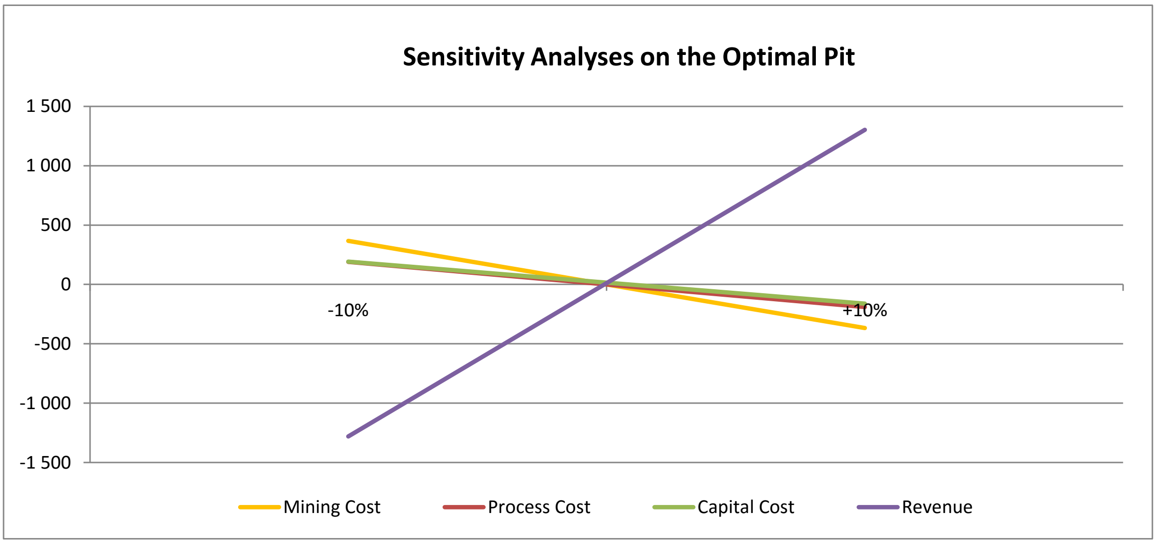



*Figure 40. Sensitivity on the optimal pit shell*

It is seen that revenue has the largest impact on the optimal pit shell by quite a margin. The revenue input is based both on the base diamond price for the four facies types and the 3% annual price escalation as determined by LMDC. This suggests that if there are any material movements in diamond price and-or expected price escalation, the pit optimisation process should be reviewed. Processing costs and capital costs have a relatively low impact on the valuation with mining costs having a larger impact mainly due to waste stripping costs being a high proportion of total costs. Currently mining cost estimates have been used for the optimisation process. Once mining costs have been confirmed after the mining contractor is in place, the optimisation process should be reviewed.

*Practical pit design.*

Due to the existing pit bottom remaining from the previous mining activities, it was decided that a concentric cut would be used to start the pit design. It was then determined that with the total width remaining from the edge of cut 1 to the pit limit, a further two cuts could be planned. Due to the existing topography it was clear that the split between the cuts would be in the west-east direction as the generally flat and more open area on the west of the pit allowed access to both the north and south sections of the pit. Therefore Cuts 2 South and North followed by Cuts 3 South and North were designed following Cut 1. In an attempt to minimise waste in the early years a number of iterations were carried out on the size of Cut 1. The final Cut 1 design produced 11.1 million tonnes of ore and 5.2 million tonnes of waste at a stripping ratio of 0.5. This design also pushed out the first major waste cut, Cut 2 South until year 4.

A representative section of the pit design with the different cuts is shown below.

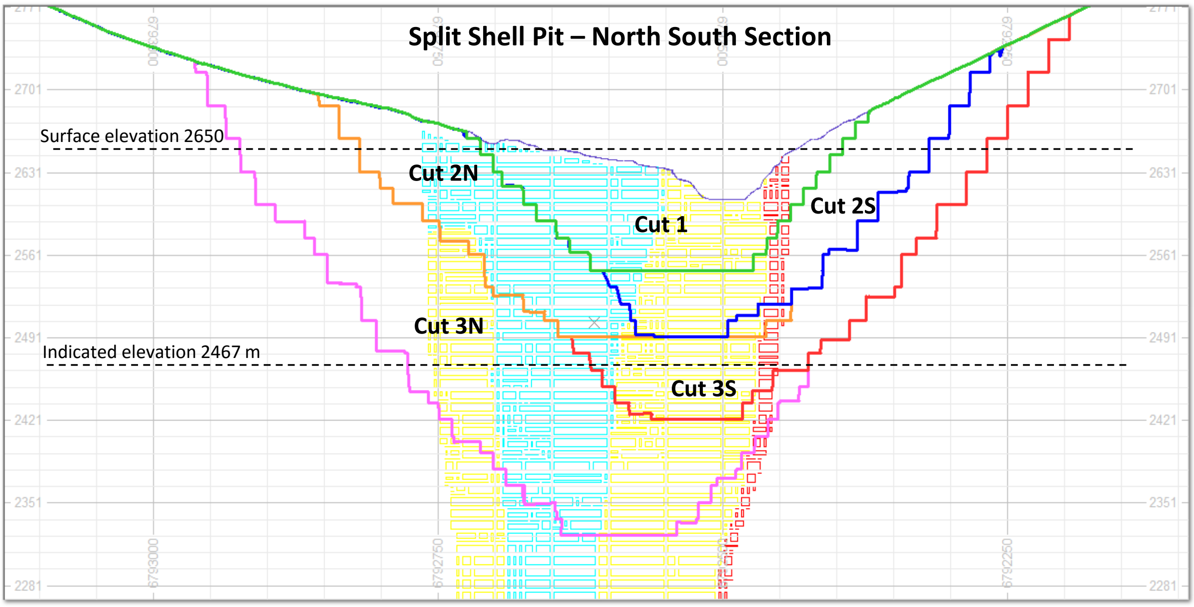



*Figure 41. Section through the split shell design.*

The overall stripping ratio for the LoM pit is 2.1 and the final pit depth is 327m from the effective surface elevation of 2650m.

The aim of the cut-back design and scheduling process was to delay waste stripping as far as practicably possible while maintaining the required ore production rates. A two month stripped reserve guideline was followed in the waste scheduling to expose ore in each new cut-back. The following production profile was produced.

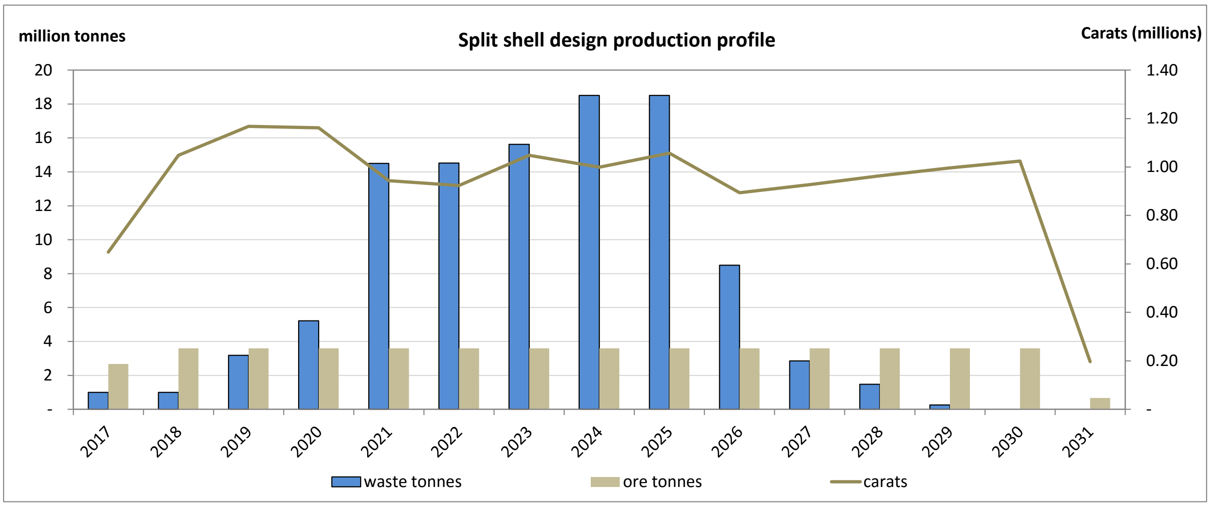



*Figure 42. LoM production profile for 28m double bench split shell design.*

The average carat production is around 1 million carats per annum. There is a waste stripping peak of 18.5 million tonnes for two years in years eight and nine. There is a total of 105 million waste tonnes in the 28m bench design compared to 117 tonnes in the concentric design. More importantly, the first four years waste total is 10.6 million tonnes compared to 15.3 million tonnes for the concentric design resulting in a material cash flow saving.

The operating philosophy is to treat all kimberlite from the delineated volume of the LoM plan in order to maximise the resource extraction. Therefore, dilution is allowed for in the plant feed as blasting patterns at the contact zone will be designed in order to remove all kimberlite which will therefore potentially include some basalt. As the pit expands and encounters more mining at the contact zones, up to 3% dilution has been allowed for.

*Geotechnical Review*

Once the final practical pit design had been completed, the entire design was reviewed by SRK in terms of overall slope angles and geological structures possibly impacting on pit stability. From this review it was noted that all intermediate and major structures mapped thus far intersect the designed ramps obliquely and are not expected to result in longer-term ramp stability issues. All the basalt and kimberlite inter-ramp and slope angles conformed to the SRK design recommendations.

The 28 metre double bench split shell design was subsequently accepted for the LoM plan for the Liqhobong operation.

*Final Year Review*

The final year of the LoM plan registered a relatively low volume of ore (670,000 tonnes) and a revision of the pit design was attempted in order to increase ore in the final year. It was attempted to maximise mining extraction considering it was the last year of the pit and long-term slope stability is no longer of the same importance as in earlier years. A number of design changes were made in order to increase the final year's production. Berm widths in the kimberlite were reduced from 11.2 metres to 5.0 metres. The ramps were narrowed from 17 metres to 15 metres from bench 2365 for the next three benches and then to 10 metres over the last two benches or effectively one way traffic flow at this point. These changes added 1.952 million ore tonnes to the final year resulting in a total of 52.052 million tonnes of ore (before dilution). 67% of the recovered carats are from the Indicated Resource of the total LoM plan. The pit depth was increased to 383 metres from surface as a result and the stripping ratio reduced slightly to 2.0. An annual summary of the LoM schedule is shown in Appendix 1.

A representative section of the revised pit is shown in the following diagram.

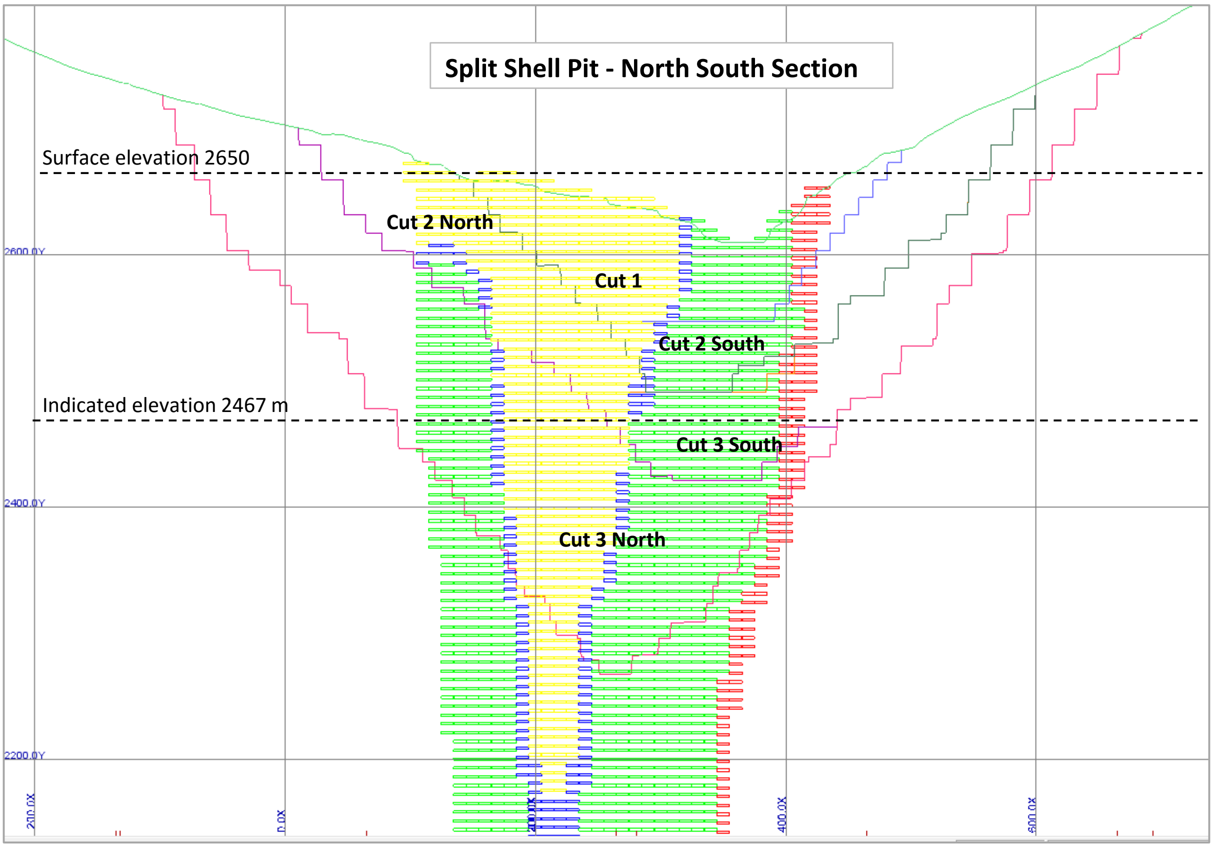



*Figure 43. Split shell pit section after final year adjustments*

With the additional tonnes in the final year the updated LoM production schedule is shown below.

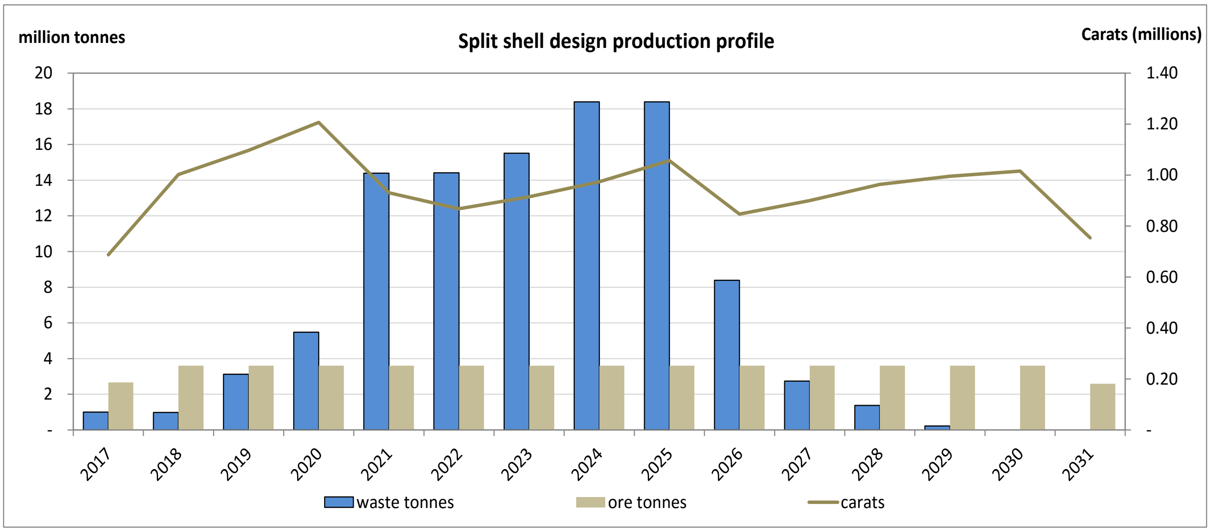



*Figure 44. LoM production profile after final year adjustment*

The currently planned open-pit continues until 2031. It is expected that as with virtually all kimberlite pipes in South Africa, once the economic life of the open-pit has been exhausted, underground mining will commence. Studies towards this end will only begin a number of years after the new open-pit commences and will be determined by the economics at the time.

5.4.3 Interaction of the Main pit with other facilities

Once mining commences and the Main pit expands over time there are two existing facilities that the pit will interact with which are described below.

*Residue Storage Facility 1*

The western side of the pit will intersect with the existing RSF1 facility which is the original slimes dam disposal site remaining from previous mining in both the Main and Satellite pits. It has been estimated from previous surveys that the depth of slimes at its deepest point is 15 metres above the natural ground level (De Swart, 2015). As the Main pit mining progresses it will intersect RSF1 and continue to move west into RSF1. The following plan view shows Cut 1 which remains east of RSF1.

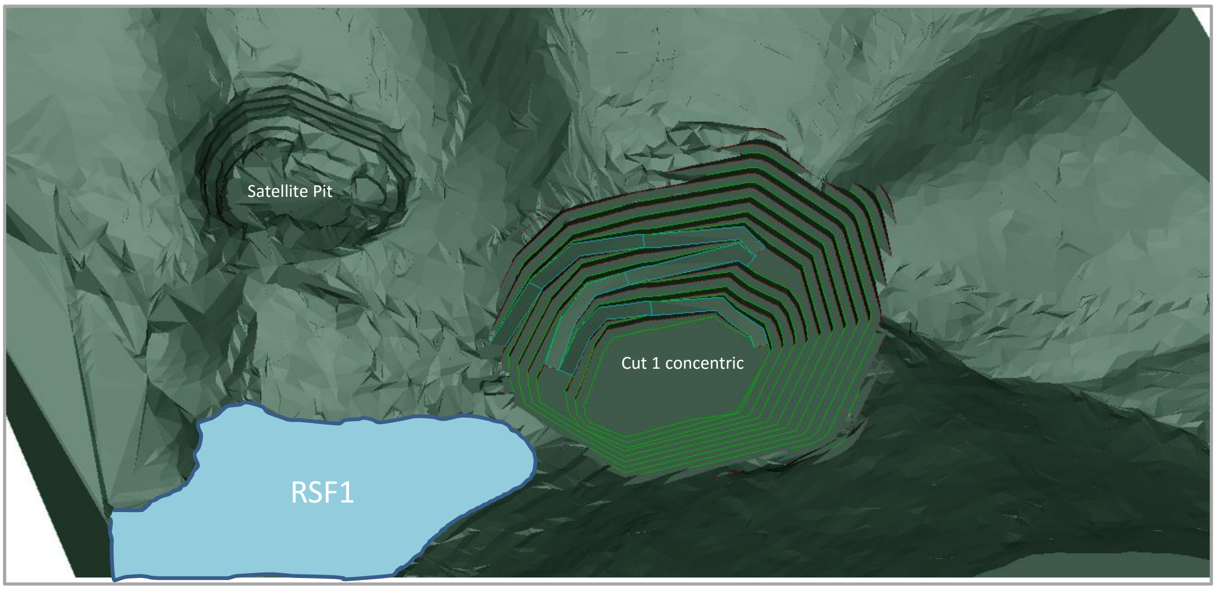



*Figure 45. Cut 1 and RFS1*

Cut 1 continues mining up to month 41 however haul roads will be needed outside the pit perimeter therefore consideration needs to be given to interaction with RSF1 from haul roads in the area. From the figure above, it can be seen that a haul road to the west of the final Cut 1 perimeter will start to impinge on RSF1, therefore it is suggested that a plan must be in place to have the slimes in RSF1 removed from this area by around month 36. Ultimately the final pit limit will expand to encroach upon approximately half of the RSF1 surface area as shown in the following diagram.

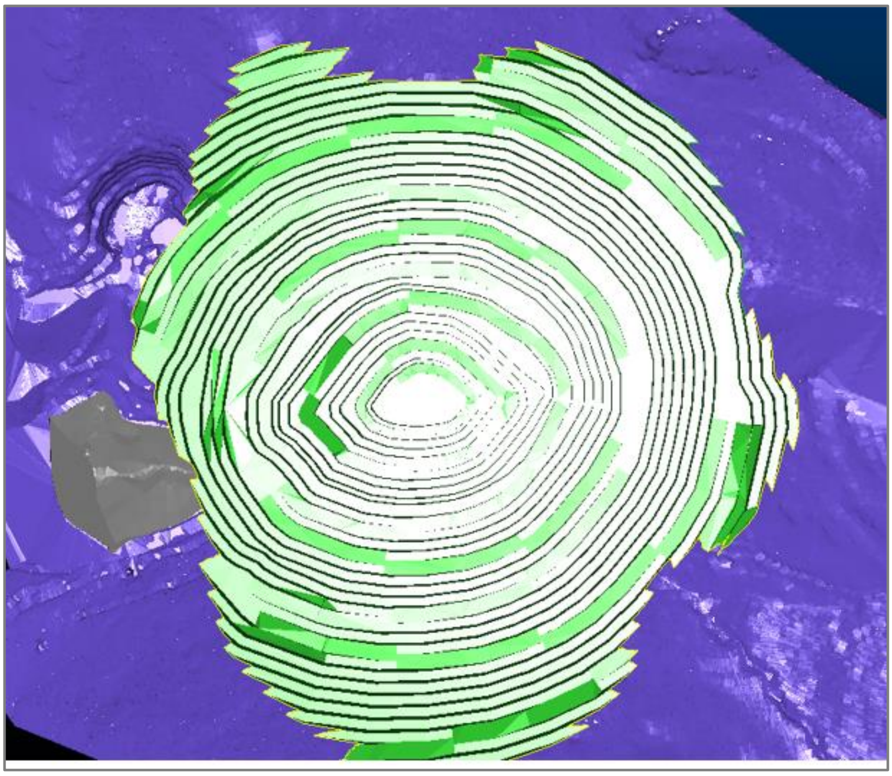


*Figure 46. The final pit limit and RSF1*

*The Satellite pit*

The Main pit intersects the nearby Satellite pit during year 8 of the operation at the 2687 bench elevation. A representative view of the pits interaction can be seen in the Figure below.
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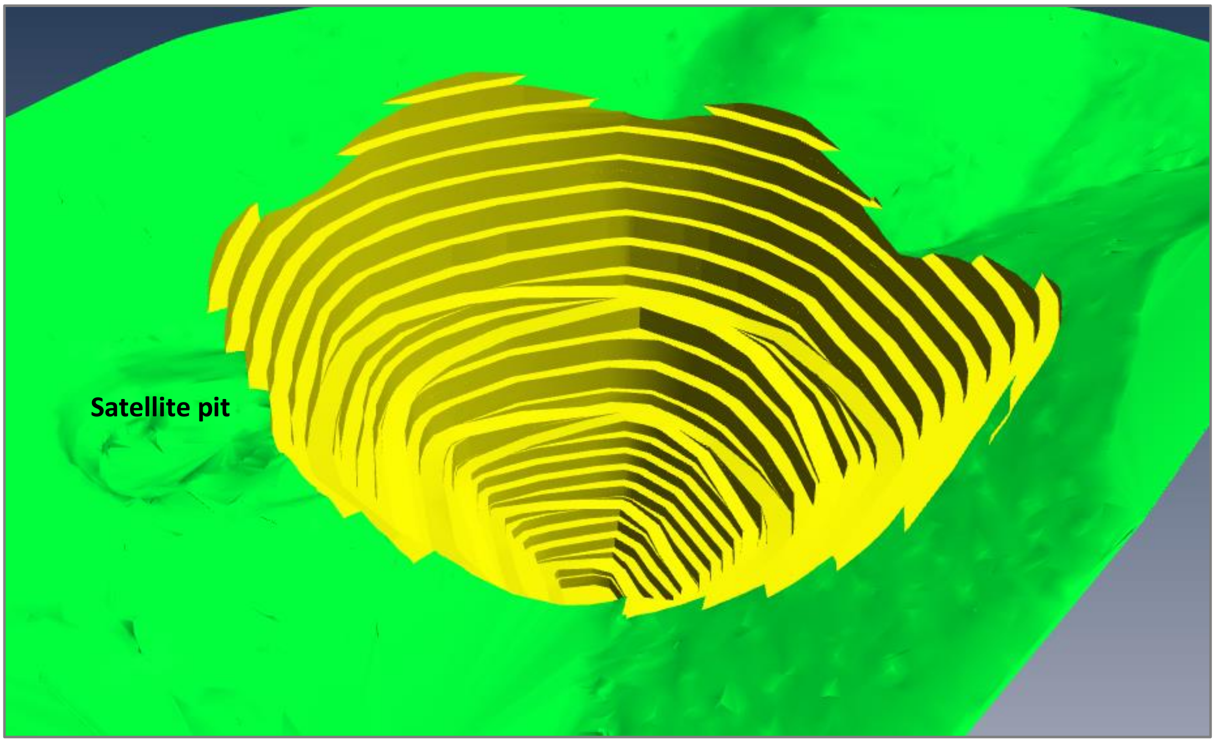



*Figure 47. Main pit showing intersection with the Satellite pit*

The Satellite pit will be used as a water storage facility at Liqhobong so at this point alternatives will have to be in place for another facility to become available. From discussions with LMDC, this scenario has been understood from the out-set and a number of plans are under review by LMDC to deal with this.

## 5.5 Staffing

### 5.5.1 Mining Contractor

Mining personal will be supplied by the mining contractor who will be responsible for the mining operations. The contractor will be carrying out drilling, blasting, loading and hauling. All relevant support activities such as haul road maintenance, pit de-watering, explosive storage control and fleet maintenance will be carried out by the contractor. A mining manning schedule was developed for this purpose and was sourced from a potential mining contractor based on the expected mine plan at the time of the initial planning (Ferreira 2014). The following numbers were presented:

| MMIC Category Summary | Year 1 | Year 2 |
|---|---|---|
| Management | 2 | 2 |
| Senior staff | 9 | 9 |
| Operators | 97 | 128 |
| Junior staff | 49 | 49 |
| Labourers | 21 | 21 |
| **Total** | **176** | **207** |

*Table 42. Mining contractor's 2 year manning schedule*

The above numbers include an additional 10% compliment for the operators category to cater for hot-seat change overs, sickness and unexpected leave requirements. This is to ensure that the mining fleet is fully operating at all times.

From year 3 onwards as waste tonnes increase, the contractor’s manning levels will increase and additional accommodation units will be built. The peak mining labour will be in year’s five to nine. Thereafter waste stripping levels will decline.

### 5.5.2 Mining and MRM

All mining technical support will be based on LMDC staff consisting of mining management, MRM, and survey. All positions will be day shift only with the exception of the grade controllers who will work on the same 12 hour shift roster as the mining contractor. Drill and blast engineering skills will be outsourced on an as needed basis. This is in addition to the drill and blast expertise that will be part of the mining contractor’s complement. An organogram of the planned structure is shown below which consists of 17 people.

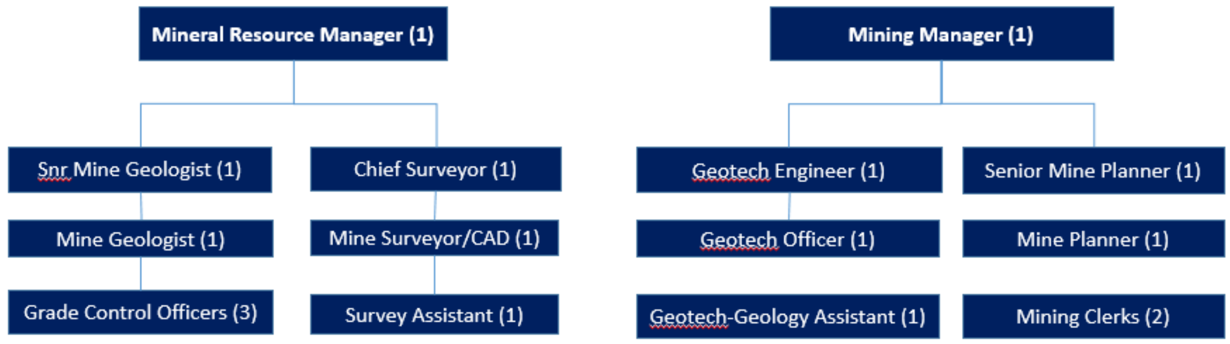



*Figure 48. Mining and MRM staffing structure*

### 5.5.3 Treatment Plant

The treatment plant will be staffed fully by LMDC personnel. The plant itself will be run on a 2 x 12 hour shift basis with a total on-shift compliment of 17 people who will run the process up to the sort-house. The sort-house will be a day shift only operation and will have a compliment of 9 people. A Plant Superintendent, a Plant Metallurgist and a Recovery Superintendent will oversee the entire operation on a day shift basis. The total staff required to operate and maintain the plant over 24 hrs, 365 days within the legal work hour limitations, and using the various shift configurations, are 120 people.

An organogram of the staffing structure is shown below.

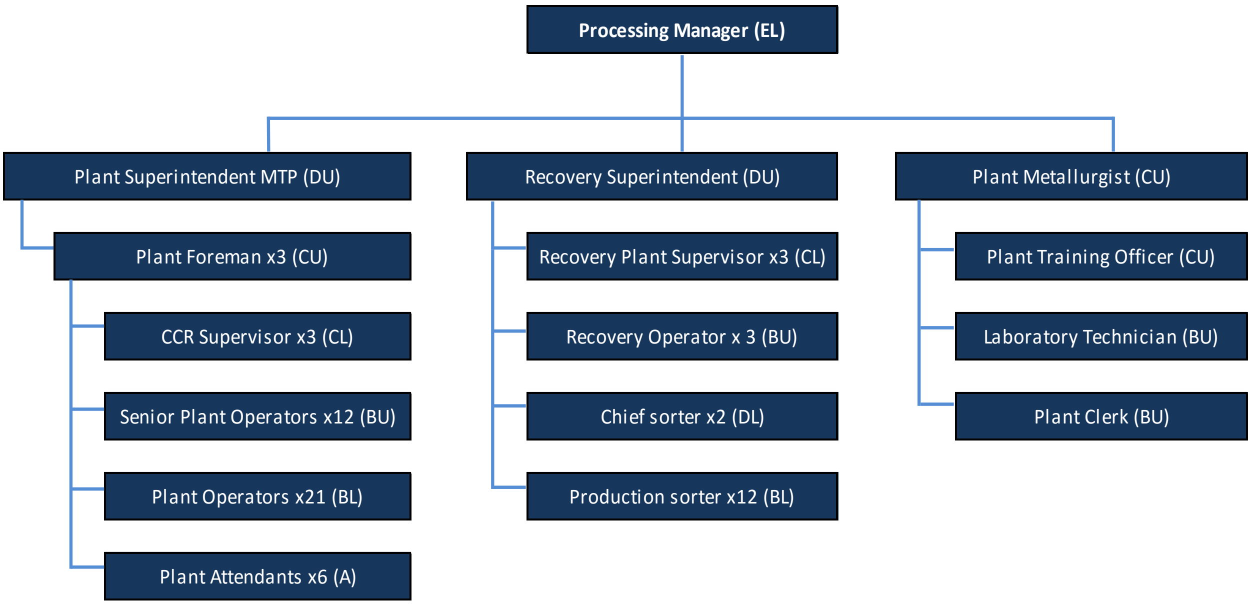



*Figure 49. Treatment Plant staffing structure*

In addition to the LMDC treatment plant staff, the tailings, slimes disposal and waste rock impoundment wall construction will be managed and operated by a contractor with a total team of 55 people. This will include five management and supervisory staff on a day shift basis and 13 operators per shift on a three shift basis.

### 5.5.4 Engineering

The engineering staff compliment will be all employed by LMDC and will cater for plant and general site maintenance. Engineering will be carried out on a day shift roster. Breakdowns on the night-shift will be dealt with on a call-out basis. An organogram of the staff compliment is shown below.

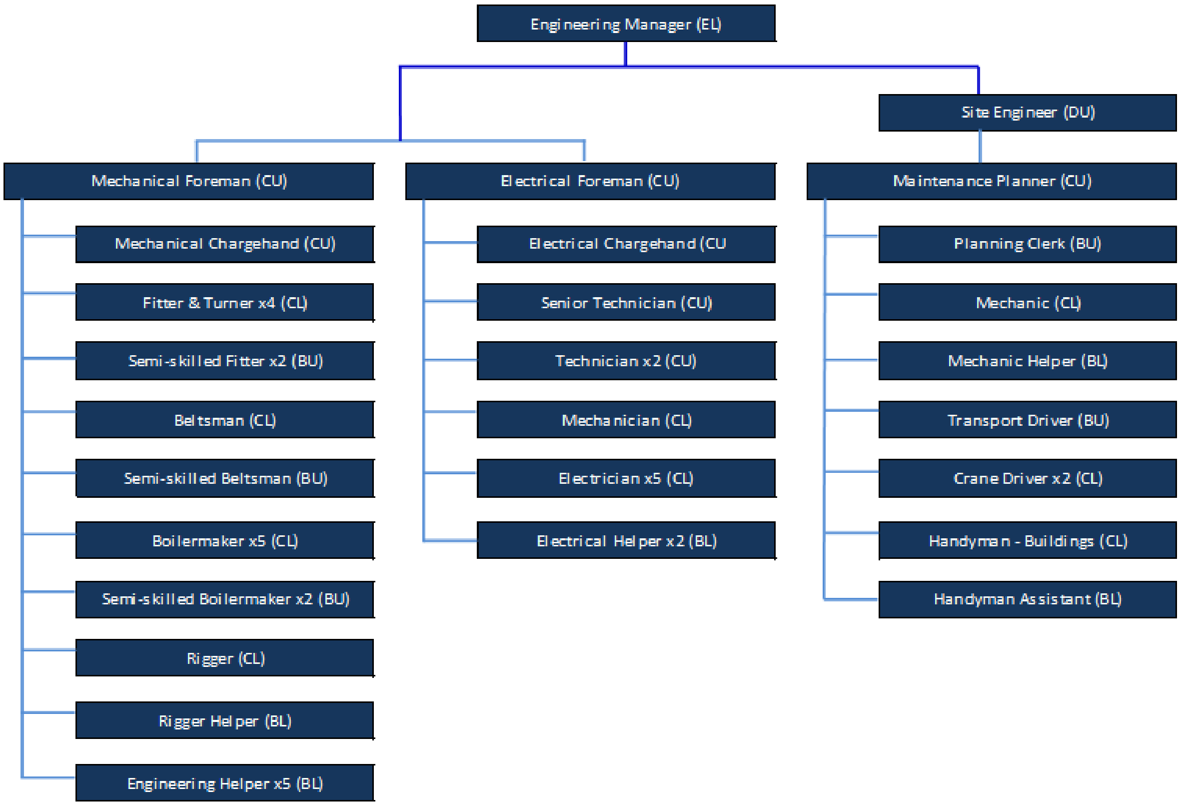



*Figure 50. Engineering staffing structure*

### 5.5.5 Supporting services

Supporting services consists of Finance and Administration, Human Resources, SHE and Security. The Finance and Administration department consists of 20 staff of which eight will be based in Maseru including the Financial Manager. Human Resources will have a compliment of seven, all based on site. The SHE department will consist of 9 people including two paramedics and two occupational nurses. Security will consist of 23 LMDC staff including 7 security officers for the Red Area. The Blue area will be manned by a contracting company with a complement of 13 people.

## 5.6 Economic Criteria

This section of the report describes and justifies all economic criteria that have been used in the compilation of the Liqhobong Mine Diamond Resource and Reserve.

### 5.6.1 Capital costs

*Construction capital*

The construction capital cost estimate is defined at a DFS level estimate comprising detailed and semi-detailed cost estimates with minor costs items being factored costs. The estimates were produced by DRA which is the EPCM contractor for the project. The estimates were compiled from firm vendor quotations and budget quotations and in-house data provided by DRA and are at an accuracy of +10% to -10%. Following the completion of the designs and associated cost estimates, a value engineering exercise was carried out to further optimise designs where possible and streamline costs.

Budgeted construction capital costs are defined as per the following breakdown.

| Capital breakdown | Rands |
|---|---|
| Civils and Earthworks | 235 227 980 |
| Treatment plant and recovery | 506 686 599 |
| Residue storage facility | 298 174 476 |
| Electrical | 92 262 307 |
| Buildings and general site | 83 154 476 |
| Water storage facility | 26 550 000 |
| Mining preparation | 12 000 000 |
| Pre-production | 40 367 016 |
| EPCM costs | 123 854 995 |
| Owners team costs | 63 435 416 |
| Contingency | 74 688 718 |
| Estimating accuracy allowance | 66 669 592 |
| Escalation | 87 177 739 |
| **Total** | **1 710 249 314** |

*Table 43. Construction capital cost breakdown*

The EPCM costs includes project management, detailed engineering design, procurement and on-site construction management and commissioning. The combined amounts of contingency, estimating accuracy allowance and escalation are 22% of the base capital amount. The capital budget will be spent over three years starting in June 2014.

Subsequent to the completion of the capital budget and a year into construction a number of delays were encountered that resulted in a project over-run cost of R156 million being added to the total capital outlay. This over-run is included in the project NPV calculations.

*Stay in business capital*

Stay in business (SIB) capital excludes mining fleet replacement as this is catered for by the mining contractor and built into the mining costs. Therefore SIB capital covers the treatment plant and associated facilities and general site infrastructure replacement costs. Due to operations beginning with the newly completed treatment plant and infrastructure it was deemed that lower SIB capital would be required during the beginning of the operation. It was estimated that R12 million per annum would be required over the first three years. From year four onwards an amount of R32 million rand per annum would be allocated including R22 million for the treatment plant (calculated as 10% of treatment plant operational costs at R62 per tonne or R223 million per annum). R10 million per annum is allocated for general site and infrastructure SIB capital.

#### 5.6.2 Mining Costs

As the Liqhobong expansion project was in the construction phase at the time of conducting the LoM planning, there were no actual site operating costs available for the planning inputs. Previous mining operations at LMDC were at a much smaller scale feeding a 50 tph plant with minimal waste stripping and therefore these mining costs are not comparable for use in the current planning situation. Operating costs from similar operations using mining contractors were assessed as a guideline for LMDC mine planning purposes (Letseng Diamond Mine in Lesotho and Karowe Diamond Mine in Botswana). Waste mining costs from Letseng have been quoted at R29 per waste tonne in 2015 terms (Letseng 2015). Karowe had declared mining cash costs as at 2014 of US$3.02/tonne for ore and US$2.97 per tonne for waste (Lucara 2014). With an US$ exchange rate of R11.00, these numbers equate to R33.22 per tonne for ore and R32.67 per tonne for waste respectively. As Karowe is a new mine employing a mining contractor and with similar shallow mining depths and tonnages when compared to LMDC, these numbers are considered suitable as a comparison for LMDC planning purposes. Unit costs of R33.50 per tonne for waste and R35.00 per tonne of ore were selected for the Whittle pit optimisation exercises.

To account for the mining cost increase with depth, a calculation was carried out which resulted in an estimated R0.44 per tonne per 10m bench. The cost was calculated based on increased fuel consumption for the additional haul distance and the increased cycle time in terms of the contractor's cost. The calculation is shown below for a 10m bench height.

| Mining Cost Adjustment Factor (MCAF) | | units |
|---|---|---|
| **Fuel cost for distance traveled due to increased depth** | **CAT 777** | |
| Bench height | 10 | metres |
| Distance travelled per additional bench (both ways) | 200 | metres |
| Average travel speed | 18.5 | km/hr |
| Average travel speed | 5.1 | metres/sec |
| Time taken to travel on additional bench | 38.9 | sec |
| Time taken to travel on additional bench | 0.011 | hr |
| Specific fuel consumption | 65 | litres/hr |
| Fuel consumption over additional distance | 0.70 | litres |
| Fuel cost per litre | 10.17 | R/litre |
| Fuel cost over additional bench | 7.15 | Rands |
| Tonnes moved per truck | 90.00 | |
| Rand per tonne per bench | 0.08 | |
| | | |
| **Contractor Haul cost** | | |
| Unit contract haul rate | 5.46 | R/tonne |
| Average haul distance for unit contract rate (both ways) | 3 000 | metres |
| Additional haul distance per bench | 200 | metres |
| Additional haul distance as % of base distance | 7% | |
| Additional haul cost per bench per tonne | 0.36 | Rands |
| | | |
| **Total MCAF per tonne per bench** | **0.44** | **Rands** |

*Table 44. Cost adjustment factor for depth*

Using a 14m bench for the second split shell design increased the MCAF to R0.62/bench. The cost of fuel is at May 2015 with prices quoted for LMDC delivery and the R5.46/tonne is the estimated haul cost only from a mining contractor as per a preliminary cost proposal as at September 2014.

### 5.6.3 Treatment costs

For treatment costs, a number of R61 per tonne ore processed was estimated for the LMDC planning process. This was based on a calculated number from the LMDC Metallurgical team using costs and inputs relevant to an owner operating model which is planned for the operation. A summary of the estimated cost is shown below.

| Processing operating costs per tonne | R/tonne |
|---|---|
| Labour | 12.21 |
| Reagents & Fuel | 6.82 |
| Electricity | 12.45 |
| Process Consumables | 1.19 |
| Maintenance Spares & Consumables | 12.85 |
| RSF & tailings management contract | 15.27 |
| **Total** | **60.79** |

*Table 45. Treatment plant operating unit cost*

#### 5.6.4 Overhead costs

An annual amount of R60 million was allocated for overhead costs. Overhead costs consist of the various LMDC departmental costs listed below:

- Human Resources
- Mining management, technical services and MRM
- Safety Health and Environment
- Security
- Corporate overheads

#### 5.6.5 Rehabilitation costs

A mine closure study was carried out and an amount of R43 million in present value terms was estimated for the LoM closure. A monthly budget of R216,000 per month (R2.6 million per annum) has been allocated as a cash contribution to an investment fund that will deliver R43 million at the end of the open-pit operation.

#### 5.6.6 Diamond Prices

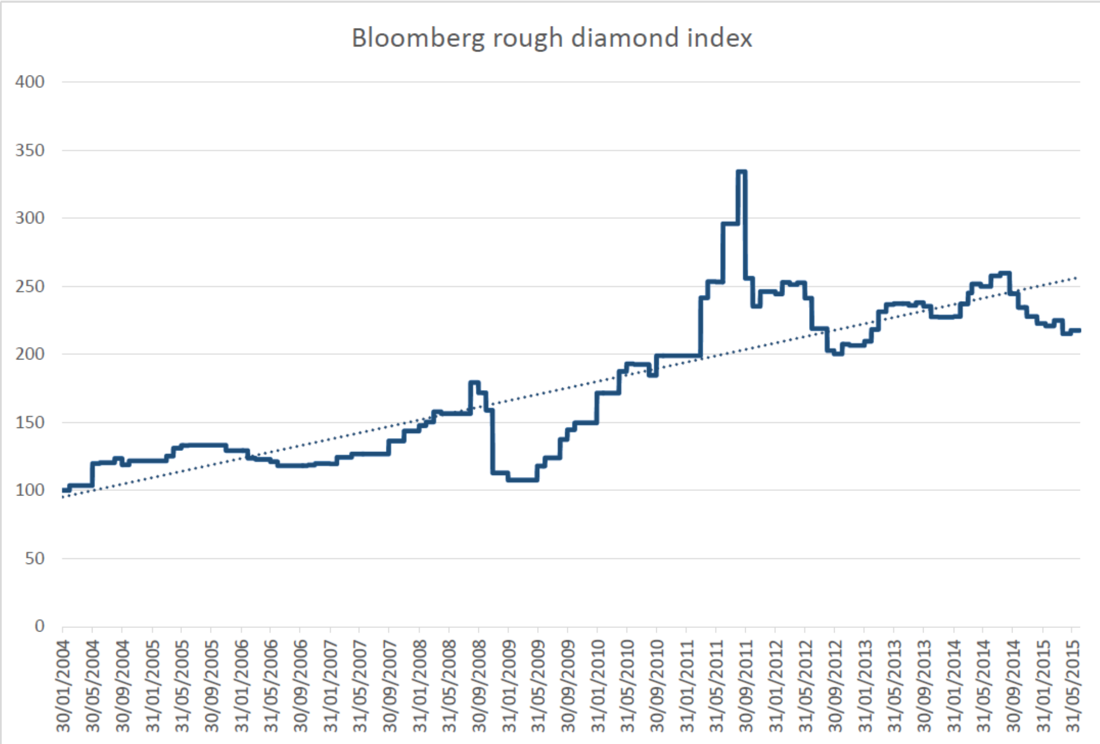
Diamond prices as at August 2014 for use in the Whittle Optimisation were determined by First Element and modelled by Z Star as described earlier in this report in section 4.5. The prices determined for the different facies are shown below;

- K2: $134/ct
- K4: $115/ct
- K5: $133/ct
- K6: $115/ct

#### 5.6.7 Diamond price escalation

LMDC has selected 3% annual real price escalation for diamond pricing which is based on historical trends and independent projections. As an opening statement on the outlook for the diamond market, a report, Prospects on the Diamond Industry (McKinsey 2014) quoted “even under the most pessimistic demand scenario and the most aggressive supply scenario, the fundamentals of the industry will likely be positive with demand outpacing supply growth”. The report paints four global scenarios all with GDP growth in the 3% to 4% range until 2025 and suggests that diamond pricing will mirror global GDP growth.

The Bloomberg Rough Diamond Index has tracked composite prices since 2004 and has recorded an average annual nominal growth of 7.1%. The average annual US CPI over this period was 2.3% resulting in an average annual real price increase of 4.8%. A chart of the index is shown below.




*Source: Bloomberg*

*Figure 51. Bloomberg Rough Diamond Index*

Data compiled by the Kimberley Process from 2004 indicates an average annual price growth of 4.8% (based on a simple averaging of global pricing trends), in-line with the Bloomberg data. A table of the data is shown below.

| $/ct | 2004A | 2005A | 2006A | 2007A | 2008A | 2009A | 2010A | 2011A | 2012A | 2013A |
|---|---|---|---|---|---|---|---|---|---|---|
| Australia | 16 | 17 | 19 | 20 | 22 | 20 | 25 | 28 | 29 | 32 |
| Botswana | 83 | 90 | 94 | 88 | 101 | 81 | 117 | 170 | 145 | 156 |
| Russia | 57 | 67 | 67 | 69 | 68 | 67 | 68 | 76 | 82 | 82 |
| Canada | 130 | 118 | 106 | 97 | 152 | 135 | 195 | 236 | 192 | 181 |
| South Africa | 76 | 85 | 91 | 93 | 96 | 144 | 132 | 211 | 145 | 146 |
| Zimbabwe | 180 | 141 | 32 | 46 | 55 | 21 | 40 | 56 | 53 | 52 |
| Rest of World | 56 | 62 | 67 | 68 | 61 | 56 | 72 | 93 | 84 | 115 |
| Total | 64 | 66 | 69 | 72 | 78 | 69 | 90 | 116 | 99 | 108 |
| Annual average compound from 2004 (%) | 0.0% | 2.8% | 3.8% | 4.0% | 5.1% | 1.5% | 5.9% | 8.9% | 5.6% | 6.0% |

*Table 46. Annual diamond price increase by country*

Projections from WWW Diamond Consultants indicate that demand will outstrip supply considerably over the following 10 years as mine production remains generally flat and then declines as a number of operations close down. Demand is expected to continue to grow primarily from developing markets and a steady US market. The gap between supply and demand is forecast to grow into the 2020's as shown in the chart below.

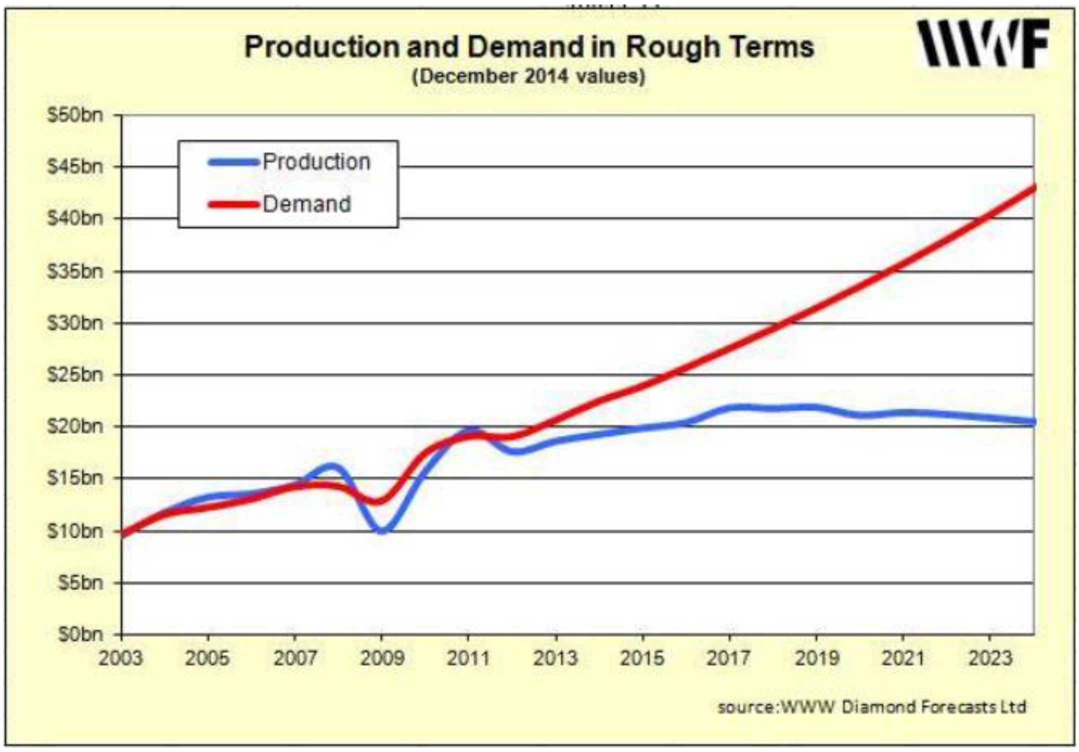



*Figure 52. WWW supply-demand projections*

Based on the above supply – demand scenario, WWW has projected prices forward based on recent price indices. The various scenarios are shown below.

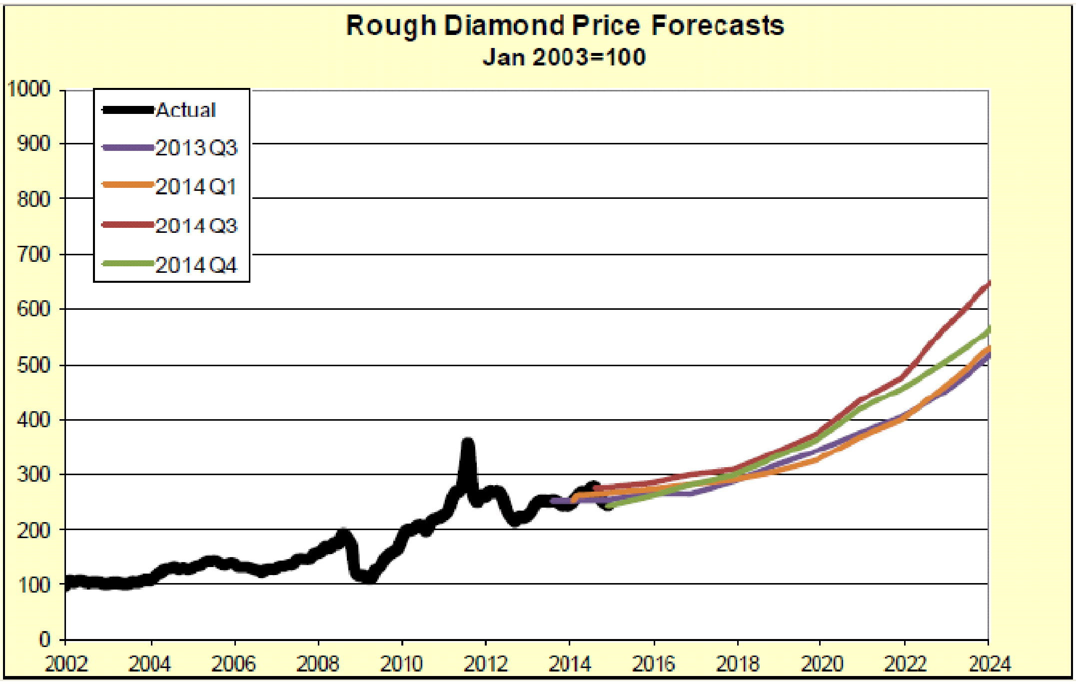



*Figure 53. WWW rough diamond price forecasts.*

Using the lowest growth trend from the four scenarios, the forecast price index to 2024 equates to an average annual growth of 6.9% nominal. Assuming an annual inflation rate of 2.5% this suggests real annual growth around 4.5%. We would consider this estimate to be on the optimistic side and therefore believe that 3% represents a more realistic growth trajectory.

5.6.8 Selling and marketing costs

A selling and marketing fee of 1.87% of diamond revenue was used for planning and financial inputs. This is determined from an agreement with a diamond marketing company, First Element which has been engaged by LMDC to manage the diamond sales process. This fee includes managing the diamond stock control system in the sort-house on-site, transporting diamonds to Antwerp, cleaning, secondary sorting, conducting diamond valuations and managing eight sights a year.

5.6.9 Discount rate

An 8% discount rate was used for the Pit Optimisation process and subsequent financial modelling.

5.6.10 Government Royalties

Under the Mining Agreement, an 8% Royalty based on revenue is applicable to LMDC. The Lesotho Government subsequently amended the Agreement to allow for a 4% royalty payable until the Company had paid US$20 million in royalties following which the 8% royalty would come into force.

For the purposes of the planning inputs, it was estimated that the 4% Royalty would be in place for the first 48 months of the operation thereafter reverting to the 8%.

5.6.11 Company Tax

Tax calculations for the financial modelling are based on a 25% tax rate throughout the life of the operation. Taxable income is calculated by allowing for capital expenditure to be carried forward including assessed losses accumulated prior to the start of the Project. Construction and ongoing SIB capital expenditure is expensed in the year that it is spent for the calculation of taxable income.

## 5.7 Mine Plan Reconciliation

The table below compares the 2015 mine plan outputs to those produced in the updated 2013 DFS.

| | Waste Tonnes (millions) | Stripping Ratio | Ore Tonnes (millions) | Grade (cpht) | Carats (millions) | LOM Revenue ($/ct) | LOM $/tonne | Exchange rate (R/$) | LOM R/ tonne | Overall $ revenue (millions) | Overall R revenue (millions) |
|---|---|---|---|---|---|---|---|---|---|---|---|
| **2013 Mine Plan** | 122,6 | 2,3 | 53,7 | 32,1 | 17,1 | 146 | 46,90 | 10 | 469,01 | 2 505,34 | 25 053,39 |
| **2015 Mine Plan** | 105,1 | 2,0 | 52,1 | 27,3 | 14,2 | 165 | 45,05 | 13,25 | 596,85 | 2 344,10 | 31 059,29 |
| **% Difference** | -14,2% | -11,6% | -3,0% | -14,9% | -17,1% | 12,8% | -4,0% | 32,5% | 27,3% | -6,4% | 24,0% |

*Table 47. Mine plan reconciliation*

The new mine plan shows a reduction in overall waste tonnes and stripping ratio compared to the 2013 plan. The large decrease in grade and carats is largely as a result of the boart diamond removal, BCO change and volume reduction described in the Diamond Resource section. The escalated average revenue over the life of mine has increased by approximately 13% from US$146/ct to US$165/ct.

# 6. TREATMENT PLANT AND ASSOCIATED INFRASTRUCTURE

## 6.1 Process Design

The processing plant design was focused on flexibility and simplicity to achieve a practical and cost effective extraction system. The following key factors guided the design considerations:

- Optimisation of revenue (and not necessarily carats)
- Maximising diamond recovery
- Minimising power consumption
- Minimising water consumption
- Minimising footprint (within the constraints of enclosing the plant due to harsh climatic conditions)
- Use of proven technology

Information from the pilot plant operations, and from the formal ore dressing studies undertaken on bulk kimberlite samples and drill core have allowed for an adequate understanding of the processing characteristics applicable. Ore dressing studies were conducted on two run of mine bulk samples, two drill core samples and various bulk samples. Various vendors were involved in conducting the metallurgical test work and these are summarised below:

<u>Mintek</u>

- Particle size distributions
- Scrubbing test work
- Heavy Liquid Separation test work (HLS densitometry)
- Comminution Characterisation:
    - Bond crushability work index
    - Bond abrasion index
    - Uni-axial compressive strength
    - Densities
    - HLS product sizing

<u>IMS</u>

- HPGR test work
- Cone crusher test work

<u>Paterson & Cooke</u>

- Kimberlite slurry flow behaviour tests

<u>Pastethick & Associates</u>

- Thickener test work

<u>Flow Electronics</u>

- Diamond and particle x-ray luminescence evaluation

The ODS work concluded that the Liqhobong kimberlite is easily scrubbed. The crushability test work and the uniaxial compression tests indicate the kimberlite is relatively soft. The samples also indicated low abrasiveness. Density profiling indicated that no significant problems are expected in relation to DMS performance. Luminescence profiling also did not highlight any specific problems with regards to X-Ray recovery. A summary of the ore dressing study is available in Appendix 2.

The design outcome was a conventional processing circuit with a combination of jaw and cone crushers, coarse and fines dense medium separation and X-ray fluorescence technology for final concentration. The processing methodologies proposed for the new plant are similar to those in current use throughout the southern African diamond mining industry, utilising tried and tested technology that is well supported in Lesotho. This will be a typical diamond plant and can be expected to produce to predicted production specifications.

Transportation of the product from site will be by helicopter and adequate security measures have been designed to ensure that the product is exported safely from site.

Maximising potential for recovery of large diamonds has been addressed by considering the known diamond SFDs and selecting the process cut-off sizes accordingly.

Water recovery processes ensure that water consumption is optimised and residue disposal methods are similar to those employed in Lesotho and elsewhere in Southern Africa.

A series of LIMN simulations were conducted by PJ Rider & Associates to test the mass deportment and expected revenue recovery of the design flowsheet by using the design parameters and in-situ grade information, The outcome of the simulation (for sixteen different ore size distribution scenarios) confirmed that the flowsheet will be able to process tonnages effectively at a rate of 500tph. It also confirmed optimum extraction efficiencies in terms of liberation potential and grade recoveries for the cut-off sizes selected.

## 6.2 Process Description

The ROM ore will be processed through a plant with a top cut-off size of 40mm, and bottom cut-off size of 1.25mm, comprising of the following unit processes (refer to Figure 47):

- Ore receiving and primary crushing
- Scrubbing and screening
- Secondary crushing
- Coarse DMS
- Fines DMS
- Tertiary crushing
- Final recovery and sort-house
- De-gritting and slimes thickening/process water reclamation
- Tailings disposal

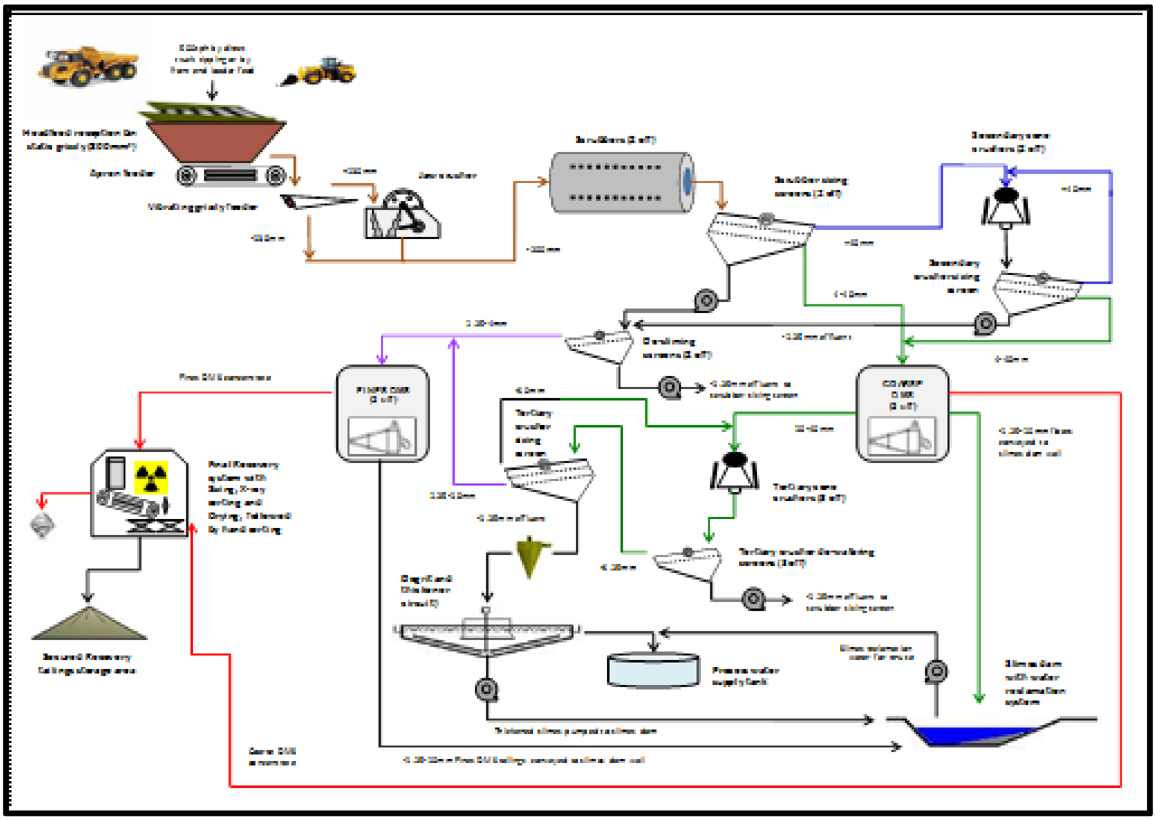


*Figure 54. Process flow schematic*

Primary Crushing will utilise a jaw crusher with a gap setting of 120mm, and will be located midway between the pit and the plant, with crushed material conveyed 290m to the plant. A crushed ROM stockpile could also be created as an operational buffer in the future and allowance has been made in the design of the front end to accommodate this

Primary Washing and Screening will be in two parallel scrubbers with sizing screens directing the >40mm oversize to the secondary crusher bin conveyor, the 4-40mm material discharges onto the Coarse DMS feed bin conveyor, and the <4mm material to the Scrubber De-sliming Screens.

Secondary Crushing will utilize two cone crushers and will operate dry and in closed circuit, choke fed and utilising surge pan feeders so as to optimise crushing efficiency and minimise diamond breakage. Oversize reports back to the feed bins, whilst +4-40mm material is conveyed to the Coarse DMS surge bin and fines (<4mm) are pumped to the scrubber de-sliming screens.

DMS Feed Preparation Section collects the <4mm material as well as the +1.25-10mm Tertiary Crusher product and feeds this to the Fines DMS Surge Bins.

Coarse DMS consists of two identical DMS modules that treat the high value +4-40mm material and operates as an open circuit, with the floats drained, washed and >10mm oversize conveyed to the Tertiary Crusher. +1.25-10mm float material is conveyed to the tailings system. The product is discharged onto the double deck sinks drain, rinse and sizing screen before being sized into three size fractions namely middles (4-8mm), coarse (8-16mm) and extra coarse (16-40mm) for processing through the Coarse DMS Recovery Module.

The fines DMS operates as above with two DMS modules, producing three size fractions namely fines (1.25-3mm), coarse (3-6mm) and extra coarse (6-12mm) fractions for processing through the Fine DMS Recovery Module.

The tertiary crushing section treats the nominally >10mm material from the Floats Screens top deck oversize of the Coarse DMS Modules, as a closed loop, with the +1.25-10mm material discharged to the fine DMS surge bin and oversize is re-crushed.

The two coarse recovery modules makes use of a dedicated middles X-ray machine, whilst the coarse/extra coarse X-ray machine operates alternatively on each concentrate feed, fed by a tube feeder. The X-ray machines are double pass, operated wet and require filtered water. Concentrate and tailings are dewatered and the concentrates report to relevant glove boxes by tube feeders, whilst the tailings report to the coarse recovery tailings conveyor.

The two fines recovery modules operate similarly, with the distinction that fines fraction concentrate from both modules is treated through an extra single pass wet recon X-ray Machine for further concentration before reporting to the drier and glove boxes.

The sort-house has four glove boxes dedicated to fines fraction, two to the middles/coarse fractions from the fines recovery modules, and one glove box for middles and one for coarse/extra coarse from the Coarse Recovery Modules. A dedicated glove box is used to de-falsify as well as size and weigh the product from the other glove boxes.

The Degrit and Thickener circuit receives the underflow from the Tertiary Crusher Sizing Screen and the Scrubber De-sliming Screens and pressure-feeds it to the Degrit cyclones. Underflow from the cyclones is dewatered with screen oversize reporting to tailings and undersize to the Thickener Feed Box for flocculent addition. Overflow from the thickener reports to the Process Water Dam, with underflow to Slimes Disposal Sump for pumping to the slimes dam.

Tailings disposal operations make use of tailings from the coarse and fine DMS circuits in the construction of the slimes dam wall. Recovery tailings are stockpiled.

## 6.3 Process Water Management Strategy

The water management strategy is to minimise the impact of water use on the environment by both reducing plant intake of fresh water through improved process technology that retains water in the plant, use of domestic waste water and harvesting of storm water. This approach fits well into the national water resources management plan which seeks to minimise the footprint of mining activities on available water supply resources which are a significant revenue earner for Lesotho.

The site water usage plan was formed around the following principles:

- Diverting Liqhobong stream water upstream of the mine and runoff from the pit using diversion trenches/drains and bunds;
- Capturing runoff from around the Main pit into the Satellite pipe pit and using this water as process water in the MTP as well as for dust suppression on haul roads.

The key physical elements of the water management strategy are:

- Satellite Pit storage facility (primary water supply);
- Process Water Dam;
- Temporary pit water storage (sump) and
- Residue Storage Facility (RSF) and associated Seepage Dam
- The construction of a water storage dam in the Liqhobong valley immediately below the waste rock dump is under investigation (to supplement stored water)

Rainfall runoff on the Project Site will be managed by a series of diversion trenches and drains, pit storage and seepage dams.

## 6.4 Product Security

The product security philosophy is consistent with industry standard security systems and includes:

- Limited personnel access to material (hands-off approach)
- Limiting access to only those staff that are absolutely necessary
- Minimizing access to equipment through physical barriers (grating and guarding) to prevent unauthorized access.
- A process design and layout (in as far as was practical) to fail safe, spill safe and limit access to product when failed

The following major security components were incorporated in the design:

- Access to the Mine Lease Area will be controlled by a 1.8m high perimeter fencing around the entire lease, and a high security Access Gate with turnstile card reader, sliding gate and boom, security office and layby for non-plant deliveries
- The facilities outside the Plant area will consist of administration and management offices, accommodation and messing, general purpose workshops and stores
- The Plant area access will be controlled by perimeter fencing, access control with turnstile and card readers, and a security complex for monitoring personal interviews, personal searches and X-ray searches.
- The Plant Red Area constitutes the recovery and sort-house sections and with access through the Blue Area with a perimeter fenced no-man's land around access point.
- Separate access into the Export Zone for diamond export, complete with a double fence and secure entrance.
- Diamonds are transported from site using a helicopter for onward courier to the market
- The operation will make extensive use of CCTV technology in collaboration with Firestone's independent security consultants, who have also been involved in the design of the recovery and sort-house modules, the export room and safe.

## 6.5 Production Planning Parameters

The production plan is based on 365-day per year continuous operations. The table below summarises the process planning parameters:

- Hourly treatment rate (nameplate capacity) 500tph
- Plant availability 90%
- Plant utilisation 90%
- Overall process utilisation 81%
- Manned hours per annum (365 days/24hrs) 8 760hrs
- Effective production hours 7 096hrs
- Annualised treatment capacity 3.6 Million tons

## 6.6 Process Operating Costs

The forecast Processing cost (inclusive of plant maintenance, coarse tails disposal and slimes dam management costs) is projected at 60 Maloti per ton of ore treated. The breakdown of unit cost is shown in Table 49 on page 90.

# 7. MINE INFRASTRUCTURE AND UTILITIES

## 7.1 Design Considerations

There are various unique aspects associated with the mine site that have influenced the location and design of the buildings. When developing the plant layout, consideration of the topography and the weather had to be taken into account. The cold weather conditions in winter require that the main unit processes be located in buildings. The layout has been configured in such a manner that major unit processes have been located in the same area and can be enclosed in the same building, for example the tertiary crushing, scrubbing and de-sliming sections. The plant layout is also relatively compact so as to minimise the terracing required, which in turn reduces the capital cost (Figure 48).

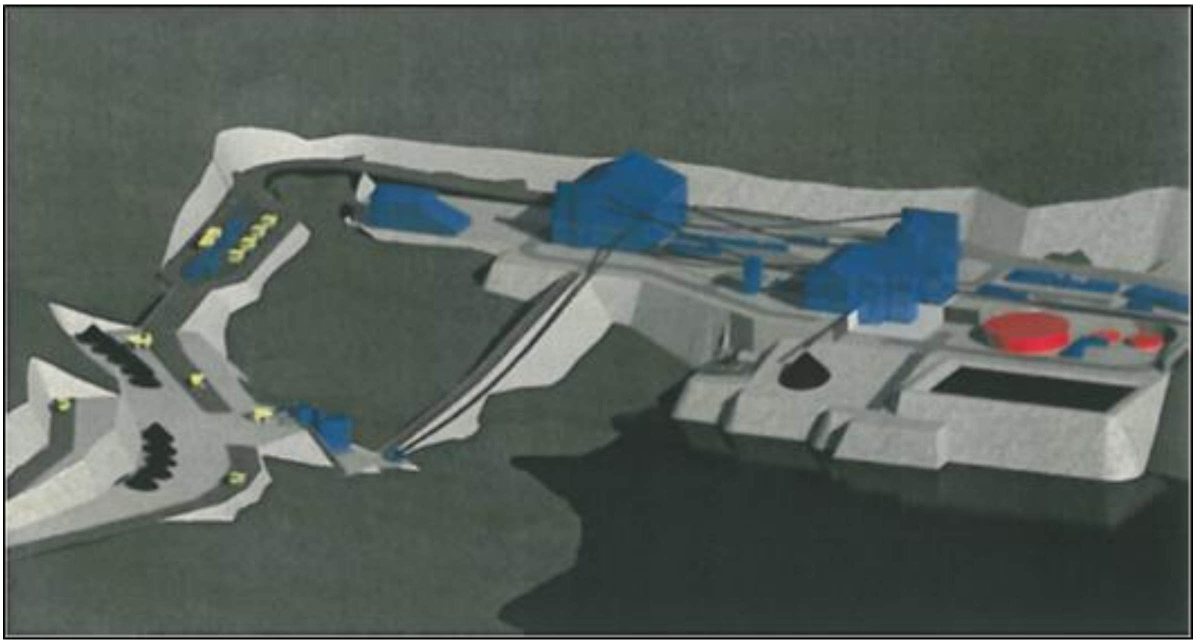


*Figure 55. Aerial view of plant and RoM area*

The topography has been used to enable the recovery plant to be integrated below the DMS sinks screens. This allows the recovery section to be gravity fed. Other practical considerations related to the design of the site layout are:

- Longer thinner buildings following the contours favoured over units crossing contours.
- Double storey units were considered in certain applications to minimise terracing.
- Covered verandas that offer shelter and protection whilst moving around the buildings were incorporated.
- The roof pitch on all buildings was set at 25 ° to help prevent snow build-up.
- All buildings are suitably insulated and heated where appropriate.

## 7.2 Power Generation and Site Electrical Reticulation

The Mine's expected installed capacity without Power Factor Correction (PFC) is 9.1 MVA; with PFC it reduces to 8.6 MVA. The project incorporates a 132 MVA Grid Power line that was constructed from Ha Lejone to the Liqhobong substation, and which will also has the potential to serve surrounding communities. A co-funding and power sharing agreement have been concluded with Storm Mountain Diamonds. The treatment plant reticulation has the following key elements:

- Emergency power has only been allowed for critical items;
- Separate emergency power has been provided for the mine village;
- Minimal power has been provided to the Northern slope - only for dewatering of the satellite pit through floating barge pumps;
- MCCs and mini-substations will be containerised to allow for pre-commissioning prior to transport, ease of installation and protection.

The level of instrumentation has been selected to ensure that the available skills local to the mine can service and maintain the provided systems. The plant will be controlled from a central control room and will be via various SCADA stations, complete with Historian® functionality for trending analysis and fault finding. The plant electrical design load is 8.03 MW. Electricity consumption is projected to be 7.5 MVA per month (0.025 kVA per ton of ore treated).

## 7.3 Roads and Terracing

The primary access to the Liqhobong Mine for all heavy vehicles (i.e. 12m tri-axle, and 12m low beds with maximum load of 32,000 kg) is through the Moteng Pass and Mothae Junction-Kao Mine route. This route takes advantage of lower gradients and road improvements already in place. Deugro (South Africa) Pty. Ltd. was commissioned to identify all upgrades needed on this route. The Ha Lejone route was also assessed and stretches requiring upgrading were identified by Deugro, who found the route to be limited to medium sized 6m vehicles with maximum loads of 13,000kg. This route is periodically compromised by high water levels over the low bridge crossing the Malibamatso River.

On the mine site, the main road from the entrance gate to the primary tip and workshop complex will be 14m wide to allow EMV vehicles to pass each other, and will follow contour lines to reduce cut and fill earthworks. All other roads will be 7m wide.

## 7.4 Accommodation and Offices

The accommodation blocks were placed approximately 1km from the process plant, close to the entrance to the mine. This was primarily to separate them from the process plant and mine, so as to reduce the noise levels in the accommodation area but secondly also due to the general lack of available space. The accommodation was also placed above and away from the slimes dam wall, to limit the effect of noise and dust that will be generated during the on-going construction of the RSF wall. The accommodation facilities include both skilled and unskilled labour housing blocks, security housing block, supervisor lodging blocks, and shared and single management units. Recreation blocks include the main kitchen and dining hall, secondary dining hall in the Plant Blue Area, social and communal TV areas, laundry, gym and computer area. Office blocks include three buildings for senior mine management, general mine management and a health clinic, are all positioned between the accommodation area and the process plant.

## 7.5 Maintenance Complex and Fuel Storage

The EMV workshop will be located adjacent to the process plant. The preliminary design incorporates a conventional structural steel building with 5 working bays large enough to accommodate a CAT 777. The facility also includes a stores area, offices and oil storage and compressor rooms.

The fuel storage facility and refuelling station has been located between the EMV workshop and the primary crusher tipping area, close to normal haul routes but separate from other mine traffic. The current total storage capacity envisaged is 400,000 litres which is planned for two weeks of operations at peak operating conditions.

## 7.6 Explosives Storage and Mixing

The site for the magazine and emulsion storage plant is ~200m east of the Satellite pit, well away from all traffic, operations and lodgings. The emulsion storage facility has been designed by AEL Mining Services (South Africa) and will provide on-site storage of 100 tonnes of emulsion initially, allowing an increase in capacity in later years up to 200 tonnes. Explosive accessories and detonators will be stored separately in two magazines.

## 7.7 Potable Water and Sewage Treatment

Once the construction phase is complete and the mine is in production, the potable water for the mine site will be supplied from the Satellite Pipe dam, which will be pumped to the potable water filtration system rated at 100m$^3$/day. Treated potable water is pumped to two 10m$^3$ tanks 35m above the process plant terrace, allowing gravity feeding to the accommodation, office and process plant areas.

As the mine is not located in an existing municipal catchment area, all waste water must be collected and treated before being discharged. The water quality must be suitable for discharge and has been specified to meet or exceed the Department of Water Affairs requirements. The waste water treatment plant has been positioned outside of the plant area, such that when sludge waste and screen waste is removed off site, it does not pose a security risk. The treatment plant is modular, fully automated and uses the “Biological Nutrient Removal Hybacs Process”. It recovers all domestic water and processed water from the waste water treatment plant which will be gravity fed to the slimes dam.

## 7.8 Residue Storage Facility (RSF3)

The positioning and design of the slimes dam and tailings dumps were a major mine design component for the project given the topography of the site. Turnkey Civil Lesotho (Pty) LTD (TCL) was appointed by LMDC to carry out the necessary activities and tasks, in accordance with the specified requirements and scope of work, to present a design report for the new Residue Storage Facility (RSF).The RSF will be developed as a single compartment valley type storage facility (De Swardt, 2015). Downstream impoundment embankments will be constructed with tailings and waste rock and the basin created will be filled with slimes and grits. The overriding principle of the design is to create a safe and stable residue storage facility to minimise risks to personal safety and health and property.

Waste rock will be delivered to the embankment walls by the mining fleet, dumped and then dozed into the designed shape as part of the ongoing embankment wall construction. Coarse tailings will also be used to provide a skin cover on the waste rock as an integral aspect of the embankment wall design. Four lifts have been planned that will increase the capacity of RSF3 to enable the deposition of the slimes and grits over the planned LoM of 15 years. However RSF3 has been designed to cater for a potential 17 year mining life. Waste rock in addition to the requirements for the four lifts described above will be trucked and dumped further downstream of the embankment walls onto a waste rock dump (WRD). The slopes of the WRD will be terraced with engineered benches at fixed vertical intervals and the entire WRD will be incorporated into the overall RSF3 embankment structure. The following diagram outlines the progressive construction phases of RSF3 incorporating the WRD (Figure 49).

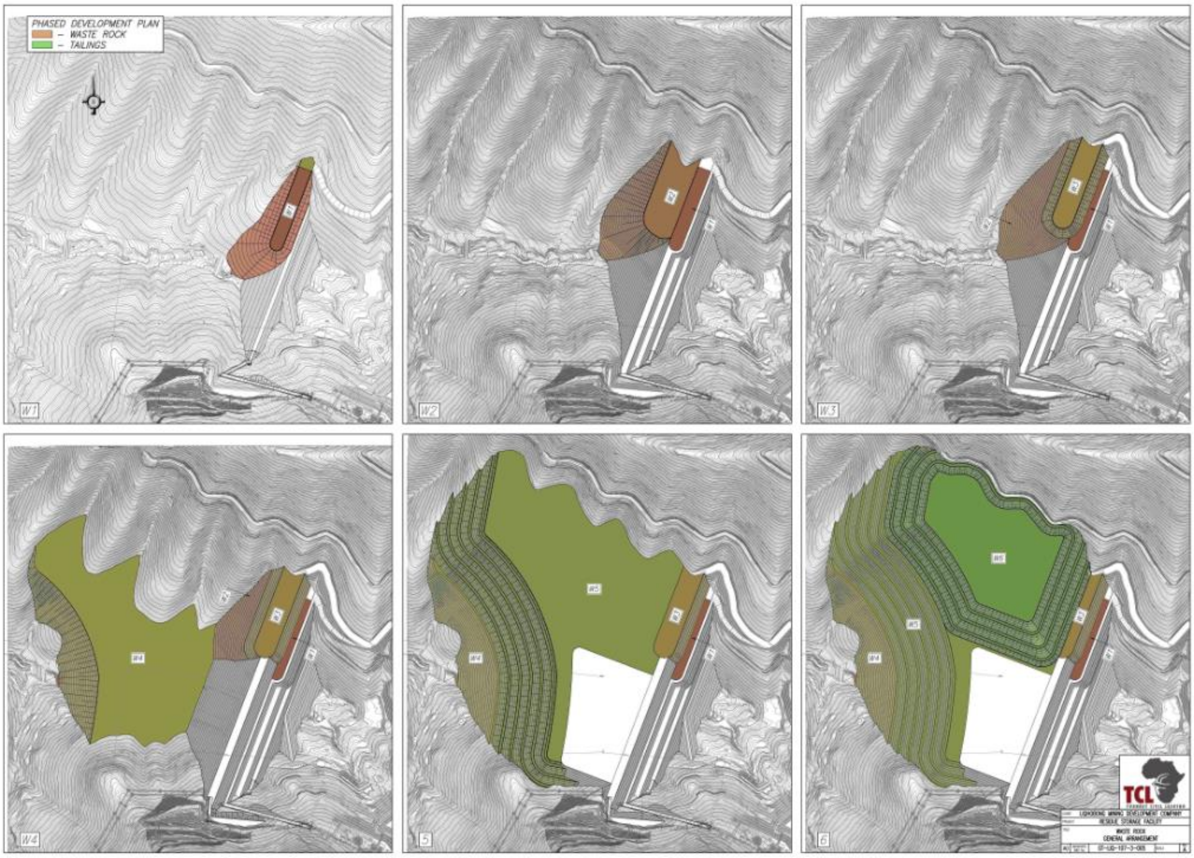



*Figure 56. Phases of RSF3 construction over LoM*

The material transport systems can be summarised as follows:

- The slimes and grits will be pumped to the RSF basin as a high-density slurry stream through two pipelines. Two associated slurry distribution pipelines will then divert the streams to designated delivery stations. The slurry stream is expected to segregate when discharged into the RSF basin.
- The tailings will be transported with a conveyor system. Initially the tailings will be placed in the impoundment embankments utilising a mobile conveyor, spreader and dozer. Excess tailings will then be placed in a designated dump utilising a mobile conveyor and finger conveyors.

The slurry (slimes and grits) will initially be placed behind an engineered starter embankment. A downstream construction method will be implemented above the starter embankment crest elevation. Vertical freeboard will be constructed through the implementation of cross-valley impoundment embankments.

The supernatant water will be decanted from the RSF basin, utilising a decant barge. Excess water will therefore only be temporarily stored in the RSF basin during high rainfall periods. The decant pipeline diverts the return draft to the return water dam (RWD). The RWD is lined with a 2 mm thick HDPE geo-membrane in order to minimise seepage losses and to ensure long-term structural stability.

Seepage will report to the Seepage Dam located downstream of the impoundment embankments and WRD. Initially, temporary Seepage Dams will be utilised before the final permanent Seepage Dam is implemented.

# 8. ENVIRONMENTAL AND SOCIAL CONSIDERATIONS

Formal mining operations commenced at Liqhobong in 2003 by Kopane Diamonds. The earlier operations focussed on kimberlite extraction from the Satellite Pipe. The focus of this current study is to expand the Main Pipe operations with the construction of a 500 tph processing plant and associated infrastructure.

The original Environmental Impact Assessment (EIA) was undertaken by SRK Consulting in 2000 to address the Kopane operations. After Firestone acquired the property, a comprehensive Environmental and Social Impact Assessment (ESIA) was undertaken by Loci Environmental in 2012 to address the proposed expansion. Based on the ESIA, LMDC compiled an Environmental Management Plan (EMP) that was submitted to the Lesotho Department of Environment (DoE). The DoE issued an Environmental Clearance Certificate for the expansion project during August 2102.

On 6 February 2014 the DoE granted exemption to LMDC with regard to the following permits:

- Pollution License
- Effluent License
- Noise Permit
- Waste License
- Ionising Radiation license

The exemption was granted because the DoE does not have procedures or regulations in place to issue the permits although they are stated as a requirement in the 2008 Environment Act.

In April 2013, a Resettlement Action Plan (RAP) was completed for the mine expansion. The RAP focuses on the resettlement of 22 households as a result of the mine expansion. Since the RAP, the number of households to be relocated increased to 26. The construction of the new houses is currently in progress and scheduled for completion by the end of 2015.

While there is no legislated requirement to have a financial provision in place to address closure and rehabilitation, the Mines and Minerals Act contains a reference to adequate and ongoing financial provision for compliance with obligations. LMDC is currently in the process of finalising their rehabilitation and closure plans.

# 9. GOVERNMENTAL CONSIDERATIONS

As stated earlier, the Liqhobong Mine is operated by Liqhobong Mining Development Company (Pty) Limited, which is 75% owned by Firestone and 25% owned by the Government of Lesotho. During April 2014, Firestone and its subsidiaries entered into a Revised Mining Lease Agreement with the Lesotho Government, represented by the Honourable Minister of Mines, which shall govern the terms under which LMDC shall mine diamonds at the Liqhobong Diamond Mine for an initial period ending on 30 June 2024. It is expected that the Mining Lease Agreement will be renewed as the agreement allows two consecutive ten year extension periods, provided that LMDC has complied with the requirements of the Agreement and any other applicable legislation.

The Agreement grants LMDC mining rights for diamonds over an area of approximately 7.6 $km^2$ as defined in section 1.3 of this report.

The Mining Agreement was negotiated with the government led by the Prime Minister Mr Tom Thabane at the time. Construction on the project commenced mid-2014. Some political instability followed including an attempted coup attempt and it took direct intervention by South Africa and some intense diplomacy by Deputy President Cyril Ramaphosa to restore democratic order. The political settlement brokered by Mr Ramaphosa resulted in an early election that was won by current Prime Minister Pakalitha Mosisili in February 2015.

Subsequently, the former Lesotho army chief Lt-Gen Mahao was killed by soldiers near his home during June 2015. Lt-Gen Mahao had been accused by the new Lesotho government of plotting a military coup, and the official line is that he resisted and drew his firearm when soldiers were sent to arrest him. As a result of the murder, the Southern African Development Community leaders intervened and decided to establish an independent commission of inquiry headed by a judge from Botswana to investigate the circumstances leading to the murder. The investigation is currently in progress.

Given the above, the current political climate in Lesotho is tense and the government coalition is fragile. A good working relationship between LMDC and the key governmental departments such as labour, mining and environment is a key component to moving the project forward at the required rate. The biggest challenge and risk facing the Project at the moment is the slow awarding of work permits to the contractors that are currently building the mine.

Notwithstanding the above, Firestone and LMDC management have made good progress in building relationships with the key ministries and are confident that the Project construction will continue at the required rate.

## 10. MARKETING

The general consensus is that the long term fundamentals of supply and demand governing the diamond industry is positive due to ageing mines and limited new production areas. During 2014 McKinsey published a paper that considered a number of economic scenarios and concluded that even under the most pessimistic demand scenario and the most aggressive supply scenario, the fundamentals of the industry will likely be positive, with demand outpacing supply growth. At the same time, the industry is set for rapid change under any scenario as a result of various trends, such as the expected accelerated or further increase in mining costs, the pressure on the midstream to professionalise and the continued shift in demand to emerging markets (Goodman, Bratt and Brantberg, 2014).

First Element Diamond Services conducted 15 tenders on behalf of Firestone from December 2010 to November 2013 which led to the sale of 393,946 carats totalling US$31,349,846 in revenue. During that period a total number of 53 “Special Stones” larger than 10 carats in size were recovered of which the largest was an off-white 74 ct stone. In the graph below the special stones are plotted from small (10 ct) to large (74 ct) and shows the variance in $/ct per stone which is a function of shape, colour and size. The highest $/ct achieved was $13,486 for a white makeable stone of 26 carats followed by an intense yellow fancy stone of 11,5 carats that sold for $8,895/ct.

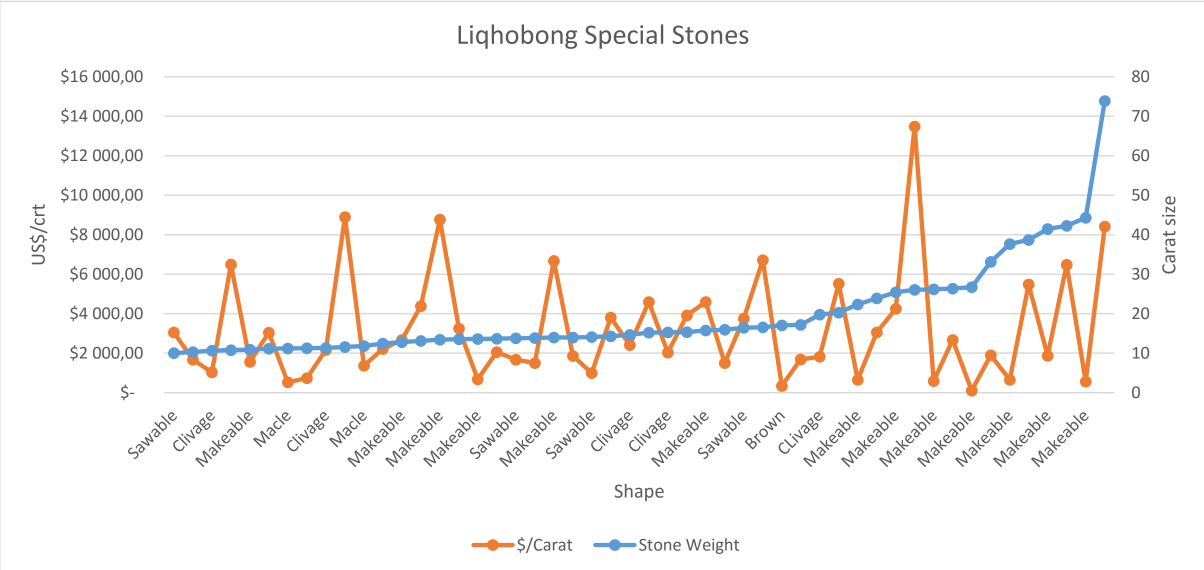



*Figure 57. Liqhobong Main Pipe diamonds recovered by Pilot Plant*

There are various sales options available to Firestone including off-take agreements for the high quantity lower value goods. The most common sales method employed by most junior and mid-cap diamond mining companies is to sell their diamonds by auction or tender. It is currently envisaged that Liqhobong diamonds will be sold in Antwerp during 8 tenders per annum. Typically each tender will allow a period where the diamonds on offer can be viewed in Antwerp by pre-approved buyers. Thereafter sealed bids can be submitted through an auction platform ensuring the highest prices are achieved especially for the exceptional stones. Typically special stones will have a reserve price set up front. Smaller “run-of-mine” stones are combined and offered for sale as parcels or boxes.

Based on the expected supply and demand outlook as referred to earlier in the report and in the sections above, Firestone Diamonds believes that diamond production from LMDC will be readily sold into the market at the prices estimated, notwithstanding possible short-term market fluctuations.

## 11. RISKS

Below is an extract from the company risk log highlighting the top five risks currently faced by the Project.

- Political risk – relates to instability within the current coalition government and paucity in issuance of work permits for expatriate contractors.
- Water supply risk – current water storage facilities on the mine may be insufficient in dry months to satisfy processing plant requirements. Investigations are underway to build a water storage dam downstream of the new RSF wall.
- Community risk – relates to the current relocation program and the compensation for lost communal grazing land that has to be addressed.
- Schedule and budget risk – the project is currently due for completion by the end of 2016 but can be delayed by a number of unforeseen incidents such as excessive rain, delay in contractor work permits, political instability etc.
- Site infrastructure risk – the rate at which the accommodation is being erected has a direct bearing on the amount of contractors that can work on site and thus is a risk if delayed.

## 12. DIAMOND RESOURCE AND RESERVE STATEMENTS

The Diamond Resource reflected below are the gross numbers inclusive of Diamond Reserves and from a LMDC company perspective (100% ownership). Firestone Diamonds PLC's ownership is 75%. The Diamond Resource estimate was independantly prepared by Z Star Mineral Resource Consultants by the competent persons listed in the compliance statement 13.1 and is SAMREC compliant.

**Diamond Resource Statement for Liqhobong Main Pipe as at 30 September 2015 (including Reserves)**

| Diamond Resource Category | Depth from and to | Source | DIAMOND RESOURCE | | | | | General Information | |
|---|---|---|---|---|---|---|---|---|---|
| | | | Volume in $m^3$ (Millions) | SG ($tonnes/m^3$) | Metric Tonnes (Millions) | Grade (CPHT) | Carats (Millions) | Resource Bottom cut-off (mm) | Value (US$/ct) |
| Indicated | Surface (2650 masl) to 2467 masl | Main Pipe - K2 | 7.843 | 2.59 | 20.302 | 24 | 4.802 | | 134.00 |
| | | Main Pipe - K4 | 1.407 | 2.63 | 3.698 | 29 | 1.064 | | 115.00 |
| | | Main Pipe - K5 | 3.840 | 2.65 | 10.169 | 32 | 3.292 | | 133.00 |
| | | Main Pipe - K6 | 0.457 | 2.62 | 1.195 | 31 | 0.375 | | 115.00 |
| Total Indicated Resource | | | 13.547 | 2.61 | 35.364 | 27 | 9.533 | | |
| | | | | | | | | | |
| Inferred | 2467 masl to 2127 masl | Main Pipe - K2 | 4.855 | 2.61 | 12.667 | 22 | 2.847 | 1.25 | 134.00 |
| | | Main Pipe - K4 | 0.018 | 2.60 | 0.046 | 29 | 0.014 | | 115.00 |
| | | Main Pipe - K5 | 12.207 | 2.66 | 32.522 | 30 | 9.883 | | 133.00 |
| | | Main Pipe - K6 | 1.055 | 2.68 | 2.829 | 29 | 0.810 | | 115.00 |
| Total Inferred Resource | | | 18.135 | 2.65 | 48.064 | 28 | 13.553 | | |
| | | | | | | | | | |
| TOTAL Diamond Resource | | | **31.681** | **2.63** | **83.428** | **28** | **23.086** | | |

The Diamond Reserve is stated after application of the relevant modifying factors as listed. The Diamond Reserve is based on the independent mine plan work for concentric and split shell designs conducted by the competent persons listed in the compliance statement 13.2 and is SAMREC compliant.

## Reserve Statement for the Liqhobong Diamond Mine as at 30 September 2015

| Diamond Reserve Category | Depth from and to | Source | Country | DIAMOND RESERVE | | | General Information | |
|---|---|---|---|---|---|---|---|---|
| | | | | Metric Tonnes (Millions) | Grade (CPHT) | Carats (Millions) | Reserve Bottom cut-off (mm) | Value (US$/ct) |
| Probable | Surface (2650 masl) to 2467 masl | Main Pipe | Lesotho | | | | | |
| K2 | | | | 20.686 | 23.2 | 4.795 | 1.25 | 134 |
| K4 | | | | 3.771 | 28.2 | 1.063 | 1.25 | 115 |
| K5 | | | | 10.371 | 31.7 | 3.290 | 1.25 | 133 |
| K6 | | | | 1.218 | 30.7 | 0.374 | 1.25 | 115 |
| TOTAL Diamond Reserve | | | | 36.046 | 26.4 | 9.523 | | 131 |

| MODIFYING FACTORS AND ASSUMPTIONS | | |
|---|---|---|
| **Factor Name** | **Units** | **Values** |
| Avg Resource Diamond price | USD/ct | 132,00 |
| Diamond price escalation | % | 3,00 |
| Production factors: | | |
| - Plant recovery factor | % | 100,00 |
| - Mining dilution | % | 2,00 |
| Financials | | |
| - Discount rate | % | 8,00 |
| - GOL royalty | % | 8,00 |
| - R/US$ exchange rate | % | 11,00 |
| - Sales and marketing costs | % | 1,87 |

## Inferred material in the LoM plan at at 30 September 2015 (this material is not categorised as a Reserve)

| Inferred section of LoM plan | Depth from and to | Source | Country | INFERRED | | | General Information | |
|---|---|---|---|---|---|---|---|---|
| | | | | Metric Tonnes (Millions) | Grade (CPHT) | Carats (Millions) | Reserve Bottom cut-off (mm) | Value (US$/ct) |
| Inferred | 2467 to 2295 masl | Main Pipe | Lesotho | | | | | |
| K2 | | | | 5.120 | 21.9 | 1.121 | 1.25 | 134 |
| K4 | | | | 0.026 | 28.7 | 0.007 | 1.25 | 115 |
| K5 | | | | 11.394 | 29.9 | 3.411 | 1.25 | 133 |
| K6 | | | | 0.519 | 28.8 | 0.150 | 1.25 | 115 |
| TOTAL Inferred section | | | | 17.059 | 27.5 | 4.689 | | 133 |

## Total LoM plan at at 30 September 2015

| LoM Plan | Depth from and to | Source | Country | TOTAL LoM PLAN | | | General Information | |
|---|---|---|---|---|---|---|---|---|
| | | | | Metric Tonnes (Millions) | Grade (CPHT) | Carats (Millions) | Reserve Bottom cut-off (mm) | Value (US$/ct) |
| Indicated + inferred | Surface (2650 masl) to 2295 masl | Main Pipe | Lesotho | | | | | |
| K2 | | | | 25.807 | 22.9 | 5.916 | 1.25 | 134 |
| K4 | | | | 3.796 | 28.2 | 1.071 | 1.25 | 115 |
| K5 | | | | 21.765 | 30.8 | 6.702 | 1.25 | 133 |
| K6 | | | | 1.737 | 30.2 | 0.524 | 1.25 | 115 |
| TOTAL LoM Plan | | | | 53.105 | 26.8 | 14.212 | | 131 |

# 13. COMPLIANCE STATEMENTS

## 13.1 Diamond Resource Statement of Competence and Compliance

The estimates presented in this Report are considered to be a true reflection of the Diamond Resource of the Liqhobong Main Pipe as at 30 September 2015 and have been carried out in accordance with the principles and guidelines of the ***South African Code for the reporting of Exploration Results, Mineral Resources and Mineral Reserves, 2009 (SAMREC code)*** and complies with the Firestone Diamonds Plc Policy for Reporting of Diamond Resources and Reserves. The estimates were prepared by or under the supervision of Competent Persons as defined in the ***SAMREC code***.

**Lead Competent Person – Diamond Resource**

Mr Paul Bosma has sufficient experience relevant to the style and type of deposit under consideration and to the activity which is being undertaken to qualify as a Competent Person as defined in the ***2009*** Edition of the ***SAMREC code***. Mr Bosma consents to the inclusion in the report of the matters based on this information in the form and context in which it appears.

| Name | Job Title | Registration | Years Experience | Signed |
|---|---|---|---|---|
| P. Bosma | Group Mineral Resource Manager | SACNASP 400259/04<br>GSSA 962504 | 22 years industry experience<br>(14 years diamond experience) | |

**Competent Persons**

The following independent Competent Persons employed by Z Star Mineral Resource Consultants were involved in the preparation of the Mineral Resource and have appropriate experience in their field of expertise with regards to the activity that they are undertaking and consent to the inclusion in the report of the matters based on the relevant technical information in the form and context in which it appears.

| Name | Job Title | Years Experience | Professional Affiliation | Responsibility | Signed |
|---|---|---|---|---|---|
| C. Lohrentz | Resource Analyst | 8 years geological modelling experience | SACNASP 40224/12 | Geology model | |
| D. Bush | Principal Resource Analyst | 25 years diamond resource estimation | SACNASP 400071/00 | Resource and Revenue Estimate | |

## 13.2 Diamond Reserve Statement of Competence and Compliance

The estimates presented in this Report are considered to be a true reflection of the Diamond Reserve of Liqhobong Diamond Mine as at 1 August 2015 and have been carried out in accordance with the principles and guidelines of the ***South African Code for the reporting of Exploration Results, Mineral Resources and Mineral Reserves, 2009 (SAMREC code)*** and complies with the Firestone Diamonds Plc Policy for Reporting of Diamond Resources and Reserves. The estimates were prepared by or under the supervision of Competent Persons as defined in the ***SAMREC code***.

**<u>Lead Competent Person – Diamond Reserve</u>**

Mr S. Hunter has sufficient experience relevant to the style and type of deposit under consideration and to the activity which is being undertaken to qualify as a Competent Person as defined in the ***2009*** Edition of the ***SAMREC code***. Mr Hunter consents to the inclusion in the report of the matters based on this information in the form and context in which it appears.

| Name | Job Title | Registration | Years Experience | Signed |
|---|---|---|---|---|
| S. Hunter | Group Mining Lead | Pr Eng (Reg No. 200 00185)<br>Member of the AusIMM | 26 years industry experience<br>(9 years diamond experience) |  |

**<u>Competent Persons</u>**

The following independent Competent Persons were involved in the preparation of the Mineral Reserve and have appropriate experience in their field of expertise with regards to the activity that they are undertaking and consent to the inclusion in the report of the matters based on the relevant technical information in the form and context in which it appears.

| Name | Job Title | Years Experience | Professional Affiliation | Responsibility | Signed |
|---|---|---|---|---|---|
| Mark Gallagher | Mining consultant | 33 years industry experience | Member of the SAIMM | Pit optimisation<br>Mine design | Original report signed |
| Tisa Chama | Mining consultant | 25 years industry experience | Member of the SAIMM | Pit optimisation<br>Mine design<br>Scheduling | Original report signed |

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## APPENDIX 1. Annual LoM plan

| LMDC LoM Plan | Totals | Year 1 | Year 2 | Year 3 | Year 4 | Year 5 | Year 6 | Year 7 | Year 8 | Year 9 | Year 10 | Year 11 | Year 12 | Year 13 | Year 14 | Year 15 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Ore tonnes treated** | **53.105** | 2.663 | 3.618 | 3.654 | 3.672 | 3.709 | 3.709 | 3.709 | 3.709 | 3.709 | 3.709 | 3.709 | 3.709 | 3.636 | 3.600 | 2.589 |
| **Recovered grade - cpht** | **26.8** | 25.80 | 27.70 | 30.02 | 32.86 | 25.08 | 23.40 | 24.66 | 26.26 | 28.49 | 22.84 | 24.26 | 25.98 | 27.37 | 28.22 | 29.11 |
| **Carats recovered** | **14.212** | 0.687 | 1.002 | 1.097 | 1.207 | 0.930 | 0.868 | 0.915 | 0.974 | 1.057 | 0.847 | 0.900 | 0.964 | 0.995 | 1.016 | 0.754 |
| **Total Waste** | **104.412** | 1.000 | 0.982 | 3.126 | 5.477 | 14.390 | 14.411 | 15.515 | 18.395 | 18.395 | 8.389 | 2.741 | 1.371 | 0.221 | - | - |
| | | | | | | | | | | | | | | | | |
| **K2 - Ore Tonnes** | **25.862** | 1.573 | 2.050 | 1.979 | 1.462 | 2.312 | 2.894 | 3.186 | 1.578 | 0.536 | 1.755 | 2.162 | 1.725 | 1.234 | 0.964 | 0.453 |
| **K2 - Ore Grade** | **22.9** | 22.8 | 23.5 | 24.7 | 27.7 | 21.0 | 23.0 | 23.3 | 21.2 | 21.7 | 23.2 | 21.7 | 21.6 | 22.0 | 22.4 | 22.6 |
| **K2 - Carats** | **5.919** | 0.359 | 0.483 | 0.488 | 0.405 | 0.486 | 0.665 | 0.742 | 0.335 | 0.116 | 0.408 | 0.468 | 0.373 | 0.272 | 0.216 | 0.102 |
| | | | | | | | | | | | | | | | | |
| **K4 - Ore Tonnes** | **3.792** | 0.789 | 0.555 | 0.284 | 0.202 | 0.469 | 0.551 | 0.122 | 0.212 | 0.001 | 0.512 | 0.081 | 0.015 | 0.000 | - | - |
| **K4 - Ore Grade** | **28.3** | 28.4 | 30.8 | 35.4 | 28.5 | 25.5 | 25.1 | 27.7 | 27.0 | 28.0 | 27.7 | 28.1 | 28.4 | 29.0 | - | - |
| **K4 - Carats** | **1.071** | 0.224 | 0.171 | 0.101 | 0.057 | 0.120 | 0.138 | 0.034 | 0.057 | 0.000 | 0.142 | 0.023 | 0.004 | 0.000 | - | - |
| | | | | | | | | | | | | | | | | |
| **K5 - Ore Tonnes** | **21.709** | 0.228 | 0.909 | 1.337 | 1.770 | 0.909 | 0.264 | 0.390 | 1.376 | 2.852 | 1.443 | 1.425 | 1.779 | 2.265 | 2.627 | 2.137 |
| **K5 - Ore Grade** | **30.9** | 34.0 | 34.3 | 36.7 | 37.9 | 34.9 | 24.6 | 34.6 | 30.7 | 29.8 | 20.6 | 27.8 | 29.9 | 30.2 | 30.3 | 30.5 |
| **K5 - Carats** | **6.698** | 0.08 | 0.31 | 0.49 | 0.67 | 0.32 | 0.06 | 0.13 | 0.42 | 0.85 | 0.30 | 0.40 | 0.53 | 0.68 | 0.80 | 0.65 |
| | | | | | | | | | | | | | | | | |
| **K6 - Ore Tonnes** | **1.742** | 0.073 | 0.104 | 0.055 | 0.239 | 0.020 | - | 0.011 | 0.543 | 0.320 | - | 0.041 | 0.190 | 0.137 | 0.008 | - |
| **K6 - Ore Grade** | **30.1** | 35.6 | 35.3 | 33.0 | 31.1 | 34.4 | - | 34.4 | 29.2 | 28.4 | - | 29.0 | 28.7 | 28.5 | 28.6 | - |
| **K6 - Carats** | **0.524** | 0.026 | 0.037 | 0.018 | 0.074 | 0.007 | - | 0.004 | 0.159 | 0.091 | - | 0.012 | 0.055 | 0.039 | 0.002 | - |

## APPENDIX 2. Ore Dressing Study summary information

### Crushing Ore Characteristics

| Criteria | Unit | Min | Expected | Max |
|---|---|---|---|---|
| Ore Types | TKB Kimberlite Facies K1 – K6 | | | |
| Country Rock | Basalt and Amygdaloidal Basalt | | | |
| Kimberlite Particle Density | t/m$^3$ | 2.5 | 2.65 | 2.8 |
| Basalt Particle Density | t/m$^3$ | 2.53 | 2.72 | 2.77 |
| Basalt Ore Dilution (max) | % mass | 0 | | 10 |
| Bulk Density | t/m$^3$ | 1.5 | | 1.6 |

The surface kimberlite is highly weathered in nature. It is expected that by year 4 of continuous operations most weathered kimberlite will have been treated.

The following crushability test-work data was measured during the ore dressing study campaign:

| Material | Bond Abrasion Index Ai [g] | | Bond Work Index [kWh / t] | | UCS [MPa] | |
|---|---|---|---|---|---|---|
| | Min | Max | Ave | Max | Ave | Max |
| K2 Kimberlite | 0.0048 | 0.0074 | 6.4 | 9.2 | 51.3 | 65.5 |
| K4 Kimberlite | 0.0045 | | 5.9 | 8.2 | 92.2 | 105.5 |
| K5 Kimberlite | 0.014 | 0.0173 | 6.3 | 10.5 | 54.9 | 95.2 |
| K6 Kimberlite | 0.012 | | 7.1 | 8.9 | 68.9 | 78.3 |
| Basalt | 0.0966 | | 12.2 | 15.8 | 141 | 158.8 |

### Scrubbing

The process design involves two scrubbers that accept material from the primary crusher section. Scrubbability testwork indicated that all ore types are easily scrubbed and that less than two minutes residence time at a solids to water ratio of 1:1 by weight will be required.

**Densitometry**

Densimetric analysis of the different ore types have confirmed that >99 percent of all ores fall within the 2.32 to 3.10 SG density range. No problems are therefore expected from a DMS concentrate yield perspective.

**Luminescence Profiling**

Pilot plant operations with XRF technology applied on the various ore types have confirmed that no specific problems with luminescent gangue materials are to be expected.

**Thickener and Slurry behaviour**

The following slimes characterisation parameters were established from rheology and thickener design test work.

| Parameter | Unit | Expected | |
|---|---|---|---|
| Undiluted Rise Rate (maximum) | m/hr | 2.5 | |
| Optimum feed solids concentration | % mass | 10 | |
| Thickener Solids SG | $t/m^3$ | 2.50 | 2.6 |
| Slimes Pumping Yield Stress | Pa | 20 | 150 |
| Thickener underflow % solids | % mass | 32 | 50 |
| Thickener underflow density | $t/m^3$ | 1.24 | 1.44 |
| Co-disposed density | $t/m^3$ | 1.37 | 1.495 |