Value Creation in an Open Cast Strip Mine Through Geotechnical Processes

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Abstract. Various geotechnical investigations are conducted to define rock mass properties which assist geotechnical engineers in understanding the level of variability within the rock mass and allow for uncertainty to be minimised during the design of rock slopes. Middelburg Mine Services previously experienced various slope failures and related rock falls, leading to injury, property damage, and production losses. Investigations into slope failures revealed that, although detailed geotechnical models form the basis for slope design and slope stability analysis, design within Middelburg Mine Services was not based on such models. As a result, the design and stability analysis processes are often highly dependent on local experience and rules of thumb. A new pit, the KE project, was used as a case study to determine whether conducting geotechnical investigations improves slope design and stability analysis processes. After collecting geotechnical data, the study entailed implementing kinematic, limit equilibrium and numerical modelling stability analysis methods. The probability of failure from stability analysis formed a basis for the design risk assessment. A risk-based design demonstrated that the original KE design was very conservative and that the subsequent optimised design is within the acceptable safety and economic risk evaluation criteria. In addition, this case study demonstrated that value is created by conducting geotechnical investigations, allowing for a risk-based design for highwall and low-wall slopes. This value created was translated to significant cost-saving and improved productivity.

Keywords: Slope design · Total probability of failure · Risk analysis

1 Introduction

Over the last five years at Middelburg Mine Services (MMS), it has been observed that, although ongoing efforts were made to manage the risk of falls of ground and slope failures, slope failure incidents continued to occur. These incidents adversely impacted safety and productivity and led to reserves being sterilised. To reduce the occurrence of such incidents, in the absence of a detailed geotechnical model of the mine, conservative designs have been implemented with negative economic consequences. To address this issue, research has recently been carried out to establish a detailed geotechnical model as the basis for risk-based slope designs (Netshivhazwaulu, 2022). This paper presents the

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resulting case study of the Klipfontein East Pit of MMS, identifying the economic value that has resulted from the availability of the geotechnical model and the implementation of the risk-based designs.

2 Literature Survey

Past research in this field includes that by Butcher et al. (2001), who described a geotechnical database relevant to highwall planning and geotechnical classification at South African collieries whilst developing a methodology for the safe cleaning of highwalls. Bye (2003) demonstrated that the unforeseen risk could be reduced by improving geological and geotechnical knowledge and enhancing safety and productivity in an open-pit mining environment. He demonstrated that the application of a 3D geotechnical model resulted in considerable productivity and financial benefit. Little (2006) expanded on the research of Bye (2003) and described a geotechnical strategy for an open pit centred around a detailed geotechnical model. This strategy contributed to improved safety and profitability of the open pit operation. Through a review of slope performance and an update of the geotechnical model, Ekkerd (2011) demonstrated the viability of steeper pit slope angles, which were then included in the mine's strategic business plan. The integration of face mapping data from laser scanners into the geotechnical database for an open pit iron ore mine was shown to improve data confidence (Russell, 2018; Russell and Stacey, 2019). At the same mine, it was demonstrated that an ongoing risk-based approach, which included a synthetic rock mass model, led to a higher-confidence geotechnical model (Bester et al, 2019). This approach resulted in major value-creation for the mine.

Methods of slope stability analysis in an open pit environment are well established (Wyllie and Mah, 2004; Read and Stacey, 2008). Stability analysis methods commonly make use of limit equilibrium and numerical modelling approaches. The use of risk evaluation methods in slope design is also well-developed (Steffen, 1997; Terbrugge et al, 2006; Contreras, 2015; Tapia et al, 2007). Risk has even been proposed as a rock engineering design criterion (Stacey et al, 2007). Kanda and Stacey (2016) investigated the influence of various factors on the results of such stability analyses and on the evaluation of risk. Chiwaye and Stacey (2010) compared the use of limit equilibrium and numerical modelling in open-pit mining risk evaluation.

Bieniawski (1992) established design principles for rock engineering and a design methodology. This has been linked to a strategic design approach (Stacey, 2009). One of the steps in the methodology involves the collection of geotechnical data. A methodology for constructing a reliable geotechnical model was suggested by Fillion and Hadjigeorgiou (2016): use appropriate guidelines for data collection; collect data with a strategy for subsequent analysis; by collecting additional information, the level of confidence in the data increases and the precision range can be reduced; and with additional data collection, the 'epistemic' uncertainty due to lack of knowledge may be reduced to give a better understanding of the true variability (aleatory uncertainty) of the geotechnical data. It was further suggested by Fillion & Hadjigeorgiou (2019) that while creating guiding principles for choosing the best drill hole spacings, geological and structural complexity should be well-defined.

In an attempt to understand the reliability of the geotechnical model, it was proposed by Steffen (1997) that the confidence of the slope design model should be classified according to a similar classification as used for resource classification:

- A Proven Slope Angle - Detailed structural mapping of the rock fabric is implied and can be extrapolated with high confidence for the affected rock mass. In this case, the strength characteristics of the structural features and in-situ rock are determined by appropriate testing procedures to allow reliable statistical interpretations. Data reliability should be such that an analytical model can carry out the design to a confidence of 85%.
- A Probable Slope Angle - Testing (small sample) for the physical properties of the in-situ rock and joint surfaces will be carried out. Similarly, groundwater data will be based on water intersections in exploration holes with few piezometer installations. This category allows for simplified design models to be developed where sensitivity analyses can be carried out.
- A Possible Slope Angle - corresponds to applying typical slope angles based on experience in similar rocks. Quantification will be based on rock mass classification and reasonable inference of the geological conditions within the affected rock mass.

Fillion and Hadjigeorgiou (2019) suggested that, for the geotechnical model, confidence level criteria should be used at various phases of a project. The confidence levels (variation) for the geotechnical model described in Read and Stacey (2008) are:

- Conceptual stage: $\leq$ 50% variation;
- Pre-feasibility stage: 30%–50% variation;
- Feasibility stage: 15%–35% variation;
- Design and construction stage: 10%–20% variation; and
- Operations stage: $\leq$ 10% variation.

The confidence levels described above are aimed at understanding the reliability of the geotechnical model to minimise uncertainty and the associated risk. All risks in mine planning, including geotechnical risks, must be communicated quantifiably and transparently (Stacey et al., 2007).

3 Case Study: Klipfontein East Pit, Middelburg Mine Services

Middelburg Mine Services is located in the Witbank Coalfield (Fig. 1), historically one of the crucial coalfields for South African power generation. According to Jeffrey (2005), more than 70% of coal reserves in South Africa are found in the Witbank, Highveld and Waterberg Coalfields. The colliery is 100% owned by Seriti Power (Pty) Ltd (formerly South32 SA Coal Holdings Pty Ltd). The coal production from MMS is of national interest because it is supplied to the Eskom Duvha Power Station, adjacent to the mine lease area.

The sedimentary coal-bearing strata in the Witbank Coalfield occur within the Vryheid Formation of the Ecca Group (Middelburg Mine Services, 2019). The colliery employs open-cast strip mining methods to exploit the coal reserves, using both dragline and truck and shovel operations. Klipfontein East (KE) Pit, which is the life-extension

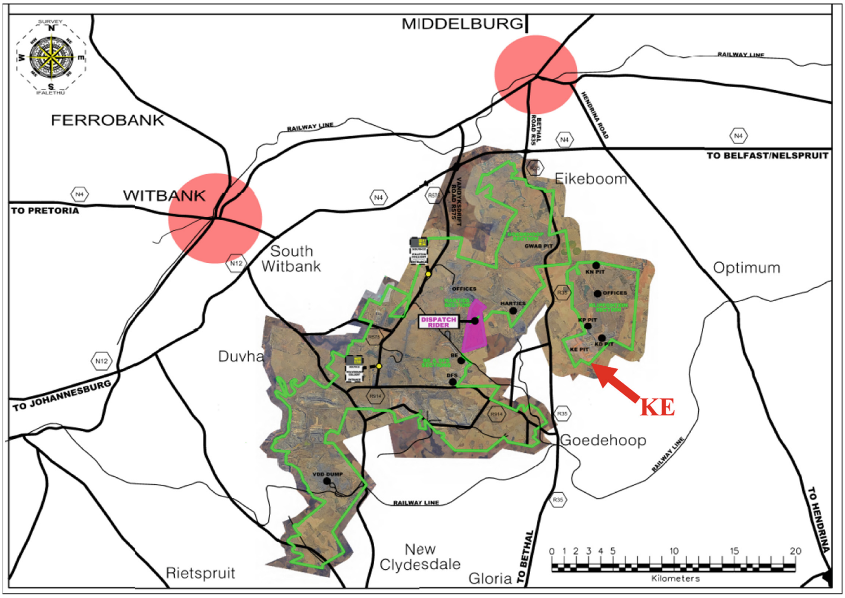

Fig. 1. MMS locality plan

project of MMS, is situated in the eastern part of the mine. The KE mining project comprises a strip-mining layout 2.7km in length and 834m in width, as shown in Fig. 2.

Fig. 2. KE pit layout

Two economic coal seams are planned to be mined, the #4 Seam and the #2 Seam, each with an average seam width of 4m. The planned mining method is the open-cast strip method. Draglines remove the burden, exposing coal on one strip and then dumping the material on the previous strip, creating spoil pile heaps. The exposed coal is then drilled and blasted before it is loaded by front-end loaders and hauled by a fleet of dump trucks to the stockpiles or the crusher.

In the planning stage of KE Pit, the slope and the pit configuration were designed based on the design engineers' experience and the operational rules of design as recorded in the Code of Practice to Combat Rock Falls and Slope Instabilities COP_010 (Seriti-MMS, 2021a). The main design requirements were specified as follows (Seriti-MMS, 2021a):

- Soft material must be pre-stripped before drilling of burden to leave not more than 3m of softs over competent burden. This soft material will be trucked to a designated waste dump;
- A smooth wall blasting technique (pre-splitting) must be practised on all competent rock masses;
- The catchment width between #2 Seam and #4 Seam must be at least 16m; and
- Spoil piles must be placed at least 5m away from the edge of the active strip.

The slope configurations within KE Pit were designed to meet the above requirements and are shown in Figs. 3 and 4.

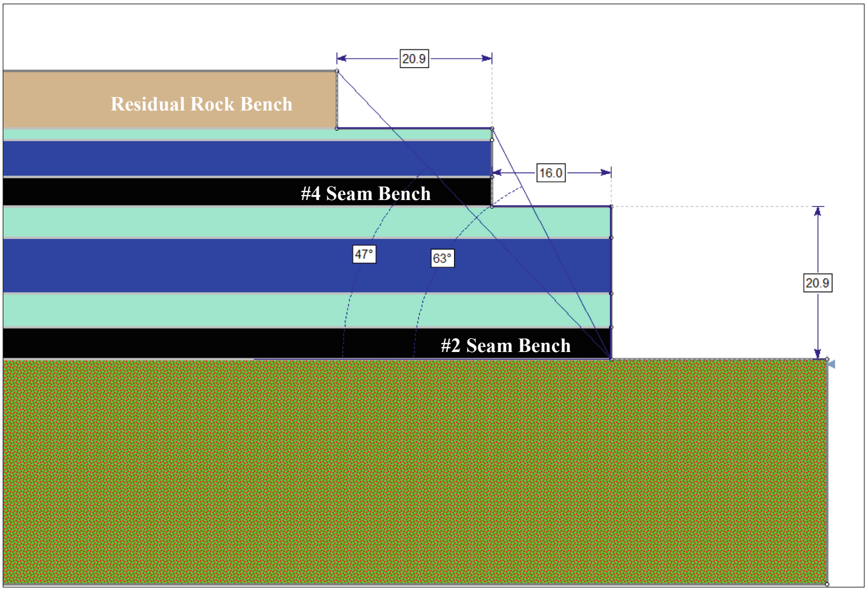

Fig. 3. KE highwall configuration

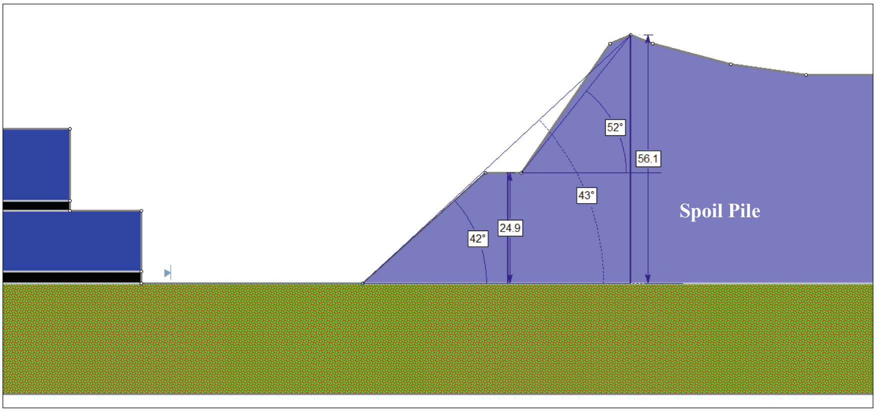

Fig. 4. KE spoil slope configuration

3.1 Economic Considerations

The design of open-pit slopes is concerned mainly with defining optimal slope angles. A slope angle may be regarded as optimal when the angle is steep enough to meet safety requirements and minimise economic impacts as far as practically possible. Generally, the steeper the slope angle, the more economical the open pit mine will be, owing to the reduced volume of waste to be removed. In the case of open-cast strip mining, steeper slopes result in a reduced volume of broken waste to be handled by shovels and draglines owing to the increased volume of waste material that is cast into the void of the previous strip by the blast. A smaller catchment width will achieve this economic benefit and, in addition, will reduce the volume of soft material removal required and increase the volume of waste blasted directly onto the spoil pile. However, increasing the slope angle will increase the risk of failure, and it is therefore essential that geotechnical investigations and analyses be carried out to justify the steeper angle. Geotechnical investigation and design activities for the KE Pit will be described in the following sections.

3.2 Geotechnical Database

As part of the KE project, the geotechnical database within MMS was reviewed to determine whether opportunities exist to improve the project design. The following is a brief review of the findings from this database.

Soft Material Strength: the strength of soft material or residual rock has historically never been measured within MMS. The depth of soft material was estimated from the exploration borehole core - the length of the unrecoverable core was regarded as the depth of soft material. The resulting thickness of soft material generally varies between 3m and 9m.

Rock Strength: laboratory testing of rock core samples is conducted to determine rock strength. The Geotechnical database contains laboratory test results carried out for MMS. A summary of laboratory results for 187 rock samples is shown in Table 1. However, none of the samples is from the KE project area.

Table 1. MMS rock property summary

Rock Type	Density (t/m^3)	UCS (MPa)	Elastic Modulus (GPa)	Poisson's Ratio	Brazilian Tensile Strength (MPa)
	Mean	Mean	Mean	Mean	Mean
Shale	2.44	79.54	11.72	0.25	5.61
Shaley Sandstone	2.53	89.71	14.96	0.29	7.75
Sandstone	2.41	75.42	20.03	0.36	5.75
LaminatedShale/Sandstone	2.5	89	13.6	0.27	7.51

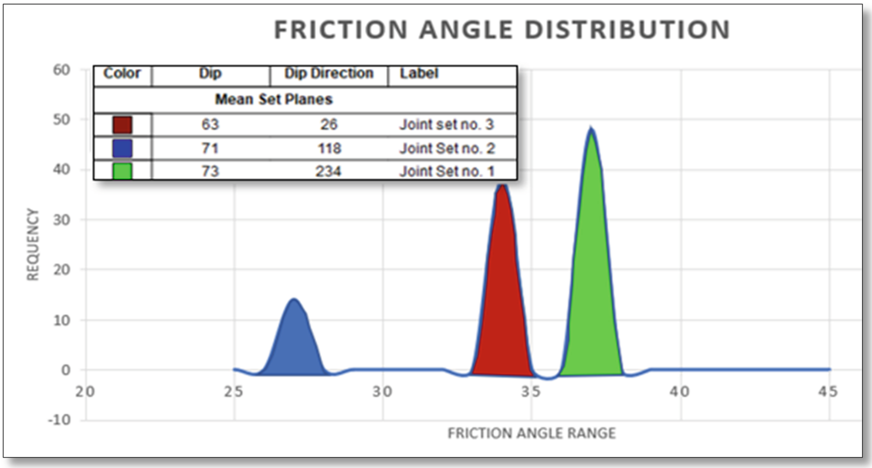

Fig. 5. Major Joint set data

Major Joint Sets: joints were historically mapped on exposed highwalls, and joint orientation data were recorded within the database. The database contains information from the northern part of MMS only, of which the KE Project is part. Figure 5 shows the major joint set orientations and friction angle distributions.

Rock Mass Strength: rock mass classification has gained wide acceptance as a tool for estimating the strength of jointed rock masses (Hoek and Brown, 2019), necessary as an input parameter for numerical modelling. However, the MMS geotechnical database does not have any rock mass classification data recorded.

3.3 Data Collection

In the rock engineering design process, geotechnical data collection is an essential step. This includes interpretation of borehole information, field mapping, rock mass classification, laboratory testing, and monitoring instrumentation. The strength of soft material and rock mass classification has never been determined within MMS. To rectify this situation for KE Pit, a data collection project was instituted to minimise the epistemic uncertainty in the design. KE Pit is currently mining within the box cut where the highwall (HW) is buffered and is not available for meaningful mapping. However, photogrammetric mapping using a Maptek Laser Scanner will be initiated when the highwall is exposed to improve the geotechnical model.

3.3.1 Soft Material Strength

The investigation to collect soft material or residual rock strength properties included conducting Standard Penetration Tests (SPT) (Grider et al., 2003), using Dynamic Probe Super Heavy Tests (DPSH). DPSH involved forcing a cone probe into the ground using a drop hammer and determining the number of blows per 100mm. The DPSH test is continuous and is performed from the surface until refusal. As indicated by Byrne and Berry (2008), penetration tests are relatively inexpensive and rapid as no core or sample is recovered. A geotechnical contractor was appointed to conduct these field tests using a TG63–100 automatic Pagani Rig with an 80kg drop hammer. At each location, the testing continued to refusal. 33 DPSH tests were distributed across the KE Pit project area on the undisturbed ground. The number of blows (hammer drops) was correlated with the material's load-bearing capacity during the tests. Load Bearing Capacity is a concept relevant to foundation strength in construction and is a function of the effective friction angle of the soil material (Grider et al., 2003).

The results from the investigation determined that the refusal depth throughout the project area was 6.8m at a 90% probability of occurrence. The refusal depth does not indicate the depth of soft material, but the depth of material with strength low enough to be penetrated by the DPSH cone. This means that the residual rock strength is higher at depths greater than 6.8m. The results indicate that, for a load-bearing capacity of 200kPa within the soft material, there is a 90% probability that the depth horizon will be less than 4.3m. The foundation loading imposed by the dragline is distributed through a circular tub with an 18m diameter, which gives a stress of 140kPa. This means that the Factor of Safety (FoS) of the foundation at 4.3m will be 1.4. Similarly, for a load-bearing capacity of 300kPa, there is a 90% probability that the depth horizon will be less than 5.8m, with a dragline pad FoS of 2.1. From these results, it can be seen that a suitable foundation depth is significantly less than the depth of soft material indicated by the interpretation of core drilling data. The depths above also include topsoil material with a thickness of between 1.5m and 2m, which is removed before mining and preserved for rehabilitation purposes.

The results of the penetration testing indicate that the strength of the soft material at KE Pit is relatively high at shallow depth. Based on this, it may be concluded that the soft part of the slope will be stable for the short life of the pit, and therefore that only topsoil needs to be pre-stripped. Figure 6 shows a hazard plan generated for each mining block where the strength of 300kPa was targeted. All areas where the depth of targeted strength is less than 3m and greater than 5m are shown in green and red, respectively. The probabilities of not achieving the 300kPa load-bearing capacity in these depth ranges are also shown in Fig. 6.

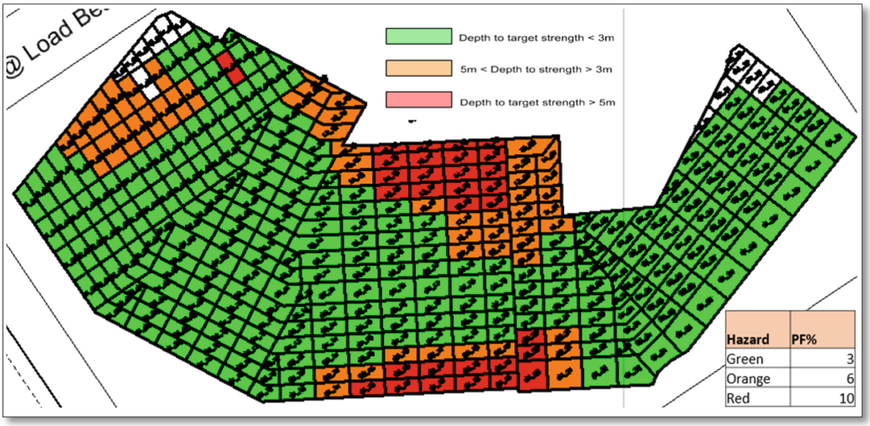

Fig. 6. Soft material hazard plan

3.4 Slope Design Analyses

The approach described by Read and Stacey (2008) regarding confidence in geotechnical information shown in Table 2 was adopted. Based on the following information available, the KE geotechnical model is classified to be between the confidence levels Design and Construction and Feasibility, as highlighted in Table 2:

- Field testing was conducted at the site, and laboratory test data from samples collected within Klipfontein was used.
- Detailed structural mapping has not been carried out at the site. The general joint orientations used are from the same operation, although in different locations.
- Specific rock mass parameters not obtained directly from the site, are required to validate the rock mass strength.

3.5 Slope Stability Analyses

The slope stability analyses conducted were based on the *Feasibility-Design Model* variability. Both limit equilibrium and numerical modelling software were used for deterministic analyses, but the former only for probabilistic analyses. The Limit Equilibrium program Slide2 (RocScience, 2021) was the favoured tool for analysis. A non-circular path was analysed using the Janbu Simplified method (Janbu, 1954). The analysis was conducted on two slope configurations: the original KE configuration based on operation rules; and the configuration in which the catchment was changed from 16m to 10m. A deterministic numerical analysis using program RS2 (RocScience, 2022) was carried out only for a 10m catchment width rock mass model because of the prolonged model run time required. The results of the analyses are shown in Table 3.

Table 2. Model confidence level (Read and Stacey, 2008)

Model Category	Variation	Basis
Conceptual	$\geq 50\%$	Data is inferred from reports and regional data from other mines in similar environments.
Pre-feasibility	30–50%	Experience in similar mines, but using data inferred from other mines. Limited borehole data, mapping on proposed sites.
Feasibility	15–35%	Based on increased density of sampling during pre-feasibility
Design and Construction	10–20%	Based on specific data obtained to validate data used in the pre-feasibility stage. Detailed mapping, field testing and laboratory testing
Operational	$< 10\%$	Additional mapping on exposed walls, performance monitoring data

Table 3. Limit Equilibrium and Numerical Model analyses

Configuration	Overall Mean FoS	Critical SRF	Overall PoF (%)	Slide2 Failure volume (m^3)	RS2 Failure volume (m^3)
16m Catch	2.7	NA	0.0	0.0	0.0
10m Catch	2.0	2.1	0.3	542	480
Spoil 16m Catch	1.1	NA	9.7	414	NA
Spoil 10m Catch	1.0	NA	18.2	414	NA

3.5.1 Kinematic Analysis

When geological structures control slope failure, the failure mechanism may be planar, toppling or wedge failure. To conduct kinematic analyses, four zones were considered corresponding with different slope orientations (see Fig. 7). Deterministic and probabilistic kinematic analyses were conducted, and the results are shown in Table 4.

3.5.2 Probability of Failure

Kinematic and limit equilibrium models were used to compute the PoF. According to Contreras (2015), the PoF calculated from a geotechnical model typically only accounts for the uncertainty in the material properties and can be referred to as model probability of failure (PoF_{Model}). To determine the total PoF, the wedge failure PoF was used because it yielded the highest PoF values. The final results of the stability analyses are shown in Table 5.

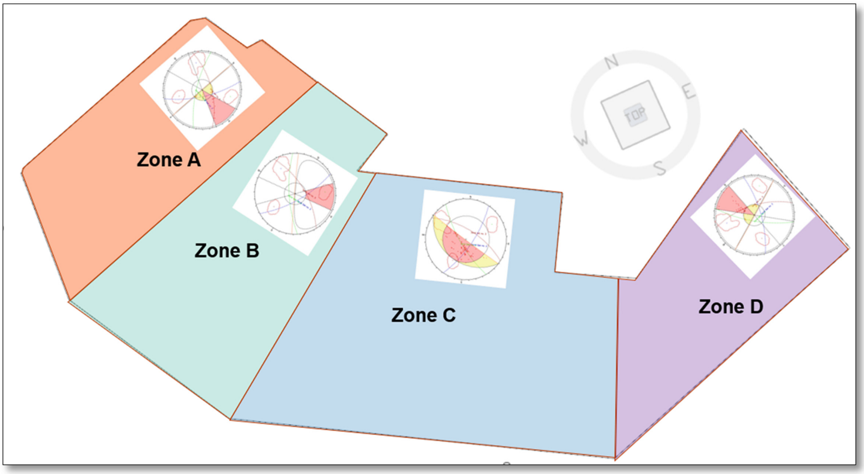

Fig. 7. Slope zones for kinematic analyses

Table 4. Kinematic analysis results

Slope Zone	POF %		
	Toppling	Planar	Wedge
A	4.3	18.0	42.0
B	14.0	18.5	44.0
C	9.4	14.0	31.1
D	13.1	0.0	27.4

Table 5. Model PoF values for all highwall slope zones

Domain	Slide2 PoF%	Kinematic PoF%	Total Model PoF%	Failure volume (m^3)/(m)
Zone A	0.3	42.0	0.12	480.0
Zone B	0.3	44.0	0.13	480.0
Zone C	0.3	31.1	0.09	480.0
Zone D	0.3	27.4	0.08	480.0
Spoil 16m catch	9.7	NA	9.7	414.0
Spoil 10m catch	18.2	NA	18.2	414.0

3.5.3 Total PoF

A total probability of failure (PoF_{total}) must incorporate sources of uncertainty that may exist within the slope but are not accounted for in the geotechnical model (Contreras, 2015). The origins of this uncertainty may include groundwater, abnormal changes in geology, mining inefficiencies, seismic events and changes in geometry. Contreras (2015) described a method for evaluating the total probability of failure when there is exposure to atypical conditions. The atypical condition can be analysed on an annual basis as described by the following equation:

$$q_{atypical} = 1 - (1 - P_{atypical})^{\frac{1}{n}}$$

where:

$P_{atypical}$ is the probability of occurrence of a particular uncertain atypical situation leading to slope failure, associated with a defined mine duration in years (n); and.

$q_{atypical}$ is the annual probability of occurrence of such an atypical situation.

The probability of failure of a slope, given the eventuality of atypical conditions ($PoF_{model}|_{atypical}$) can be evaluated together with the slope stability model. Results from this are expressed as a factor ($f_{atypical}$) of the model probability of failure, which was assessed under normal conditions, as follows:

$$PoF_{model|atypical} = PoF_{model} \times f_{atypical}$$

The final probability of failure due to an atypical condition ($PoF_{atypical}$) can be calculated for a particular year (i) of the mine plan as follows:

$$(PoF_{atypical})i = PoF_{model|atypical} \times (1 - \left(1 - p_{atypical}\right)^{i})$$

The total probability of failure PoF_{total} for given uncertainties such as groundwater, geology, and mining is provided by the following equation:

$$PoF_{total} = 1 - (1 - PoF_{model})x\left(1 - PoF_{groundwater}\right) - \left(1 - PoF_{geology}\right) - (1 - PoF_{mining})$$

The approach described above has indicated that even though the calculated PoF_{model} is zero, the likelihood of failure must never be taken as zero due to the epistemic uncertainty. This approach will be applied to the KE pit, where four atypical conditions were identified: blasting, geometry, geology, and groundwater. It was concluded by Contreras (2015) that, to manage subjectivity, expert judgment must be considered. The following is the estimation of these atypical conditions for KE Pit:

- Blasting: this includes surface blasting leading to poor smooth wall conditions or high peak particle velocities causing ground vibrations. There is no certainty that this will not happen, and history during the operation of Klipfontein and blasting of the box cut reveals that it is likely. Therefore, for the life of the KE project, the probability of failure due to this condition is estimated at 15%. This estimation was based on the fact that the slope failure and rockfall database for the last five years within MMS identifies that frozen highwall material due to blasting inefficiencies contributed 11% to total failure (SERITI-MMS, 2021a).

- Geometry: this includes the deviation from planned geometry, which can be observed for both highwall and spoil slopes. During the establishment of the KE box cut, deviations from the scheduled geometry were already observed and recorded (Mokoena, 2021). The recorded variations allowed the estimation of atypical probability at 10%.
- Geology: this includes unplanned geological structures such as dyke and the loss of rock mass strength due to abnormal geological conditions. The probability of failure due to this is estimated to be 7%.
- Groundwater: this includes the probability of failure because of groundwater seepage. The rockfall database within MMS has revealed that groundwater contributes to failure in both highwall and spoil slopes. This probability is estimated at 10%.

The operating life of the KE pit is planned to be between 2021 and 2027, a total of 7 years (SERITI-MMS, 2021b). The calculated PoF_{total} per year in Tables 6 and 7 is based on Zone B of the rock mass model with 10m catchments. Zone B was selected because it has the highest PoF_{model}.

Table 6. PoFtotal for 10m catchment _HW Zone B

Year	2022	2023	2024	2025	2026	2027
n	2	3	4	5	6	7
PFmodel	0.13%	0.13%	0.13%	0.13%	0.13%	0.13%
PFmodelblasting	0.2%	0.2%	0.2%	0.2%	0.2%	0.2%
PFmodelgeometry	0.2%	0.2%	0.2%	0.2%	0.2%	0.2%
PFmodelgeology	0.2%	0.2%	0.2%	0.2%	0.2%	0.2%
PFmodelground	0.2%	0.2%	0.2%	0.2%	0.2%	0.2%
PFBlasting	0.9%	1.3%	1.8%	2.2%	2.6%	3.0%
PFGeometry	0.5%	0.8%	1.0%	1.2%	1.5%	1.7%
PFGeology	0.4%	0.6%	0.8%	1.0%	1.2%	1.4%
PFground	0.5%	0.7%	0.9%	1.1%	1.4%	1.6%
PFTotal	**15.2%**	**16.1%**	**17.0%**	**17.9%**	**18.8%**	**19.7%**

Table 7. Input factors_PoFtotal for 10m Catchment_HW Zone B

LOM	7.0		
	q	P	f
Blasting	0.023	15%	1.5
Geometry	0.015	10%	1.3
Geology	0.010	7%	1.5
Ground	0.015	10%	1.2

Table 8. PoFtotal for 10m catchment _Spoil

Year	**2022**	**2023**	**2024**	**2025**	**2026**		**2027**
n	2	3	4	5	6		7
PFmodel	18.2%	18.2%	18.2%	18.2%	18.2%		18.2%
PFmodelblasting	0.91%	0.91%	0.91%	0.91%	0.91%		0.91%
PFmodelgeometry	1.82%	1.82%	1.82%	1.82%	1.82%		1.82%
PFmodelgeology	1.092%	1.092%	1.092%	1.092%	1.092%		1.092%
PFmodelground	0.91%	0.91%	0.91%	0.91%	0.91%		0.91%
PFBlasting	3%	4%	5%	7%	8%		9%
PFGeometry	3%	4%	5%	7%	8%		9%
PFGeology	2%	3%	4%	6%	7%		8%
PFground	2%	2%	3%	4%	5%		5%
PFTotal	**9%**	**13%**	**17%**	**21%**	**24%**		**28%**

Table 9. Input factors_PoFtotal for 10m catchment Spoil

LOM (years)	7.0		
	q	P	f
Blasting	0.0149	0.1	0.05
Geometry	0.0073	0.05	0.1
Geology	0.0103	0.07	0.06
Ground	0.0088	0.06	0.05

Tables 8 and 9 show the PoF_{total} for the spoil slope, corresponding with the highwall slope with 10m catchments. Again, $P_{atypical}$ and $f_{atypical}$ were adjusted for the spoil pile slope because the impact of the described uncertainties is low on spoil pile slopes.

3.6 Risk Analysis

Risk has been defined as the product of the likelihood and the consequence of an event. In slope engineering, the risk of failure has various possible consequences: safety, economic, reputational, environmental, and legal. Only the safety and economic impacts will be considered here.

3.6.1 Safety Risk Analysis

For the safety impact of a slope to be realised in the form of injury or fatal injury, the failure of the slope must coincide with the spatial and temporal presence (spatio-temporal coincidence) of personnel in the slope location. This suggests that slope failure can occur without any impact on personnel safety. Therefore, in order to assess personnel safety risk, personnel exposure levels to slopes must be known. Contreras et al. (2006) indicated that personnel exposure is established from that of equipment because equipment operators make up the most significant proportion of people exposed to slope failures in strip mining. This exposure can be expressed in terms of spatial factors and temporal factors. Furthermore, equipment exposure to the highwall can also be estimated from equipment utilisation data.

KE Pit plans to mine, on average five million tons of run of mine (ROM) coal as its annual production. All this coal would have been exposed by a dragline. This coal will be mined through three ramps, each established on the northern edge of Zones A, C and D, as shown in Fig. 7. It is expected that the entire 2.7 km strip will be mined in a year. The Spatio-Temporal Coincidence or exposure factors are estimated following the approach adopted by Contreras et al. (2006). This approach uses idealised pit geometry and actual statistical performance data for all types of equipment at risk. The exposure factors are estimated from planned equipment operational time, cycle time and operational distances. Table 10 indicates the planned operational time per annum for the main equipment within KE Pit.

The temporal (cycle) exposure factors in Table 10 are based on the times each item of equipment spends travelling in, or working while standing within, a risk area. The only items of equipment that have travel times as part of their operational cycle are the coal dump trucks. These trucks will spend a maximum of 30 min completing one operational cycle; loading coal from the loading face in the pit, travelling with coal to the stockpile or crusher; dumping coal, and then travelling back to the loading face in the pit. The total cycle time in various zones is partitioned: 13 min for travel within the pit (where 6min is along both HW and Spoil dump), 10 min for travel outside the pit, 5 min at the loading face and 2 min at the crusher. The rest of the equipment spends all operational time exposed to the highwall and spoil failure risk.

KE Pit has been planned with three ramps, each ramp responsible for accessing 800-900m of linear strip advance independently. The length of each ramp ranges from 350m to 400m. The spatial exposure factors correspond with the slope length worked from a single ramp compared with the total length of the pit or cycle worked in a year. This factor was calculated based on the loading-hauling circuit, which is 12km long,

Table 10. Cycle exposure factors

Equipment	Fixed Part of Cycle			Travel Part of Cycle		
	Fixed (In-pit loading face)	**at HW**	**at Spoil**	**Travel**	**at HW**	**at Spoil**
Coal truck	**20%**	**0.4**	**0.6**	**80%**	**0.16**	**0.125**
Dozer	**40%**	**0.6**	**0.4**	**0**	**0**	**0**
Drills	**100%**	**0.67**	**0.33**	**0**	**0**	**0**
FEL	**100%**	**0.5**	**0.5**	**0**	**0**	**0**
Dragline	**100%**	**0**	**0.1**	**0**	**0**	**0**

Table 11. Spatial exposure factors

Spatial	Cycle	at HW	at Spoil
	Fixed	**29.60%**	**29.60%**
	Travel	**11.60%**	**6.60%**

and from the strip length of 2.7km. The results of the spatial exposure factors are shown in Table 11.

Where:

Fixed = length of slope accessed from one ramp/length of the strip;
Fixed at HW/Spoil = 800m/2700m;
Travel = (Travel length along HW or Spoil)/total travel length.
Travel at Spoil = 800m/2700m and Travel at HW = 1400m/12000m.

The temporal coincidence factors were calculated as a ratio between mining blocks that are at risk of slope failure and the total number of blocks mined per year. This ratio will correspond with the total fraction of time the equipment is exposed to the risky zone. The total number of mining blocks which are at risk is equivalent to the number of blocks that are accessed by a single ramp, which is eight blocks. It is expected that the equipment used will be fixed in each block along the strip during each year's production advance. One year's advance is equivalent to the advance of the entire strip length. This means that it will take twelve months to mine out a single strip that is 2.7km long. Results for temporal exposure are depicted in Table 13.

Where:

Fixed = number of mining blocks at risk/total number of mining blocks mined in one year.
= 8 blocks/27 blocks.
Travel = Time equipment is exposed/total cycle time.
Travel at HW = 13 min/30 min and Travel at Spoil = 6 min/30 min.

The exposure for specific equipment was calculated based on multiplying all relevant factors in Tables 10 to 13 according to that equipment's fixed and travelling condition as exposed to each slope. The exposure for personnel to safety impact is equivalent to that of the equipment. The results shown in Table 14 are for the fixed condition. Only fixed condition results are depicted because they have higher personnel exposure factors. A sample calculation shown below is for the drills for a highwall exposure condition. For

Table 13. Temporal exposure factors

Temporal	Cycle	at HW	at Spoil
	Fixed	**29.60%**	**29.60%**
	Travel	**43.70%**	**25.00%**

Table 14. Personnel exposure analysis - fixed

Equipment	No. of people	Exposure HW (%)	Exposure Spoil (%)
Coal truck	5	1.2	1.3
Dozer	2	0.6	0.4
Drills	4	**3.3**	**1.6**
FEL	1	3.8	3.8
Dragline	2	0.0	0.3
Blasting	5	0.6	0.3

coal trucks with both the travel and fixed portion of their cycle time, the same calculation is repeated for both parts, and their products are added.

Calculation:

$$\text{Exposure} = \text{temporal factor} \times \text{spatial factor} \times \text{cyclefactors} \times \text{annual planned operational time.}$$

$$\text{Exposure} = 0.296 \times 0.296 \times 0.67 \times 0.57$$

The highlighted factors are selected as the critical exposure factors because they are high, with the greatest number of people exposed.

A simple event tree, as described by Terbrugge et al. (2006), was used to evaluate the probability of multiple fatalities based on the critical exposure factors indicated in Table 14. Figures 8 and 9 represent the event tree analysis for the highwall slope and spoil slope, respectively. In addition, the highest probability of failure was used as a basis for designing the KE project during this study. In this fault tree analysis, the corresponding probability of multiple fatalities was determined to be $1.56 \times 10^{-4}\%$ and $1.08 \times 10^{-4}\%$, respectively.

Where:

$$\text{Probability of Fatality} = \text{PoFtotal} \times \text{probability of ineffective monitoring} \times \text{probability of people e}$$

$$\text{Probability of fatality} = 19.7\% \times 0.12 \times 0.033 \times 0.2 \times 0.01 = 1.56 \times 10^{-4}\%$$

where:

$$\text{Probability of Fatality} = \text{PoFtotal} \times \text{probability of ineffective monitoring} \times \text{probability of people e}$$

$$\text{Probability of fatality} = 28\% \times 0.12 \times 0.016 \times 0.2 \times 0.01 = 1.08 \times 10^{-4}\%$$

3.6.2 Acceptability Criteria

The calculated probability of fatality was plotted against safety acceptability criteria in Fig. 10. The results indicate that the probability of multiple fatalities from rock mass and spoil design is within the ALARP consensus region and recommended slope design zone.

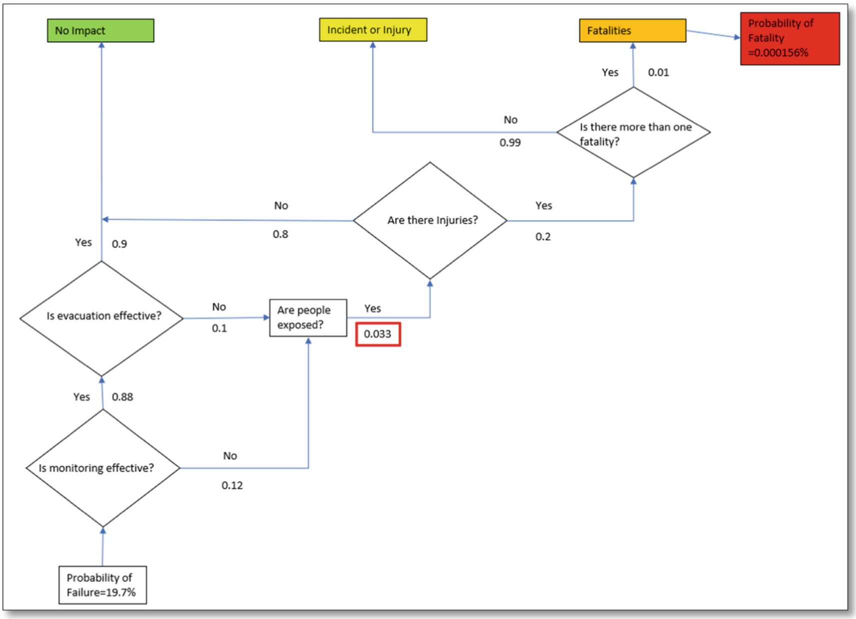

Fig. 8. Event tree analysis probability of fatality along HW

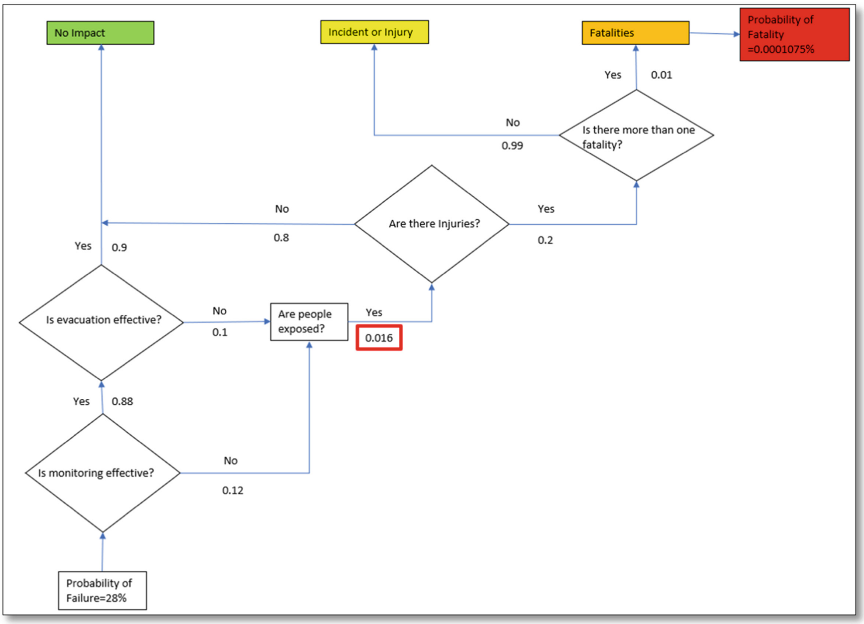

Fig. 9. Event tree analysis probability of fatality along spoil slope

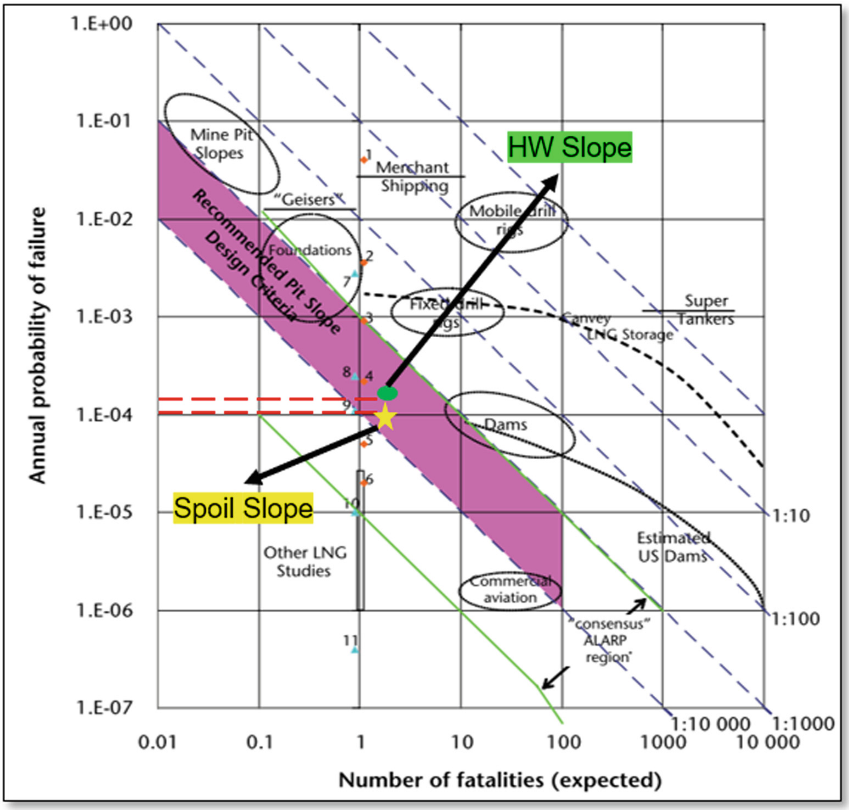

Fig. 10. Application of risk acceptability criteria (Terbrugge et al., 2006)

3.6.3 Economic Risk Analysis

The economic evaluation for the KE pit is presented in terms of possible costs incurred during slope failure due to equipment damage, loss of productivity and clean-up of failed material. In three years, MMS incurred damage to the excavator and shovel, owing to rock slope failure, with an estimated cost of ZAR4 239 414 (South32-MMS, 2020), corresponding with an annual value of ZAR1 413 138. When slope failure occurs in strip mining within MMS, it generally leads to a two-day loss/delay of coal production, estimated at ZAR 320 000. In addition, the cost of removing the resulting waste material with the truck fleet is R41/m^3.

Based on the fall of the ground database, the maximum lateral failure lengths on the highwalls, spoils, and dumps are 91m and 71m, respectively, and the corresponding costs are summarised in Table 15.

A risk map approach defined by Contreras (2015) and based on the event tree methodology first described by Tapia et al. (2007), was adopted for KE pit economic risk analysis. This analysis determined a 5.5% probability of a cost impact of ZAR6.65million. Based on the risk levels defined in the MMS risk management framework, this risk is tolerable.

Table 15. Total cost per slope failure event

Slope	Volume per linear metre (m^3/m)	Plausible Failure volume for 91m and 71m length (m^3)	Cost (ZAR)
HW	480	43 680	3 524 294
Spoil	414	34 080	3 130 694

It may therefore be concluded that KE Pit design with a 10m catchment results in a tolerable risk within the operation.

3.7 Creating Value Through Design

To demonstrate the value associated with a revised KE Pit design with a 10m catchment, and with soft material not pre-stripped, a comparison of the relevant data is shown in Table 16.

The value created by the revised design is driven mainly by the two factors: the reduction in soft material stripping; and the blast volume gain. Soft material stripping entails removing a layer of soft material, before the drilling of the burden, and placing the material in the dedicated waste dump. This material remains in the dump until required for the rehabilitation, when it is loaded and transported back to the pit. The same volume would therefore be handled twice at an indicative rate of ZAR40/m^3.

Blast gain in strip mining is a vital productivity parameter, and it is based on the muck pile profile after blasting. It is defined as the volume of the material which will not be handled by either the dragline or dozer because it would have been cast in its final position. The gain difference for the two designs is 6%, translating to a volume of 4567m^3 per mining block, which corresponds with a saving of ZAR23/m^3 and ZAR32/m^3 for

Table 16. Differences between 10m catch and 16m catch design

Parameter	Original design (16m Catch)	Proposed design (10m Catch)
Volume of soft material removed	4 371 088m3	0m3
Catchment	16m	10m
Blast Gain	17%	23%
Geotech study required	No	Yes
Rock mass model FoS	2.7	2
Rock mass model PoF	0.0%	0.10%
Spoil model FoS	1.1	1
Spoil model PoF	9.70%	18.20%

Table 17. Cost benefit comparison

Parameter	**16m Catch Design Costs (ZAR)**	**10m catch Design Costs (ZAR)**
Soft Stripping	349 687 040.00	0
Blast Gain	90 175 415.00	0
Geotechnical study	0	1 346 763.00
Monitoring Instruments Capital	0	4 531 230.00
Total	439 862 455.00	5 877 993.00
Cost-Benefit	**R433 984 462.00**	

the dragline and dozer respectively. Therefore, the cost-benefit can be calculated for 359 mining blocks that are part of the KE pit layout.

Table 17 indicates the difference in cost between the original and revised designs. By implementing the 10m catchment design following the risk analysis, there is a projected cost-saving of some ZAR0.43billion through the life of mine (LoM). The investment in the geotechnical design work, and monitoring equipment is some ZAR6 million, but the projected return is some 70 times the investment cost.

It is to be noted that, in line with the rock engineering design process (Stacey, 2009), the monitoring of the slopes is extremely important to confirm that performance corresponds with design expectations.

4 Conclusions

The shift from underground to open pit coal mining in South Africa brings an increased risk of rock fall and slope failure accidents, making it essential to implement robust pit slope design and stability analyses. This paper presents a case study of Middelburg Mine Services' life extension project, KE Pit, which used a risk-based geotechnical model for design. Previous design processes relied on local experience and rules of thumb rather than detailed geotechnical models, highlighting the need for a comprehensive approach. The geotechnical model was developed using various techniques, including field geotechnical testing, laboratory testing of rock samples, and structural mapping. The confidence level model by Read & Stacey (2008) was adapted to minimise geological uncertainty. The design work employed limit equilibrium, numerical and kinematic highwall stability analyses, and limit equilibrium analyses for spoil slopes.

Furthermore, safety and economic risk analyses were also conducted. As a result, the geotechnical investment in the project was ZAR6 million, with a projected return of ZAR433 million, representing a 70-fold return on investment. Therefore, the case study has demonstrated that operations have value when they invest in geotechnical studies and the adaptation of a risk-based slope design process.

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