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Rockfall Risk:

Quantification of the consequences of rockfalls

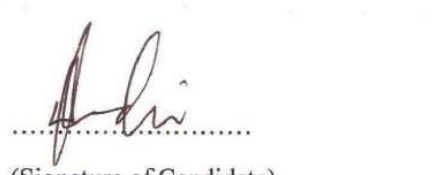
Lawrence Rwodzi

A dissertation submitted to the Faculty of Engineering and the Built Environment, University of the Witwatersrand, Johannesburg, in fulfilment of the requirements for the degree of Master of Science in Engineering.

Johannesburg, 2011

DECLARATION

I declare that this dissertation is my own unaided work. It is being submitted for the Degree of Master in Science in Engineering to the University of the Witwatersrand, Johannesburg. It has not been submitted before for any degree or examination to any other University.



(Signature of Candidate)

4th day of APRIL (Year) 2011

At JOHANNESBURG

ABSTRACT

Though the significance of investing in safety spending is appreciated, it is generally viewed as an expensive overhead cost. In lean economic times particularly, this mind-set places productivity first before safety. However, because of compromised safety conditions, there may be rockfalls. The consequences of rockfalls are numerous, some direct and others indirect. While these losses can be significant, there appear to be no records of evaluation of such losses on the mines. The financial evaluation of risk mitigation systems is therefore currently based only on the cost of implementing a support system and does not take into account the losses resulting from accidents or failure of the system.

When a fall of ground occurs in a stope the method of remediation depends largely on the size of the rockfall. Very small rockfalls will be simply cleaned up and the area made safe, larger rockfalls will require the installation of additional support, or the panel may need to be re-established in the case of the largest rockfalls. Losses are incurred as a result of dilution, re-supporting, reduced productivity and loss of sweepings. Pillars and remnants may also be left behind, which reduces the available reserves. The findings of this research suggest that the full distribution of rockfall sizes should be considered.

In this dissertation, an investigation into rockfall risk has been carried out. This is based on the definition of risk, where risk is the product of the probability of occurrence of a rock fall and the consequences it causes. The research was conducted on two case study operations, AngloGold Ashanti (Mponeng Mine) and Implats (Impala Platinum 12 Shaft). On each operation the probability of occurrence of rockfalls was determined from the mine's rockfall database and where joint data was available; a key block generating software, JBlock, was used.

Extensive studies were then carried out to determine the consequences and costs of rockfall damage and as a result, a generic methodology of calculating these was developed. This methodology is presented in this dissertation. This methodology enables the calculation of risk in monetary terms, a language well understood by management. Management can therefore be involved in the design of risk mitigating systems as they can set the acceptable risk criteria. The ability to calculate risk and set acceptable risk criteria implies that while productivity is maintained, safety is not compromised.

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I would like to thank the Management and staff at Mponeng Gold Mine and Impala Platinum Mine for their cooperation, and for affording me the opportunity to use their operations as case studies.

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NOMENCLATURE

Acceptable risk: The residual risk remaining after controls have been applied to associated hazards that have been identified, quantified to the maximum extent practicable, analysed, communicated to the proper level of management and accepted after proper evaluation.

Consequence: The outcome of an event expressed qualitatively or quantitatively, being a loss, injury, disadvantage or gain. There may be a range of possible outcomes associated with an event.

Cost: Of activities, both direct and indirect, involving any negative impact, including money, time, labour, disruption, goodwill, political and intangible losses.

Frequency: A measure of the rate of occurrence of an event expressed as the number of occurrences of an event in a given time.

Hazard: A potential occurrence or condition that could lead to injury, damage to property, delay or economic loss.

Risk assessment: The overall process of risk identification, risk analysis and evaluation.

Risk identification: The process of determining what can happen, why and how.

Risk Analysis: A structured process that identifies both the likelihood and the consequences of hazards arising from a given activity or facility.

Risk Evaluation: Involves the comparison of the results of a risk analysis with risk acceptance criteria or other decisions. If the risk does not meet the risk criteria, the risk is treated, that particular risky option discarded, or risk control options sought.

Risk acceptance: An informed decision to accept the consequences and the likelihood of a particular risk.

Risk- benefit analysis: Evaluation of risks and benefits of some activity or agent usually based on economic consideration.

Risk Reduction Measure: An action that can be adopted to control a risk by either reducing the likelihood of occurrence or by mitigating the consequences of an occurrence.

Risk Management: The process by which decisions are made to accept known risks, or to implement actions to reduce unacceptable risks to acceptable risk levels.

1 Introduction

1.1 Background

Worldwide mining is known for its hazardous conditions. It is therefore not surprising that a large number of accidents occur in the mining industry. South Africa is one of the countries with the most mining activity in the world. It hosts some of the world's deepest mining operations. However, it is also on record for having one of the worst safety records. In 2007 and 2008, South Africa had 220 and 171 fatalities, USA had 26 and 23, Canada 8 and 6, and Australia had 11 and 4 respectively (Khuzwayo, 2010). At the Mine Health and Safety summit in 2003, the Government, organised labour and South Africa's Chamber of Mines agreed that the local mining industry should achieve safety performance equivalent to international benchmarks by 2013. The figures cited by Khuzwayo (2010) indicate that South Africa has a long way to go in achieving these targets.

The statistics reported above are supported by the prevalence of mine accidents that fall into various categories. A study of accidents between 1999 and 2007, Figure 1-1, showed that fall of ground accidents are the most dominant. Other major causes include transportation, mining and machinery related accidents.

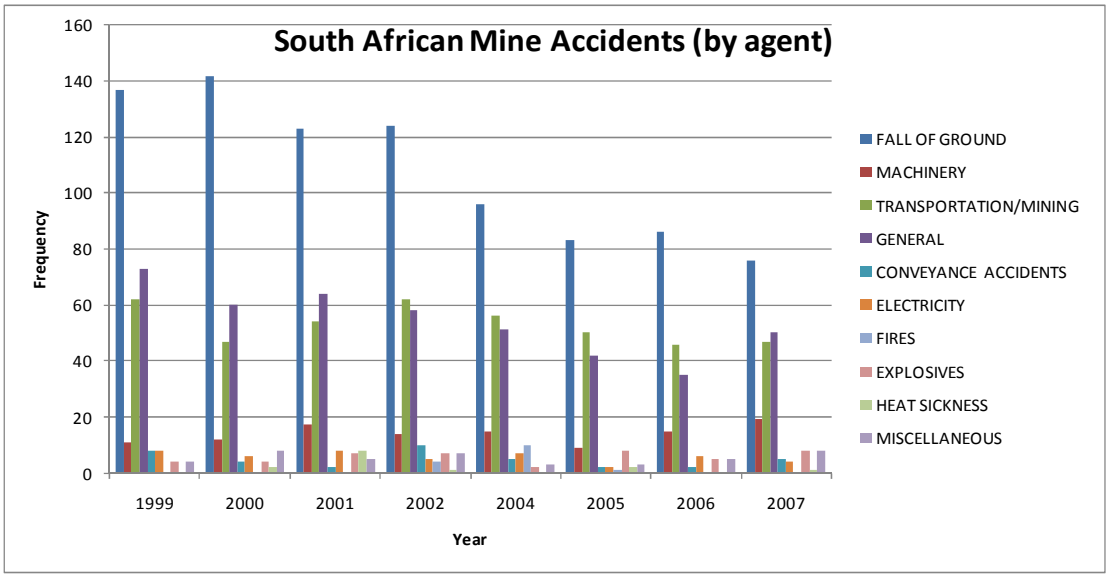


Figure 1-1: South African mine accidents by causing agent (SAMRASS Database, 2008)

Within the rock, related accidents category, the analysis of the fatal accidents showed that gravity induced falls are more prominent compared to rockbursts (Figure 1-2). An analysis of the frequency of rock related injuries (Figure 1-3) also showed that gravity related injuries are more prominent. While rockbursts are hazardous, based on these statistics, it can be concluded that rockfalls carry the highest risk to personnel. A comparison of 2005 accident statistics from different mining sectors showed that gold and platinum operations were the worst affected by rockfall fatalities (Figure 1-4).

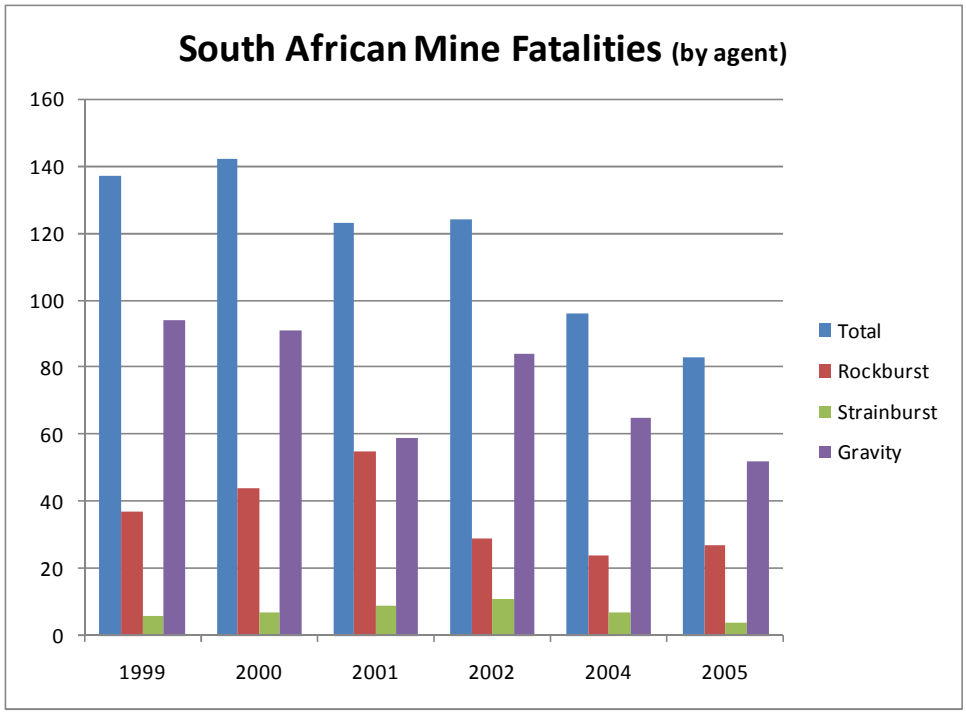


Figure 1-2: Fatal rockfall accidents analysis

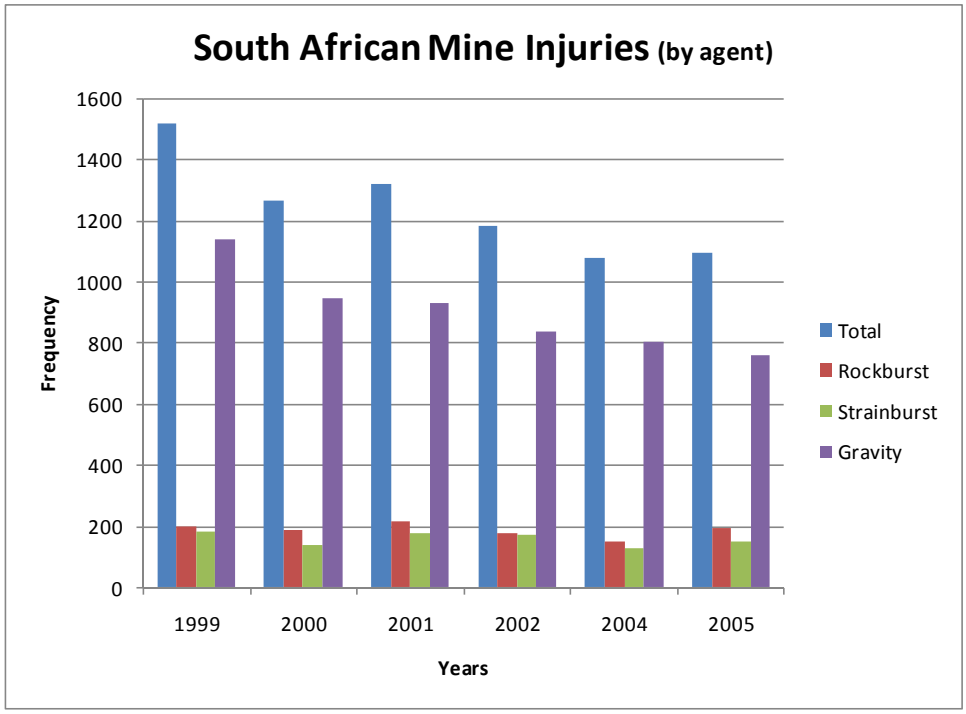


Figure 1-3: Rockfall related injury accidents analysis.

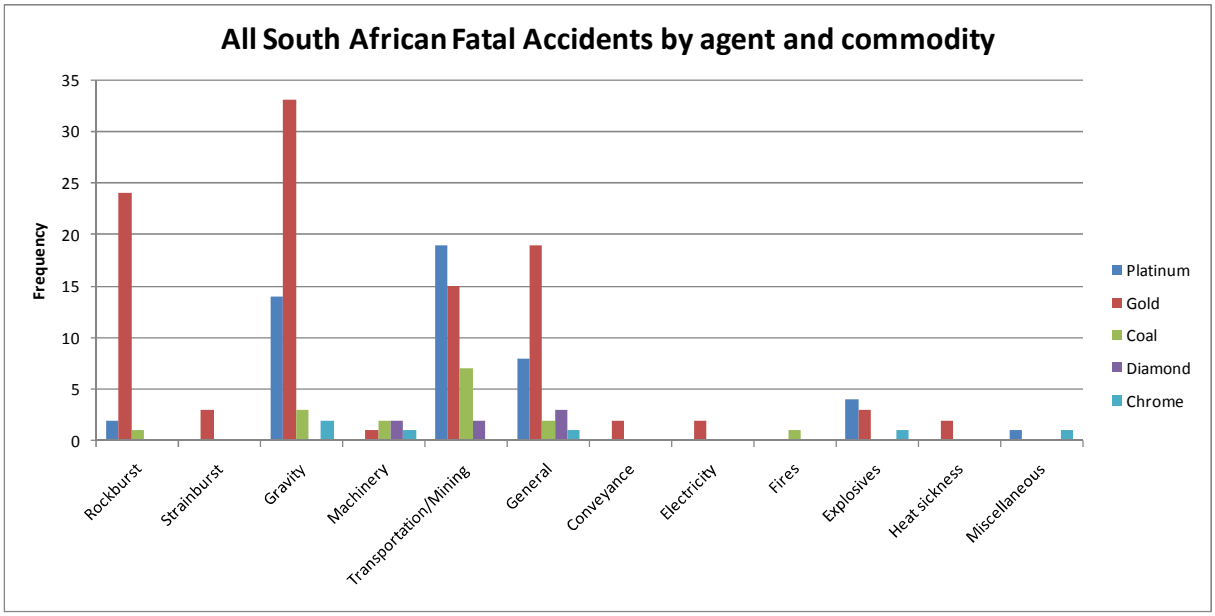


Figure 1-4: Accident frequency by agent and commodity

Rockfall accidents do not only result in personnel injuries or fatalities but may result in other consequences such as equipment damage, destruction of mine workings, production delays, abandonment of working areas and temporary mine closures, amongst others. Some consequences require remediation action to enable the continuation of the mining operation. Remedial action involves, among others, cleaning up rockfalls, repairing or replacing equipment, and re-establishing or rehabilitating panels. Such consequences come with significant economic repercussions. Adequate support design prevents rockfall accidents thereby minimising consequences.

A Safety in Mines Research Committee (SIMRAC) 'Elimination of rockfalls' study conducted by Joughin et al (2009) revealed that during support design, the current common industry practice is to consider only the capital and working cost of implementing the design. The financial impacts of rockfall consequences in the given design are not considered. In addition, proposed changes to support systems that would mean an increase in the support budget are often met with resistance and reluctance from management. The reason being mining is a profit driven business; therefore management is not willing to make commitments that it perceives to have a direct negative financial impact. This gives rise to the perception in the South African mining industry that safety is a necessary but expensive overhead cost. The result is that safety is put in direct opposition to production, hence the former being compromised for the latter. Not only is this a mindset problem, but, also a financial dilemma, as there is no comparison by management of the costs associated with the accidents to the overall cost of implementing the support strategy changes.

To develop an appreciation for the potential costs of rockfall accidents, an evaluation of the risk associated with each support design in a defined geotechnical zone needs to be determined. Risk is the product of the probability of failure and the consequences of failure. To calculated the risk, first the probability of failure of an excavation and support design has to be determined. This is done using software (JBlock) or historic rockfall data. The financial consequences of rockfalls associated with

each support design are then determined. Adding the cost of implementing the design in each zone to this calculated risk in that zone gives an indication of the total risk that is associated with each support design. Through this process risk is therefore be an integral part of the support design process.

The primary aim of the research described in this dissertation is to develop a methodology to identify and quantify the consequences of rockfalls in narrow tabular mining operations. The methodology is then applied to two case study operations, Impala 12 Shaft and Mponeng Mine, to determine the financial consequences of rockfalls in the stopes. The total costs involved in the rockfall consequences and support design are determined in each case study.

The developed methodology provided essential input to risk modelling software being developed for the Safety in Mines Research Committee (SIMRAC) under the SIMRAC SIM 060201 "Rockfall Elimination" research programme. The software enables calculation and comparison of the total costs of rockfall consequences of one support design against another. This risk-cost approach to mine support design is demonstrated in Chapter 6 of this dissertation.

1.2 Definition of the problem

While mining companies claim a "zero tolerance" approach to accidents, nothing can be 100% reliable and safe (Wong, 2005), as proven by accident statistics, (SIMRAC Report 2005/2006). Owing to the uncertainties that prevail within the rock engineering environment, there always exists a probability that an underground excavation will be less stable than expected (Bawden, 2008; Christian, 2004; Einstein, 2003 and Stacey, 2006). Instability results in rockfalls or mine working collapses which are either gravity or seismically driven. There is therefore a need to rethink the "zero tolerance" approach whilst considering uncertainty.

Rock Engineers carry out the mine support designs on the mines (Stacey, 2006). They are generally risk averse and focus on technical excellence. The designs are technical and include no input from management on the design objectives (Stacey, 2006). However, Wong (2005) stated that making things safe costs money. If it costs money to make things safe, then it is appropriate to assess the trade-off between the support cost and the cost of the consequences of instability. This brings the concept of risk into design. Considering risk analysis in support design enables management's direct involvement in the design process. Management stipulate acceptable risk criteria for each design. Rock Engineers will therefore design the excavations and support to satisfy the specified risk criteria.

Research has shown that current mining practice in South Africa takes into consideration only the capital and working costs of installing support in an excavation. The risk-reward relationship associated with non-performance of support is often not analysed in the mines. There is generally no assessment by managers of the consequences of support failure on the value of money, future profits, and costs of production (Baecher and Christian, 2003).

There is a knowledge gap regarding a clear understanding of the concept of risk, the relevant risks to mine support design and the acceptance thresholds set by the management (Christian, 2004 and Summers, 2000). The financial risk implication associated with rockfall accidents is not quantified. Quantification of such risk allows effective management aimed at protecting the mining company, its people, assets and profits against the physical and financial consequences of risk. It also allows integrated mine planning at the operations level. Further, defined acceptable risk criteria will enable motivation for increased investment in safety, as designs have to meet the set criteria.

The research described in this dissertation seeks to quantify, in monetary terms, the direct and indirect consequences associated with rockfalls in a given geotechnical environment in an underground mine. The focus is primarily on:

- Support designs: support standards, performance, quantities and costs,
- Extent and consequences of rockfalls,
- Rockfall investigations and recommendations/remedial actions, and
- Quantifying the costs of the consequences associated with rockfalls.

Mining is a business and therefore it is imperative that decisions be made based on risk criteria that are acceptable in both the business and ethical contexts. The cost associated with rockfalls suggests that there can be significant value generation by reducing risk through safety spending.

1.3 Objectives

The research described in this dissertation seeks to meet the following objectives:

- To highlight the direct and indirect consequences associated with rockfalls in narrow tabular mining operations,
- To develop a methodology for quantifying the costs of the consequences,
- To apply the methodology on selected mine operations as case studies,
- Demonstrate an evaluation of the overall risk-cost analysis on two different support systems under consideration for implementation.

1.4 Research methodology

All rockfalls above specified sizes are reported to the DMR, this is a requirement by the Mine Health and Safety Act (1996). The Department collates these statistics into a database, the SAMRASS Database. This database is accessible to various public and private institutions. Hence, it was a starting point in the literature survey of this research. The same Act also requires mines to maintain a rockfall database, which should be analysed continuously and used in support design and risk analysis. This design and analysis are presented in the Code of Practice (COP) on all South Africa mining operations. Different mines have different database formats. In this research, an evaluation of the available databases on gold and platinum mining operations was carried out. The two operations with the more comprehensive databases were targeted as case studies.

A period of six to eight weeks was spent on each operation by the researcher during which a detailed study was undertaken with the key focus areas being to:

- Understand the detail in the mining cycles and rock engineering practices.
- Identify support standards, compliance and costs in the demarcated ground control districts.
- Study fall of ground cases, which involved accident investigations, interviews with Mine Captains, Shift Bosses, Miners, Mine Engineers, Safety Officers, Rock Engineers and Strata Control Officers on their experiences with FOG's, the resulting consequences and the remedial actions taken in individual cases.
- Consolidate individual cases into categories to obtain a generic approach to consequences, remedial action strategies and the time factor involved in the different scenarios.
- Obtain cost information from the costing/accounts department. Of key importance were revenue, variable operating costs, equipment repair/replacement costs, statistics on the production tonnages, and determination of profit margins.

After this study, a methodology to determine the costs of the different consequences due to rockfalls was constructed for each operation. Experienced engineers involved in the 'Rockfall Elimination' project then tested and reviewed the methodology for its validity and applicability. Software was then developed based on this methodology. The software development is however, not part of this dissertation, but illustration of the use of the software on one of the case studies is presented.

1.5 Content of dissertation

In this dissertation, Chapter 2 reviews risk in the mining industry. This involves the concept of risk, its role, appreciation of risk and acceptable risk criteria in the mining industry, and geotechnical design. Chapter 3 evaluates the consequences of failures due to rockfalls. This chapter also reviews available means of financial quantification of such consequences. A proposed methodology for quantifying the consequences of rockfalls is then presented in Chapter 4. In Chapter 5, the methodology is tested using Mponeng Mine and Impala 12 Shaft as case studies. This methodology has been coded into software by other researchers. Chapter 6 demonstrates the application of the software to evaluate the cost of consequences at the Mponeng Mine. Chapter 7 concludes the dissertation.

2 A review of risk in the mining industry

2.1 Concept of risk

Engineering structures are designed for varying conditions, with varying life spans and for varying purposes. For example, dams amongst other civil engineering structures are designed for long life service. As such, they should be very reliable; therefore, conservatism is often practised in their design. Rock slopes in open pit mines on the other hand, are transient short to medium term structures where conservatism is not sought. Underground structures include both long life (shafts, haulages) and short life structures (orepasses, slushier gullies).

In any business operation, it is imperative that structures satisfy four criteria: safety, serviceability, economics and aesthetics (Einstein, 1996). All these factors, however, carry technical and non-technical uncertainties. Uncertainties affect the performance of structures in relation to the criteria. While some designs are very conservative, conservatism may negatively affect the economics. In addition, design is carried out for variable conditions. Variable conditions give rise to uncertainty. Variability and uncertainty in the overall business objective give rise to the concept of risk.

What is risk? Vanmarcke et al (1982), Einstein (1996), Summers (2000), Einstein (2003), Baecher and Christian (2003) and Tapia et al (2007) agree on the definition of risk. Risk in engineering is the product of the probability of occurrence of an event and the associated consequences of the event. Not only is risk expressed in technical terms but it also can be combined with economic consequences and hence expressed in economic terms.

Casagrande (page 1, 1965) in his paper, “Role of the calculated Risk” describes “calculated risk” as being:

- “The use of imperfect knowledge, guided by judgement and experience to estimate the probable ranges for all pertinent quantities that enter into the solution of a problem”,
- “The decision on an appropriate margin of safety, or degree of risk, taking into consideration economic factors and the magnitude of losses that would result from the failure”

Risk is also a major concern from a safety and health perspective in any industry. Foster et al (1998) describe risk from the safety perspective as being the likelihood of occurrence of a hazard.

Given the above definitions and perceptions, risk can be expressed in both technical and economic terms. The engineering definition of risk is however, applicable in both cases. For example, from the safety perspective, where little work has been done on quantifying the consequences, the risk is expressed as severity frequency rates. Similarly, in open pit mining risk analysis risk is expressed in financial terms.

The discussion above would be incomplete without a description of the types of risk. These types of risks include systematic, non-systematic, financial and operational/technical risk.

Systematic risks are risks relating to financial and capital markets. They relate to market uncertainty and the effect it has on the cost of capital. It is the risk of collapse of an entire financial system or market. Such risk of failure has cascading effects for example the failure of banking systems in 2008 that cascaded into global recession. In contrast, failure of individual entities or groups has no inter-linkages or inter-dependencies. Knowledge of such inter-dependencies is important to enable the engineer to adjust discount rates in terms of the capital assets pricing model, the after tax cost of debt and so on.

Non-systematic risks are directly related to the inputs of the business or investment. They must be dealt with directly at source, namely appropriate diversification strategies. It is not appropriate to deal with these through adjustments to discount rates; this would be an indication that the practitioner does not understand the specific risks and hence applies a fudge factor.

Financial risk is the chance that a project or financial venture will not perform according to expectations, for example, collapse of a venture due to the inability of capital repayment from cash flow. This is the probability of actual return on the investment being less than the expected return. Such risk should be accounted for in the cost of capital.

Operational/Technical risks: these are risks due to the execution of the project. Examples include legal risks, fraud risk, physical or environmental risks. In mining, such risks include geological risk, geotechnical risk, sampling risk, assaying, grade, production profiles, design, capital, operating, construction and completion, and environmental risks.

The research in this dissertation, as explained in the problem statement, focuses on geotechnical risk in underground mining. Hence, no further consideration of the other types of risk is given in this dissertation.

2.2 Risk in the mining context

Mining is a high risk and capital intensive business. Investors and shareholders therefore desire high returns on such investments. A mining project life begins in the mineral exploration phase, and its life proceeds as illustrated in Figure 2-1. In the prospect evaluation phase, prefeasibility and feasibility studies determine the value of the resource, mining methods, support systems, and environmental issues. If the project is feasible, the mine is developed to the pilot project phase. The monetary value of the property increases from exploration to the production phase. This is largely a function of the investment made in obtaining more knowledge on the deposit. The more the information obtained on the project the better the risks are understood and therefore managed. Risk is reduced by extensive and good quality studies. As such, time and money are spent in prefeasibility and feasibility studies.

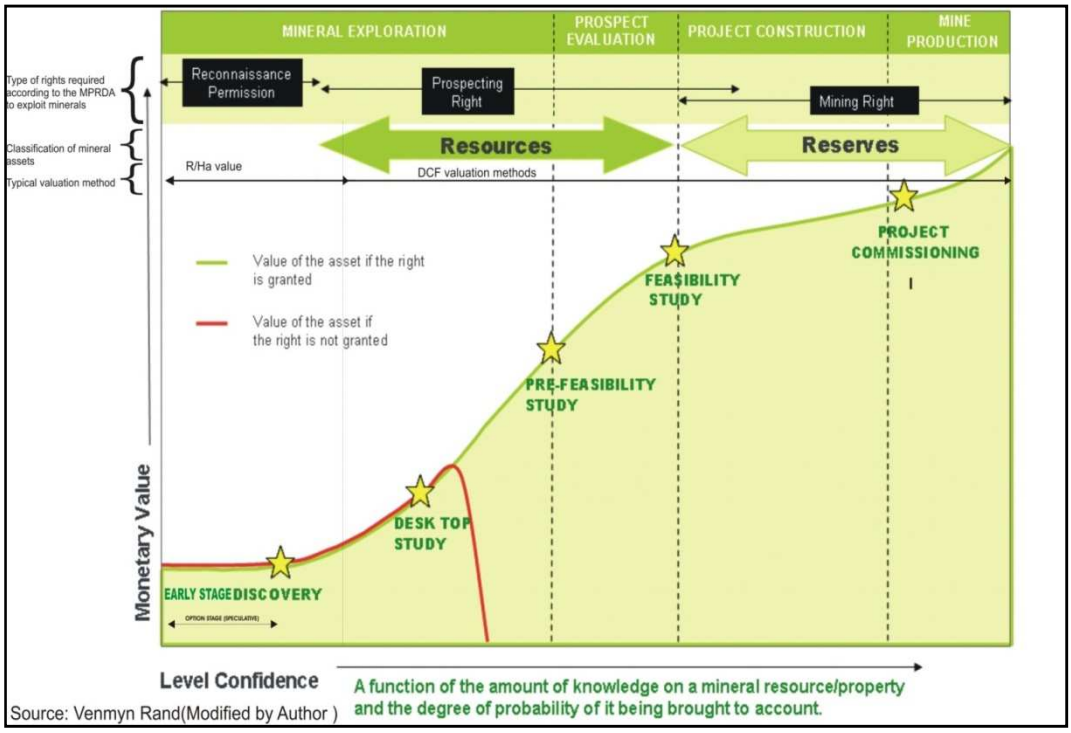


Figure 2-1: Value chain of a mineral project (Modified after Venmyn Rand, 2008)

Geotechnical information is usually limited at the feasibility stage and more information is acquired as the mine is developed. Figure 2-2, adapted from Terbrugge et al (2006), illustrates the potential sources of uncertainty that affect the mine plan. At the planning stage, trade-off studies are made between the stability of the openings, ground support costs and production efficiency (Horsley and Medhurst, 2000). Geotechnical input is therefore a critical aspect to the performance and hence profitability of the mine. In this light, Stacey (2007) questioned if enough geotechnical data is collected to carry out mine designs.

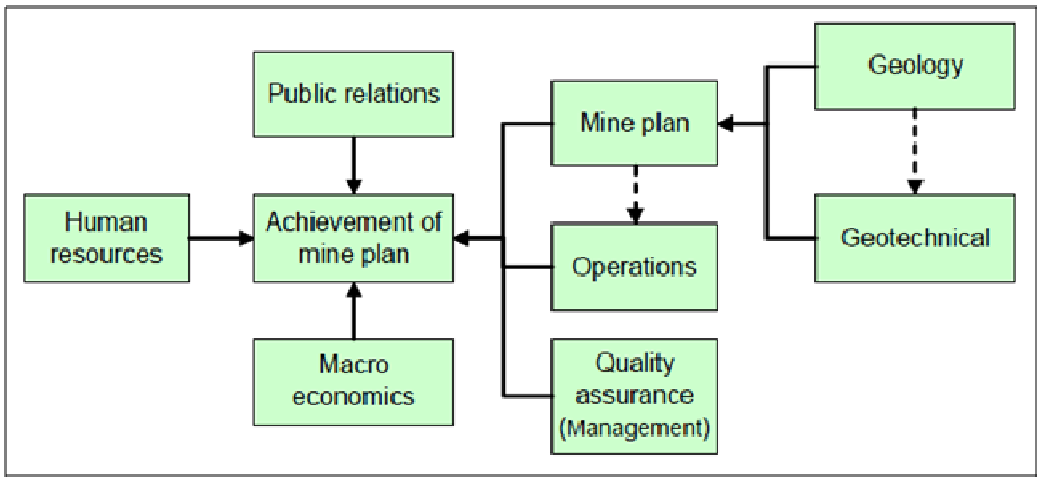


Figure 2-2: Sources of uncertainty affecting the mine plan (Terbrugge et al, 2006)

In open pit mining, the Factor of Safety (FOS) was the common basis of design up until the 1980's. The Probability of Failure (POF) then became popular. Figure 2-3 demonstrates the application of the FOS and POF in risk analysis for two data sets, A (blue) and B (red).The peak of the normal distribution curves indicates the FOS for the data set. From the graph, the FOS for B (FOS_B) is higher than the FOS for A (FOS_A). An SRK internal report (No. 2004-ha62), suggests that the FOS offers a subjective decision which is not transparent and incommunicable, and not always satisfactory. The FOS only tells whether a structure is stable or not, it does not indicate when failure will occur (if it will occur). This weakness is resolved through applying the POF concept to the data set. The area under the curve to left of FOS = 1 indicates the POF of each slope (Figure 2-3). The POF for set B (POF_B) is higher than that for set A (POF_A). Therefore, while set B has a higher FOS, it also has a high POF, thus indicating higher overall risk of failure. This indicates that conservativeness in design that is based only on the FOS is inappropriate. The POF on the other hand provides transparency and an easier way of communicating the decision. It indicates the likelihood of failure occurring. The challenge with this approach is selecting the permissible POF, hence the acceptable risk of the design.

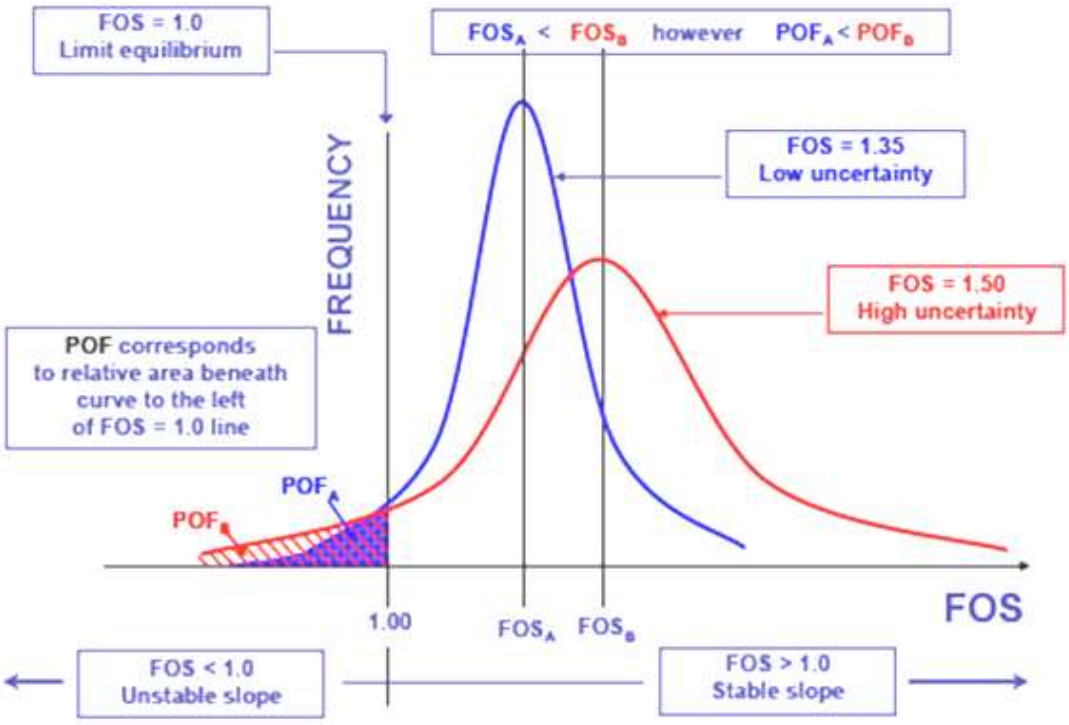


Figure 2-3: Uncertainty in slope design (after Tapia et. al 2007)

To answer this question the Risk/Consequence (R/C) analysis has been adopted in slope design. R/C analysis does not replace the FOS or POF criteria, but enhances them by allowing the designs to be carried out to specified acceptable risk criteria set by management. R/C therefore allows management to have an appreciation of the consequences of failure and the benefits of defining acceptable risk. This approach has been successful in several cases, and Contreras et al (2006), Tapia et al (2007), Terbrugge et al (2006) and Bawden (2008), discuss typical examples.

Underground support design is based on various methods for example rock mass classification methods (Q, N', MRMR), support resistance, energy absorption and analytical models. These methods, though widely used and well understood, are very technical and empirically derived. The R/C concept is not applied in these designs (Brummer and Kaiser, 1995; Davies, 1997; Stacey, 2006; Joughin, 2008). The consequences of support failure are not evaluated against the corresponding design.

Risk analysis has been practised in other areas of underground mining. Bawden(2008) cites discussions on; seismic risk assessment (Owen et al, 2002); crown pillar risk assessment (Carter and Miller, 1995), mine pillar risk assessment (Pine and Thin, 1993), risk assessment of underground excavations (Pine and Arnold, 1996) and rock risk analysis in coal mines (Duzgun and Einstein, 2004). However, these authors do not analyse risk inclusive of a quantitative evaluation of the consequences of failure. As such, these authors do not consider risk in financial terms.

Brummer and Kaiser (1995) and Davies (1997) considered cost of consequences in their work on seismic damage and general mining risk analyses respectively. Brummer and Kaiser (1995) determined risk from uncontrollable losses, namely falls of ground as the product of the probability of failure and cost of consequences. This cost is apportioned from a cost matrix. Davies (1997) proposed the Potential Problem Analysis method to risk evaluation. In this method, risk is evaluated in monetary terms. However, in this approach, an assessor subjectively determines the cost of the consequences. The value of the losses can therefore be easily over or understated. The probabilities of occurrence are also generated qualitatively, making the method subjective.

Brummer and Kaiser (1995) concluded that it is a challenge to assign a cost to a rockfall collapse. Horsley and Medhurst (2000) suggest that such an approach requires a well-defined geotechnical environment or stability analysis problem and that the resulting probabilistic outcome should take into account the actual relationship to mining cost impacts. These issues will be addressed in the risk analysis approach proposed in this dissertation.

2.3 Risk/Consequence analysis in design

Current underground excavation designs are based on the traditional "non-risk" design approach with the following stages:

- collect the geotechnical data required for design of the underground excavation
- design to a factor of safety (FOS) or probability of failure (POF) criterion
- provide the resulting support specifications to the mine planners for their design and economic calculations
- Apply monitoring procedures to determine the adequate performance of the supported excavation according to the expectations of the rock engineer.

Swart et al (2000) suggested that stope design is carried out using empirical, analytical and observational methods. Empirical methods involve rock mass classification methods and engineering judgement. The classification methods include the Q system (Barton et al, 1974), Modified stability Number 'N' (Potvin, 1998, Potvin and Hadjigeorgiou, 2001); and MRMR by Laubscher (1990). Analytical methods involve closed form solutions, numerical methods and structural analyses (Swart et al, 2000). Observational methods, formalised by Peck (1969) rely on monitoring ground movement to measure instability. This method serves more as a check for the empirical and analytical methods as the support and rock mass should behave as per design.

These methods of design are driven by technical excellence (Terbrugge et al, 2006) and generally ignore the likelihood that the design may fail, thus ignoring the impact of failure (Davies, 1997). There is no link or input into these designs by management. However, part of the responsibility of management in the strategic mine planning process is the identification of strategies that maximise returns to shareholders at an acceptable level of risk, thus implying risk-consequence considerations.

A risk-consequence analysis approach would be as follows:

- determine the acceptable risk criteria for each consequence at the outset
- collect geotechnical data appropriate for the required level of design confidence
- calculate the required POF for rock support design corresponding with the acceptable risk criterion
- perform the rock support design to the required reliability and
- Establish best practice management tools for the supported excavation required.

Stacey et al (2006) suggested that such a reversal in approach to underground support design delivers a design in conformance with the business requirements of the project. This approach allows the executives to define their own risk criteria, which, for example, will consider consequences of rock falls and collapses and hence lead to the most appropriate support design. As defined by Stacey et al (2006), the following should fall within the required criteria: risk to personnel, equipment damage, economic impact, force majeure, industrial action and negative public relations. This design approach also allows the designer to identify measures that will improve the design, and reduce the risk. Such measures would include improved support, monitoring and advance warning of danger, hazard awareness training, and reduced number of workers or reduced working time.

The traditional approach is deterministic, whereas the risk approach is probabilistic. The latter allows engineers to determine the technical and financial risk for alternative mining, support, operational and other scenarios. The mining executives then choose the scenario that best matches the risk profile that is acceptable to them in terms of their operational and strategic goals. It can then be concluded that rock engineering design can be based on specified acceptable risk criteria. Whittlestone and Johnson (1993), Read (1994), Tulp (1994), and Terbrugge et al (2006) have described how this is applied to open pit slope design. They stress the importance of quantification of risk in order to come up with the appropriate mitigation strategies in monetary terms.

Applying this approach to stope support design in underground mining, Stacey et al (2006) proposed a thorough design process as shown in Figure 2-4. The process also presents a checklist to ensure that a defensible design has been carried out. Steps 1 and 2 specify the objectives of the project including the acceptable risk. Steps 5 to 7 allow the designing of various support systems and evaluating them using risk as criterion. Step 10 is an interactive part of the process where feedback and reviews for the other steps are conducted and implemented.

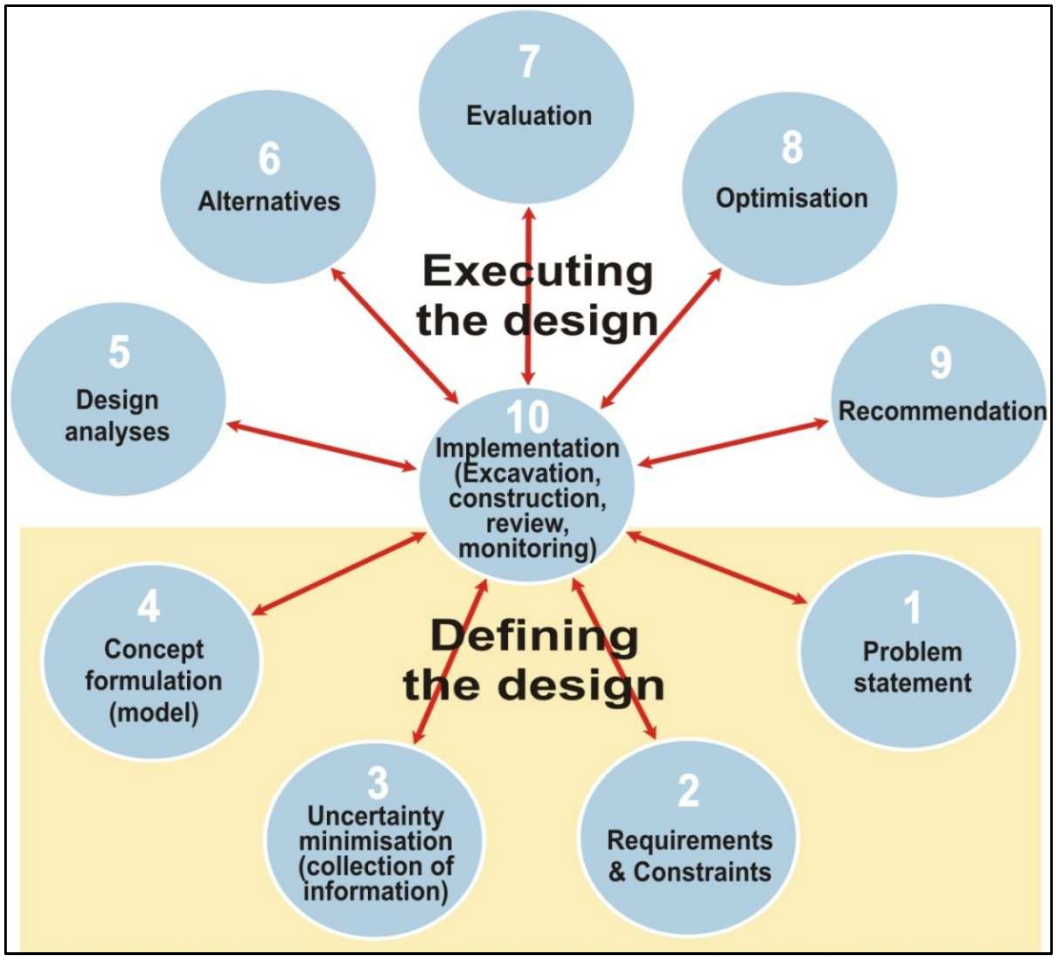


Figure 2-4: The Engineering wheel of design (Stacey et al 2006)

2.4 Risk evaluation process

Risk evaluation processes have been put forward for rock engineering applications. Terbrugge et al (2006) proposed the process in Figure 2-5 for open pit mining. Stacey et al (2006) adapted this approach for underground stope design. The general approach first determines the reliability of the design; this is expressed as the probability of failure (POF). Reliability equals one minus POF. In Figure 2-5, the overall POF is determined from the POF's in the fault tree (left side). Risk is the product of the probability of failure and the consequences of failure. In the process, event trees are then constructed to determine the risk associated with each consequence (centre frame in Figure 2-5). The resulting risks are measured against the accepted risk levels.

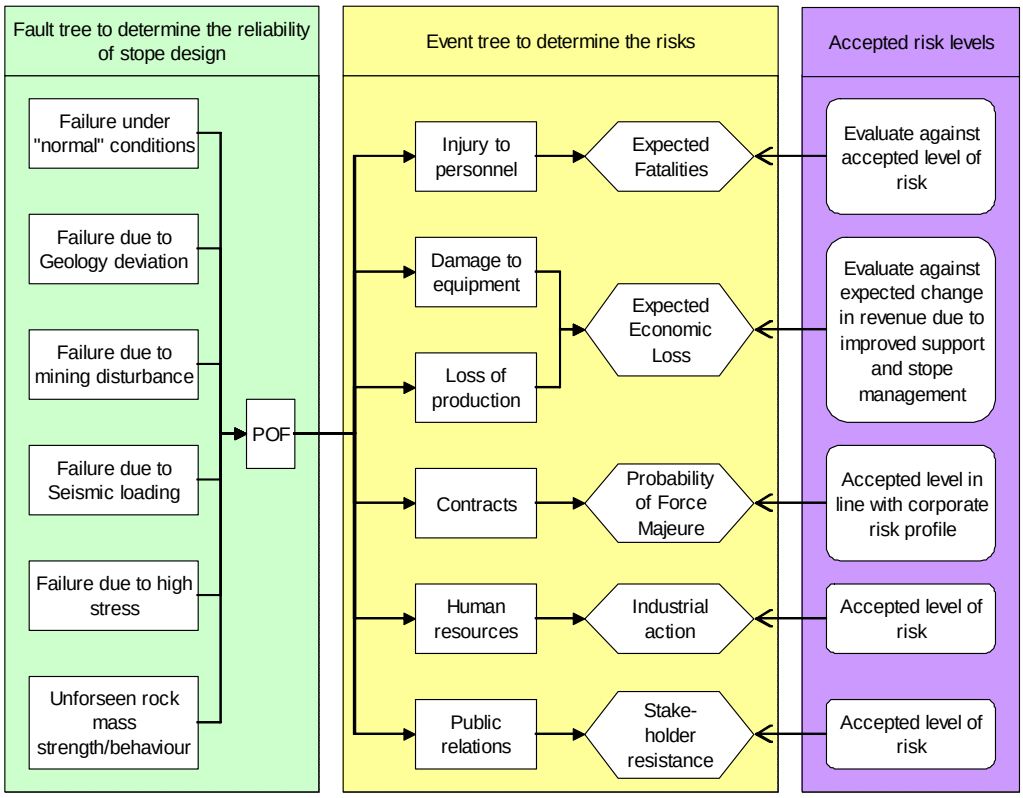


Figure 2-5: Risk evaluation process (Stacey et al, 2006)

Contreras et al (2006) and Tapia et al (2007) described the application of the risk evaluation for overall risk management at Cerrejon Mine. This approach, Figure 2-6, illustrates how they integrated risk evaluation into the management process. Once a risk model has been developed, the acceptable risk criteria are defined up front. The stope POF is then evaluated together with the associated consequence analysis. Risk calculations, as described in earlier sections of this Chapter, are then applied. The risk evaluation follows the methodology described by Terbrugge et al (2006), where the calculated risk is compared to the defined risk criteria. The risk is then either accepted or declined. If accepted, monitoring and reviewing will be ongoing as more information becomes available within the project. If declined, the design process must be repeated to ensure that there is compliance with the defined acceptable risk criterion.

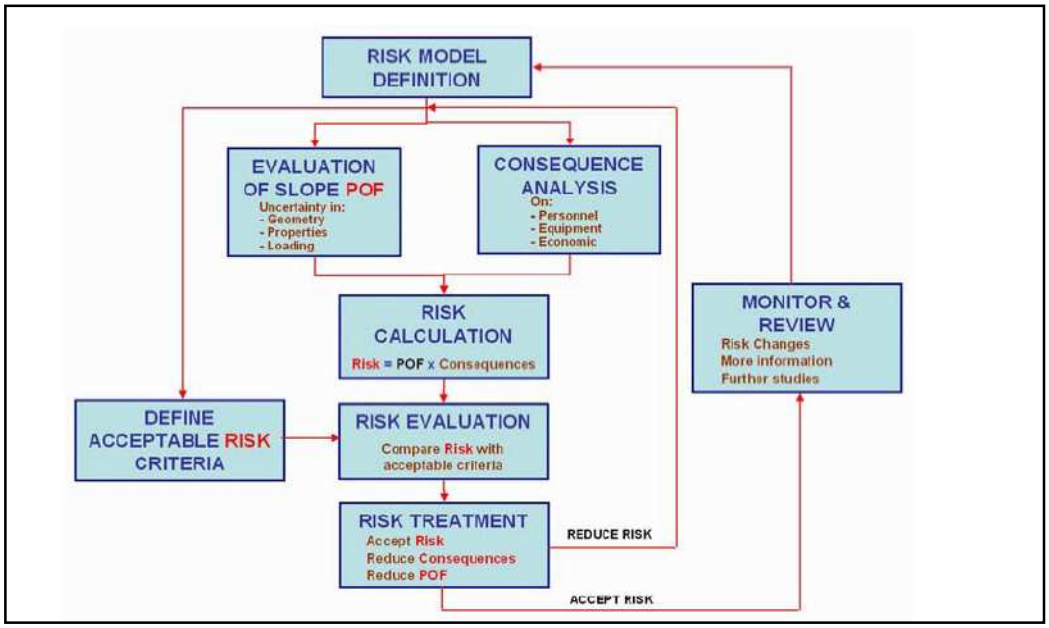


Figure 2-6: Risk management at Cerrejon mine (Contreras et al, 2006)

This approach (Contreras et al, 2006) is a comprehensive approach to design. It is an important part in the Engineering Wheel of Design proposed by Stacey et al (2006). The research described in this dissertation is based on this approach. Subsequent sections in this chapter will discuss the approaches to risk estimation and determination of the POF. Evaluation of the consequences of rockfalls in stopes will be discussed in Chapter 3.

2.4.1 Risk estimation

There are three main ways of carrying out risk estimation namely qualitative, semi quantitative and quantitative methods. Semi quantitative methods are a combination of the qualitative and quantitative methods and hence will not be discussed. The other two methods are discussed in the next subsections.

2.4.1.1. Qualitative risk estimation

This approach uses a classification system to assess the likelihood and consequences of a threat(s). Depending on the scale of the project and consequences to be examined, categories ranging from high, medium to low are defined. The probability of occurrence of hazards or failures is then assigned on a judgemental basis. Scales will differ depending on the application of the assessment. Table 2-1 shows an example of a qualitative evaluation. In a risk analysis, a team of experts brainstorms the appropriate qualitative evaluation of a probability of occurrence. They then refer to the corresponding quantitative scale in this table to obtain the numerical value of the probability of occurrence.

Table 2-1: Classes for probabilities of occurrence (Kirsten, 1999)

Classes for Probability of Occurence	
Qualitative Evaluation	**Quanitative Evaluation**
Certain	Every time (1.0)
Very High	1 in ten (10^{-1})
High	1 in a hundred (10^{-2})
Moderate	1 in a thousand (10^{-3})
Low	1 in ten thousand (10^{-4})
Very Low	1 in a hundred thousand(10^{-5})
Extremely Low	1 in a million (10^{-6})
Practically Zero	1 in ten million (10^{-7})

While qualitative risk assessments are widely applied, their application is restrictive. This is because they are only as good as the quality of the subjective judgement that goes into them. There is also difficulty in assessing dependencies and their relationships (Stewart, 2001). In this research, however dependencies and relationships are critical and subjective judgement would skew the results. Qualitative risk is therefore not applicable and hence this research discusses qualitative risk no further.

2.4.1.2. Quantitative risk estimation

This approach uses a stochastic system to assess the likelihood and consequences of a threat(s). The probability of occurrence and value of the consequences are derived quantitatively from available information and numerical simulation. This approach eliminates the element of subjective judgement also taking into account possible interdependences of hazards.

The general equation for expressing risk quantitatively is:

$$Risk = P\ (failure) \times Cost\ of\ Consequences$$

However, the disadvantage of expressing risk as given above is that it assumes that each failure is directly associated with a cost and that the risk always has to be expressed as a cost (Einstein, 1996). The limitation arising from this perception is that quantitative risk is only applicable where a monetary risk criterion is applied. A risk criterion in practice is however not restricted to monetary risk criterion. Einstein (1996) proposes the following general expression:

$$Risk = P\ [Performance] \times P\ [Consequence/Performance] \times u(X)$$

Where: *P[Performance]* is the performance uncertainty,

P[Consequence/Performance] is the conditional probability of a consequence resulting from a given performance

u(X) is the utility of a vector of attributes. *X* may include monetary attributes or just numerical attributes for example X_1 = cleanup, X_2 = repairs and X_3 replacement.

Risk quantification using this general equation has found application in various sectors. For example, in landslide risk management (Australian Geomechanics Society, 2000) these guidelines propose the calculation of risk to property in monetary terms and risk to individuals in numerical terms as follows:

The risk to property *R(Prop):*

$$R\ (Prop) = P\ (H) \times P(S: H) \times V\ (Prop: S) \times E$$

Where:

R(Prop) is the risk (annual loss of property value)

P(H) is the annual probability of the hazardous event (the landslide)

P(S: H) is the probability of spatial impact by the hazard (namely, of the landslide impacting the property, taking into account the travel distance) and for vehicles, for example, the temporal probability

V (Prop: S) is the vulnerability of the property to the spatial impact (proportion of property value lost)

E is the element at risk (the value or net present value of the property)

For loss of life, the individual risk is determined as:

$$R\ (DI) = P\ (H) \times P(S: H) \times P\ (T: S) \times V\ (D: T)$$

Where:

R(DI) is the risk (annual probability of loss of life (death) of an individual)

P(H) is the annual probability of occurrence of the hazardous event (the landslide)

P(S: H) is the probability of spatial impact by the hazard (for example of the landslide impacting a building (location) taking into account the travel distance) given the event, that is exposure

P(T:S) is the temporal probability (for example of the building being occupied by the individual) given the spatial impact

V(D:T) is the vulnerability of the individual (probability of loss of life of the individual given the impact).

Joughin (2008) has adapted this method for determining individual risk for use in underground mining. He equates individual risk as:

$$Risk_{(H)} = POF \times Exposure \times Spatial\ coincidence \times Cost\ of\ consequence\ (H).$$

Exposure is the ratio of the area of the rockfall to the total area considered. Spatial coincidence refers to the likelihood of men being in the same area as the rockfall when it falls. Typical consequences of rockfalls on personnel include medical costs, wages and compensation, Section 54 production losses,

SIMRAC levies, investigation and legal costs. The total risk is the sum of risks of the different consequences, expressed as:

$$Total\ risk = \sum (Risk_{(H1....Hn)})$$

Where H_1 to H_n are the different consequences. Joughin's methodology is applied in this research.

In summary, the purpose of Quantitative Risk Assessment (QRA) is to calculate the value of the risk and to enable better risk communication and decision-making (Lee and Jones 2004). As cited by the Australian Geomechanics Society (2000), Ho et al (2000), and Lee and Jones (2004) the steps in any risk consequence approach should answer the following questions:

- What are the possible dangers/problems? [Danger identification]
- What are the possible consequences and/or elements at risk? [Consequence/Elements at risk identification]
- What might be the degree of damage in elements at risk [Vulnerability Assessment]
- What is the probability of damage? [Risk Quantification/Estimation]
- What is the significance of estimated risk? [Risk Evaluation]
- What should be done? [Risk Management]

From this discussion it can be concluded that risk has two components: the probability of occurrence and the consequences. In order to evaluate the risk associated with rockfalls; both components have to be evaluated and quantified. Section 2.5 will describe the determination of the rockfall annual frequency of occurrence. The consequences of rockfalls will be discussed in Chapter 3.

2.5 Rockfall frequency analysis

The previous sections have shown that the POF is a critical input into the overall risk equation. This section reviews alternative approaches for determining the POF of rockfalls in underground mines.

Determination of the POF can be done by objective or subjective approaches. Similar to quantitative risk estimation, the objective approach uses available statistical data. The subjective method requires the use of judgement. The subjective approach, however, leads to a qualitative risk assessment and hence will not be discussed any further. Joughin (2008) states that there are three ways of objectively obtaining the probability of failure: use of analytical models, use of historic rockfall data and use of rock mass classification methods. This section describes these approaches.

2.5.1 Analytical methods

Swart et al (2000) discussed the use of analytical models to design stope layouts and support. The models determine the factor of safety and the probability of failure of a design. An FOS of less than 1.0 indicates failure while an FOS greater than or equal to 1.0 indicates stability. The probability of failure is the probability of the factor of safety being less than 1.0.

To determine the factor of safety, the input parameters, capacity and demand are first determined through investigation. As these are variable, where data are adequate, probability distributions are fitted to the data. Where there are limited data, an appropriate assumed distribution is fitted. Common distributions are the normal and triangular distributions. Using simulation techniques like Monte Carlo or Point Estimate method by Rosenblueth (1975), a distribution of FOS can then be determined. Simulations are an interactive process in which random numbers from the input distributions are picked and the factor of safety calculated. This process is done a specified number of times. The resulting FOS then generates the overall FOS distribution.

Analytical models are based on different rockfall mechanisms. Joughin (2008) describes the common rockfall mechanisms associated with keyblock and beam behaviour, and explains how the POF is determined in each**.** The POF of rockfalls is therefore a sum of the probabilities of failure for each mechanism. Keyblock and beam behaviour will be discussed in the following sections.

2.5.1.1. Beam models

Beam models include the elastic beam theory and the Voussoir beam model (Diederichs and Kaiser, 1999 and Brady and Brown, 2006). In both models, to determine the annual frequency distribution of rockfalls of a given size, an assumption is made. The assumption is that the rockfall will cover the full span of the excavation; hence, the excavation is divided into units that represent the full span of the excavation multiplied by the face advance increment. The annual face advance divided by the face advance increment thereby gives the number of units to be considered in one year. This can then be simulated to determine the annual frequency distribution of rockfalls.

Limitations of the use of the elastic beam theory are that it is only applicable to laminated strata, and it is limited to geotechnical areas where vertical or sub vertical or inclined joints do not cross the laminations (hence does not take into consideration the tensile strength of joints). It is therefore applicable mostly to South African coal mines where laminated unjointed shales occur in the roof. The Voussoir beam model is, however, applicable where sub vertical joints occur.

2.5.1.2. Keyblocks

Blocks are bound by geological structures, joints, parting planes, curved joints, faults, and stress fractures. The blocks can form on the excavation surface or beyond the exposed walls. Depending on the location, size, shape and interaction between the structures, the blocks can fall out. This logic has been built into a software package, JBlock by Esterhuizen (2003). The software analyses keyblock

stability based on discontinuity networking and limit equilibrium methods. Joughin (2008) describes how the key blocks are generated and how the probability of occurrence of rockfalls is then calculated.

Several authors have used this software for keyblock stability analysis and support design. Examples include Joughin and Stacey (2006), Stacey and Gumede (2007), Dunn et al (2008) and Joughin et al (2009). JBlock output includes:

- a list of possible block sizes where the stable and non-stable blocks are indicated,
- a relationship between the failed blocks and the block size (area or volume),
- total simulated area,
- percentage area that is a key block,
- total number of key blocks,
- total number of failed blocks,
- average rockfalls per 1000m^2
- average failure volume per 1000m^2 exposed. Since the equivalent area mined out is known, the rockfall frequency distributions can be scaled.
- rockfall frequency distributions whereby the user has the flexibility to change the input formats,
- actual excavation geometry to enable modelling of the equivalent area mined,
- support codes to allow the integration of this software with the risk model.

JBlock (Esterhuizen, 2003) is therefore a useful tool in this research and its application will be discussed in Chapter 6.

2.5.2 Historical rockfall data

Several mines collect data on rockfalls and store the information in databases. Duzgun and Einstein (2004) used the databases of 12 mines in the Appalachian region to determine the probabilities of occurrence of rockfalls. They applied the time intervals between falls (TBF) and the numbers of falls per annum (NOF) approach. These are based on analysing the time between falls of ground and the number of falls occurring over a given time. The weakness with these approaches is that they take no consideration of the size and location of the rockfalls. Stacey and Gumede (2007) established that different sizes of rockfalls have different probabilities of occurrence. This implies that in a rockfall risk study the different sizes have different probabilities of occurrence, different impacts and hence different risks.

Joughin (2008) used several South African mine databases to determine statistical distributions based on the number of rockfalls of given sizes per year. The distributions formed the basis for determining the probability of occurrence or annual frequency of occurrence of rockfalls of given sizes. This method has proved to be applicable where the mine has a comprehensive database that is regularly updated. However, the research showed that most mines do not maintain a regular rockfalls database. Joughin's method takes into consideration the size of rockfalls. It has direct application to the research in this dissertation as will be discussed in later sections of this chapter.

2.5.3 Rock mass classification methods

The stability of excavations in rock masses and the required support can be determined using rock mass classification methods.Geotechnical mapping or borehole logging is used to obtain the required input parameters for these classifications. The input parameters however exhibit variability. Dunn et al (2008) fitted distributions to the geotechnical parameters to express such variability. However, the wider application of these classification systems in determining probability has been through simulation. Techniques like Monte Carlo and Point Estimate (Rosenblueth, 1975 and Harr, 1987 respectively) fit a probability distribution to each of these classification systems. This probability is then applicable to risk analysis. The weakness of this approach is that it cannot determine the size or number of rockfalls expected from a given geometry (Joughin, 2008). Another shortcoming is that it is not possible to determine the extent of possible damage or the exposure of personnel or equipment using this method. This approach is therefore not applicable in quantitative risk estimation.

2.6 Consequence analysis

The probability of occurrence and the consequences have been established as the main components of quantitative risk analysis. The previous section discussed the approaches to determining the probability of occurrence of rockfalls. This section focuses on approaches for determining the consequences of rockfalls. Consequences are divided into economic and personnel consequences. Economic consequences involve all losses resulting in loss of monetary value. Personnel consequences relate to individual risk.

2.6.1 Economic consequences

Several approaches have been used to determine the economic consequences of rockfall accidents. Baecher (1981), Chowdhury and Flentje (1998), Joughin (2008), Duzgun and Einstein (2004) and Marx (1996) have different approaches to quantifying the cost of the consequences of excavation damage. Chowdhury and Flentje (1998) put forward a formula to determine total cost;

$$Total\ cost\ = C\ (budget)\ x\ P\ (failure)\ + C\ (consequences)$$

C(budget) refers to the budgeted cost of mining. *P(failure)* is the probability of the structure failing. *C(consequences)* refers to the cost of consequences associated with the failure.

Duzgun and Einstein (2004) suggested a relative cost approach. This approach relates the attributes of damage to cost by means assigning a unit cost to one of the attributes then expressing the other attributes relative to this unit cost Baecher (1981).

$$Rj = P(Roof\ fall)x \sum_{i=1}^{n} u(Xi) * Cj$$

In the equation *Rj* is the total risk cost, *u(Xi)* is a utility representing the consequences. *Xi* is the number of roof falls where *i* = 1,2, 3…n. *Cj* is the cost of possible consequences, where *j* = 1,2,3…*m* are the different consequences.

While this approach provides relative costs for attributes that are difficult and or unethical to cost, it has its shortcomings. The weighting is subjective based on the user's engineering judgement and experience. Another point is that any consequence that is not realised is taken to be zero. Attributes considered include the cost of cleanup, equipment damage, disruption and cleanup in operation, injury and fatality.

Joughin (2008) suggested a breakdown of the consequences into direct and indirect impacts. Indirect impacts are a result of the direct impacts. The cost of each consequence is determined on an activity basis. Using this method the effect of location and size of the rockfall is considered in determining the associated losses. The strength in this methodology is that the consequence cost drivers can be determined; hence, the impact of individual rockfalls can be assessed.

2.7 Acceptable risk

In any job or operation, there exists a risk. Some risks are intolerable, some are tolerable and some are acceptable. Intolerable risk is an imminent threat and results in the job or operation being stopped for safety and or economic reasons. However, where careful controls can be implemented to reduce the risk, such risk is known as tolerable risk. Tolerable risk is reduced to acceptable risk levels. Different industries have different perceptions on acceptable risk.

2.7.1 Acceptable risk in mining

In 2003, the Biennial Mine Health and Safety Summit held for the South African Mining Industry agreed on targets and milestones to be attained by 2013. A milestone goal of reducing fatality and injury frequencies to reach parity with international standards was set. This is to be achieved by a 20% annual reduction of the injury and fatality rates across all commodities. Mines have however set on a "zero accident" rate target. However, Wong (2005) suggested that nothing could be 100% reliable, that human beings, one day, will invariably make a mistake. Stacey (2006) agreed with Wong and therefore describes the zero rate target as idealistic but not realistic. Stacey therefore emphasizes the need to define what risk is acceptable.

The question then posed is "What is acceptable risk?" Rowe (1979) discusses a risk as being acceptable when those affected are generally no longer (or not) apprehensive about it. In defining acceptable risk, Baecher and Christian (2003) quoted the four conclusions drawn by Starr (1969):

i. The public is willing to accept 'voluntary' risks roughly 1000 times greater than 'involuntary' risks;
ii. Statistical risk of death from disease can be taken as a psychological yardstick for establishing the level of acceptability of other risks;
iii. Acceptability of risk is proportional to the third power of benefits;
iv. Public awareness of the benefits of an activity influences the societal acceptance of risk.

Otway and Cohen (1975) criticised these conclusions, but, according to Baecher and Christian (2003), accepting that acceptable risk exhibits regularities is important.

Starr and Whipple (1980) noted that, "implicit in the term, acceptable risk is acceptable to whom?" To answer this question, Duzgun and Lacasse (2005) define it as the level of risk that requires no further reduction. When risk is accepted by the society, no further reduction measures are required for it. However, where careful controls can be implemented to reduce the risk, such risk is known as tolerable risk.

In mining companies, acceptable risk criteria standards have been set for project level/global level decisions. Individual project risk analysis has not yet found universal application (Summers, 2000). He suggests that this is because of the following:

- Misunderstanding of the types of risk and risk analysis,
- Confusion with the terminology ,
- The erroneous belief that risk analysis is of value only in safety.

Summers (2000) and Coulthard et al (2005) suggested that limited application of risk analysis could be a result of the following:

- Lack of awareness,
- Lack of expertise,
- Perception that risk analysis is unnecessary,
- Analysis paralysis,
- Time and cost pressures,
- Difficulty in quantifying risk,
- Mistrust of results,
- Lack of understanding by senior management.

The common result of these perceptions is that there is a general lack of trust in risk analysis and there is an unwillingness to accept risk analysis as a management tool. Thus, no acceptable risk criteria are set in mining projects.

In mining projects, typical guidelines for acceptable risk would include acceptable criteria for fatalities, likely damage to equipment, production losses or other incidents of significant impact and failure resulting in a force majeure declaration. Steffen and Rigby (2005) suggest that such criteria will determine the degree of conservatism to be applied to design. Several authors, Summers (2000), Duzgun (2005), Terbrugge et al (2006), Steffen (2007), Stacey (2006, 2007) and Joughin (2008) agree that defining acceptable risk criteria is essential for the management and success of a project.

2.7.2 Risk acceptance criteria

Techniques to risk acceptance include the F-N curves and As Low as Reasonably Practicable (ALARP). Other techniques also exist which typically compare the probability of failure with risks that are seen as the 'Acts of God' or largely inevitable events. However only the common approaches will be discussed in this research, as they are more widely used and comprehensive.

2.7.2.1. F-N curves

The F-N curves are a common way of expressing societal risk in various industries. The method is a graphical representation of the relationship between the annual probability of an event causing N or more fatalities, and the number of fatalities, Figure 2-7. The graph shows zones of negligible, ALARP and Intolerable risk for different industries. The commonly applied benchmark is the annual frequency of a fatality by natural causes of a teenager aged between 10 and 14. This annual frequency is known to be 10^{-4} and it represents the divide between voluntary and involuntary risk (Steffen, 2007). This age group is rated as the safe age group; hence, the annual frequency of one or more fatalities on a mining operation should be 10^{-4} for the design to be as safe as practically possible.

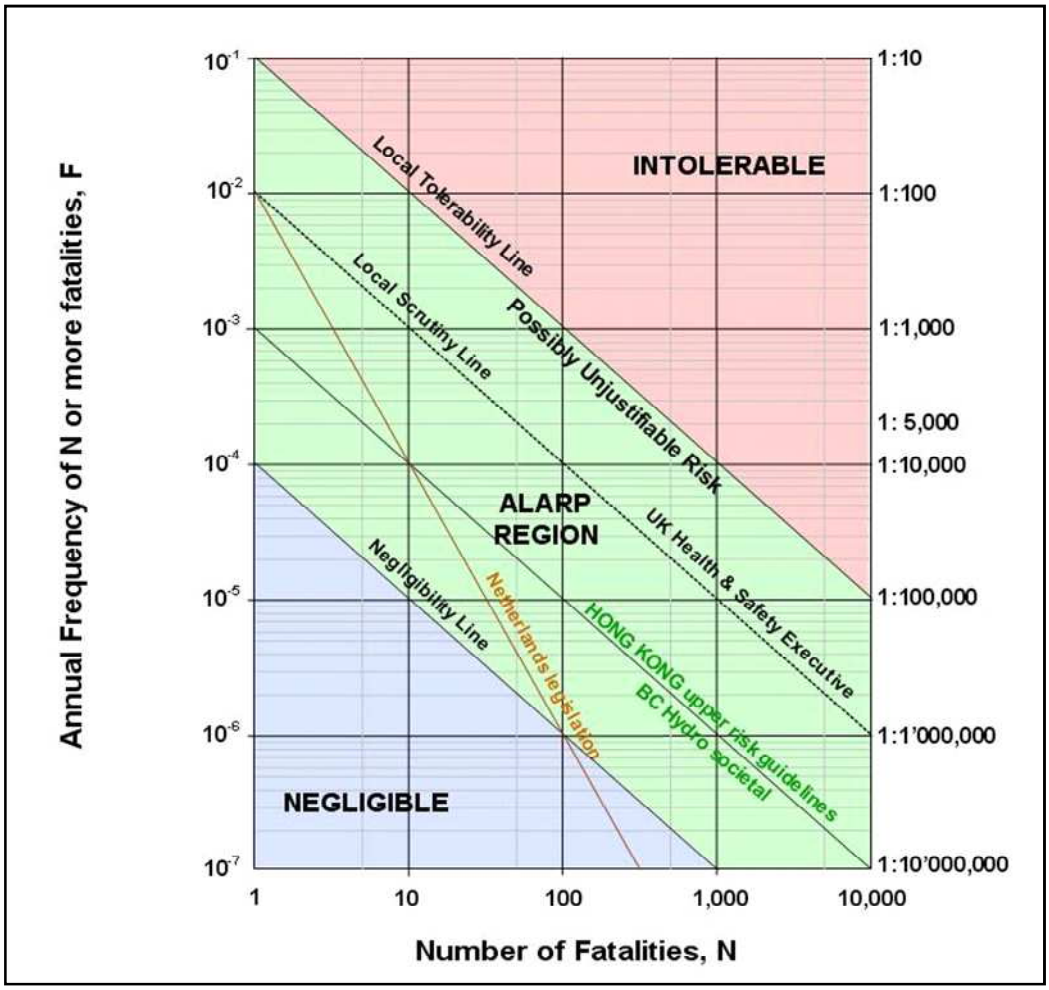


Figure 2-7: F-N Graph for bench marking safety impact (Contreras et al, 2006)

According to Ball and Floyd (1998) the shortcoming of this approach is that different annual frequency tolerance ranges are used depending on company or national criteria, thus making the approach applicable based only on a set company criterion. This implies that comparison is applicable only to companies and countries with similar criteria.

2.7.2.2. The ALARP principle

The Edward versus National Coal Board case (HSE, 1999 – Annex 3, paragraph 2) which established this principle as law in Britain in 1949 best explains the ALARP principle. It says, "it is stated that the case established that a computation must be made in which the quantum of risk is placed on one scale and the sacrifice, whether in money, time or trouble, involved in the measures necessary to avert the risk, is placed on the other; and that, if it be shown that there is a gross disproportion between them, the risk being insignificant in relation to the sacrifice, the person upon whom the duty is laid discharges the burden of proving that compliance was not reasonably practicable". A diagrammatic representation of this principle is illustrated in Figure 2-8.

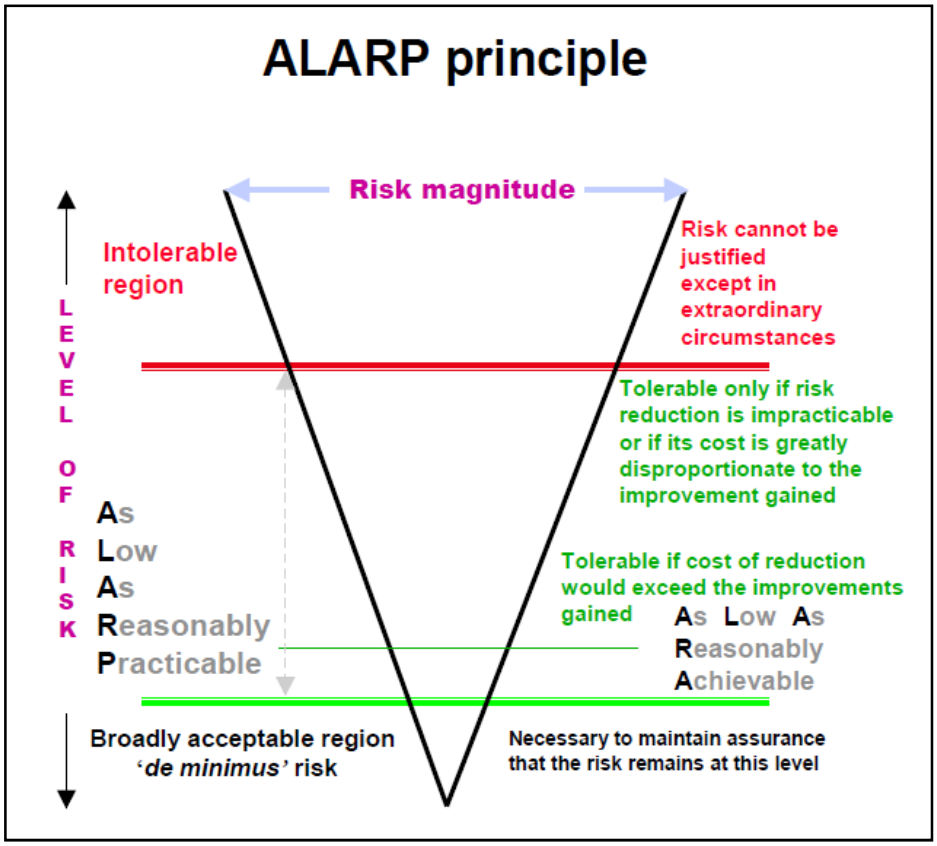


Figure 2-8: Major hazard facilities regulations – guidance notes

In this research dissertation, the two approaches to acceptable risk have been integrated into a risk cost-benefit analysis approach. In the analysis, the benefits of risk mitigation are weighed against the total costs of losses (including the risks imposed).

2.8 Summary

In this chapter, various aspects of risk were discussed. The concept and definitions of risk were explored. Then the significance of risk in the mining concept was expounded. The importance and applicability of risk/consequence criteria in design was then discussed. Evaluation of rockfall probability of failure, frequency, economic and personnel risk methods were also described. Various viewpoints on what acceptable risk is were explored. Chapter 3 will discuss the evaluation of the consequences of rockfalls. This is a critical input to the risk equation.

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3 Evaluation of consequences of failure

3.1 Introduction

It is evident that the frequency of rockfall related accidents occurring in the mining industry are very high. The consequences of these rockfalls are numerous, some direct and others indirect, and resulting losses can be significant (Udd, 1982; Marx, 1996; Hebblewhite, 2003; Stacey et al; 2006, Joughin, 2008). However, research by the author proved that there are no records of regular evaluation of such losses on the tabular mines of South Africa. It showed that when there are rockfall accidents on a mine, injuries or damage to equipment, accident reports are often compiled. Aspects such as production loss are however not reported. The financial benefit of risk mitigation systems is therefore currently based only on the cost of implementation. The financial losses resulting from the accidents are not taken into consideration.

This chapter will review the consequences associated with rockfalls and collapses in narrow tabular mine stopes. First, the types and mechanisms of rockfalls are described. Then the approaches to evaluating consequences of rockfalls including personnel injuries, excavation damage and equipment damage are discussed.

3.2 FOG types and mechanisms

It is important to discuss the types of rockfall hazards and failure mechanisms. Fall of ground (FOG) or rockfalls can be either gravity falls or rockbursts. Rockbursts are caused by seismic activity. Seismic activity is the release of energy that is stored in a rock that is exposed to very high stresses. Such high stresses are typical of deep and ultra deep mines (Jager and Ryder, 1999). Gravity induced falls are associated with the rock mass failing along planes of weakness and discontinuities, for example joints, parting planes. Daenhke et al (2001) emphasize the importance of the influence of discontinuities, density of fracturing, frictional strength of bedding surfaces, joints and mining induced fractures on excavation stability.

In an ongoing SIMRAC project, SIM060201 Track A, comprehensive research is being carried out on the fall of ground mechanisms and precursors. The work highlights the type of rockfalls occurring in the different commodities, mechanisms of failure, causes of failure and the precursors. A list of these FOG types and mechanisms is shown in Table 3-1. The summary of the preliminary findings are attached in Appendix 3.

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Table 3-1: FOG types and mechanisms (SIMRAC Track A, in progress)

FOG Types	FOG Mechanisms
Joint/ fracture release surfaces	Loss of cohesion
Joint/joint release surfaces	Reduced friction coefficient
As above plus bedding parallel partings	Change of confining force
Backbreak	Shear forces
Panel/back collapse	Tensile failure
Wedge failure	Loosening
Curved joint failure	Buckling
Fall from & into weak layer in Hanging wall(Basal Reef shale)	Toppling
Flow of fractured rock mass (WAF)	Fracturing
Master bedding plane fault	Beam failure
Beam failure	Movement of thrust structures (dome)
Presence of stress fracturing, fall-out of slabs, sidewall, hangingwall pillars	
Faults/dyke contacts	
Toppling from brows or previous FOG	
Collapse of brow	
Uncontrolled Goafing	
Sloughing, steep ore bodies	
Failure of middling	
Sill related falls	
Crown/sill pillar instability	
Failure of stress damaged rock, Carbon Leader	
Collapse of gothic arch and guttering	
Cross bedding	
Man assisted FOG	
Blasted support	
Rigging	
Scrapper damaged support	
Decline in support performance with time and closure	
Overhanging face and sidewalls high stoping width	
Seismically induced rockfalls	
Rockbursts	

An FOG is an undesired occurrence in any excavation as it is a direct deviation from the intended performance of the excavation. The following sections discuss the consequences of such occurrences and a review of the determination of the associated costs. The focus will be on personnel injuries, damage to equipment and damage to/loss of excavations in stopes.

3.3 Direct and indirect costs of rockfalls

Consequences fall into two categories: direct and indirect. Indirect are a result of the direct consequences. Joughin (2008) in Table 3-2 illustrates these consequences. This list was drawn up from intensive research and workshops involving experienced mining personnel from across the South African mining industry. These direct and indirect consequences will be discussed in subsequent sections of this chapter.

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Table 3-2: Direct and indirect costs (Joughin, 2008)

Direct Consequences	Indirect consequences
Injuries and fatalities	Excessive individual risk exposure for personnel (Evaluate risk on F-N graph)
	Temporary mine closure imposed by the Department of Minerals and Energy (Section 54) –(revenue from lost production less variable costs) up to 5 days of partial or full mine closure
	Medical and rescue operation costs
	Wages and compensation
	Investigation and inquiries – cost of professional time
	Re-training – cost of re-training new employees
	SIMRAC levies
	Legal costs – (determined from precedent practice)
	Insurance premiums – (Increase due to accident record)
	Industrial action – difficult to quantify
	Stakeholder resistance (reputation, share price and cost of capital)- difficult to quantify
Damage to equipment and machinery (mobile and fixed)	Loss of production. - only production affected by equipment loss
	Cost of re-deployment of machinery and personnel to maintain production
	Replacement costs - large rockfalls
	Cost of repairs – depends on extent of damage (size of rockfall)
Damage to excavations (access excavations and stopes)	Loss of reserves - Net present value of lost reserves
	Loss of production (revenue from lost production less variable costs) if there is no alternate source of production
	Replacement of access excavations–large rockfalls
	Rehabilitation - proportional to size of rockfall and importance of excavation
	Dilution (in stopes)
	Re-deployment of machinery and personnel to maintain production
	Clean up operations. – (depends on size of rockfall)
	Insurance premiums – (Increase due to claims)
	Stakeholder resistance (reputation, share price and cost of capital)- difficult to quantify

3.4 Personnel injury and fatalities

Mine accidents have an impact on several stakeholders including the individual and his family, the mining company, government, public and the economy (Udd, 1982; Davies and Teasdale, 1994 and Marx, 1996). The degree of impact varies with the stakeholder.

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In this section, the impact of personnel injury on the mining company will be described. The fact that no formal accountability or documentation of accidents costs is carried out on the mines could imply that managers are not fully aware of the additional costs brought by accidents. Adams (2004) estimated total cost of injuries and fatalities in the gold mining sector to be approximately R330 million in 2003. They estimated the cost for a single fatality to be R 1 million. However, it is agreeably unethical to place a value on the pain and suffering borne by persons suffering accidents and by their family, let alone value the changes to lifestyle which may result from the accident (Udd, 1982). The severity of the impact of the accident is also usually not proportionate to the agent causing the accident. For example, a broken limb caused by a mobile machinery accident would have the same impact as a broken limb from a rockfall accident.

This section will review the determination of some of the consequences incurred by the mine because of injuries or fatality on the mine.

3.4.1 Temporary mine closure

The Mine Health and Safety (MHS) Act (1996), Section 54, stipulates that an official has the power to stop any operations he/she considers unsafe for the workers. The average shutting down period after a fatal accident is 5 days (SRK Workshop, 2007). In this period the DMR can shut down the whole mine or a particular section thereof.

An example of the impact of a section 54 declaration is a fatal accident at Moab Khotsong underground mine in 2009. Here Julia Schoeman (AngloGold Ashanti Spokeswoman) told Reuters that, "It will affect production, approximately 45 kg of gold (1 447 ounces) per day will be lost" (March 17, 2009. http://www.fin24.com). The DMR has shut several mines under section 54 over the years. Loss of production is the most significant impact of such closure.

3.4.2 Medical costs

There is a relationship between the nature and degree of injury and the medical treatment required for an injured person. Severe injuries may require hospitalisation and surgery whereas minor injuries are treated as outpatient cases. Marx (1996) determined the cost of hospitalisation and outpatient visits at the Western Deep Levels Hospital. In his methodology, he looked at the hospital invoice of each accident investigated and used the following formula to determine the total medical costs:

$$Medical\ Costs = Hospital\ Invoice + Outpatients\ Cost + Trans\ port\ Cost$$

In the formula the hospital invoice is the total cost of all attention and medication given to the individual while in hospital (if they were hospitalised). The outpatient cost represents the cost of consultation and medication for minor injuries or reviews after hospitalisation.

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The transport cost is the cost of transportation from the scene on the accident/ shaft bank to the medical station or hospital.

The Rand Mutual Assurance (RMA) in accordance with the Workmen's Compensation Act 30 of 1941 provides compensation benefits to Chamber of Mines members. This Act ensures a guarantee that employees are compensated for occupational injuries and diseases, medical expenses, disablement and fatalities. It operates on a non-profit basis, whereby the employers pay the full premium to it to cover their employees. When accidents occur, the employer pays for the employee's medical expenses and he can then claim funds used from RMA. Depending on the nature and degree of injury, the employee can then claim compensation from RMA. Member employers are refunded on an annual basis for any unused funds.

3.4.3 Wages and compensation

When an employee is injured, during the period he is absent from work he is paid wages. The amount and method of payment however differs with the severity of the injury as stated in Schedule 4 of the Compensation for Occupational Injuries and Diseases Act (COIDA), 1993 (No. 130 0f 1993, amended by Act 61 of 1997).

Compensation includes all the benefits set out in COIDA to which an employee/claimant may become entitled. The payment of compensation depends on the facts of each case. Examples of the types of compensation include, inter alia:

- Permanent disablement
- Temporary total disablement
- Temporary partial disablement
- Medical treatment
- A disability pension
- A fatal pension or lump sum payment to dependants (more about these concepts is presented later)

Compensation is always in monetary terms. It is aimed at restoring, as far as possible, the financial loss the employee or his dependant/s has suffered following his injury, illness or death in a work-related accident. Compensation does not include a claim for other types of loss or damage, such as property loss or damage, loss of future earnings, loss of amenities and of life.

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3.4.3.1. Calculating temporary total disablement

This is defined as the compensation to an employee who is incapable of working because of a workplace injury. Compensation, excluding the medical expenses, is directly proportional to the employee's earnings at the time of the accident. All such wages as paid by the employer to the employee during the period of injury is reimbursed to the employer by RMA. Appendix 2 describes the various degrees of compensation for disabilities.

3.4.3.2. Calculating scheduled and unscheduled injuries

The Compensation for Occupational Injuries and Diseases Act (COIDA), First Schedule (Annex 2) describes compensation for scheduled injuries and the corresponding permanent disability percentages. A panel of doctors, who will determine the degree of permanent disability, assesses unscheduled injuries.

3.4.3.3. Calculating fatality compensation

A man killed in industrial accidents in South Africa has a working life expectancy of approximately 20 years. Hence COIDA considers 6 000 working days to be the total number of days lost for any injury or permanent total disability.

The overall wages and compensation cost according to Marx (1996) is worked through the following steps:

$$Net\ Cost = Direct\ Labour + Labour\ Overhead - Payment\ to\ mines + Effect\ on\ insurance$$

Where: $Direct\ Labour\ Cost = Daily\ Wage\ Rate \times Days\ of\ normal\ work$

And;

$$Labour\ overhead\ cost = Labour\ overhead\ cost\ per\ shift \times days\ of\ normal\ work$$

However, for the RMA affiliated mines, the RMA pays the wages and compensation costs on behalf of the mine. The mines in turn pay an insurance premium to the RMA. This premium is determined from the payments and claims history of the company. Unaffiliated mining companies have to pay the employees directly.

3.4.4 Investigation and enquiries

In the event of an accident resulting in death or serious harm to any persons on a mine, the Mine Manager shall report such an occurrence to the DMR. The DMR, in accordance with the Minerals Act Section 28, shall then investigate any such accident. It is the duty of the Mine Manager to make the report and facilitate the resources necessary for the carrying out of the investigation.

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The government bears the salaries costs for the DMR transport and other administrative costs. Whereas the mine bears the costs for the resources it provides. It also caters for the salaries and wages of the persons involved in the investigation inclusive of both the internal and external persons. The cost of the time spent by the mine personnel on the investigation can be significant. This will vary according to the severity of the accident in terms of the number of people involved and the duration of the enquiry.

To obtain an indication of the cost, Marx (1996) considered the total cost to the company per worker for the time spent on the investigation. He determined the cost of investigation as:

$$Cost\ of\ Investigation = \sum(Time\ spent \times Personnel\ hourly\ rate)$$

$$Cost\ of\ resources = \sum Costs\ to\ facilitate\ execution\ of\ DMR\ duties$$

This method however focuses on the investigation involving mine personnel only but does not consider the cost of hiring external expertise to carry out an independent review. Such review costs can be significant (SRK Workshop, 2007).

3.4.5 Re-training

Marx (1996) discussed the quantification of retraining costs. His approach takes into consideration the cost of retraining employees that were part of the accidents or the new employees replacing the injured. Most mines however run a skills training centre, and this centre is financed by the mine's budget. Training is an on-going exercise on the mines, hence taking the cost of refresher training of crew involved in a rockfall accident as an additional cost would be double accounting hence over estimating the cost.

3.4.6 SIMRAC levies

The Safety in Mines Research Advisory Committee (SIMRAC) was established in terms of Regulation 35.1 of the Mine Health and Safety Act, No. 50 of 1991, Section 26 (9) (a) to advise the Director-General of the Department of Minerals and Energy. The committee was also given the responsibility of the Safety Research Account which was created under the same Act. This innovation was to fund and administer the industry related research and development.

After reportable and fatal accidents are reported to the DMR, a statutory levy is imposed on all mines for the reportable accidents. These levies makeup the funds in the Safety Research Account. This fund is used in research projects whose aim is to minimize and prevent such accidents from re-occurring, through research and development.

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Each reportable accident is converted to allocated days lost as per the DMR Accident Classification Code Book. The days lost is a function of the nature of the injury and the part of the body injured**.** The total number of days lost for a three-year period ending June 30th of each year is then calculated.

SIMRAC has an annual budget that is based on Safety and Health Research needs. This budget is divided by the total number of days lost to determine the amount payable per allocated day lost. Hence, the resultant per day fee times the number of days lost on that particular mine is the resulting cost of the SIMRAC Levy. This amount is paid for three consecutive years on all reportable accidents. A fatality is equivalent to 6 000 days lost.

Total Levy=Allocated days x Rate/day

3.4.7 Intangible costs

Udd (1982) and Joughin (2008) agree that it is impossible to estimate the indirect costs such as altered production plans, industrial action, and stakeholder resistance even though these could have a significant impact. Marx (1996) does not discuss these issues. After a major accident, morale is low amongst workers, and the Workers Union and Government are taking a strong stance against the mining company. This has a negative impact on the image of an employer, but such effects are difficult to quantify.

3.5 Operational consequences

Operational consequences involve equipment and excavation damage. Chicken (1996) described them as follows: direct costs are associated with physical assets such as plant and equipment (such as load and haul, ventilation system, utility vehicle and drill rigs). Indirect costs are less tangible and include production down time, lost man shifts, haulage stoppage time, stockpile shortage and inaccessibility of mine workings. The direct and indirect consequences are illustrated in Table 3-2.

Often a restricted view of the cost of risk is taken, and this leads to a grave misunderstanding of the economic significance of risk. Chicken (1996) described the factors leading to a restricted view on risk as including:

- A lack of awareness of all the components of the overall cost.
- Unquestioning beliefs in the conventions of costing risk in a particular industry (Civil, nuclear power, mining).
- The influence of regulatory requirements.
- Those costing the risk may have limited responsibility for cost.

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3.5.1 Damage to equipment and machinery

Damage to equipment and machinery can have significant impact. The direct and indirect consequences associated with such damage are illustrated in Table 3-2. Loss of production is incurred when there is equipment downtime due to the damage. Damaged equipment can be either replaced or repaired.

3.5.1.1. Plant and equipment

Equipment is classified as fixed, non-moving and mobile equipment. Plant will include piping systems, ventilation, electrical networking systems. Variations of the equipment used on the mine depend on the mining method, degree of mechanization, level of flexibility built into the mining plans and schedules.

The degree of damage will depend on the location, size and extent of cause of the damage. Remedial action includes either repairing or replacement of the damaged equipment. Marx (1996) showed that seismic activity generally causes more damage. He adopted an average of R 2704 per seismic damage for combined plant and equipment costs and R 250 per gravity incident. It should be noted that this study was conducted between 1994 and 1996 on deep level conventional mining.

Seismic data were obtained from the mine's seismologist and the above average determined. No data on gravity FOG damage were kept on the mine hence the figure suggested was an estimated average from the mine personnel.

3.5.2 Damage to excavations

When a fall of ground occurs in the stope, there is disruption of the mining activities. The direct and indirect consequences associated with such damage are illustrated in Table 3-2. The damage can affect part of the total excavation. While all such losses are incurred on the mines, research showed that no records of such losses are kept.

3.5.2.1. Rehabilitation costs

Marx (1996) described ways of determining rehabilitation and opening up costs. He considers the cost of labour involved in the processes and the loss of production incurred there from, the sum of which would give the total cost of rehabilitation. The drawback with his method is that he does not consider the quantity and cost of the resources (materials and machinery) involved in the rehabilitation process. No consideration of the relationship between the size of the fall of ground and the material required for the necessary remedial action is given. This therefore underestimates the total cost.

`

In determining the labour costs, he ignores the relationship between the size of the FOG and the labour, time and resources required. He does not elaborate on the typical remedial actions for stope or development FOG. It will be more accurate to consider remedial action as being cleanup for small FOG's, re-supporting for intermediate and large falls, and re-establishing of tunnels or re-raising for stopes.

3.5.2.2. Loss of production

Scott (1982), Marx (1996) and Duzgun and Einstein (2004) show that the cost of loss of production is significant. Any blasts lost on the panels directly affect targeted productivity. Marx (1996) assessed the cost of a FOG to the mine, collecting data over a two-year period, 1993 to 1994. A total of 185 rockfall accidents, both seismic and gravity induced were analysed. These are inclusive of reportable injuries and fatalities. For each accident, he defined cost drivers, and through interaction with the mine personnel, derived the costs associated with each driver. All the data were then statistically analysed by determining the total, average, median costs, the standard deviation and the skewness of each activity (Table 3-3).

Table 3-3: **Cost of accidents (Marx 1996)**

Consequence	Total cost	Average cost per accident
Investigation	231 896.00	1 253.49
Admin	236 750.00	1 279.73
Plant and Equipment	257 294.00	1 390.78
Rescue	440 113.00	2 378.99
Levies	942 121.53	5 092.55
Medical	1 013 365.00	5 477.65
Wages and compensation	1 608 429.40	8 694.21
Training	7 843 582.43	42 397.74
Production loss	33 381 404.25	180 440.02
Total Cost	**45 954 955.61**	**231 896.00**

However, Marx does not consider the costs and impacts of the following:

- Mine closure imposed by the Department of Mines and Energy (DMR) according to Section 54 of the Mine Health and Safety Regulations,
- Re-establishing time of the mine excavations,
- The cost of risk mitigation and other safety initiatives.

`

3.6 Summary

From the methods discussed in this chapter, it is evident that the process of calculating costs is intensive and cumbersome. It requires a lot of information from various departments at an operation. There is a lack of a coherent methodology to determine the costs of the consequences on the operations. As such, engineers do not consider such losses when carrying out designs. There is therefore no method of determining a set acceptable risk criterion on financial terms. Management therefore has no direct input in the design process. This neglects the treatment of risk in support design as an integral part of the business planning process.

Based on this logic, it is fundamental to have a methodology for determining the cost of the losses that is readily used by the mine. The designers can apply it to determine the risk of a design. Setting this against the acceptable risk criteria defined by management, this approach will ensure thorough risk based design.

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4 Formulation of models

4.1 Introduction

In Chapter 3, various methods by which rockfall consequences are evaluated were reviewed. From this study, it was concluded that there exists no comprehensive methodology or software to determine the monetary value of these losses. This research has shown that when a fall of ground occurs in a stope the method of remediation depends largely on the size and location of the rockfall. Very small rockfalls are generally cleaned up and the area made safe. Larger rockfalls require the installation of additional support, or the panel may need to be re-established. It is apparent that such remediation costs money, therefore the size and location of rockfall can be used in determining the cost of the consequences.

A methodology to quantify the costs of these consequences in narrow tabular underground stoping is proposed in this chapter. However, this proposed methodology is limited to quantifying only stope damages and personnel injuries. Equipment damage will not be included in this discussion as little information could be obtained on the subject when the field research was conducted.

In this Chapter, a description of the general risk-cost approach to stope support design is given first. The overall approach to quantifying the annual cost of consequences of rockfalls is then discussed. Then the methodology for the determination of the cost of expected consequences due to rockfalls is proposed.

4.2 Model concept

The level of risk in a stoping operation can be related to the costs of risk control and expected cost of consequences from rockfalls. Figure shows the relationship between the risk and associated cost. The cost of risk control is the cost of support design, implementation, monitoring and evaluation in the stope. The Expected Annual Cost of Consequences (EACC) is the product of annual frequency of rockfalls in that stope and the costs of the associated consequences. The sum of the cost of annual risk control and EACC gives the total cost of the risk for that particular stope design.

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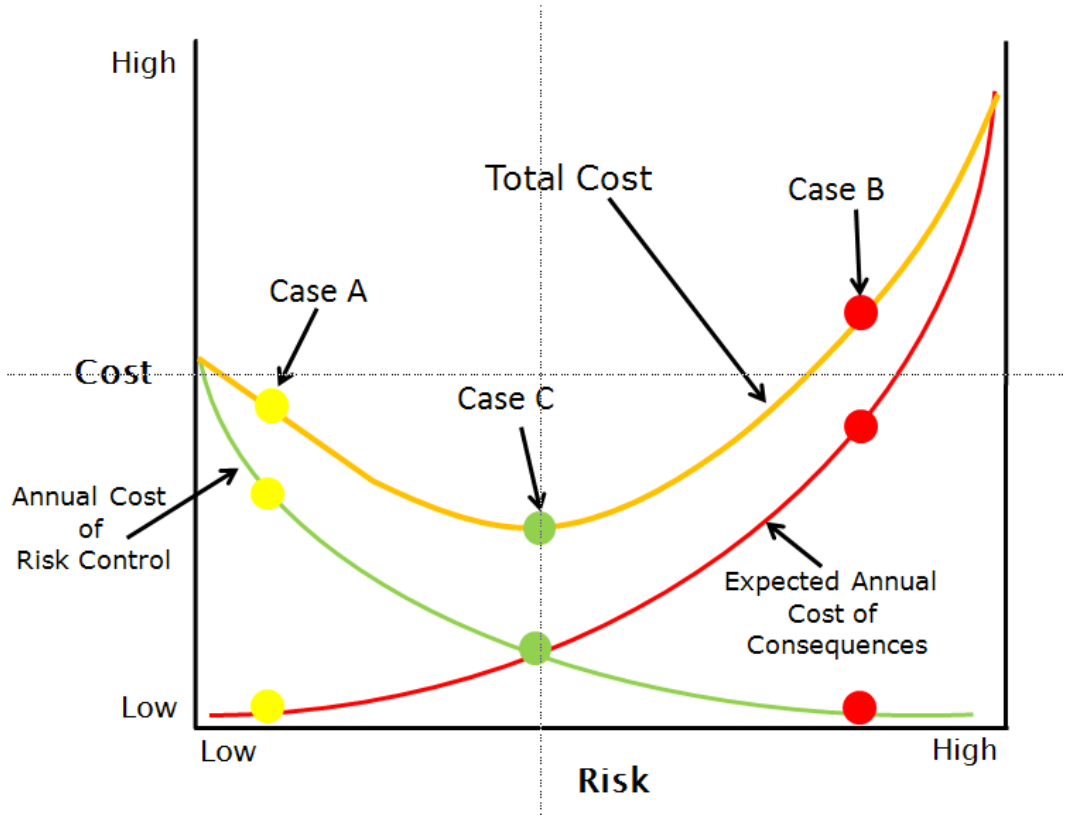


Figure 4-1: Model concept (Viner, 2002)

Figure 4-1 suggests that:

- when more investment is made in risk control, the expected cost of losses is significantly lower (Case A).
- when minimal investment is made in risk control (Case B), the expected losses can be severe.
- based on this risk cost approach, the design with the lowest total cost (compared to the other options) can then be considered the optimum design (Case C). However due to practical considerations the optimum design does not necessarily have the lowest point on the Total Cost curve. It can be a lower cost compared to the other options.

This design analysis can be carried out for different support systems on a single panel or for a whole ground control district. Current practice is that practioners carry out stope design based only on the cost of risk control. However, the total risk - cost approach gives a more comprehensive basis for comparing the risk associated with each support system.

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This approach to design will help answer the questions:

- how much support is too much support?
- What are the impacts of installing minimal or substandard support?
- What is the cost of failure of a given support system?

The risk cost approach is not a new concept. Duzgun and Einstein (2004) illustrated its application in coal mines. They explained the concept by using three scenarios, the status quo, over designed and the optimal support design. In each option a probability of occurrence of rockfalls, with an annual frequency of occurrence (t), is determined. The monetary value of each consequence of the rockfall (k) is then determined. Therefore, the expected cost (EC) of the consequences for each option is expressed as:

$$\boldsymbol{EC} = \sum_{\boldsymbol{n=1}}^{\boldsymbol{m}} (\boldsymbol{k_n}\ \mathbf{x}\ \mathbf{t})$$

Where k_1, k_2….k_n. represents the different consequences.

Relating this concept to Figure 4-1, the status quo is represented by Case B, a scenario whereby there is minimal support in the stope and high rockfall risk. The overdesigned option is represented by Case A, an over designed support system with very low rockfall risk. Case C represents the optimal design, an optimal risk cost based support design strategy, which balances the cost of risk control with the cost of the expected losses.

However, the work by Duzgun and Einstein (2004) has two shortcomings. Firstly, it does not take into account the size and location of rockfalls in determining the probability of occurrence and impact of different rockfalls. Secondly, the criteria applied to determine the cost of consequences is based on the user's judgement. This leaves room for subjectivity, thus over or understating the actual cost.

These shortcomings will be addressed by the approach proposed in this research. The research focuses on the determination of the direct and indirect costs, which is *k* in Duzgun and Einstein (2004). This subject will be addressed in more detail in the following sections.

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4.3 Expected annual cost of consequences: Overview

Individual rockfalls result in multiple consequences. As discussed earlier, these will vary depending on the size of the FOG, location, monitoring systems in place, evacuation procedures, exposure of men and equipment. Such variability in the factors affecting the resulting consequences is best captured in the form of an event tree. Figure 4-2 shows a simple event tree illustrating the direct consequences resulting from stope rockfalls. This illustration is drawn from

Table 3-2.

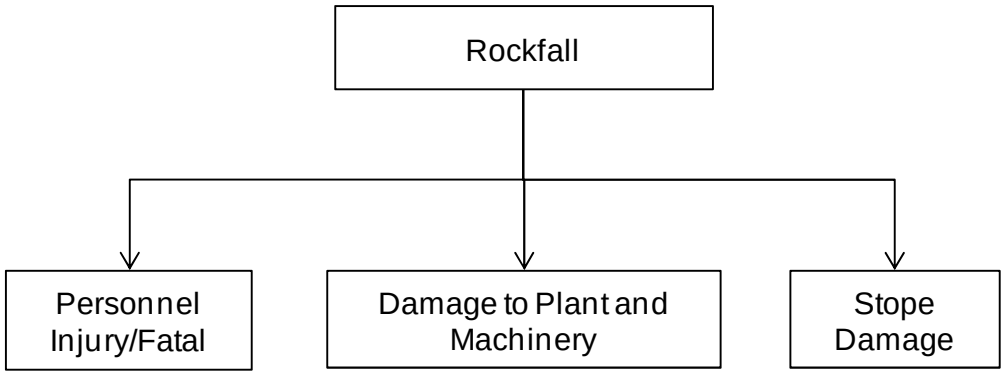


Figure 4-2: Direct consequences

By expanding the direct consequences in Figure 4-2 the possible indirect consequences are deduced. The extensions to each branch will be discussed separately and in more detail in the sections that follow.

The general approach to determine the expected cost of consequences on a mining operation is as follows:

- Through a study of the mining operation and brainstorming with experienced personnel, detailed event trees for each direct consequence in Figure 4-2 are constructed. The final tree will indicate the possible indirect consequences.
- Determine the distributions of the annual frequencies of rock falls of different sizes using historical rockfall data or numerically generated data (Esterhuizen, 2003).
- Quantify each indirect consequence; these will vary depending on the size of the FOG, location, monitoring systems in place, evacuation procedures, exposure of men and equipment.
- The product of the annual frequencies of occurrence and the cost of the consequence associated with each occurrence is the annual expected cost of that particular consequence.
- Summation of annual expected costs results in the total expected annual cost of losses.

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The following sections discuss rockfall distribution determination, evaluation of stope damage, equipment damage and personnel injuries and fatalities. The final section will discuss the approaches to risk cost mitigation.

4.4 Rockfall distributions

Rockfalls can occur in varying shapes, sizes, and at varying locations in a stope. It is a statutory requirement for any mine to maintain a rockfall database. This database enables the determination of the 95% fallout height, panel risk rating, and measure of support performance. Research has shown that some mines maintain an extensive database whereas on others, the databases are incomplete or in some non-existent. An extensive database is required for the derivation of rockfall distributions.

To quantify the consequences of rockfalls, the proposed methodology requires the following input from the rockfall databases:

- rockfall length, width, and height,
- location of the fall,
- complete and consistent recording of rockfalls,

From these records, annual frequencies of rockfalls will be determined.

4.4.1. Annual frequency of rockfalls

Rockfall lengths are grouped into bins and the frequency of falls in each bin determined based on the rockfall database or the application of numerical analysis, (Esterhuizen, 2003). Where no rockfall records are applicable, stochastic probabilistic methods can be applied to joint properties to determine keyblocks and the corresponding probability of failure of such blocks. Tyler et al (1991), Beauchamp et al (1998), Esterhuizen (2003), Gumede (2006), Stacey (2007) and Dunn et al (2008) are authors that have used this approach. The JBlock software also outputs block dimensions and the frequency of failure of blocks of a given size. This information is essential in the research as shall be discussed later in this chapter.

Both methods, the historical database and stochastic methods provide a rockfall frequency vs. rockfall size relationship which can be determined over a given period of mining. Figure 4-3 is an example of a relationship between the frequency of occurrence of rockfalls and the length of the rockfalls parallel to the mining face. From the graph, it is observed that the smaller size rockfalls have larger frequencies of occurrence. This observation was also noted by Gumede (2006) and is consistent with observations in industry.

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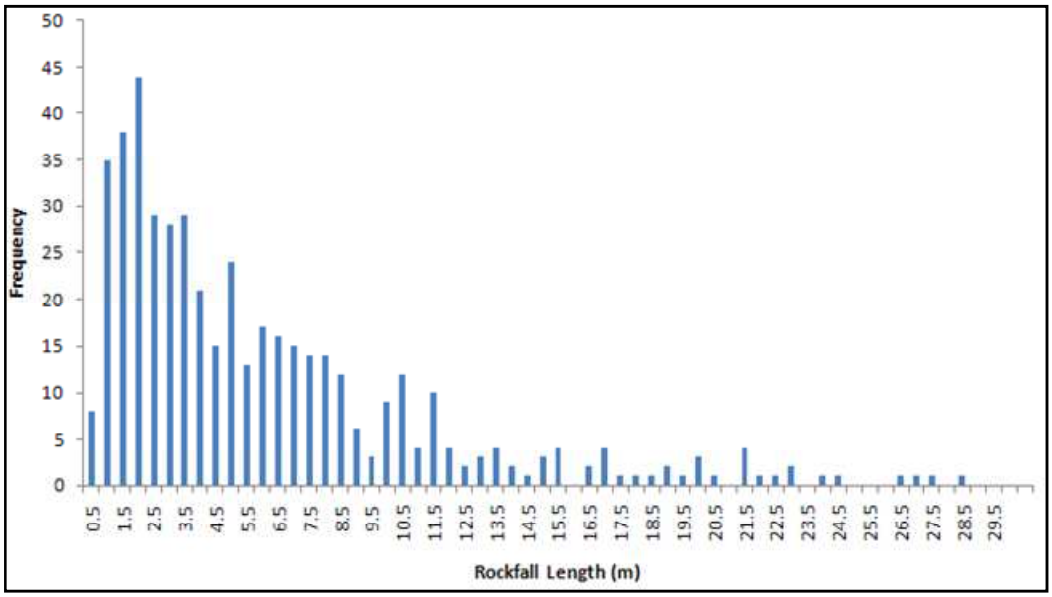


Figure 4-3: Rockfall frequency

4.5 Stope damage losses

The distribution of rockfalls described in the previous section occurs at any location within the panel. As a result, stope damage can occur with varying consequences. This section will discuss the stope damage consequences and a generic event tree will be developed for all the possible remedial actions and consequences. Lastly, the procedure to evaluate the resulting consequences in monetary terms is proposed.

4.5.1 Stope damage event tree

When a rockfall occurs in a stope a decision must be made whether to stop or continue mining, and under what conditions mining can continue. Typical remedial recommendations may include altering or increasing the support, reducing the mining span or leaving a pillar adjacent to the collapse. These choices are generally based on logistical and economic demands, and all have different cost and risk implications.

After consultations with mining and rock engineering personnel on several mines, a generic event tree, Figure 4-4, was constructed. The tree illustrates the series of choices that are made relative to size and location of a collapse.

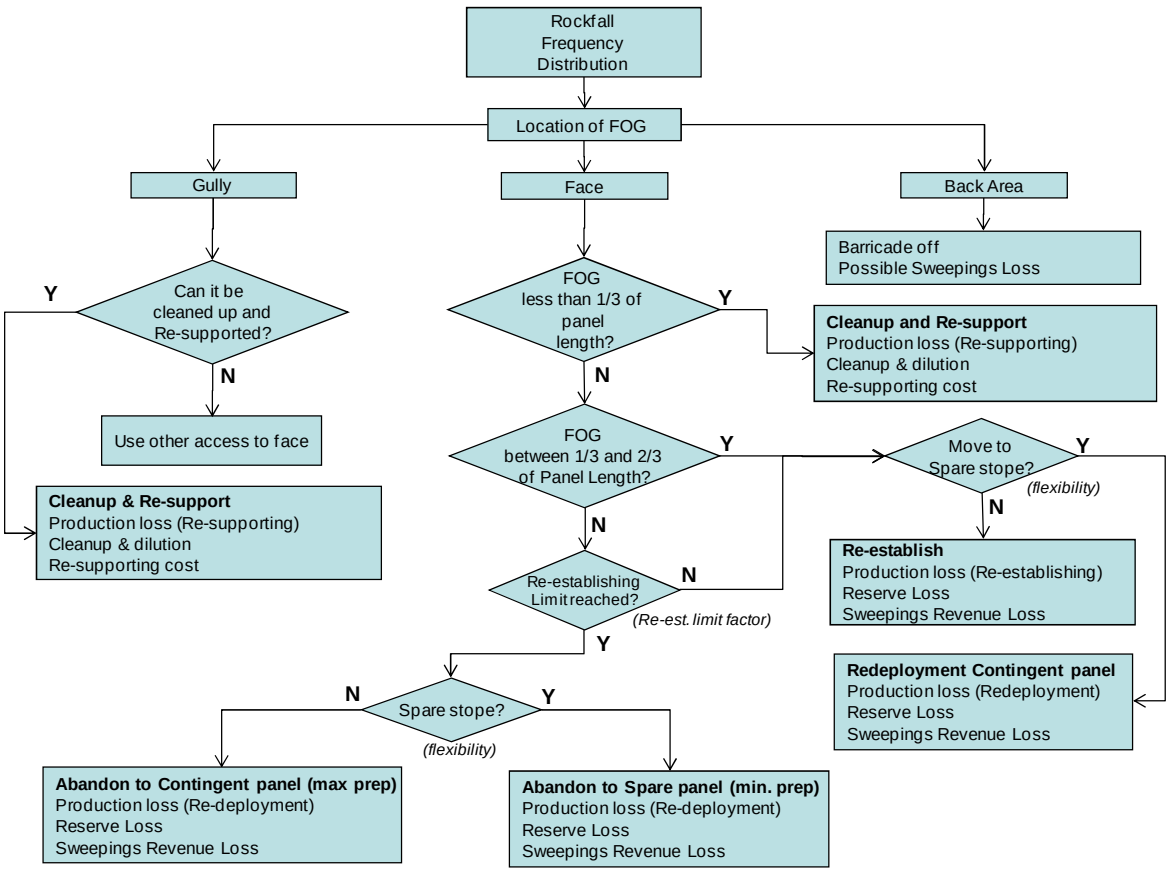


Figure 4-4: Stope damage event tree

As shown in Figure 4-4, in a stope, a rockfall will occur in the gully, face area or back area. Falls in the gully, where possible, are cleaned up and the area re-supported, otherwise another access to the panel face is sought. Falls in the back area are barricaded off. These two areas are however, not of interest in this research as studies showed that the occurrence frequency is minimal.

This research focuses on rockfalls in the stope face area. The falls occurring in this area are classified into categories based on the dimensions of the fall and the associated consequences. The section below describes the applicable classes.

Classifying the extent of stope damage based on the face length of the rockfall relative to the panel face length is common practice on mines. Apart from the condition of the rock mass in the stope, the length of the fall is used to determine the corresponding remedial action adopted. Legislation also requires that falls of specified lengths be reported to the Inspector of Mines.

`

4.5.2 Remediation action classifications

The remedial action classifications are based on industry practices of handling falls of ground. Four generic classes were identified and these are discussed further below.

4.5.2.1. Class 1: Very small rockfalls

Class 1caters for very small rockfalls whose maximum dimensions, Rf_l or Rf_w (Figure 4-5), are less than 1m in length. The main assumption is that these rockfalls are waste rock, hence mixing of waste rock with ore results in dilution. Cleaning them up and hoisting them to surface with the ore results in an extra cost of processing waste being incurred. This cost is the dilution cost.

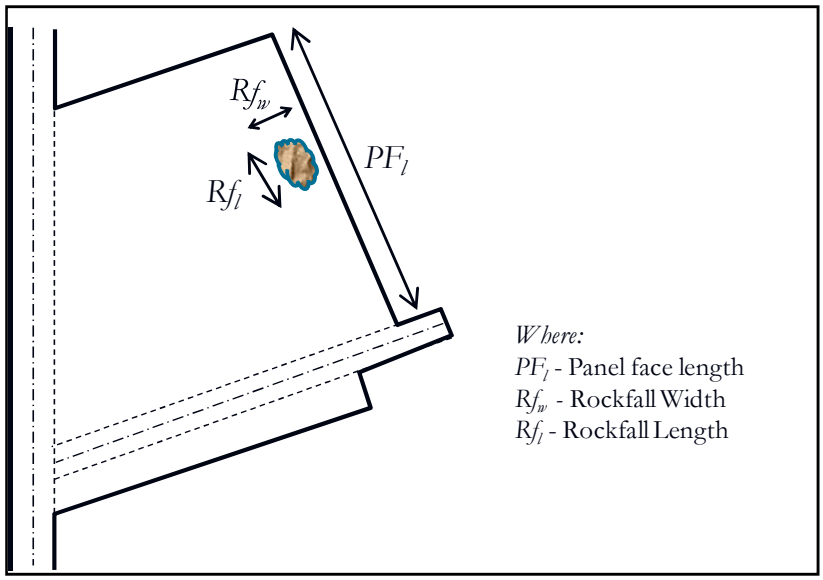


Figure 4-5: Class 1, small rockfalls

4.5.2.2. Class 2: Rockfalls with lengths less than one third panel face length

Class 2 represents those rockfalls whose dimension (Rfl) parallel to the panel face is less than one-third the panel face length (Pfl), Figure 4-6. This fall is cleaned up resulting in the waste rock being hauled with ore, hence causing dilution. The face area will also have to be re-supported; this results in a re-supporting cost. The cleaning up and re-supporting will result in production downtime, which results in production profit loss. These consequences are illustrated in Figure 4-4.

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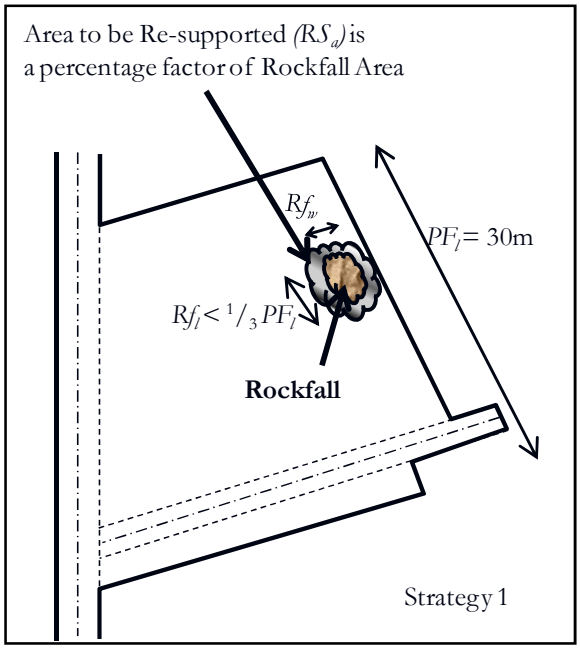


Figure 4-6: Class 2, cleanup and re-support

4.5.2.3. Class 3: Wide end re-establishing

In this category, the rockfall length *(Rf_l)* is between a third and two thirds of the Panel Face Length (*Pf_l),* as shown in Figure 4-7. The panel will be re-established and the re-establishing end width *(Rw_b)* on breast is calculated as:

$$Rw_b = Pf_l - (Rf_l + 1m)$$

An allowance (1m) is given to allow for safety while re-establishing. The re-establishing raise width on dip *(Rw_d)* is the rock engineer's recommendation. Rw_b and Rw_d are mine standards for deep South African gold mines.

As shown in Figure 4-4, resulting consequences are production loss due to re-establishing, possible redeployment of personnel, and loss of revenue from sweepings lost. Availability of a spare stope, which is a contingent panel, will result in lesser magnitude of losses than when there is no spare stope.

There are circumstances however when the fall occurs in the top of the panel. When this occurs, re-establishing can be done from an adjacent panel hence Rw_b is 0m and Rw_d is equal to Pf_l.

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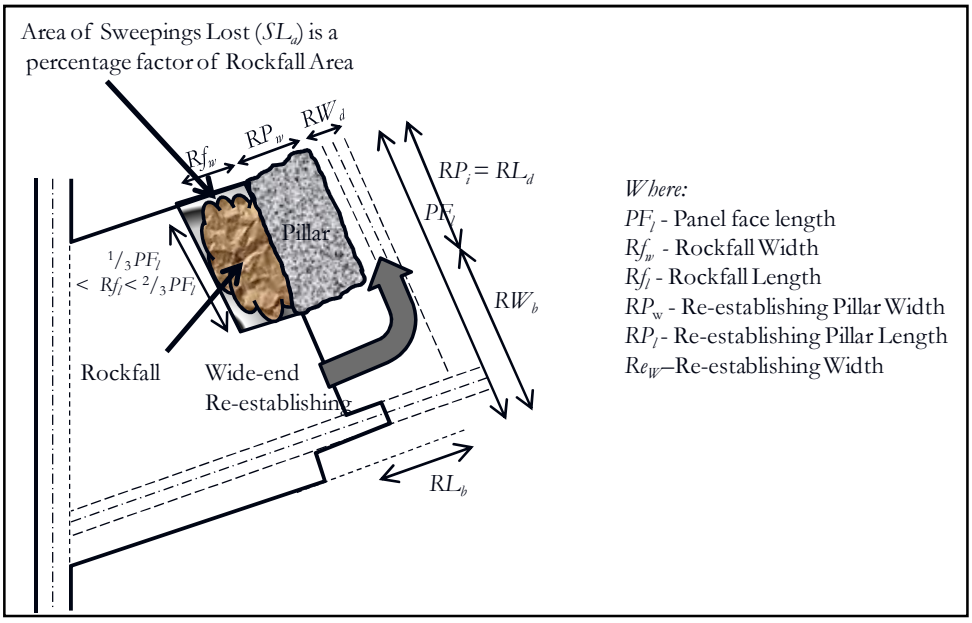


Figure 4-7: **Class 3, re-establishing**

4.5.2.4. Class 4: Re-establishing and abandoning

In this category the rockfall length, Rf_l, is more than two thirds of the Panel Face Length (Pf_l), Figure 4-8. The panel is either re-established or abandoned based on the following criteria:

- If the fall occurs when the panel has advanced not more than the re-establishing limit, see 4a in Figure 4-9, then the panel will be re-established.
- If the fall occurs when the panel has advanced more than the re-establishing limit, see 4b in Figure 4-9, the panel is abandoned.

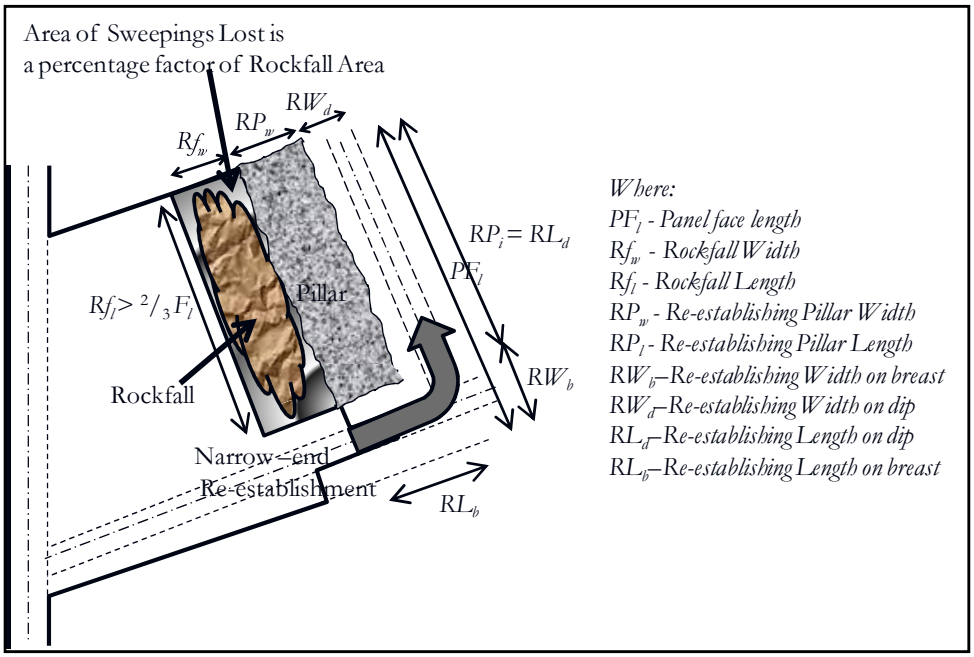


Figure 4-8: Class 4, large rockfalls

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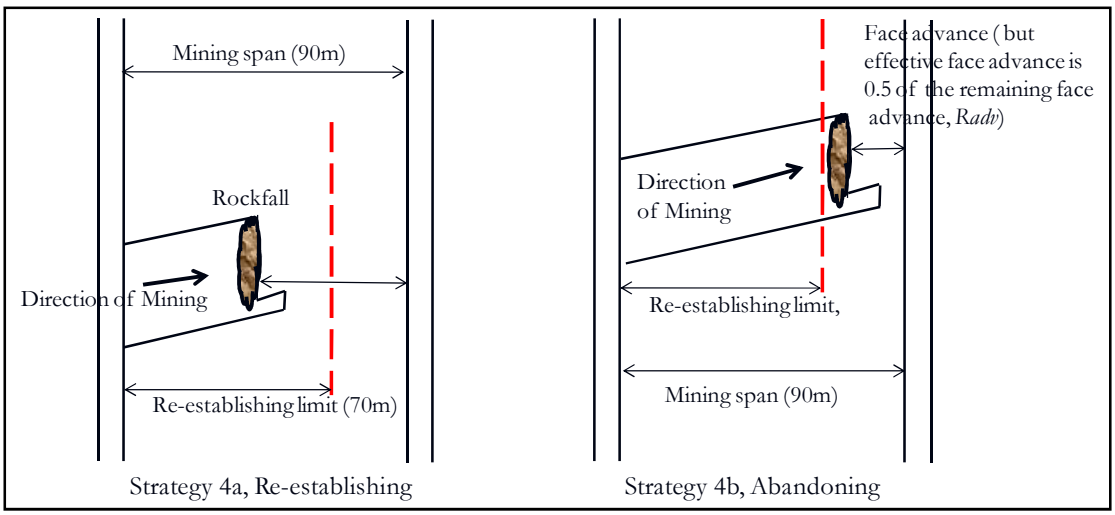


Figure 4-9: Re-establishing or abandoning criteria

For re-establishing purposes, Figure 4-8, the re-establishing end width on breast *(Rw_b)* is

$$Rw_b = Pf_l - (Rf_l + 1m)$$

The Rock Engineering department determines the re-establishing raise width on dip (Rw_d). The recommendation is based on rock engineering principles and experience.

As illustrated in Figure 4-9, the effective abandoned face advance is half of the remaining face advance. If, for example, the abandoning maximum limit is 20m, the average possible face advance is therefore 10m. This logic is applied in the determination of the area of reserve loss.

4.5.2.5. Summary of classes

Table 4-1 gives a summary of the resulting consequences for the different rockfall sizes.

Table 4-1: **Summary of strategies**

	Class 1	Class 2	Class 3	Class 4a	Class 4b
Cleanup and dilution Loss	Yes	Yes	No	No	No
Re-supporting Cost	Yes	Yes	No	No	No
Sweepings Revenue Loss	No	No	Yes	Yes	Yes
Production Loss					
Cleanup and Re-supporting	Yes	Yes	No	No	No
Re-establishing	No	No	Yes	Yes	No
Re-deployment	No	No	No	Yes	Yes
Reserve Loss	No	No	Yes	Yes	Yes

`

4.5.3 Evaluation of cost of stope damage

The relationship between stope rockfall remediation and the rockfall size was discussed in the previous section. This relationship was then used to formulate remediation strategies. Table 4-1 summarised the strategies to handle stope damage, and possible consequences are indicated for each strategy. This section will describe the evaluation of the costs of the consequences.

The overall expected cost of stope damage is a function of several cost factors. These include cleaning up, re-supporting, sweepings loss, loss of reserves and the production loss due to redeployment, re-establishing or abandoning a working place. The general equation that expresses this function is given as:

Total Expected Cost=Rockfall Annual Frequency of Occurrence x Cost of Consequence

The annual frequency of occurrence of rockfalls has been discussed in Section 4.4. The cost of the consequences of stope damage can be determined from the formula:

Cost of Consequence = [Cleanup and Dilution Cost + Re-supporting + Sweepings Revenue Loss+ Production Loss+ Reserve Loss (Given on NPV basis)]

The cost of consequences will be discussed separately for classes 1 and 2, 3 and 4.

4.5.3.1. Strategies 1 and 2

These classes of rockfalls consist of small rocks and as shown, they result in dilution, re-supporting and production downtime.

4.5.3.1.1. Cleanup and dilution cost

This loss applies to classes1 and 2. Small rockfalls are cleaned up with the ore, hoisted and taken through to the processing plant. The rockfall is assumed to be waste rock, hence it results in dilution. The transportation and processing of waste material is the dilution cost. It results in less mineral value being output from the processing plant. The actual cleaning up cost is largely attributed to the cost of labour. However, no additional labour is required to clean up falls in this category as the panel crew can handle them. The labour cost is therefore viewed as a fixed cost and hence is not considered in this methodology.

It can be argued that if there were no rockfalls, there would be less labour required for the same production output. However, it is important to note that the labour requirement on a mine is neither directly dependent nor determined by the number of rockfalls. Labour cost is a fixed cost that is dependent on the labour requirements for the overall mining cycle.

`

Total dilution cost is determined as:

Total Dilution Cost (R)=Dilution cost (R/t) x FOG Size (t)

Where dilution cost is determined as:

Dilution Cost (R/t)=(Vertical Cost+ Horizontal Cost+ Processing Cost)/ Tonnes mined and processed.

Classes 3 and 4cater for large rockfalls where no cleanup is done hence there is no dilution cost incurred. The inputs are obtained from the mines accounts department or alternatively, annual reports can be used.

4.5.3.1.2. Re-supporting cost

This cost does not apply to class 1 because the rockfall is very small (less than 1m). It is assumed that when a class 2 rockfall occurs, the area has to be re-supported in order to make the panel safe. This can be done by installing the same support pattern, increasing the support density or using a different support system. The costs of storage, transportation and installing are however not included in the re-supporting cost unless a new support system which required new infrastructure and personnel was being applied. As discussed in the cleanup and dilution cost section, the labour in transporting and installing support is also considered a fixed cost. As such, the unit cost for re-supporting is considered to be only the purchase cost of the unit.

The total Re-supporting Cost equation is given as:

Total Re-supporting Cost = Area to be resupported x Support Density x Cost per installed unit

Where;

Area to be re-supported = Area of FOG x Re-supporting Area Factor

The area to be re-supported is assumed, after talking to experienced mining personnel, to be the area of the rockfall plus a factor to allow for safe areal coverage. Discussions with personnel from different mines suggested a typical example to be 50%.The support density and types at re-supporting either can remain the same or are changed at the Rock Engineer's discretion. The cost per unit or per square metre is determined in the risk mitigation cost section.

`

4.5.3.1.3. Cleaning-up and re-supporting production loss

The time spent cleaning and re-supporting will vary depending on the size of the fall of ground and the type and availability of the recommended support units. Such time loss results in the production loss. Therefore, Cleaning-up and Re-supporting Production Loss is determined as:

Production Loss = Total days required to clean and Re-support x

Profit Loss(R/m^2) x Panel tonnage/day

In the equation above, Profit Loss *(R/m^2)* refers to the difference between expected profit and actual profit for the lost production. That is the difference between revenue and transport, processing and cost of consumables (explosives, support etc, which would not be expended if no production takes place).

Profit Loss=Revenue-(Cost of Consumables + Transport + Processing Cost)

Clean up time is the time required to cleanup a FOG of a given length along the face. Typical times are presented in Table 4-2, which were obtained from interviewing experienced mining personnel. On average, it takes 0.1 days to clean 1m (length on face) of rockfall.

Table 4-2: FOG length vs. cleanup size relationship

FOG Length on face	2m	5m	10m	15m	20m	30m
Cleanup and Re-support (days)	0	0.5	1	3 - 5	-	-

4.5.3.2. Strategies 3 and 4

The categories cater for the larger rockfalls as discussed in Section for 4.5.2. The consequences associated with these categories are sweepings loss, reserve losses, production losses from re-establishing and redeployment.

4.5.3.2.1. Sweepings loss

Rockfalls classified under classes 3 and 4 are large, hence the affected area is usually inaccessible or unsafe to sweep. As a result the sweepings are not claimable. Sweepings are broken ore carrying high mineral grade, therefore are a source of revenue. Profit could have been made from the sweepings; this loss is therefore termed the sweepings loss and is determined as follows:

Total Sweepings Loss = Sweepings loss per m^2 x Expected Sweepings Area

`

Where,

Sweepings Loss is determined from:

Sweepings Loss= Unadjusted Sweepings Loss x Height Factor x Density Factor

Where,

Sweepings Loss/m^2 = (Revenue - Processing Cost)/ Area mined

Height Factor = sweepings height/stoping height

Density Factor = Sweepings Loose Density/In situ Rock Density

Expected Sweepings Area is determined by the following:

Expected Sweepings Area = FOG Area (m^2)+ (FOG Area (m^2)x Sweepings Area Factor)

Sweepings area factor is a percentage area relative to the FOG size where sweepings are unrecoverable. Discussions with mine personnel resulted in an approximate value of 20% being selected for this factor.

4.5.3.2.2. Flexibility

Flexibility is the measure of availability of spare stopes for each panel crew and is expressed as a percentage. It is based on the number of spare panels that each crew has per given section. Rock Engineers on most mines suggested a value of 80%.

4.5.3.2.3. Production loss

This is the loss due to less or no productivity over a period. For class 3 and 4 rockfalls, this can be because of the following:

- Production loss due to re-establishing,
- Production loss while crews are redeployed,

The production loss discussed in this section is only that loss resulting from the remediation processes listed above. It is appreciated that when a rockfall occurs, the panel will in some instances be required to lie unmined pending investigations. Such periods are included in the overall production loss. Temporary mine closure by the DMR under Section 54 of the Mine Health and Safety Act of 1996, will be discussed under personnel injuries section.

`

The total production loss is determined as shown in the equation below:

Total Production Loss= (Redeployment Production Loss x Flexibility)

+

(Re-establishing Production Loss x (100% - Flexibility))

4.5.3.2.4. Redeployment production loss

This class refers to the case in which a panel crew has to be re-deployed after a collapse or when it has to wait for in-loco inspections. Such time is production downtime. This down time is assessed in terms of the productivity (m^2) per man per day. The total cost per m^2 is the actual cost that the mine incurs as productivity loss (per man) per given downtime.

Research has shown that crew sizes are variable. There is no readily available information on crew complements that have reported for duty on any given day. As a result, an alternative method of determining the loss of productivity was investigated.

The alternative approach takes into consideration the number of crew(s) affected, total number of days in redeployment and the total tonnage required from the crew(s) in those given number of days.

Redeployment production loss is expressed as:

Redeployment Production Loss = Total days to redeployment x Production Loss X

Panel t/day

Where:

Total days to redeploy = No. of crews affected by FOG x days to relocate

`

4.5.3.2.5. Re-establishing production loss

Jager and Ryder (1999) give a design rationale and example of current industry practice of re-establishing of panels. This is done by either open raising within the confines of the existing excavation or by re-raising beyond the collapsed panel. Typically, the deep level mines use well supported wide ends, whereas shallow to intermediate mining for example in the Bushveld Complex use narrow end re-raises. Hence the size of the re-establishment has an effect on the actual production loss realised.

Some operations have a re-development team; this only stands in for re-establishments hence allowing the panel crew to move to (redeployment) spare/contingency panels when a collapse has occurred. In operations without such teams, re-establishment is done by the panel crew themselves.

Important factors in determining this loss are:

- Time taken to re-establish,
- Productivity per day during the re-establishing,
- The target productivity per day of the panel, this is determined at the mine planning stage/meetings. Normal conditions productivity is predefined and is the panel call,
- The expected revenue and
- Flexibility of the operation

The time taken can be determined in two ways; directly from experienced personnel or from calculations based on mining geometry. Both methods will be described below.

4.5.3.2.5.1. Direct input from experienced personnel.

Mine personnel were interviewed to obtain the typical time taken to carry out re-establishment or redeployment activities. Table 4-3shows the extent of the damage (classified as 1 to 5), typical remedial action and the estimated time required to carry out the action plan at Mponeng mine.

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Table 4-3: Estimated re-establishing times and damage severity

Extent of damage	Remedial action	Material allocation	Est. time required	Comment
Minor face burst/shakedown/No damage to support (<5m), 1	Clean, re-support	As per standards	Less than 1 day	-
Minor face burst/shakedown/No damage to support (5-10m), 2	Clean, re-support	As per standards	Approx 1 day	-
Significant face burst/ FOG/seismic closure/damage to support/severe damage to excavation (30% - 50% of panel), 3	Clean and re-support OR re-establish	Increase support density	3 days OR 2-3 months	Depending on ground conditions, panel can be abandoned. Crew redeployed.
Significant face burst/ FOG/seismic closure/damage to support/severe damage to excavation (50% - 66% of panel), 4	Clean and re-support OR re-establish	Increase support density	7-10 days OR 2-3 months	Depending on ground conditions, panel can be abandoned. Crew redeployed.
Panel closed/collapsed/back break affecting whole panel, 5	Re-establish OR abandon panel	Increase support density	2-3 months OR 5days to set-up and start new panel	Depending on ground conditions, panel can be abandoned. Crew redeployed.

4.5.3.2.5.2. From calculations based on mining geometries and logistics

In this approach, the re-establishing times are determined through a four-step calculation as follows:

Step 1: Determine the total tonnage mined while north siding, for example re-establishing. This is the sum of the breasting and up-dipping tonnages:

$$Tonnage\ (up\ dipping\ or\ breasting) = \sum (width \times length \times height \times density)$$

Step 2: Determine of the total number of days required to breast and up-dip. This is determined as:

$$Total\ Required\ Days = total\ tonnage\ /\ (tonnes/blast \times daily\ blasting\ efficiency))$$

`

Where;

Blasting efficiency = No. of blasts per month days

Tonnes/blast = Stope height x Stope length x Advance/blast x Density

Step 3: From the determined total tonnage and total days required, the re-establishing period productivity, expressed as tonnes per day is then determined as follows:

Re-establishing production (t/day) = (Total tonnage/ Total Required Days)

Step 4: The production loss is then determined. For an operation without a redevelopment team, this shortfall is expressed by the following equation:

Re-establishing production Loss = Required Days x (Expected panel production

– Re-establishing production) x Production Profit x (100%-flexibility)

Where redevelopment teams are employed, the production loss is determined as follows;

Re-establishing production Loss = Redeployment Days x (Expected panel production–
Re-establishing production)x Production Profit x flexibility

4.5.3.2.6. Reserve loss

As shown in Figure 4-4reserve losses occur because of re-establishing or abandoning of the panel. The loss of reserves results in the shortening of the life of the mine. This has an overall impact on the net present value (NPV) of the operation. This effect is negative and hence is a net present cost, which has the effect of reducing the life of the mine. Strategies 3 and 4 both result in loss of reserve as shown in Section 4.5.

*Revenue Loss (Reserve) = Pillar size * Profit Loss/m^2 * Rock Density * Stope Height*

`

4.5.4 Input parameters

The required inputs to the methodology proposed in Section 4.5.3 were all summarised into the following:

i. General
   - Rock density
   - Average mineral grade
   - Swell factor

ii. Production Call
   - Mine call/crew
   - Average mine call/month
   - m^2 achieved per month
   - Tonnage achieved per month
   - Stoping height
   - Panel length
   - Daily advance

iii. Re-supporting material costs (cost/unit) both primary and secondary support

iv. Financial information
   - Revenue
   - Overall operating costs
   - Variable and fixed operating costs
   - Gross profit margin
   - General mining parameters

v. Geometry
   - Face length(m) is the full panel length
   - Advance per blast (m) is the average distance the panel advances after each blast
   - Stoping height (m) is the distance between the hanging and footwall at the face
   - Mining span (m), the maximum distance of face advance in the life of the panel, typically measured from the raise line to the stopping limit
   - Abandoning limit (m) refers to the distance from the stopping limit at which the panel can be abandoned over re-establishing it

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- Re-supporting area factor this is the percentage area around a rockfall that is re-supported after the fall has been cleaned

vi. Rock mass

- Rock density (t/m^3)
- Average mineral grade
- Swell factor
- Fallout height (m)

vii. Productivity

- m^2 achieved per month
- Tonnage achieved per month
- Working days per month
- Blasts per month

4.6 Personnel injuries and fatalities losses

The South African mining industry measures safety performance in relation to the trend in and severity of personal injury rates. The costs resulting from personal injuries as discussed in earlier chapters are not considered. Table 3-2 showed that some of the costs are quantifiable whereas others are difficult to quantify. Data on such issues is usually sensitive to divulge. Only the frequency rates on fatalities, reportable lost time injuries and shifts lost are available for the public domain or external research. Data on actual cases and the costs involved therein are usually difficult to obtain. However, the DMR maintains a database on all cases reported, SAMRASS database, by the mines as per the Mine Health and Safety Act of 1996. Some of the data used in this research came from that database.

4.6.1 Injuries event tree

When a rockfall occurs while people are working in the stope face area, there is likelihood that a person is injured. The likelihood is dependent on the effectiveness of monitoring, evacuation, personnel exposure and spatial coincidence. Stacey and Gumede (2007) suggested that 0.02m^3 rock sizes are the lower bounds for sizes that can cause serious injuries or fatality. The list of potential consequences resulting from an injury or multiple injuries was shown in Table 3-2. Figure 4-10 is an event tree illustrating the factors involved in the determination costs of consequences resulting from personnel damage. Injuries have different classes and hence different financial impacts. The potential injuries are classified in four categories namely:

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- Dressing cases are injuries in which the injured will not lose a day or more from work. Such cases are usually first aid cases.
- Reportable injuries: these are lost time injuries where the injured will lose between one and fourteen days off work.
- Serious injuries: these are severe injuries which can result in partial or permanent disability. The injured misses fourteen or more days from work.
- Fatal injuries result in the death of a person.

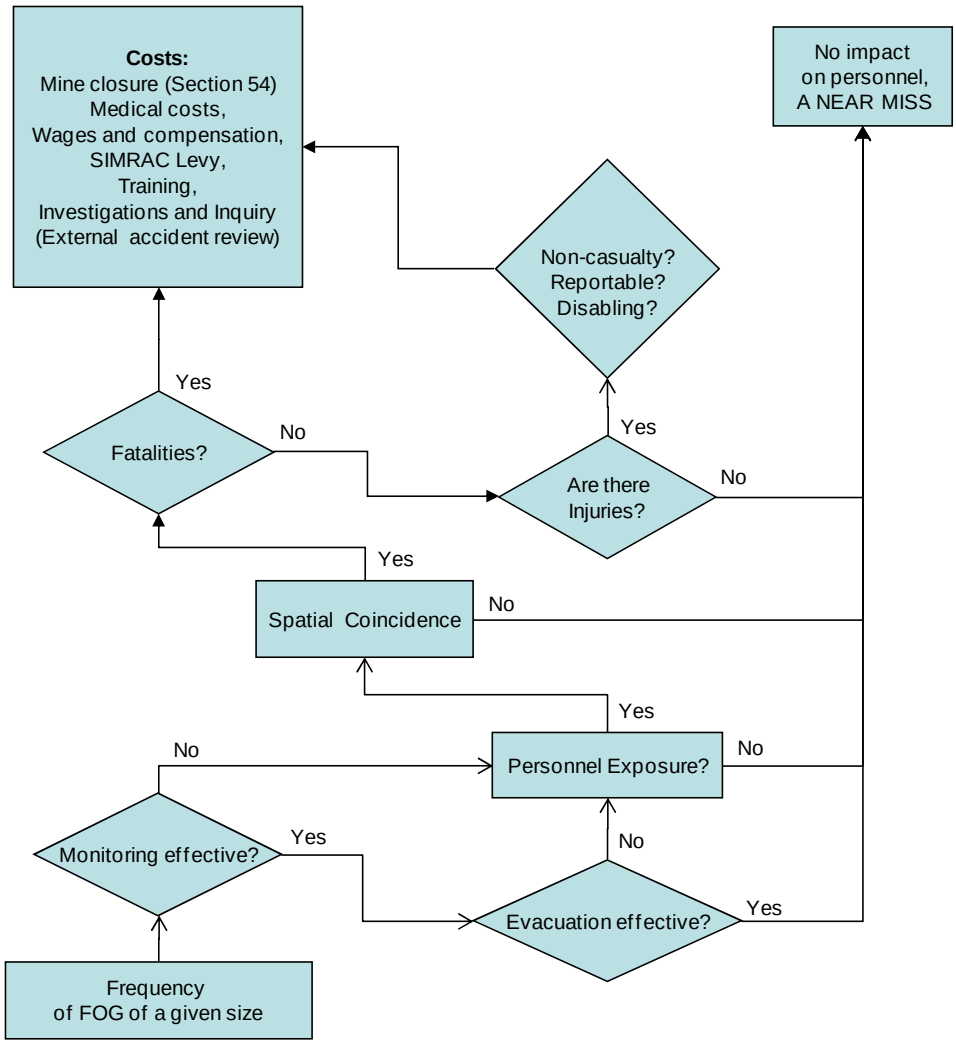


Figure 4-10: Personnel injury and fatality event tree

The components making up the event tree are discussed in the subsections that follow.

`

4.6.2 Monitoring and evacuation

Monitoring and evacuation is a concept that is practised widely in open pit mining, (Terbrugge et al, 2006; Contreras et al 2006). This could be a very effective way of reducing personnel injury or equipment damage. With the presence of monitoring systems, an effective evacuation procedure can be implemented (Joughin, 2008). Hence, in this methodology, where no reliable monitoring and evacuation procedures are in place, the multiplier is 1. Where they are present and measurable/estimated, the multiplier is given as a fraction (percentage). Monitoring and evacuation are however difficult to apply in the underground environments.

4.6.3 Exposure/Temporal coincidence

Personnel and equipment working in the potentially hazardous area have a degree of exposure. This is dependent on the time that they are exposed to the rockfall hazard. The purpose of exposure analysis is to estimate the probability of coincidence in timing of mine personnel with the rockfall. The methodology adopted is that presented by Roberds (2005) for landslides and modified by Joughin (2008).

Personnel exposure is determined as:

Exposure = (hours per day/24 hours) x ((days per month x 12 months – leave days)/365 days)

4.6.4 Spatial coincidence

The purpose of this analysis is to estimate the probability of coincidence in space of people with the rockfall. That is, the rockfall must occur in the same area that the people are present at the time of occurrence. Where one or more people are working in the area, the probabilities (of one or more, two or more etc) are multiplied. Joughin (2008) explained this concept well and his approach is therefore adopted.

The probability of one or more rockfalls occurring in a given area in one year (f_r) and the probability of one or more people occupying a given area at one time (f_p) can be determined using a binomial distribution:

$$f(x) = \sum \binom{n}{x} p^x (1-p)^{n-x}$$

Where x = number occurrences, p = probability of occurrence, n = number of trials

Spatial coincidence (Cp) for personnel is then calculated:

$$Cp = \left(1 - fr(0)\right) \times \left(\frac{Ax}{As}\right) \times \left(fp(x_p)\right)$$

`

For people $f_p(x_p)$, x_p= number of persons in a given area. For rockfalls $f_p(x_r)$, x_r =number of rockfall occurrences in a given area. The probability of one or more rockfalls in one year occurring is 1-f (0), corrected by (Ax/As).

4.6.5 Evaluation of cost of injuries

In Chapter 3 approaches to evaluating different consequences resulting from personnel injury and fatalities were discussed. This section discusses the methodology of evaluating the injury consequences.

4.6.5.1. Temporary mine/section closure (Section 54).

As discussed in Chapter 3, the DMR can instruct closure of the whole or a section of the mine after a fatal accident. This closure can result in loss of revenue because of the loss of production during the closure period. However, during the period of closure variable costs such as mining consumables, transport, and processing costs are not spent. Therefore, the total loss resulting from production stoppage is the revenue loss less the costs not incurred. To determine this loss the following formula is applied:

Production Loss=Daily Production Rate x Production Profit loss (R/t) x No. of Days closed

The daily production rate is determined from the mine call for the shutdown section. The production profit lost has been explained in previous sections and the Inspectorate determines the number of days of closure. Interviews with mine personnel gave an average Section 54 period for a fatality to be 5 days. The interviews also revealed that Section 54 is imposed for reasons other than injuries, for example hazardous working conditions or substandard practices. Such cases are not considered in the proposed methodology.

4.6.5.2. Investigation

The workers' time and the resources spent on the investigation are a cost to the mine. The time the workers spend in the investigation is unproductive time. Marx (1996) determined the cost of unproductive time from the hourly wage rate of personnel involved in the investigation and the time spent by the personnel on the investigation. However in accounting, an employee is cost to company is considered a fixed cost. Therefore, the wages paid to the employee for the total time spent in the investigation is a fixed cost. Adding this cost to the cost of labour in the company's books will be double accounting. It is therefore not considered in this methodology.

`

It is acknowledged that the accident results in additional management time being lost. This loss is however difficult to quantify as management is not paid on an hourly rate. It can also be argued that dealing with such situations is part of management's responsibility and as such, it cannot be viewed as an additional cost.

After an accident occurs, the mine may approach external consultants or reviewers to conduct an independent investigation, review or audit the mine's rock engineering systems. The cost of the work done by such consultants is an additional cost resulting from this accident. This is therefore considered under the cost of investigations in the proposed methodology.

The government caters for the Inspectorate's expenses, but it is the duty of the mine manager to facilitate the resources for the investigation. The cost of such facilities is difficult to quantify hence is not considered in this methodology.

4.6.5.3. SIMRAC levies

As discussed in Chapter 3, levies are divided into the Safety and Health Risk Levy. However, because FOG accidents result in an increase in the safety component of the levy, the model will determine only the cost resulting from the Safety Risk Levy. This is determined as follows:

Safety Risk Cost=Days lost per injury classification x Daily Safety Risk Levy

The days lost per injury classification are determined from the SAMRASS Codebook. For dressing cases, no days are lost and for fatalities, the codebook stipulates 6000days. The daily safety risk levy is determined by SIMRAC.

4.6.5.4. Medical costs

Information on injury costs is available on invoices from mine hospitals. The invoice includes hospital treatment (all services) and where applicable the outpatient costs. The model uses total costs from invoices that are then apportioned in relation to the frequency of the injury as per its classification.

4.6.5.5. Re-training

The approach suggested by Marx (1996) has been inflation adjusted to 2010 using CPI figures from www.statssa.co.za. These figures were multiplied by the ratio of the number of fatalities (and disabling) to the sum of fatalities and all injuries. The rationale for applying his method was that Marx (1996) considers the different levels of skills training required for all injuries. The normalising according to the fatal accident ratio was based on the reason that a fatal (and disabling) accident would require total re-training.

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4.6.5.6. Legal costs

Suing of the employer and the consequential legal cases can have significant costs and image implications for the mining company. Such implications are difficult to quantify, and hence will not be quantified in this work.

4.7 Damage to equipment and machinery

In any mining operation, plant and equipment are used. ` The mining method and ore body geometry amongst other factors determine whether conventional, trackless, or a hybrid method will be used. However, in any of these scenarios, rockfalls that potentially damage plant and equipment do occur and the consequences thereof can be minor or severe. As shown in Table 3-2, the consequences include production loss, redeployment, replacement or repairing costs. Figure illustrates the factors that determine the consequences of the rockfall on equipment.

Equipment is always exposed to potential FOG's. If there are effective monitoring and evacuation systems in place then the FOG may not affect the equipment. Where the monitoring system is not as effective and the equipment is exposed with some degree of spatial coincidence (Figure 4-11) then the FOG will hit the equipment. Spatial coincidence is the probability that the rockfall occurs at the exact time and place where the unit is.

A damaged unit is either repaired or replaced. In cases where a mobile production unit is damaged and a spare unit is available, the spare unit will replace the damaged one in production. Meanwhile, a decision has to be reached whether to replace or repair the damaged unit. The consequences of equipment damage are production loss, redeployment of machinery costs, replacement or repair costs (Table 3-2).

4.7.1 Proposed methodology to quantify costs of damage

The hypothetical equipment damage model for mobile equipment is illustrated in Figure 4-12. The model is for an LHD. The first section of the model, named 'FOG' requires as input the description of the FOG, its dimensions, mass and annual frequency of occurrence.

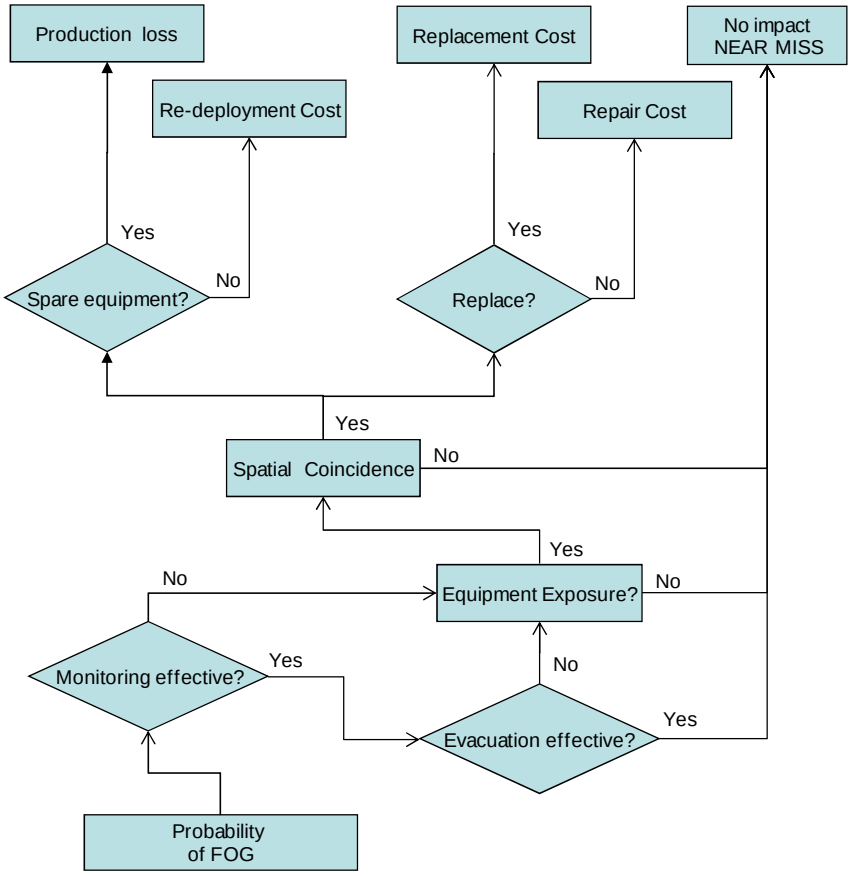


Figure 4-11: Equipment damage model event tree

Equipment - LHD

Height of fallout (m)	0.5
Rock density (t/m^3)	2.7
Repair/replace criteria	80%
Profit margin (ZAR/t mined)	1000
Replacement Cost (ZAR)	450000
Operating Cost (ZAR/t)	20
Max Machine productivity (t/day)	600
Availability	90%
Utilisation	80%

Total Expected Cost Table

FOG			Repair/Replace Costs			Flexibility	Production loss			Redeployment Cost		
Equivalent Dimension(m)	Mass(t)	Annual Probability	Typical Repair cost (ZAR)	Cost per incident (ZAR)	Expected Cost (ZAR)	Spare equipment available?	Equipment Downtime (days)	Loss per incident (ZAR)	Expected loss (ZAR)	Cost per incident (ZAR)	Expected Cost(ZAR)	Total Expected Loss (ZAR)
1	1.35	2.62E-04	20,000	20,000	5	90%	0.5	21,168	6	10,000	2	**13**
2	5.40	8.62E-04	50,000	50,000	43	90%	2	84,672	73	10,000	8	**124**
5	33.75	4.85E-03	100,000	100,000	485	50%	5	1,058,400	5,128	10,000	24	**5,637**
7	66.15	1.07E-04	200,000	200,000	21	20%	10	3,386,880	362	10,000	0	**384**
10	135.00	1.91E-05	500,000	450,000	9	10%	30	11,430,720	218	10,000	0	**227**
20	540.00	3.57E-06	500,000	450,000	2	10%	30	11,430,720	41	10,000	0	**42**

Figure 4-12: Equipment damage model

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Repairing and/or replacement costs are then determined. Repairing and/or replacement criteria are based on the management policy and/or experience, for example where the repairing cost is over 60% of the replacement cost, then replace the equipment (Impala Mines, 12# Management Report). Cost per incident is therefore the cost of the chosen remedial action. The expected cost is the product of the annual frequency of occurrence of a rockfall of the given size and the cost per incident.

Production loss is a result of a shortfall in production due to the downtime of the unit. This loss is determined on the availability of a spare unit (flexibility). Where an LHD, truck, scraper or winch is damaged, the resulting total loss in production is considered as follows:

*Loss per incident = ((100%-Flexibility) x downtime x productivity x availability x utilisation x profit margin) - ((100%-Flexibility) x downtime x productivity x availability x utilisation x operating cost)*Expected panel daily production*

Where there is flexibility, a redeployment cost is incurred. Redeployment cost is the total cost of moving the replacement unit from where it will be, such as on surface, or another section/workshop, to the production line. This cost for mobile equipment can vary from a negligible to a large amount. The redeployment loss is therefore also a function of the flexibility of the system.

Expected redeployment cost = Annual probability of occurrence x Flexibility x Cost per Incident.

The availability of spare equipment is given as a percentage. For example, if there are no spare units underground, the availability is 0%; if there is a spare unit for every say 3units, then the availability is 70%. Alternatively, the average availability of units can be applied.

The total expected cost of equipment damage is expressed as:

$$\text{Expected Cost} = \sum (\text{Cost/ Incident} \times \text{Annual frequency})$$

Data required for all the above calculations included rockfall dimensions, equipment repair and replacement history and costs, equipment downtime history, capital and operating costs, machine productivity, availability, utilisation and flexibility. However, due to the lack of availability of sufficient data on the inputs, this proposed methodology was developed no further than this hypothetical stage in this research.

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4.8 Expected cost of risk mitigation

Risk mitigation strategies include the cost of support material, installing and monitoring strategies employed on the mine to maintain stability and safety standards. It is anticipated that these costs will vary depending on the ground control district and support standards. Hence, the cost of risk mitigation strategies to be determined will be for the location of the FOG.

In the event of a FOG and the mitigation strategy requiring a change in the drilling and blasting system or the loading and hauling system, the related costs then have to be determined. Figure 4-13 below highlights the high-level cost centres. Where only support changes are employed, not all the other activity centres are included in the cost analysis.

The quantification of the cost of materials for support is conducted on all mines, Figure 4-14 shows the aspects generally considered in the calculations. Transportation logistics and installation costs are in most operations not added to the overall support unit cost. These however, are essential in determining the actual cost of implementing a support strategy as the cost of managing logistics can be significant.

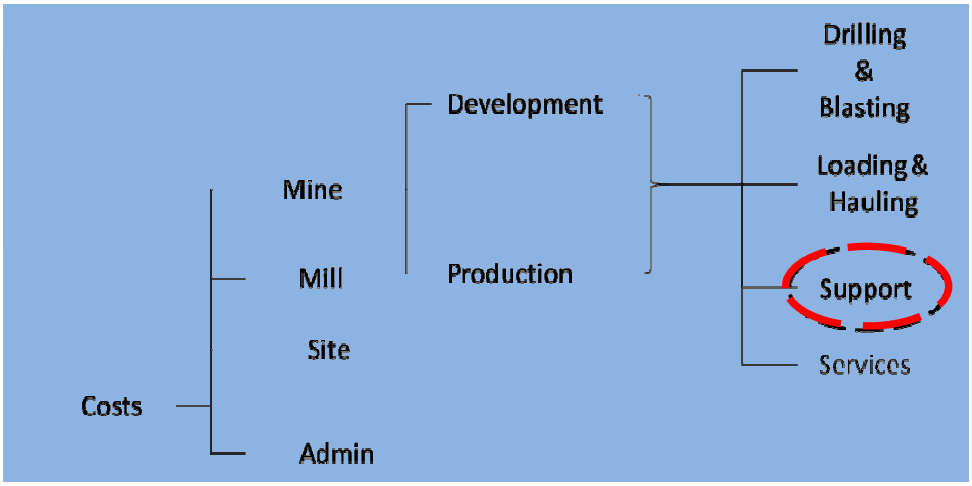


Figure 4-13: Mining cost components

In the proposed methodology, the overall cost of support (risk mitigation) is broken down into capital and operating costs. This enables the inclusion of the cost of implementing a new support regime, which requires capital. The operating cost is the monthly or annual cost of materials and consumables required in the supporting process. The Net Present Cost (NPC) is used to determine net cost over the life of the operation or the panel.

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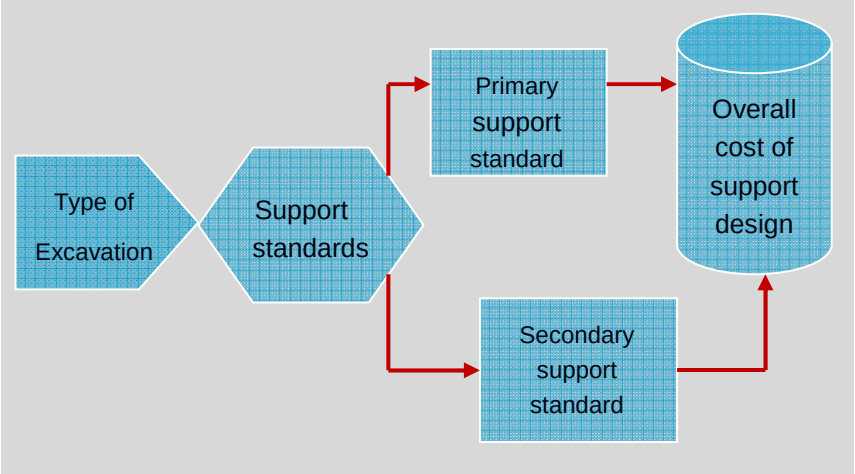


Figure 4-14: Support costs

4.9 Summary

In this chapter, a methodology to quantify the consequences of rockfalls has been proposed. First, the risk-cost approach to design was described. Then a rigorous methodology to determine the financial implications of rockfall consequences was described. This is split into three parts, stope damage losses, personnel injury and equipment damage consequences. While the stope damage and personnel injuries methodologies have been fully described, only the proposed logic for an equipment damage model is presented. This is due to insufficient information on equipment damage history and typical costs. In Chapter 5, the stope damage methodology will be applied on real case studies to test for its correctness and applicability in the mining industry's narrow tabular operations.

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5 Model application

The proposed methodology has been described in Chapter 4. In this chapter, the methodology is tested for practicality by applying it to mining case studies. Two case studies, Mponeng Mine conventional mining and Impala 12 Shaft are used. Operational, safety and financial data was obtained from the sites and input into the model.

5.1 Case study 1: Mponeng conventional

5.1.1 Background

Mponeng is a deep level gold mine, which exploits the Ventersdorp Contact Reef (VCR), with capital projects underway to access the Carbon Leader (CL) reef. The VCR is split into two sections, the VCR Lower and Upper, both mined using sequential grid mining. Breast mining is conducted on 30m long panels per level on each side of a raise and stopes are separated by 30m wide dip pillars, 180m apart. Strike gullies are used as access ways to the face, for cleaning, and material transport. These gullies are either developed concurrently with or slightly ahead of the panel face or protected with a siding or footwall lifted behind the panel face, depending on the raise line geometry (underhand or overhand). Crosscuts are situated 210m apart and haulages are between 100 and 150m above/below reef plane.

The objective of the case study is to determine the Expected Annual Cost associated with rockfalls. The rockfall data that was used is by no means a complete data set. As a result, the costs are likely to be underestimated.

5.1.2 Description of input parameters

The input parameters are categorised as; general mining parameters, support standards and costs, financial information, accident statistics and rockfall data.

5.1.2.1. General mining parameters:

A diagrammatic illustration is given in Figure 5-1.

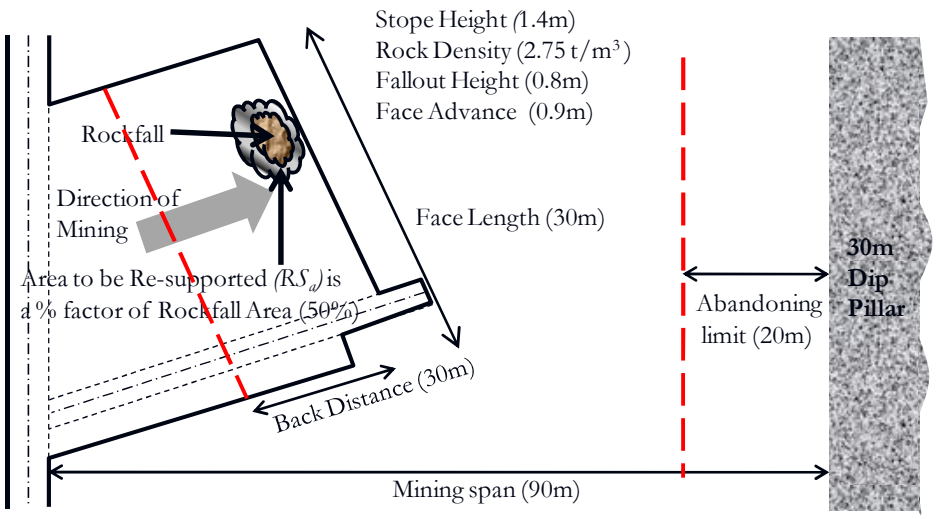


Figure 5-1: Mining parameters

The mining input parameters are categorised into:

- Geometric parameters; these define the stope geometry,
- Rock mass parameters; which define the densities and grades,
- Productivity parameters; these define the tonnes and area mined and the blasting to working days per month ratio

Table 5-1 presents a summary of the parameters. These were obtained from the mine standards and from discussions with the mining personnel.

Table 5-1: General mining parameters

Geometry	
Face Length (m)	30
Advance per blast (m)	1.0
Stoping height (m)	1.8
Mining Span (m)	90m
Abandoning limit (m)	20
Re-supporting factor (%)	50
Rock Mass	
Rock density (t/m^3)	2.76
Average mineral grade(g/t)	9
Bulk density (t/m^3)	1.83
Productivity	
m^2mined per month	29913
Tonnage mined per month	153620
Working days per month	22
Blasts per month	18

`

5.1.2.2. Support standard and cost

Mponeng has two Ground Control Districts (GCD), the VCR Lower and the VCR Upper. The typical stope support standard for the two districts is as given in Figure 5-2. Variations in the support types and combinations of elongates used in the two GCD's occur due to the yield and energy absorption requirements.

The unit costs of the individual support units (as at December 2008) are given in Table 5-2. These costs will be used in determining the cost of re-supporting in the model.

Table 5-2: Support costs

ID	Code	Name	Cost	Basis
1	E1	Wedge Props _1.0mx180mm (combo)	69.81	R/unit
2	E2	Wedge Props _1.2mx180mm (combo)	72.46	R/unit
3	E3	Wedge Props _1.4mx180mm (combo)	77.94	R/unit
4	E4	Wedge Props _1.6mx180mm (combo)	81.44	R/unit
5	E5	Wedge Props _1.8mx180mm (combo)	87.98	R/unit
6	E6	Madoda 180ST PSU	37.9	R/unit
7	E7	Stromaster 450_2.1m with headboard	250.48	R/unit
8	E8	Stromaster 450_2.4m with headboard	263.61	R/unit
9	E9	ST Combo Headboard_400mmx400mm	35.35	R/unit
10	E10	Rocprop_20t_1.0mx1.2m	473.97	R/unit
11	E11	Rocprop_20t_1.4mx1.8m	572.24	R/unit
12	E12	Rocprop_20t_2.0mx2.5m	678.58	R/unit
13	E13	Rocprop_20t_2.2mx3.2m	760.67	R/unit
14	E14	Rocprop_20t_2.3mx3.5m	817.3	R/unit
15	E15	Boulby Headboard_380mmx500mm	68.68	R/unit
16	E16	Split sets_42mmx1.5m	32.33	R/unit
17	E17	Split sets_42mmx2.1m	42.89	R/unit
18	E18	Dolly Pusher_25mm	281.69	R/unit
19	E19	Dolly Pusher_22mm	281.69	R/unit
20	E2	Backfill	36.54	R/m^2
21	E3	Backfill	42.07	R/t
22	P21	Timber Combosite Pack_110x9x9	1513.18	132 PER BUNDLE
23	P22	Timber Combosite Pack_150x9x9	1180.80	72 PER BUNDLE
24	P23	Hydrocell_1225x450mm	89.47	Each/100 per bundle

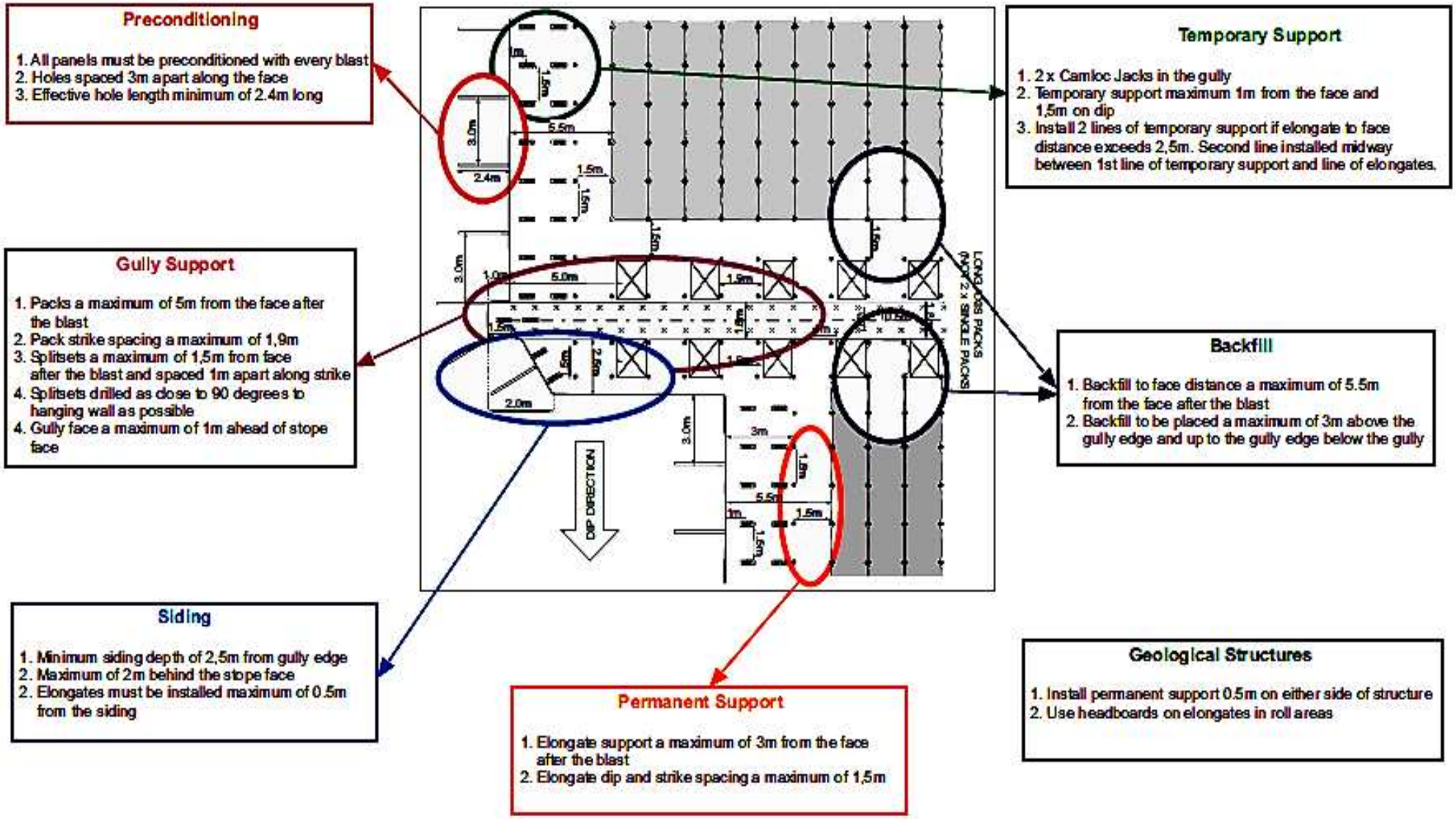


Figure 5-2: Stope support standard (Mponeng COP, 2008)

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5.1.2.3. Financial information

Financial data were obtained from the mine's finance department and discussions with various accounts, rock engineering and mining personnel. The key data obtained were revenue earned, mining consumables costs and the processing costs over the 2008 period. These data were used to determine dilution cost, production loss and the sweepings loss.

5.1.2.3.1. Dilution

Due to their size, small rockfalls are cleaned up with the ore resulting in dilution. The dilution cost is the cost of handling (horizontal and vertical transport) and processing of waste rock (the rockfall) per tonne. The cost of vertical and horizontal transport could not be extracted directly from the financial data provided. A dilution cost of R 56/t (Table 5-3) was therefore determined from dividing the processing cost by the tonnes milled per month as described in section 4.5.3.This approach is however a conservative one as it considers only the processing cost.

Table 5-3: Dilution cost

Description	Value/Quantity
Tonnage milled per month (t)	153,620
Processing Cost (average monthly) (R)	8,635,341
Dilution cost (R/t)	56

5.1.2.3.2. Production loss

Table 5-4 gives the input values for the determination of the production loss (data from the accounts department). The production loss is R 992/t. This was arrived at using the methodology explained in section 4.5.3, such as revenue less the cost of consumables and processing cost all divided by the tonnes per month.

Table 5-4: Determination of production loss

Description	Value
Tonnage milled per month (t)	153,620
Revenue (R)	180,305,000
Cost of Consumables (R)	19,288,000
Processing Cost (R)	8,635,341
Production Loss(t)	992

`

5.1.2.3.2. Sweepings loss

Table 5-5gives the finance data used in determining the sweepings loss. As described in section 4.5.3, the sweepings loss is the quotient of the difference between the expected revenue and processing cost and the area mined per month. This quotient is adjusted for density and height corrections. A sweepings loss value of R 545 per square metre mined was obtained.

Table 5-5: Determination of sweepings loss

Description	Value
Square metres mined per month	29,913
Expected Revenue (R)	180,305,000
Processing Cost (R)	8,635,341
Sweepings Loss/m^2	545

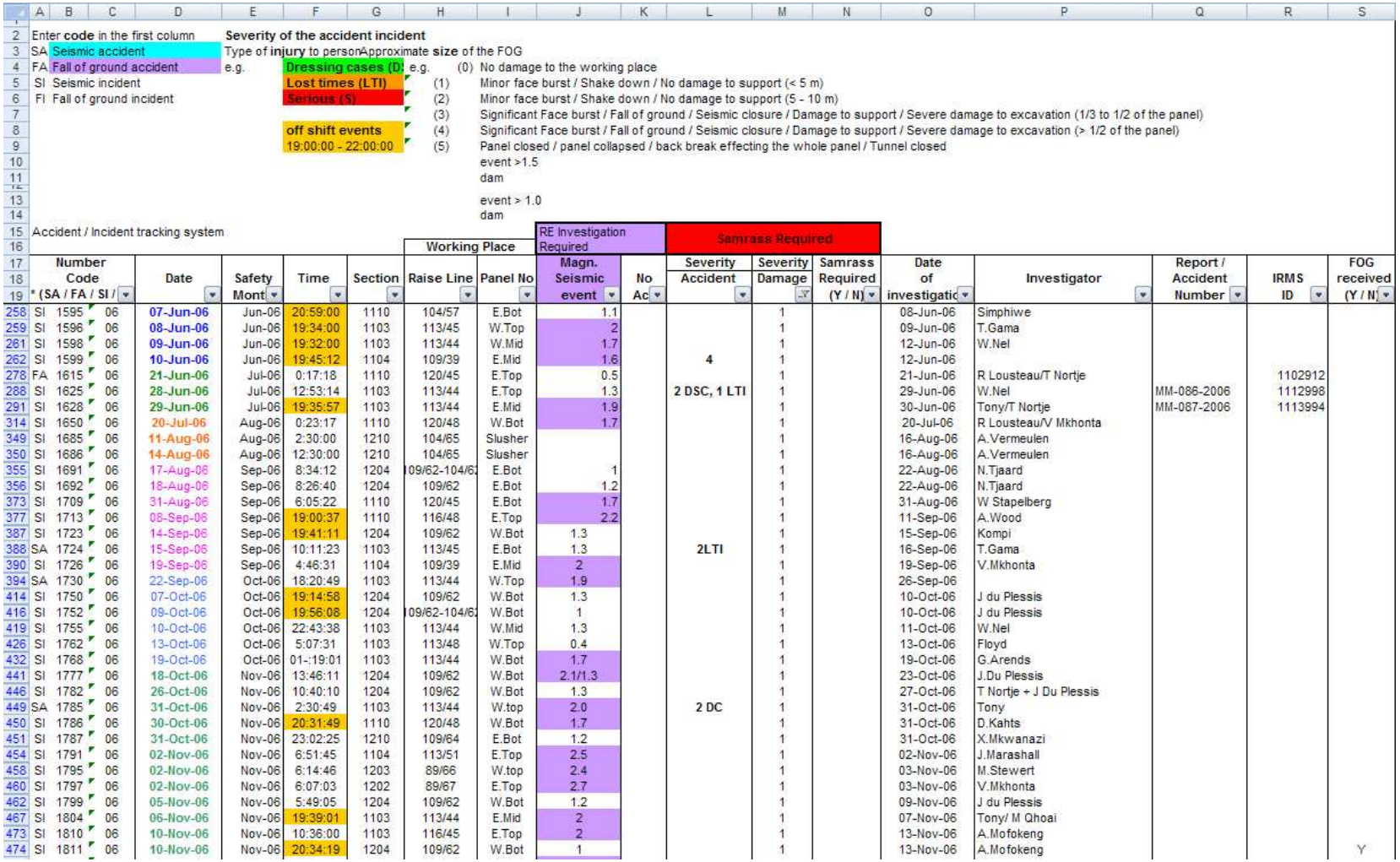
5.1.2.4. Rockfall data

The rock engineering department maintains a rockfall database in which records of rockfall accidents and incidents are kept. Figure 5-3shows an extract from the database. This database captures the section, date, time, location the magnitude of a seismic event, severity of injuries, damage and the report number. However, the rockfall dimensions are not captured. Rockfall data (dimensions and location of failure) was therefore obtained from the accident reports written by the department. The dimensions are shown in Table 5-6. This is however, an incomplete dataset as some of the reports were not obtained.

Table 5-6: Rockfall dimensions

Length (m)	Width (m)	Height (m)	Area	Dimension
39.0	1.5	1.0	58.5	8
42.0	2.0	1.0	84	9
9.0	2.2	1.0	19.8	4
2.0	2.0	1.2	4	2
35.0	5.5	1.5	192.5	14
1.0	0.4	0.15	0.4	1
9.0	3.0	0.2	27	5
13.0	4.5	0.5	58.5	8
4.0	2.0	0.7	8	3
37.0	28.0	1.0	1036	32
9.2	2.5	0.5	23	5
5.2	1.4	0.3	7.28	3
0.5	0.4	0.1	0.2	0
5.0	2.4	1.5	12	3
14.0	1.6	0.5	22.4	5
5.0	1.2	0.3	6	2
0.6	0.5	0.2	0.3	1
7.5	1.0	0.4	7.5	3
14.0	3.6	0.3	50.4	7
10.0	0.2	0.3	2	1
20.0	2.5	0.3	50	7
15.4	2.0	0.5	30.8	6
12.3	2.1	0.5	25.83	5
22.0	2.8	1.5	61.6	8
6.0	3.5	2.4	21	5
1.0	0.4	0.3	0.4	1
8.9	2.5	0.3	22.25	5
10.0	5.0	0.3	50	7

`



Enter **code** in the first column

SA Seismic accident
FA Fall of ground accident
SI Seismic incident
FI Fall of ground incident

Severity of the accident incident

Type of **injury** to person e.g.
Dressing cases (D
Lost times (LTI)
Serious (S)

off shift events
19:00:00 - 22:00:00

Approximate **size** of the FOG e.g.
(0) No damage to the working place
(1) Minor face burst / Shake down / No damage to support (< 5 m)
(2) Minor face burst / Shake down / No damage to support (5 - 10 m)
(3) Significant Face burst / Fall of ground / Seismic closure / Damage to support / Severe damage to excavation (1/3 to 1/2 of the panel)
(4) Significant Face burst / Fall of ground / Seismic closure / Damage to support / Severe damage to excavation (> 1/2 of the panel)
(5) Panel closed / panel collapsed / back break effecting the whole panel / Tunnel closed
event >1.5
dam
event > 1.0
dam

Accident / Incident tracking system

									Working Place		RE Investigation Required		Samrass Required							
	Number Code			Date	Safety	Time	Section	Raise Line	Panel No	Magn. Seismic	No	Severity	Severity	Samrass	Date of	Investigator	Report /	IRMS	FOG
	* (SA / FA / SI /				Mont					event	Ac	Accident	Damage	Required (Y / N)	investigatic		Accident Number	ID	received (Y / N)
258	SI	1595	06	07-Jun-06	Jun-06	20:59:00	1110	104/57	E.Bot	1.1			1		08-Jun-06	Simphiwe			
259	SI	1596	06	08-Jun-06	Jun-06	19:34:00	1103	113/45	W.Top	2			1		09-Jun-06	T.Gama			
261	SI	1598	06	09-Jun-06	Jun-06	19:32:00	1103	113/44	W.Mid	1.7			1		12-Jun-06	W.Nel			
262	SI	1599	06	10-Jun-06	Jun-06	19:45:12	1104	109/39	E.Mid	1.6		4	1		12-Jun-06				
278	FA	1615	06	21-Jun-06	Jul-06	0:17:18	1110	120/45	E.Top	0.5			1		21-Jun-06	R Lousteau/T Nortje		1102912	
288	SI	1625	06	28-Jun-06	Jul-06	12:53:14	1103	113/44	E.Top	1.3		2 DSC, 1 LTI	1		29-Jun-06	W.Nel	MM-086-2006	1112998	
291	SI	1628	06	29-Jun-06	Jul-06	19:35:57	1103	113/44	E.Mid	1.9			1		30-Jun-06	Tony/T Nortje	MM-087-2006	1113994	
314	SI	1650	06	20-Jul-06	Aug-06	0:23:17	1110	120/48	W.Bot	1.7			1		20-Jul-06	R Lousteau/V Mkhonta			
349	SI	1685	06	11-Aug-06	Aug-06	2:30:00	1210	104/65	Slusher				1		16-Aug-06	A.Vermeulen			
350	SI	1686	06	14-Aug-06	Aug-06	12:30:00	1210	104/65	Slusher				1		16-Aug-06	A.Vermeulen			
355	SI	1691	06	17-Aug-06	Sep-06	8:34:12	1204	109/62-104/6	E.Bot	1			1		22-Aug-06	N.Tjaard			
356	SI	1692	06	18-Aug-06	Sep-06	8:26:40	1204	109/62	E.Bot	1.2			1		22-Aug-06	N.Tjaard			
373	SI	1709	06	31-Aug-06	Sep-06	6:05:22	1110	120/45	E.Bot	1.7			1		31-Aug-06	W Stapelberg			
377	SI	1713	06	08-Sep-06	Sep-06	19:00:37	1110	116/48	E.Top	2.2			1		11-Sep-06	A.Wood			
387	SI	1723	06	14-Sep-06	Sep-06	19:41:11	1204	109/62	W.Bot	1.3			1		15-Sep-06	Kompi			
388	SA	1724	06	15-Sep-06	Sep-06	10:11:23	1103	113/45	E.Bot	1.3		2LTI	1		16-Sep-06	T.Gama			
390	SI	1726	06	19-Sep-06	Sep-06	4:46:31	1104	109/39	E.Mid	2			1		19-Sep-06	V.Mkhonta			
394	SA	1730	06	22-Sep-06	Oct-06	18:20:49	1103	113/44	W.Top	1.9			1		26-Sep-06				
414	SI	1750	06	07-Oct-06	Oct-06	19:14:58	1204	109/62	W.Bot	1.3			1		10-Oct-06	J du Plessis			
416	SI	1752	06	09-Oct-06	Oct-06	19:56:08	1204	109/62-104/6	W.Bot	1			1		10-Oct-06	J du Plessis			
419	SI	1755	06	10-Oct-06	Oct-06	22:43:38	1103	113/44	W.Mid	1.3			1		11-Oct-06	W.Nel			
426	SI	1762	06	13-Oct-06	Oct-06	5:07:31	1103	113/48	W.Top	0.4			1		13-Oct-06	Floyd			
432	SI	1768	06	19-Oct-06	Oct-06	01-:19:01	1103	113/44	W.Bot	1.7			1		19-Oct-06	G.Arends			
441	SI	1777	06	18-Oct-06	Nov-06	13:46:11	1204	109/62	W.Bot	2.1/1.3			1		23-Oct-06	J.Du Plessis			
446	SI	1782	06	26-Oct-06	Nov-06	10:40:10	1204	109/62	W.Bot	1.3			1		27-Oct-06	T Nortje + J Du Plessis			
449	SA	1785	06	31-Oct-06	Nov-06	2:30:49	1103	113/44	W.top	2.0		2 DC	1		31-Oct-06	Tony			
450	SI	1786	06	30-Oct-06	Nov-06	20:31:49	1110	120/48	W.Bot	1.7			1		31-Oct-06	D.Kahts			
451	SI	1787	06	31-Oct-06	Nov-06	23:02:25	1210	109/64	E.Bot	1.2			1		31-Oct-06	X.Mkwanazi			
454	SI	1791	06	02-Nov-06	Nov-06	6:51:45	1104	113/51	E.Top	2.5			1		02-Nov-06	J.Marashall			
458	SI	1795	06	02-Nov-06	Nov-06	6:14:46	1203	89/66	W.top	2.4			1		03-Nov-06	M.Stewert			
460	SI	1797	06	02-Nov-06	Nov-06	6:07:03	1202	89/67	E.Top	2.7			1		03-Nov-06	V.Mkhonta			
462	SI	1799	06	05-Nov-06	Nov-06	5:49:05	1204	109/62	W.Bot	1.2			1		09-Nov-06	J du Plessis			
467	SI	1804	06	06-Nov-06	Nov-06	19:39:01	1103	113/44	E.Mid	2			1		07-Nov-06	Tony/ M Qhoai			
473	SI	1810	06	10-Nov-06	Nov-06	10:36:00	1103	116/45	E.Top	2			1		13-Nov-06	A.Mofokeng			
474	SI	1811	06	10-Nov-06	Nov-06	20:34:19	1204	109/62	W.Bot	1			1		13-Nov-06	A.Mofokeng			Y

Figure 5-3: Rockfall database extract

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While no rockfall dimensions are captured in the database, the severity of damage captured was very useful in determining classes of rockfall consequence strategies. Table 5-7 shows the classification of damage severity and the description of each category. This is based on Table 4-3.

Table 5-7: Rockfall damage classification

Damage Severity	Description
0	Small FOG, no damage to the working place
1	FOG < 5m of panel face
2	FOG 5 - 10m panel face
3	Severe damage to excavation (1/3 to 1/2 of the panel)
4	Severe damage to excavation (> 1/2 of the panel)
5	Full Panel closed / panel collapsed / back break effecting the whole panel

This classification (Table 5-7) was then used in developing an event tree, Figure 5-4, which illustrates how the rockfalls are classified according to their location, dimension and consequences. While rockfalls can occur in the gully, face or back area, the gully and back area falls are not the focus of this research. Hence, only the face area falls are determined in this case study.

This event tree was constructed after:

- Interviewing experienced mining and rock engineering personnel,
- Studying past rockfall occurrences on the shaft,
- Studying the mine's rockfall database.

Table 5-8 gives a summary of the consequences application to each damage classification at Mponeng mine.

Table 5-8: Summary of strategies

Description	Class 1 & 2	Class 3	Class 4a	Class 4b
Cleanup and dilution Loss	Yes	No	No	No
Re-supporting Cost	Yes	No	No	No
Sweepings Revenue Loss	No	Yes	Yes	Yes
Production Loss				
Cleanup and Re-supporting	Yes	No	No	No
Re-establishing	No	Yes	Yes	No
Re-deployment	No	No	Yes	Yes
Reserve Loss	No	Yes	Yes	Yes

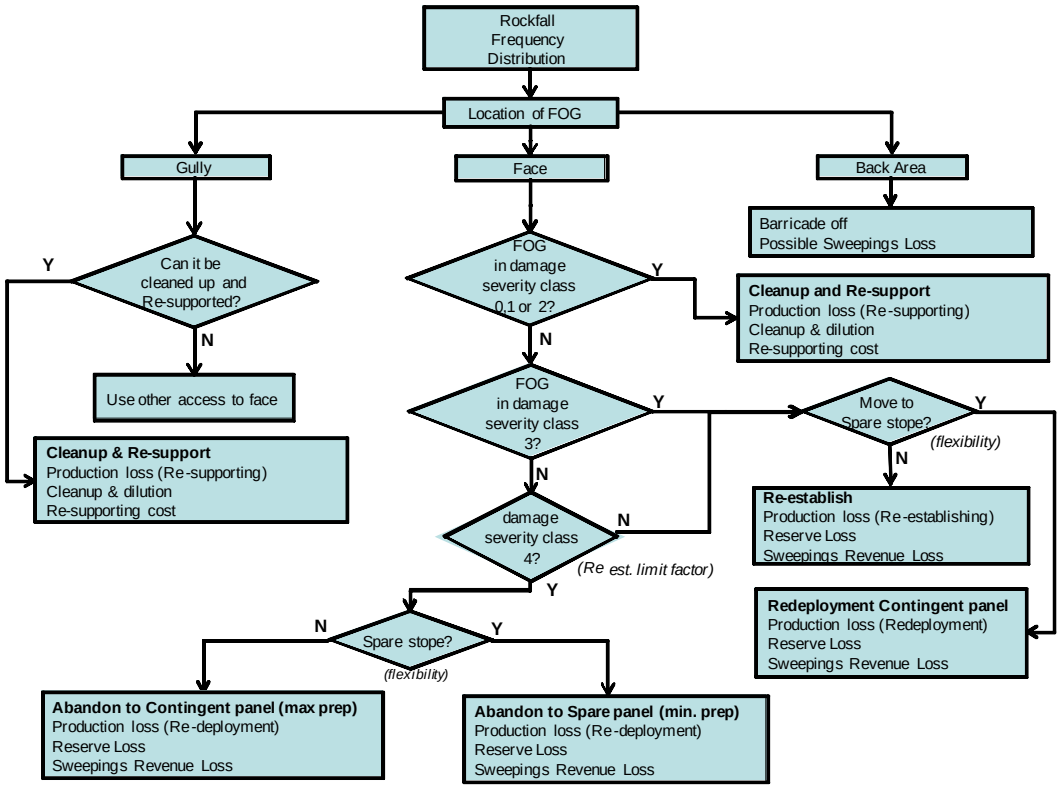


Figure 5-4: Stope damage

5.1.2.5. Accident statistics

Mponeng mine injury/fatality accidents are defined as follows:

- Dressing case (injured is treated and reports for work the following shift)
- Lost-time injury (the injured is off work for 1-14 shifts)
- Serious injury (the injured is off work for longer than 14 shifts)
- Fatality (injured dies as a result of his injuries)

Table 5-9 shows the annual distribution of injuries per injury type, these figure were obtained from the mine's Safety Department. The total represents the sum of the annual cases under the category from 2002 to 2008.The proportion refers to the percentage frequency of occurrence of an injury. This is calculated from the totals per injury category.

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Table 5-9: Accident statistics for gravity related falls of ground

Year	Dressing Cases	LTI	SI	Fatal
2002	100	31	17	0
2003	81	24	12	1
2004	47	23	17	0
2005	56	39	25	1
2006	42	25	18	1
2007	43	34	24	1
2008	31	27	5	0
Total	400	203	118	4
Proportion	55%	28%	16%	1%

5.1.3 Quantification of consequences

A summary of possible consequences resulting from the rockfalls was presented in Table 3-2. This section will focus on the quantification of the consequences. The general equation for quantifying the Total Expected Cost of Losses is expressed as follows:

$$\textit{Total Expected Cost} = \textit{Annual frequency of occurrence} \times \textit{Cost of Consequences}$$

The annual frequency of occurrence has been discussed in section 4.5.2. The cost of consequences is the sum of the costs of all the consequences associated with the occurrence. This will be discussed in more detail in the sections below.

5.1.3.1. Cleanup and dilution cost

This cost is determined as discussed in section 4.5.3.A dilution cost per tonne of R 56 was determined in Table 5-3 and the rockfall size was determined from the dimensions. The results of the cleaning up and dilution costs per given size (rockfall length) are shown in Figure 5-5. The graph shows that the very small rockfalls (Class 1 type falls) incur no cost. Falls between 1m and 10m (Class 2 type falls) do result in costs but these are relatively low with an average cost of R 1500 per rockfall. Falls above 10m in length are not cleaned up, therefore, no costs are incurred. The outlier cost (R 7 800) exists because this cost is determined from the tonnage of the rockfall and not only its length. The cost of cleanup and dilution is therefore not necessarily proportion to the length of the rockfall.

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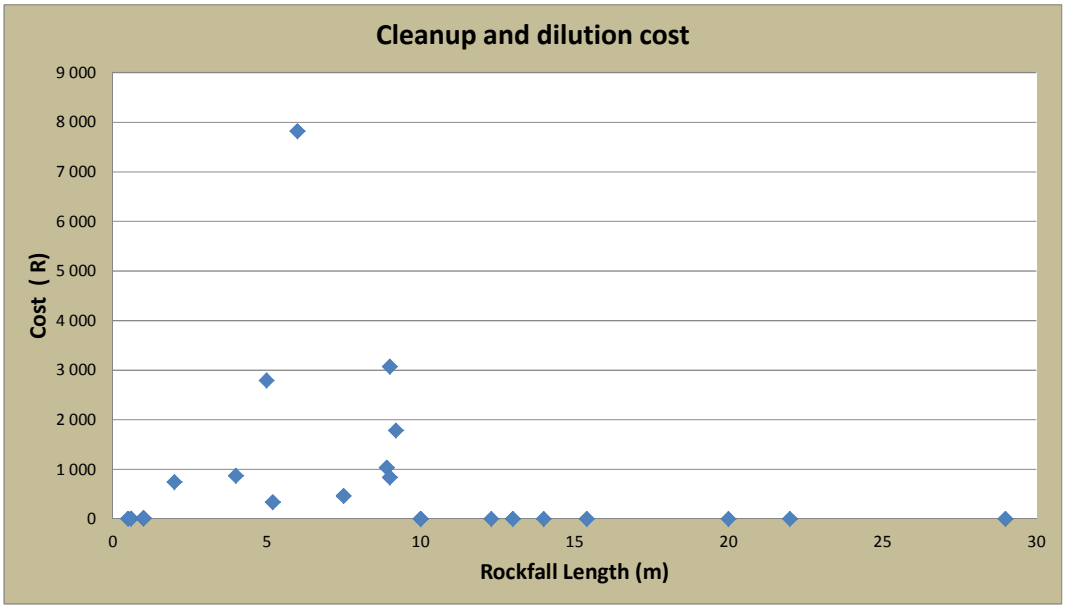


Figure 5-5: Cleanup and dilution cost

5.1.3.2. Re-supporting cost

Class1 falls are small and hence need no re-supporting. Re-supporting is conducted for Class 2 falls. The means of determining this cost is as discussed in Section 4.5.3.A value of 50% for the re-supporting areal factor is applied into this equation. This figure was obtained from discussions with experienced mine personnel. The Rock Engineering Department recommended use of the same support types at an increased density. Figure 5-6 shows that no costs are incurred for falls less than 1m in length. The cost then gradually increases with length up to R 6500 for a 9m rockfall, which is the upper bound for Strategy 2. The larger falls (length greater than 10m) are not re-supported.

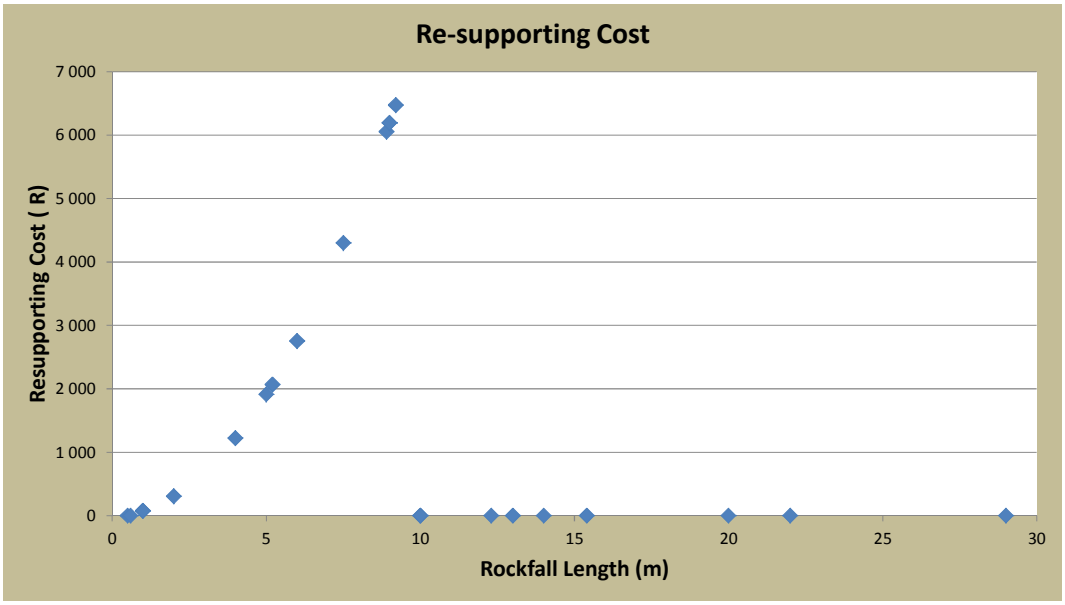


Figure 5-6: Re-supporting cost

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5.1.3.3. Production loss -cleaning-up and re-supporting

As discussed in Section 4.5.3, time spent cleaning and re-supporting depends on the area of the rockfall. Table 5-10 shows the typical time spent cleaning up and re-supporting falls of ground. This information is generic and was obtained from discussions with mine personnel.

Table 5-10: FOG length vs. cleanup time relationship

FOG Length on face (m)	2	5	10	15	20	30
Cleanup and Re-support (days)	0	0.5	1	3 - 5	-	-

Figure 5-7 shows the resulting loss incurred per given rockfall length. The graph shows that rockfalls less than 1m and those above 10m result in no production losses from cleaning-up and re-supporting. The reason is that such remediation strategy is not applied to those rockfalls. Rockfalls between 1m and 10m result in a loss. The outlier cost (R 37 000) exists because the cost is determined from the tonnage of the rockfall and not only its length. Production loss is therefore not proportion to the length of the rockfall.

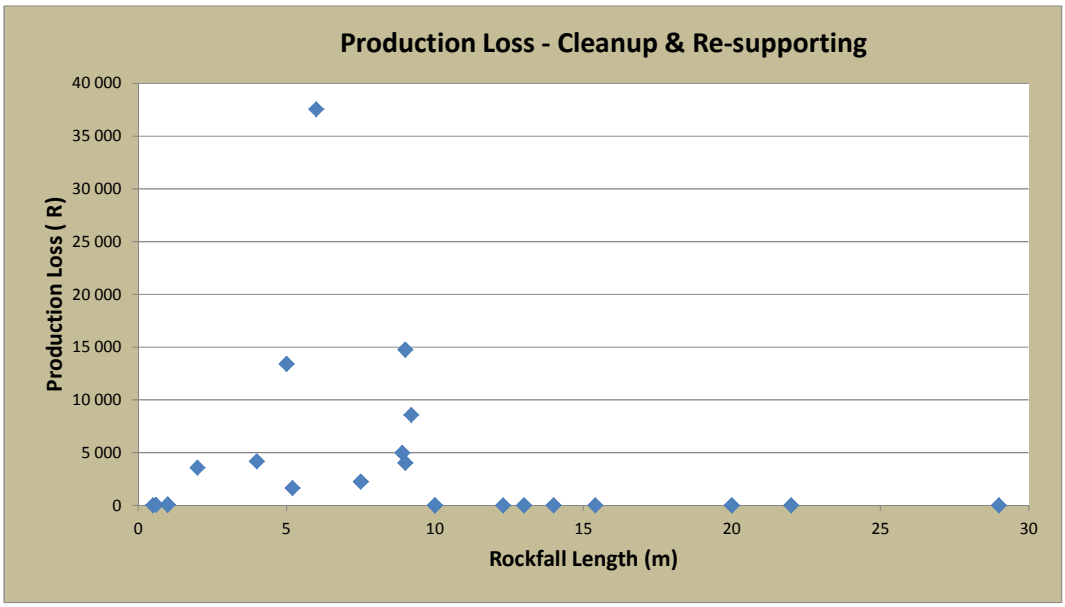


Figure 5-7: Production loss – cleanup and re-supporting

5.1.3.4. Sweepings loss

This loss is incurred in Classes 3 and 4. The methodology for its determination was described in Section 4.5.3. The sweepings loss input (calculated in Table 5-5) is R 545/m^2. The mine personnel suggested a loose density of 1.83 t/m^3 and an average sweepings height of 0.2.

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The survey department provided a sweepings area factor of 20% for use in the calculations. Table 5-11 below shows the resulting expected sweepings area loss determined from the typical rockfall area.

Table 5-11: Sweepings loss

Typical rockfall length (m)	12	14	16	18	22	28
Rockfall Area (m^2)	113	154	201	254	380	616
Expected Sweepings Area loss	136	185	241	305	456	739

Figure 5-8 shows the resulting losses incurred. There are no losses for falls less than 10m in length as falls less than 10m are cleaned up and therefore no sweepings are lost. An average of R 35000 is incurred for falls between 10m and 20m. These losses show no correlation, as they are a function of the area of the rockfall and not the length.

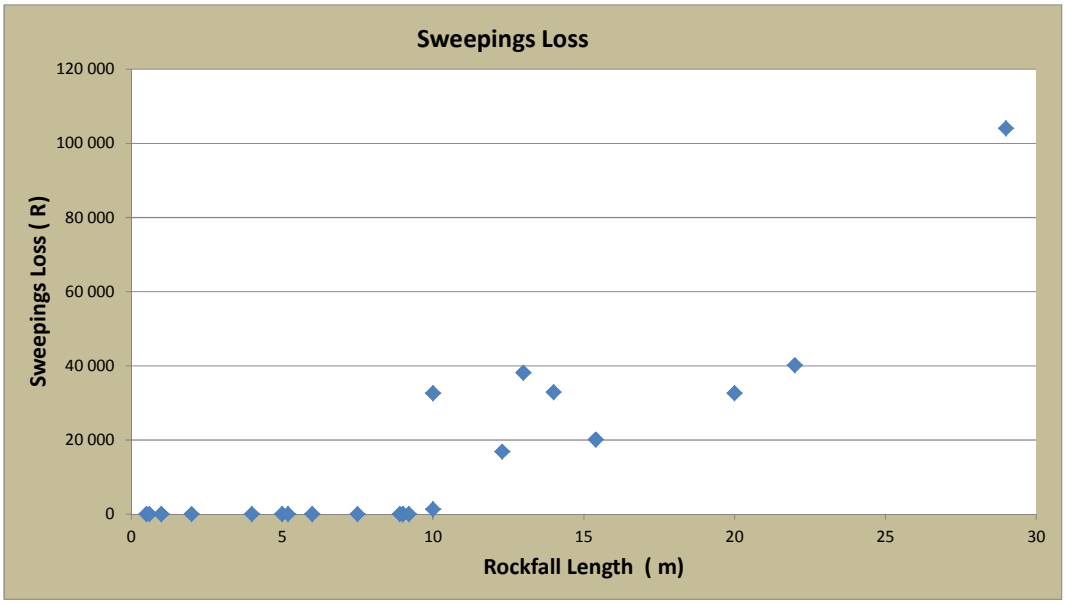


Figure 5-8: Sweepings loss

5.1.3.5. Redeployment and re-establishment cost

Redeployment costs may be incurred in Classes 3 and 4. The other Classes (1 and 2) do not incur this cost. Section 4.5.3 described the factors that influence this cost. On average a single crew works each panel. When a large rockfall occurs, the crew might have to be re-deployed to either a spare or a contingent panel. Inclusive of in-loco inspection periods, this unproductive/production downtime redeployment time on average is 5 days. This number was obtained from interviews with experienced mine personnel.

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Normal productivity per panel is 99t/day but the large rockfalls (Classes 3 and 4) result in lesser productivity during the re-establishing or abandoning period. This loss in productivity during this period is the re-establishing/abandoning loss.

Figure 5-9 shows the resulting costs of re-establishing and redeploying. As none of these is practised for falls less than 10m there is therefore no associated production loss. Re-establishing where there is flexibility results in fewer losses than without flexibility. The cost of redeployment is constant for the varying rockfall lengths as the days taken to redeploy crews are the same.

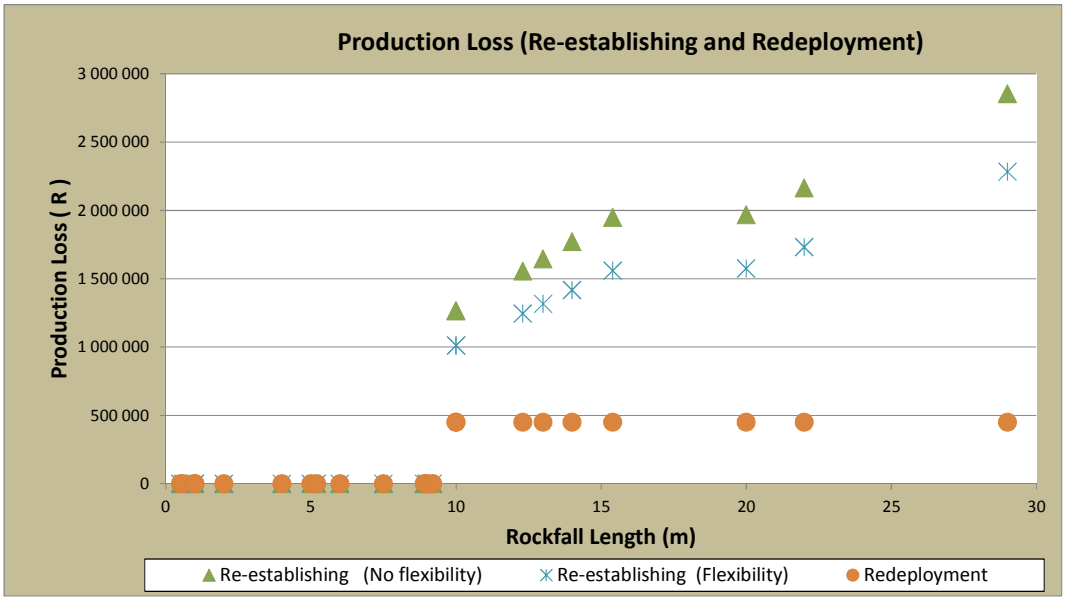


Figure 5-9: Production loss- re-establishing and redeploying

5.1.3.4. Reserve loss

Loss of reserves is not considered a production loss, but as a loss of revenue due to paying ground, being left unexploited. This has the effect of reducing the life of the mine. Loss of reserves occurs when pillars are left during re-establishing or abandoning a panel.

Figure 5-10 shows the results of the reserve losses incurred. Rockfalls less than 10m in length result in no reserve losses. Rockfalls between 10m and 16m result in an average of R 150000 worth of reserve loss. Large falls result in higher losses, R 510000 lost for a whole panel collapse.

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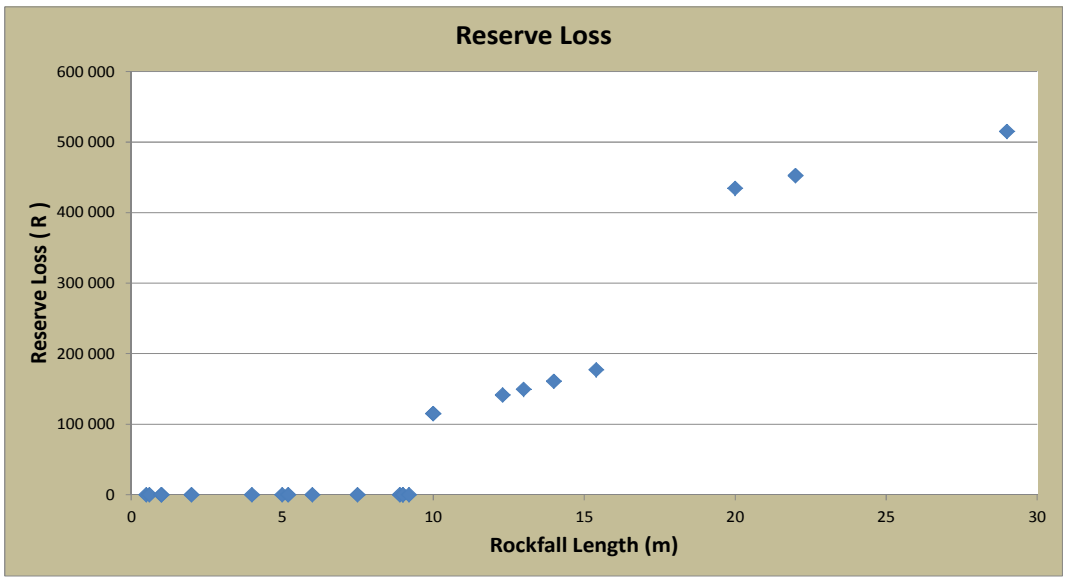


Figure 5-10: Reserve losses

5.1.4 Summary of financial consequences

Figure 5-11 shows the total expected costs per rockfall length for all the rocks (dimensions given in Table 5-6). This graph shows that rockfalls less than 10m in length cost very little in terms of damage to excavation costs. However, upwards of 10m, the re-establishing cost is the most significant costs driver. Re-establishing a 10m length panel costs approximately R 1million, this cost increases up to R 2,5million for a full panel collapse (29m). Reserves lost are about R 0.1million for a 10m fall, and increase up to R 0.5million for a 29m fall. The sweepings loss contribution to the total cost is very small.

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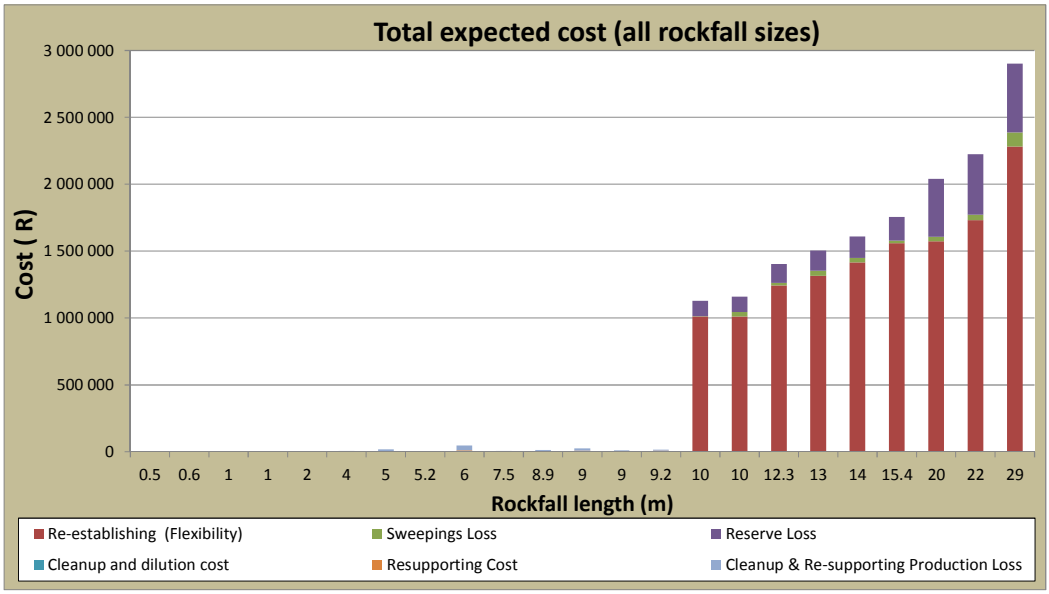


Figure 5-11: Total expected costs (all rockfall sizes)

For clarity purposes, the costs of rockfalls less than 10m are shown at a larger scale in Figure 5-12 below. No data on falls above 10m is included. Production lost due to cleaning up and re-supporting is the major cost contributor, followed by re-supporting costs and the clean up and dilution cost. The costs are not linear because individual rockfall dimensions are used is calculating the costs. For example the 6m length fall (and area of 21m^2) has a production loss of R 38 000 whereas the 9m fall (an area of 19.8m^2) is about R 14 000, but their masses are 136t and 54t respectively. An average of R 7 000 is spent on re-supporting the rockfalls.

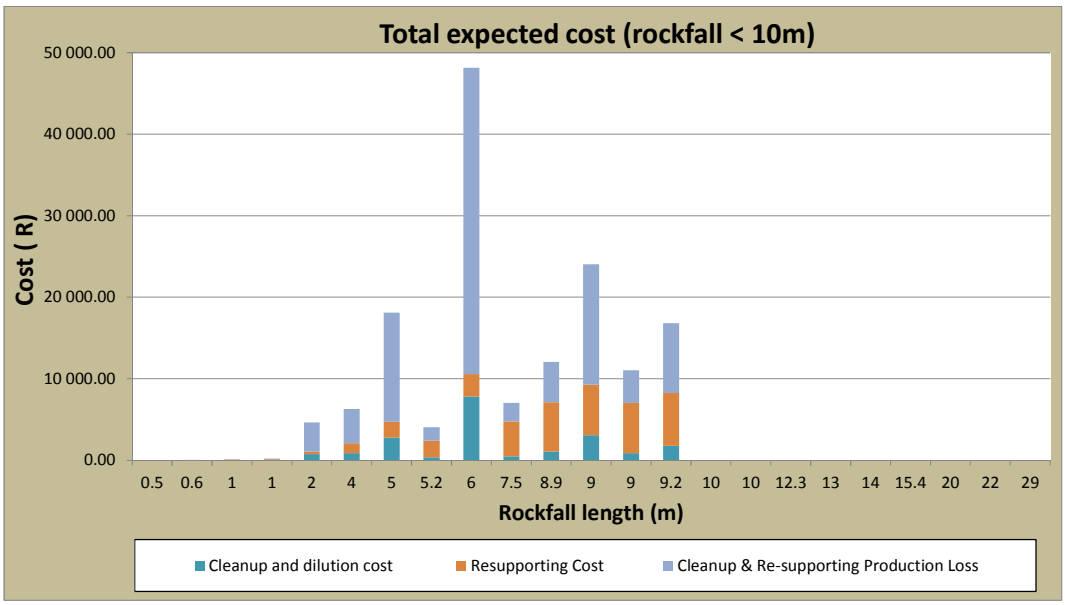


Figure 5-12: Total expected costs (rockfalls < 10m)

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5.2 Case study 2: Impala 12# conventional mining

5.2.1 Background

Impala 12 Shaft exploits the UG2 and Merensky reefs. Narrow tabular conventional mining averaging 76 000t per month is practised on all the UG2 workings and part of the Merensky (remaining upper levels). All mechanised mining is practised on the Merensky workings producing about 112 000t per month. For conventional mining, scattered breast mining is conducted on 30m panels (average) and raise lines are 180m apart, Figure 5-13. Strike gullies are used as access ways to the face, and for cleaning, and material transport. These gullies are either developed concurrently with or slightly ahead of the panel face or protected with a siding, or footwall lifted behind the panel face, depending on the raise line geometry (underhand or overhand). Crosscuts are situated 210m apart and haulages between 100 and 150m above/below reef plane.

The objective of the study is to determine the financial consequences associated with rockfalls in the conventional mining section of the mine. First the inputs are described. Classification of the rockfalls is then carried out using the proposed criteria. The consequences in each classification are then quantified. This gives the expected annual costs. Available rockfall data are used, which is by no means a complete data set, and the costs are likely to be underestimated.

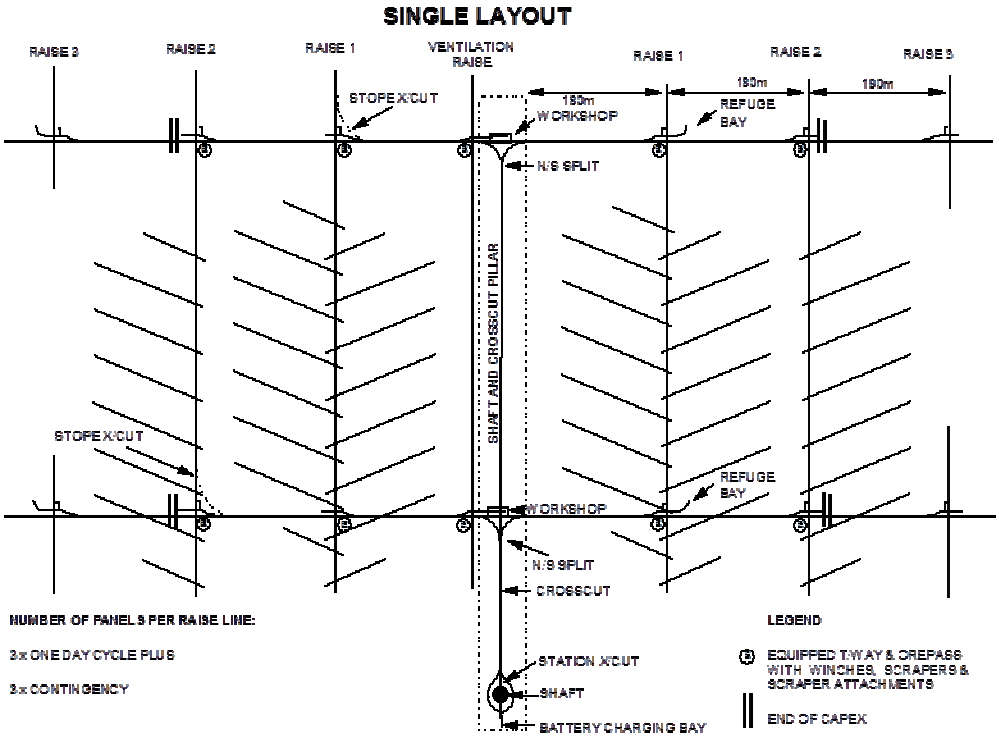


Figure 5-13: Conventional mining layout (Impala Standards, 2008)

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5.2.2 Description of input parameters

The input parameters for determining the costs are categorised as; general mining parameters, accident statistics, financial information, support standard and cost.

5.2.2.1. General mining parameters

Figure 5-14 below shows an illustration of the stope layout.

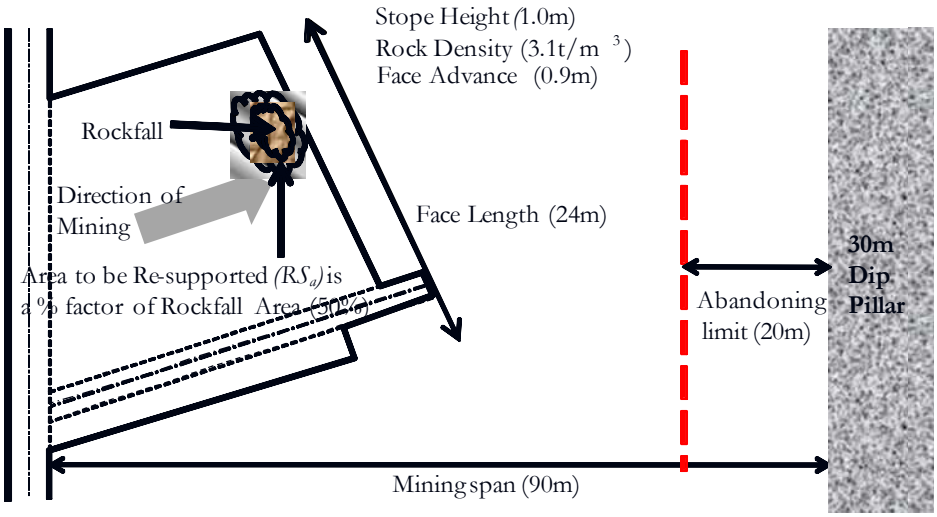


Figure 5-14: Mining parameters

The mining inputs are further categorised as geometric, rock mass and productivity parameters. The aspects considered under each parameter are given in Table 5-12. The values presented were obtained from the mining standards and discussions with the mining personnel.

Table 5-12: General mining parameters

Geometry	
Face Length (m)	24
Advance per blast (m)	0.9
Stoping height (m)	1.0
Mining Span (m)	90m
Abandoning limit (m)	20
Re-supporting factor (%)	50
Rock Mass	
Rock density (t/m^3)	3.1
Average mineral grade(g/t)	3.7
Bulk density (t/m^3)	1.83
Productivity	
m^2 mined per month	18,397
Tonnage mined per month	86,000
Working days per month	23
Blasts per month	18

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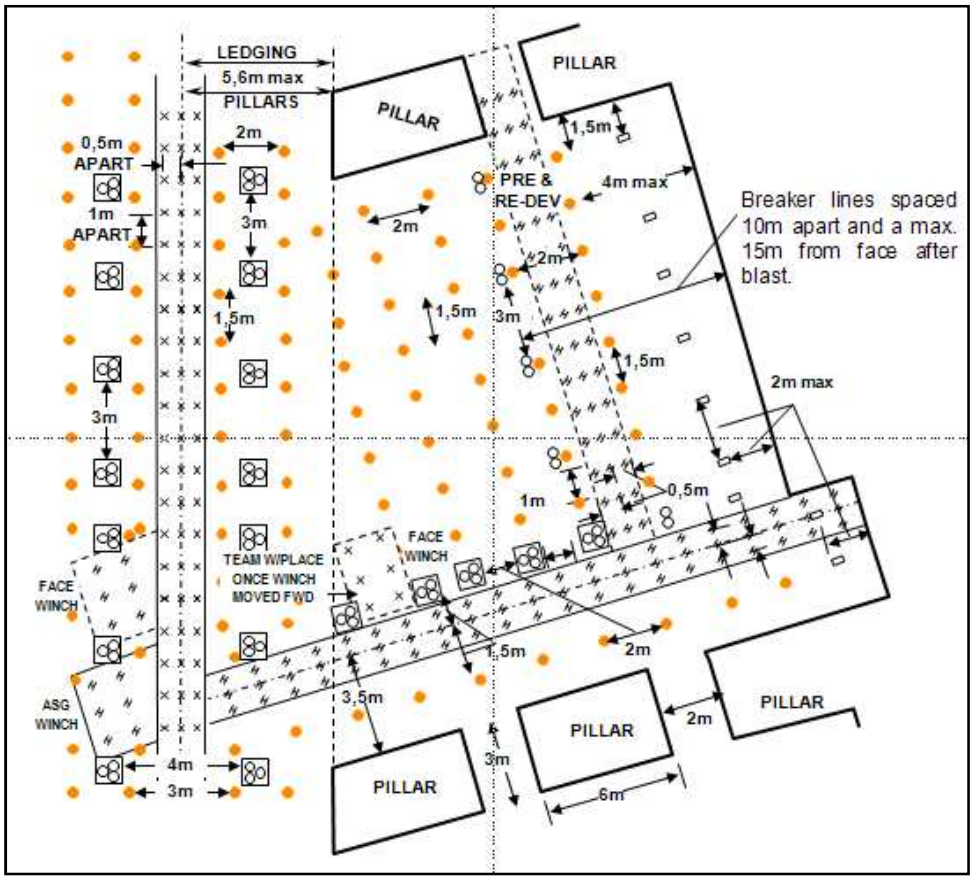
5.2.2.2. Support standard and cost

Impala has six Ground Control Districts, defined in Zones A to S. Figure 5-15gives a description of the zones (Impala Code of Practice). The table shows the risk factor, 95% potential fallout height as required by the Impala COP Guidelines (2008), support resistance and support strategy colour for each zone. Approximately 60% of the mining on the property occurs in the Red zone with conventional mining on 12 Shaft being conducted in Zone E and F (rolling reef and blocky ground).

Zone	Dominant feature	Risk Factor	Potential fallout height - back area	Potential support resistance - back area	Support Strategy colour
A	Normal Ground	Low risk from geological features	1.4 m	42.5 kN/m²	GREEN
B	Surface protection	0 - 30 m No Mining 30 - 100 m Shallow mining retrictions	1.4 m	42.5 kN/m²	BLUE
C	Curved Joints	Wedge type failure	2.2 m	66.9 kN/m²	BLUE
D	Coarse pyroxenite Spotted anorthosites	Large flat failures, extension fractures	2.7 m	82.1 kN/m²	RED
E	Rolling Reef	Associated with curved joints and domes various reefs on same panel: A, B or C	3.0 m	91.2 kN/m²	RED
F	Blocky Ground	Associated with extensive jointing, faulting, shear zones, etc. on the various reefs horizons	2.7 m	82.1 kN/m²	RED
G	Triplets or ICL < 0.3 m from UG2 Top Reef Contact	Weak beam tends to falls of ground between support units	0.8 m	24.3 kN/m²	LIGHT BLUE
H	Low-angled joints in UG2	Series of domes which could intersect into the triplets and results in major falls of ground	3.0 m	91.2 kN/m²	RED
S	Seismicity	Seismic risk associated mostly with crush type events (pillar strain bursts or pillar foundation failure) and less with slip type events (geological features)	3.0 m	91.2 kN/m²	YELLOW

Figure 5-15: Ground control districts (Impala Mine Code of Practice, 2008)

Each ground control district has mining and support standards. The support standard for the Red zone is as illustrated below in Figure 5-16 and Figure 5-17 for the UG2 and Merensky reefs.

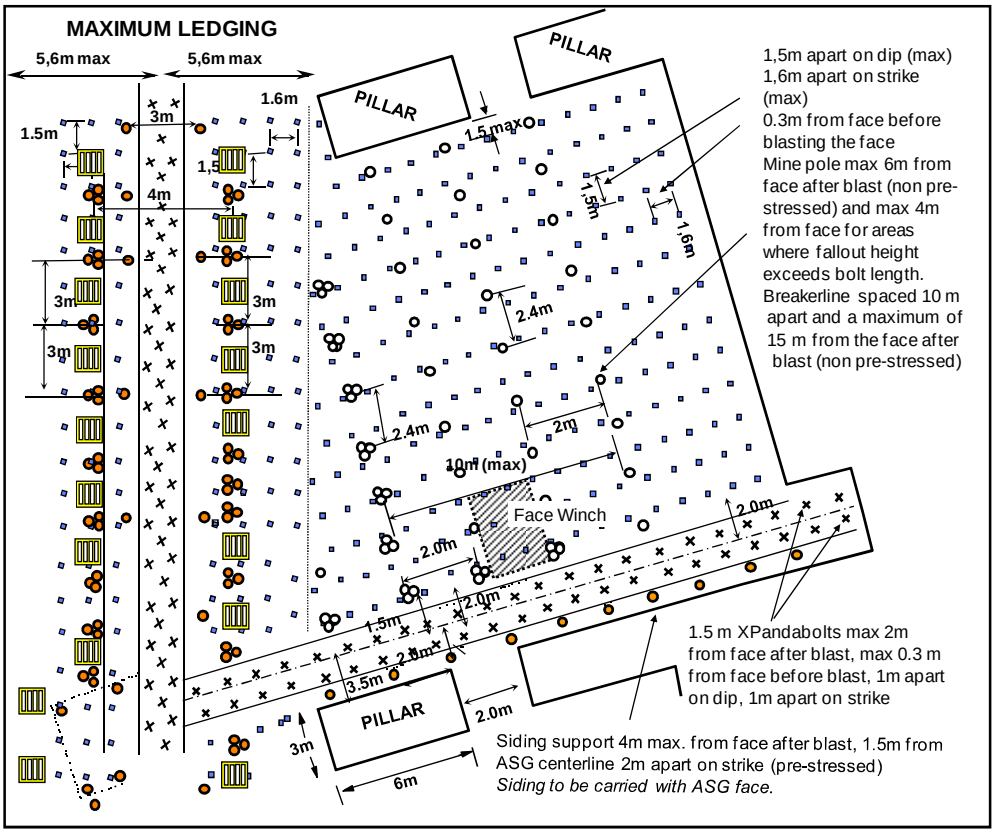


LEGEND:

×	1,5m Xpandabolt, 1m apart in direction of mining, 0,5m from center bolt minimum 55º into the h/wall plane. Complete row must be installed 30cm from face before the blast irrespective of face advance.
H	1,2m Hydra bolt 1,5m apart on dip and *daily face advance* apart on strike, capped at 1,6m. Minimum 70º into the h/wall plane.
▭	Temporary props max 2m from face and operator.
▥ ⊛	Pre-stressed matpack and cluster pack
●	Pre-stressed mine pole
O	Mine pole without pre-stress unit

SIDING If no siding is blasted, install 1 row 1,5m Xpandabolts, 1m from h/wall, 1m apart, minimum 55º. Sidewall support must be kept at least in line with panel face. Best practice is to carry the siding with the ASG face, otherwise it may be carried between 6 and 12m from the panel face. Sidings may not be down-dipped.

Figure 5-16: UG2 support standard – No in-stope bolting (Impala Mine Code of Practice, 2008)



LEGEND:

- Non-stressed mine poles
- Non-stressed clustered mine poles
- Pre-stressed mine pole
- Pre-stressed clustered mine poles
- Temporary Props
- Hydra bolt (1.2m)
- Xpandabolt (1.5m)
- Pre-stressed matpack

Figure 5-17: Merensky support standard – In-stope bolting (Impala Mine Code of Practice, 2008)

The support unit costs are given in Table 5-13. These figures were obtained from the Rock Engineering Department. As described in Chapter 4, only the cost per unit or the cost per square metre is applied in these calculations.

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Table 5-13: Support costs

ID	Code	Name	Cost	Basis
1	E1	Wedge Props _1.0mx180mm (combo)	69.81	R/unit
2	E2	Wedge Props _1.2mx180mm (combo)	72.46	R/unit
3	E3	Wedge Props _1.4mx180mm (combo)	77.94	R/unit
4	E4	Wedge Props _1.6mx180mm (combo)	81.44	R/unit
5	E5	Wedge Props _1.8mx180mm (combo)	87.98	R/unit
6	E6	Madoda 180ST PSU	37.9	R/unit
7	E7	Stromaster 450_2.1m with headboard	250.48	R/unit
8	E8	Stromaster 450_2.4m with headboard	263.61	R/unit
9	E9	ST Combo Headboard_400mmx400mm	35.35	R/unit
10	E10	Rocprop_20t_1.0mx1.2m	473.97	R/unit
11	E11	Rocprop_20t_1.4mx1.8m	572.24	R/unit
12	E12	Rocprop_20t_2.0mx2.5m	678.58	R/unit
13	E13	Rocprop_20t_2.2mx3.2m	760.67	R/unit
14	E14	Rocprop_20t_2.3mx3.5m	817.3	R/unit
15	E15	Boulby Headboard_380mmx500mm	68.68	R/unit
16	E16	Split sets_42mmx1.5m	32.33	R/unit
17	E17	Split sets_42mmx2.1m	42.89	R/unit
18	E18	Dolly Pusher_25mm	281.69	R/unit
19	E19	Dolly Pusher_22mm	281.69	R/unit
20	E2	Backfill	36.54	R/m^2
21	E3	Backfill	42.07	R/t
22	P21	Timber Combosite Pack_110x9x9	1513.18	132 PER BUNDLE
23	P22	Timber Combosite Pack_150x9x9	1180.80	72 PER BUNDLE
24	P23	Hydrocell_1225x450mm	89.47	Each/100 per bundle

5.2.2.3. Financial information

The financial data were obtained from the Impala Platinum Annual Report (2008), shaft Finance Departments and discussions with accounts personnel. Important derivatives from the data are, the revenue earned over a period, the costs of mining consumables and the processing costs. From these, the dilution cost, production loss and the sweepings loss were determined.

5.2.2.3.1. Dilution cost

While dilution cost is the cost of horizontal and vertical transport and processing the waste rock, the cost of vertical and horizontal transport could not be extracted from the financial data provided. The processing cost comprised of concentrating and smelting and the refining operations cost. These figures were obtained from the Impala Platinum Annual Report (2008). The dilution cost per tonne, shown in Table 5-14, was calculated to be R 96,67 per tonne.

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5.2.2.3.2. Production profit

Table 5-15 gives the breakdown of the variable costs. The total variable costs for December 2008 are R 8,092,000.

Table 5-14: Dilution cost

Description	Value/Quantity
Tonnage milled per month (t)	15,855,000
Concentration and Smelting Cost (R/t) Refining Operations Cost (R/t) Processing Cost (average monthly) (R)	1,057,000,000 476,000,000 1,533,000,000
Dilution cost (R/t)	96.67

Table 5-15: 12 Shaft variable costs (Impala Mine 12 Shaft Dec month end report, 2008)

	December '08				YTD December '08		
	ACT	BP'09	VAR	VAR %	ACT	BP'09	VAR %
Mining Contracts	2,799	2,242	-557	-25%	12,539	14,977	16%
Drill Steel	249	242	-7	-3%	1,034	1,598	35%
Explosives & Accessories	1,895	1,947	53	3%	11,809	12,858	8%
Support	1,367	955	-411	-43%	8,057	6,468	-25%
Winding, Scraper & Ropes	615	761	145	19%	3,395	5,035	33%
Other Mining Stores	1,167	860	-307	-36%	6,374	5,926	-8%
Total Consumables	8,092	7,007	-1,085	-15%	43,208	46,861	8%

Table 5-16 shows the parameters required to determine the production loss. The revenue and processing costs per tonne was determined from the annual report information. The consumables cost per tonne where then determined on 12 Shaft data. The resulting production profit was R 1,127/t.

Table 5-16: Production profit loss

Parameters	Value
Tonnage milled (Impala)	15,855,000
Revenue - Impala (R)	20,889,000,000
Processing Cost – Impala (R)	1,533,000,000
Revenue per tonne (R/t)	1,317.50
Processing Cost per tonne (R/t)	96.67
Tonnes mined per month (12 Shaft) (T/month)	86,000
Cost of Consumables (12 Shaft) (R)	8,092,054
Cost of consumables per tonne (R/t)	94,09
Production Profit (R/t)	1,127

`

5.2.2.3.3. Sweepings loss

The adjusted sweepings loss value is R 727 per square metre mined. Table 5-17 shows the data used in determining this value.

Table 5-17: Determination of sweepings loss

Description	Value
Square metres mined (Impala)	3,391,679
Revenue - Impala (R)	20,889,000,000
Processing Cost – Impala (R)	1,533,000,000
Loose density	1.83
Insitu density	3.1
Average sweepings height (m)	0.2
Stope height (m)	1.0
Sweepings loss (R/m^2)	727

5.2.2.4. Rockfall data

The Rock Engineering Department maintains a rockfall database which records rockfall accidents. Figure 5-18 shows an extract from the database. This database captures the section, date, dimension, volume, tonnage, and strike and dip length of the FOG. These data are comprehensive and applicable to the purposes of this research.

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Section	Date	Maximum Height	Width	Length	Area	m3	Tons	Strike length	Dip Length
126	24-Jan-07	0.8	2.3	2.5	5.8	8.6	26.7	2.5	2.3
121	11-Feb-07	2.5	24	39	936	1404	4352.4	39	24
121	22-Feb-07	2.5	30	45	1350	2025	6277.5	45	30
126	22-Mar-07	1.5	5.0	12.0	60.0	90.0	279.0	12	5
126	22-Mar-07	2.0	8.9	11.5	102.4	153.5	475.9	11.5	8.9
126	12-Apr-07	0.7	2.8	3.5	9.8	14.7	45.6	3.5	2.8
126	29-May-07	1.0	4.9	5.5	27.0	40.4	125.3	5.5	4.9
126	05-Jun-07	1.2	3.0	5.6	16.8	25.2	78.1	5.6	3
125	29-Jun-07	1.6	2.4	9.2	22.1	33.1	102.7	9.2	2.4
125	09-Jul-07	1.3	6.0	9.0	54.0	81.0	251.1	9	6
125	13-Jul-07	1.9	1.8	6.3	11.3	17.0	52.7	6.3	1.8
126	13-Jul-07	2.3	7.0	15.0	105.0	157.5	488.3	15	7
1277	01-Aug-07	0.4	2.0	4.5	9.0	13.5	41.9	4.5	2
125	14-Nov-07	1.5	4.6	16.0	73.6	110.4	342.2	16	4.6
1222	12-Dec-07	0.3	0.8	1.6	1.3	1.9	6.0	1.6	0.8
125	08-Jan-08	1.0	8.8	9.0	79.2	118.8	368.3	9	8.8
126	16-Jan-08	2.0	3.5	6.0	21.0	31.5	97.7	6	3.5
126	21-Jan-08	1.8	5.0	8.0	40.0	60.0	186.0	8.0	5
126	30-Jan-08	1.8	4.0	8.0	32.0	48.0	148.8	8	4
126	15-Feb-08	0.1	0.2	0.2	0.0	0.0	0.1	0.15	0.15
125	15-Feb-08	1.4	12.0	11.0	132.0	198.0	613.8	11	12
125	27-Mar-08	0.9	1.6	2.8	4.5	6.7	20.8	2.8	1.6
125	27-Mar-08	2.2	8.0	25.0	200.0	300.0	930.0	25	8
125	01-Apr-08	0.7	0.8	2.6	2.1	3.1	9.7	2.6	0.8
125	01-Apr-08	1.4	11.3	14.7	166.1	249.2	772.4	14.7	11.3
125	05-Jun-08	2.1	1.7	4.6	7.8	11.7	36.4	4.6	1.7
125	18-Jun-08	1.1	1.2	3.9	4.7	7.0	21.8	3.9	1.2
125	18-Jun-08	2.0	11.0	28.5	313.5	470.3	1,457.8	28.5	11
126	15-Aug-08	1.3	5.4	8.6	46.4	69.7	215.9	8.6	5.4
126	18-Aug-08	1.2	8.0	12.0	96.0	144.0	446.4	12	8
126	19-Aug-08	2.0	5.0	10.0	50.0	75.0	232.5	10	5
126	01-Sep-08	0.8	1.0	1.3	1.3	2.0	6.0	1.3	1
125	01-Sep-08	1.6	9.0	18.0	162.0	243.0	753.3	18	9

Figure 5-18 Rockfall database extract (Impala Mine 12 Shaft Rockfall database, 2008)

When a rockfall occurs in the face area of a stope, remedial strategies are undertaken to deal with the rockfall. Figure 5-19 illustrates the event tree that describes rockfall classification according to remedial strategies. The Mponeng convention of classifying the severity of damage was adopted in determining these remedial strategies. This event tree was constructed through:

- Interviewing experienced mining and rock engineering personnel,
- Studying past rockfall occurrences on the shaft
- Studying the mines rockfall database.

Table 5-18 gives a summary of the consequences per rockfall classification.

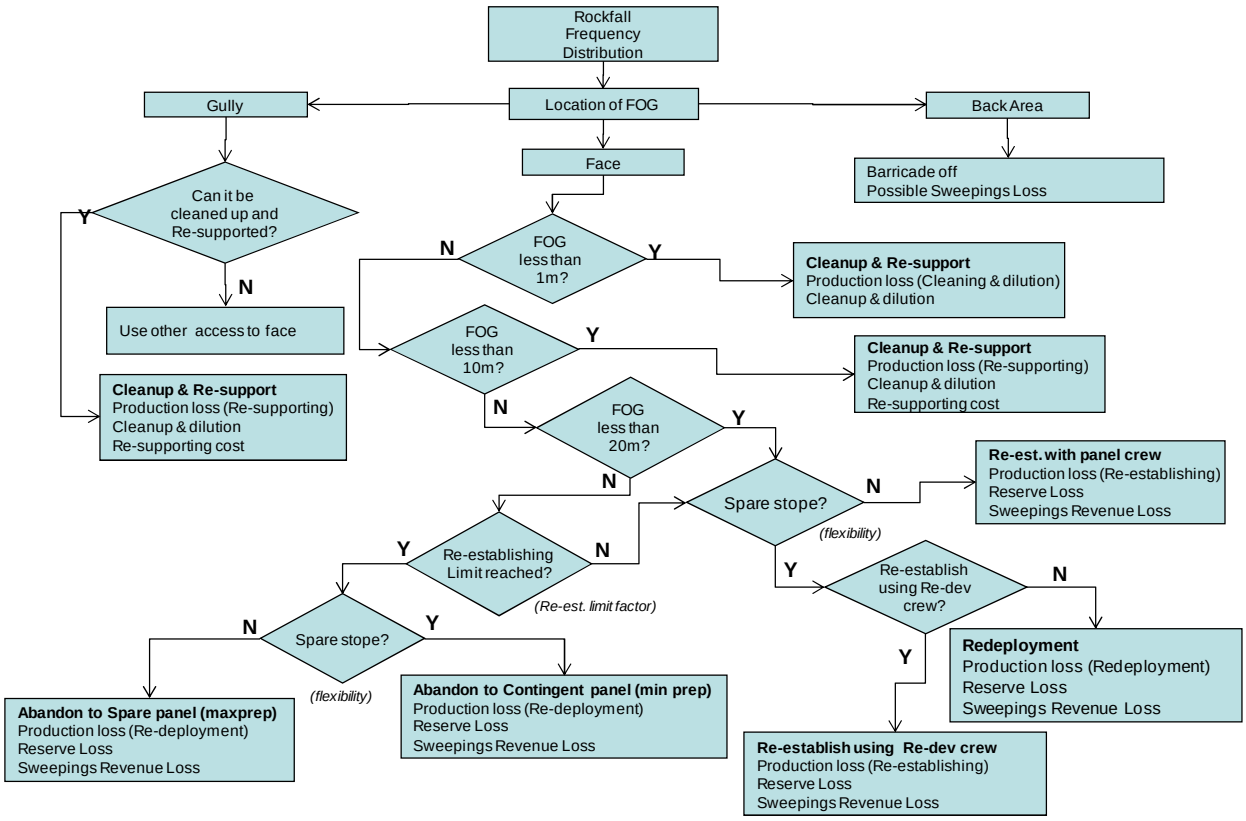


Figure 5-19: Stope damage event tree

Table 5-18: Strategies summary

Description	Class1	Class 2	Class 3	Class 4a	Class 4b
Cleanup and dilution Loss	Yes	Yes	No	No	No
Re-supporting Cost	No	Yes	No	No	No
Sweepings Revenue Loss	No	No	Yes	Yes	Yes
Production Loss					
Cleanup and Re-supporting	Yes	Yes	No	No	No
Re-establishing	No	No	Yes	Yes	No
Re-deployment	No	No	No	Yes	Yes
Reserve Loss (NPV)	No	No	Yes	Yes	Yes

5.2.2.5 Accident statistics

Injury/fatality classifications are defined as:

- Dressing case (injured is treated and reports for work the following shift)
- Lost-time injury (the injured is off work for 1-14 shifts)
- Serious injury (the injured is off work for longer than 14 shifts)
- Fatality (injured dies as a result of his injuries)

`

The statistics on the number of injuries classified per category, Table 5-19, per year were obtained from the mines Safety Department. The injury rate refers to injuries per million man-hours worked.

Table 5-19: Accident statistics for gravity related falls of ground

Year	Dressing cases	LTI	SI	Fatal
2003	174	125	71	6
2004	204	113	67	5
2005	185	111	60	3
2006	239	95	89	3
2007	189	80	94	8
2008	154	73	59	3

5.2.3 Quantification of consequences

A summary of possible consequences resulting from the rockfalls is presented in Table 5-18. This section will focus on the quantification of the consequences. The general formula for quantifying the total expected cost of losses is shown below:

Total Expected Cost

=

Annual Frequency of Occurrence of a rockfall of a given size (m^2)

X

[Cleanup and Dilution Cost + Re-supporting + Sweepings Revenue Loss+ Production Loss+ Reserve Loss (Given on NPV basis)]

5.2.3.1. Cleanup and dilution loss

Cleanup and dilution costs are incurred only in class 1 and 2. The methodology from Section 4.5.3 and a dilution cost R 96,67/t determined in Table 5-3were applied. The results of the cleaning-up and dilution costs per given size of rockfall length are shown in Figure 5-20.

The graph shows that the very small rockfalls (Class 1 type falls) incur no cost. Falls between 1m and 10m (Class 2 type falls) do result in costs with costs such as R 24 000 being incurred for a 10m rockfall. Falls above 10m in length are not cleaned up therefore no costs are incurred. Variations in the costs in relation to size occur because the cost is determined from the tonnage of the rockfall and not only its length.

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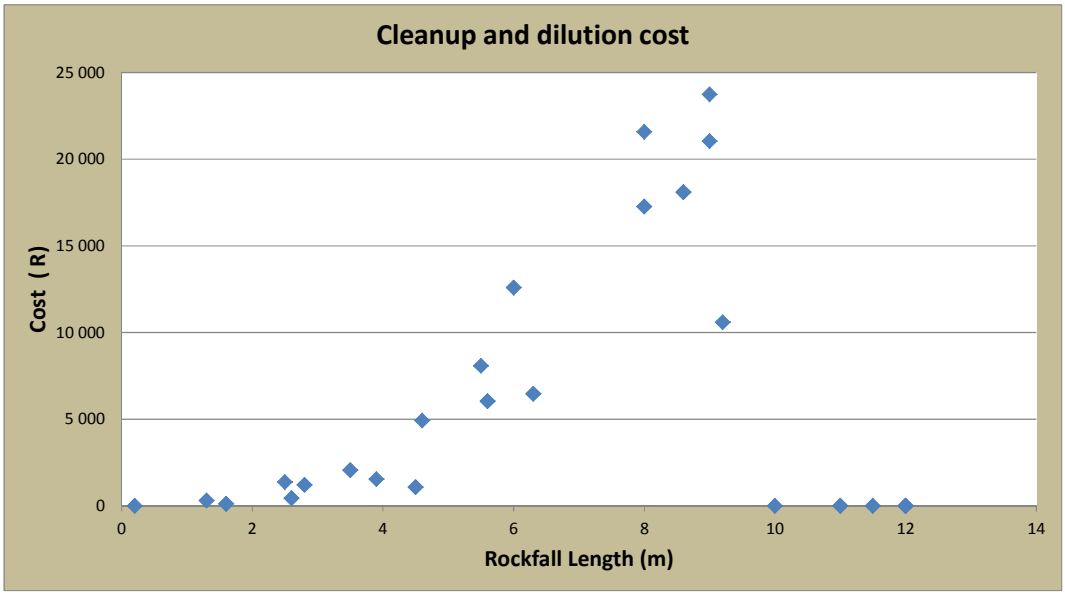


Figure 5-20: Cleanup and dilution cost curve

5.2.3.2. Re-supporting cost

Re-supporting is conducted for class2 FOG's. The methodology for determining the re-supporting cost is discussed in Section 4.5.3. The re-supporting areal factor has been taken to be 50%. This figure was obtained from discussions with experienced mine personnel.

Figure 5-21 shows the distribution of re-supporting costs related to the length of the rockfall along the face. A cost of R 6500 is incurred for a rockfall 9m along the face. The larger falls, greater than 10m, are not re-supported.

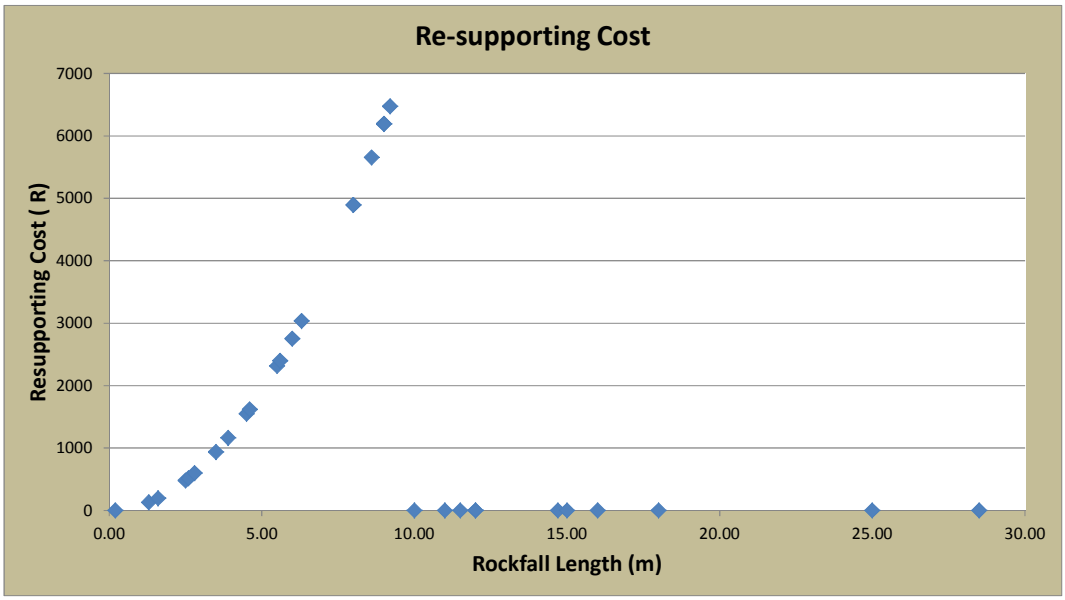


Figure 5-21:Re-supporting cost

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5.2.3.3. Production loss-cleaning-up and re-supporting

As discussed in section 4.5.3, time spent cleaning and re-supporting will vary depending on the area of the rockfall. Table 5-20 shows the typical time spent cleaning up and re-supporting. These are generic figures obtained from interviewing experienced mine personnel.

Table 5-20: FOG length vs. clean-up time relationship

FOG Length on face	2m	5m	10m	15m	20m	30m
Cleanup and Re-support (days)	0	0.5	1	3 - 5	-	-

Figure 5-22 shows the production losses resulting from the FOG's. It is noted that only falls between 1m and 10m result in cleaning up and re-supporting losses. The loss is not based only on the length of the FOG, hence the non-linear relationship of the loss to the length of the FOG.

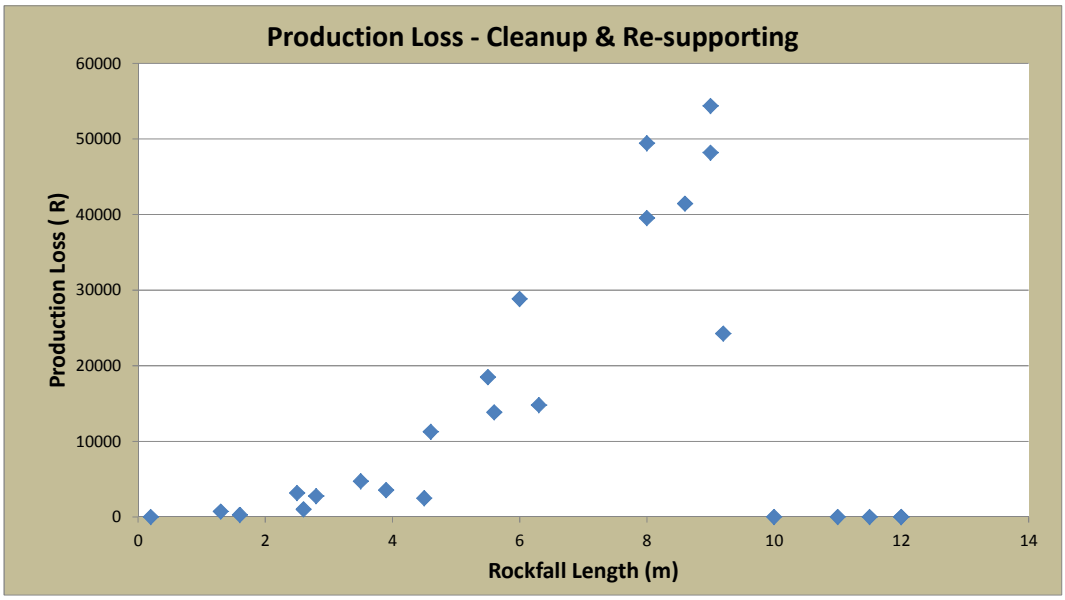


Figure 5-22: Production loss – cleanup& re-supporting

5.2.3.4. Sweepings loss

Sweepings losses are incurred in classes 3 and 4. The methodology for determining this loss is described in Section 4.5.3. A sweepings loss per m^2 input of R 747was applied. The loose density is 1.83t/m^3 and the average sweepings height is 0.2. The Survey Department provided a sweepings area factor of 20%.

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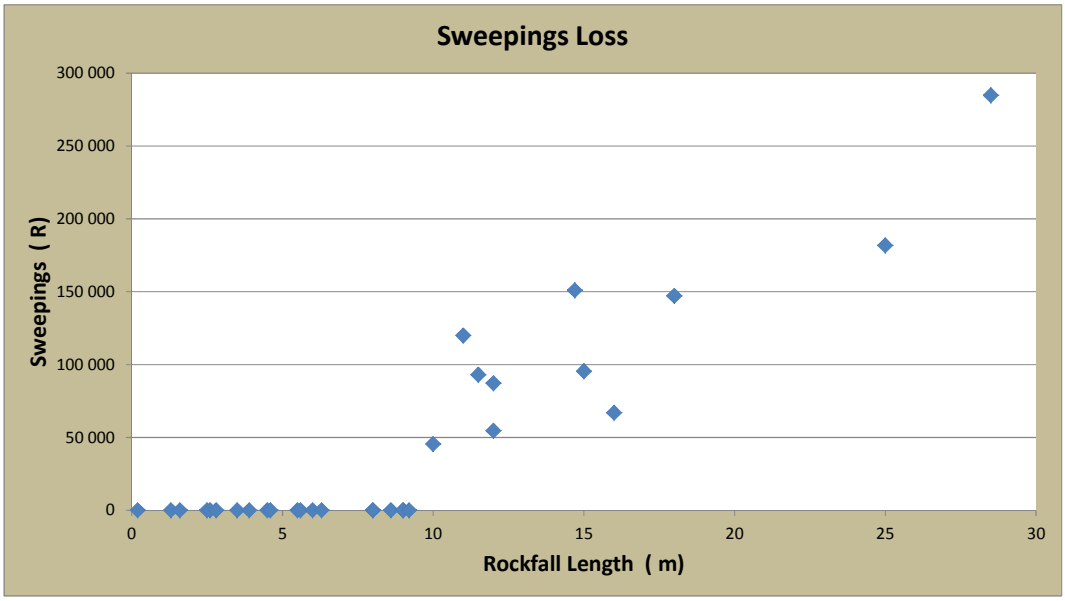


Figure 5-23: Sweepings loss

5.2.3.5. Redeployment and re-establishment cost

Redeployment costs are incurred in classes3 and 4.The number of crews affected, total number of days in redeployment and the total tonnage required from the crew(s) in those given days, as discussed in Section 4.5.3, influence this cost.

On average, a single crew works each panel. When a large rockfall (class 4) occurs, the panel crew is re-deployed to either a spare or a contingent panel. The mine has redevelopment crews, which are then called in to re-establish the panel. For class 3 rockfalls, the same panel crew does the re-establishing. Inclusive of in-loco inspection periods, this unproductive/production downtime redeployment time on average is 5days. This number was obtained from interviews with experienced mine personnel.

Normal productivity per panel is 99t/day. Large rockfalls, class 3 and 4, result in lesser productivity being achieved during the re-establishing or abandoning period. The productivity shortfall is the re-establishing loss.

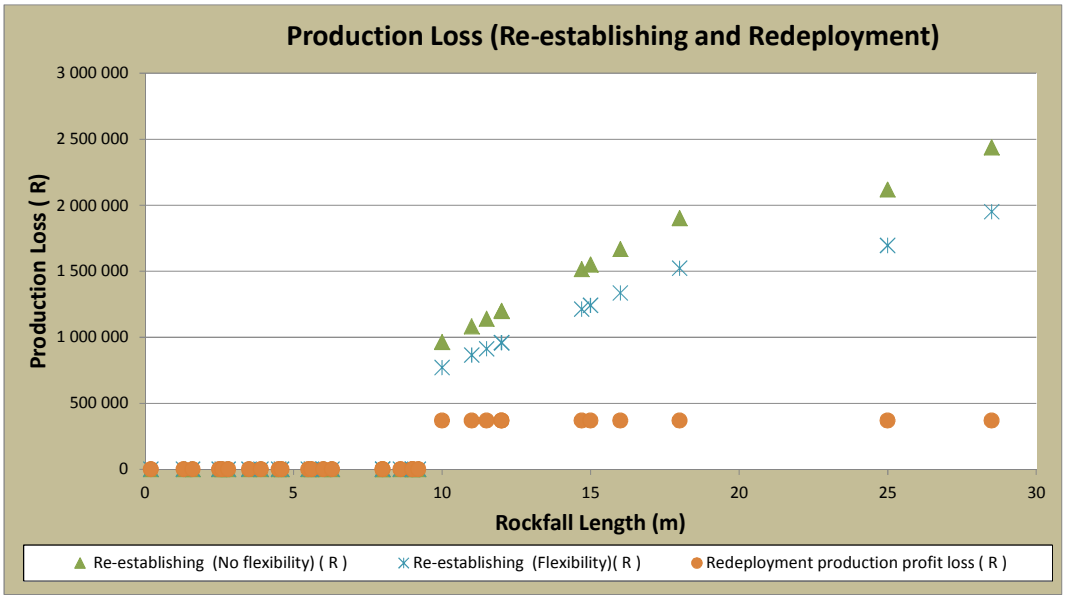


Figure 5-24: Production loss – re-establishing and redeployment

5.2.3.6. Reserve loss

The loss of reserves has the effect of reducing the life of the mine. Hence it is not quantified as a production loss, but as a present value loss of revenue. Its overall impact is on the net present value (NPV). Loss of reserves occurs when pillars are left in the process of re-establishing a panel or when a panel is completely abandoned.

Figure 5-25 shows the value of reserve losses per length of rockfalls. There are no losses for smaller falls, less than 10m. 3m pillars left for falls between 10m and 20m average R 150 000. Large 3m wide pillars can lockup R450 000 worth of reserves.

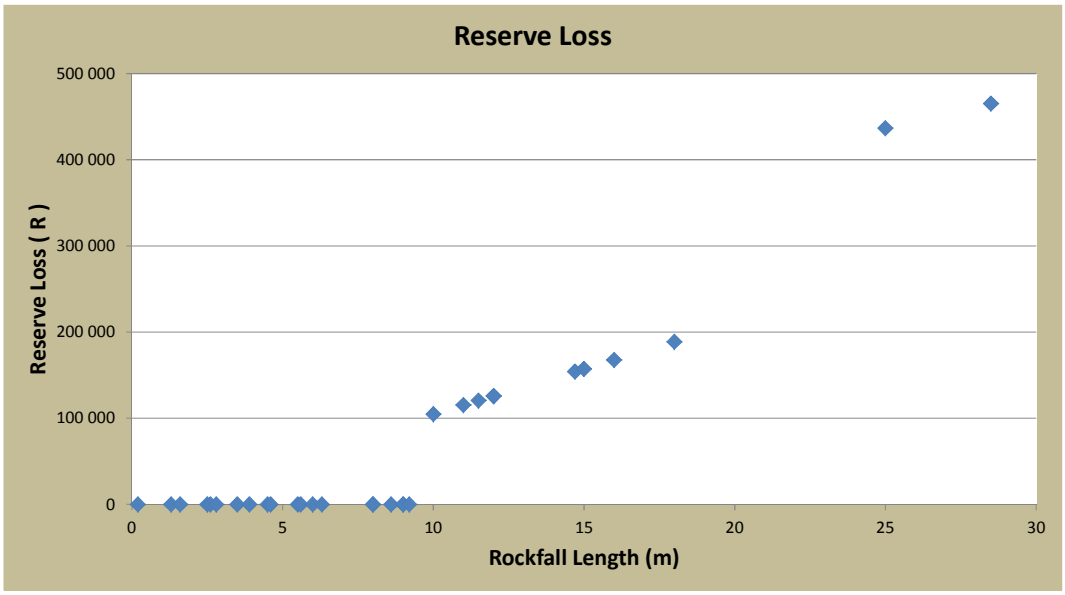


Figure 5-25: Reserve loss

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5.2.4 Summary of financial consequences

The total expected costs per rockfall length for all the rocks (dimensions given in Figure 5-18) are illustrated in Figure 5-26. From the graph it is clear that for rockfalls upwards of 10m, the re-establishing cost is the major cost driver. Re-establishing a 10m length panel costs approximately R 0.75million, and this cost increases up to R 1,9million for a full panel collapse (28.5m). Reserves lost are about R 0.08milliom for a 10m fall, and increase up to R 0.4million for a 28.5m fall. The sweepings loss contribution to the total cost is minimal, R 0.3million is realised for the 28.5m rockfall. This graph shows that rockfalls less than 10m in length cost very little in terms of damage to excavation costs, as shown more clearly in Figure 5-27.

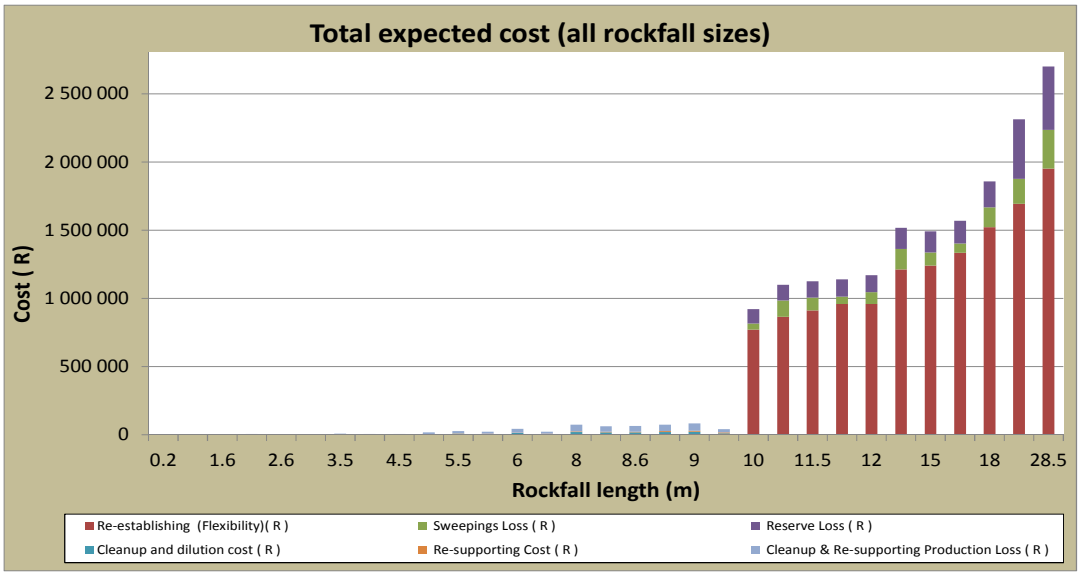


Figure 5-26: Total expected costs (all sizes)

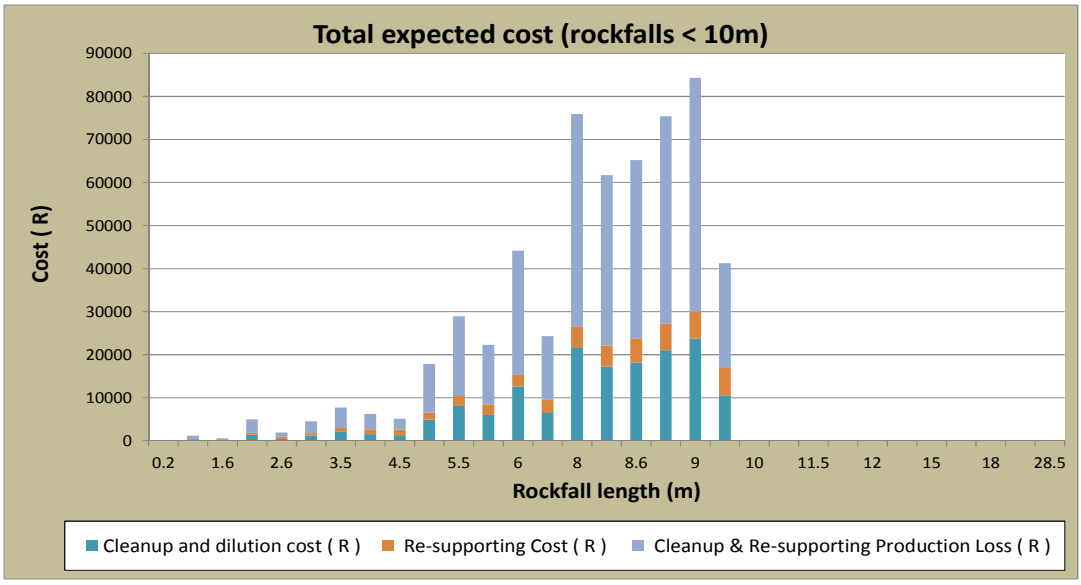


Figure 5-27: Total costs (rockfalls < 10m)

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In Figure 5-27 no data on falls above 10m is included. The costs are not linear because individual rockfall dimensions are used in calculating the costs. Production lost due to cleaning up and re-supporting is the major cost contributor, followed by clean up and dilution and then re-supporting. The production loss increases as the rockfall size increase. Typically a 9m rockfall has a production loss of R 54 000, a cleanup cost of R 22 000and a re-supporting cost of R 7 000.

`

6 Application of the methodology

In the preceding chapters of this dissertation, a methodology to quantify the costs of rockfalls has been described. To enhance this methodology's applicability to the mining industry, as part of a SIMRAC Rockfall Elimination project, the methodology was coded into risk-cost analysis software (Joughin et al, 2009). This software is a tool for quantifying the risk associated with different support designs due to gravity-induced rockfalls in underground narrow tabular stopes. The primary objective of the software is to determine the risk-cost associated with each support design (scenario) for typical narrow tabular conventional mining stopes. Similar to the methodology in Chapter 4, this calculation is achieved through calculating the:

- Annual cost of each support system (Risk Mitigation Cost),
- Annual rockfalls (dimensions) expected under each support system,
- Annual losses incurred from injuries resulting from expected rockfalls under each support system,
- Annual losses incurred from stope damage resulting from expected rockfalls under each support system,
- Reserve losses incurred from re-establishing or abandoning of panels,
- Net present cost of the sum of the risk mitigation and resulting losses for each support system over the period of interest.

Even though the development of this software is not yet complete at this stage, in order to illustrate the application of this methodology, it was considered appropriate and convenient to make use of this software in this dissertation. In this chapter, a demonstrative risk cost analysis for two support designs applicable to Mponeng Mine conventional operation is carried out. First, the two support systems are described, and then the inputs into the software explained. The discussion of the results then follows.

6.1 Support systems

The two support systems used in this analysis are, Scenario A (Figure 6-1) and Scenario B (Figure 6-2). Scenario A consists of 1.5m x 1m camlock props in the face area and 2m x 2m spaced 1.4m long pre-stressed elongates in the back area. The gully consists of timber composite packs 2m apart skin to skin along the gully shoulders and 1.4m split sets on 1m x 1m rectangular pattern. Note that no back break support has been included in this pattern.

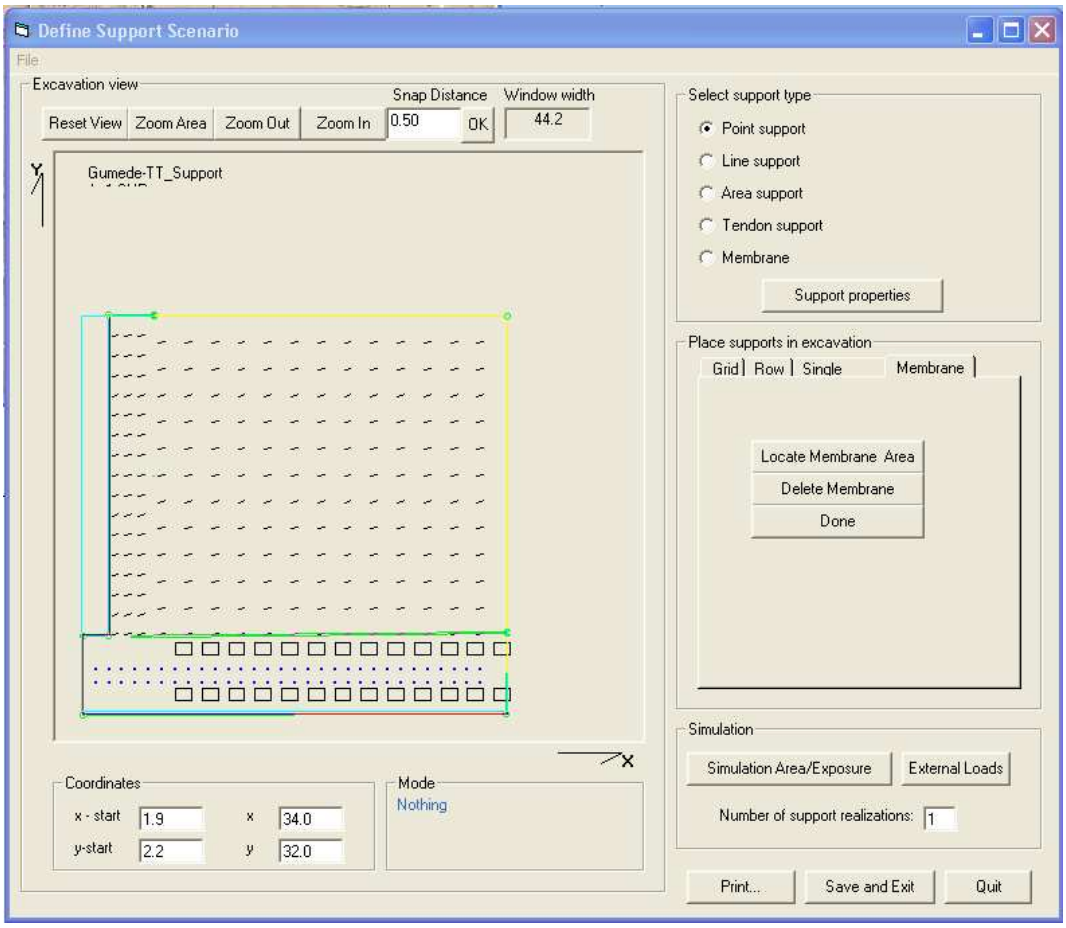


Figure 6-1: Scenario A support

Scenario B support consists of 1.5m x 1m camlock props in the face area. The back area consists of 1.5m x 1m spaced 1.4m long pre-stressed elongates and pre-stressed timber composite packs at 5m x 3m spacing. The gully consists of timber composite packs 2m apart skin to skin along the gully shoulders and 1.4m split sets on 1m x 1m rectangular pattern.

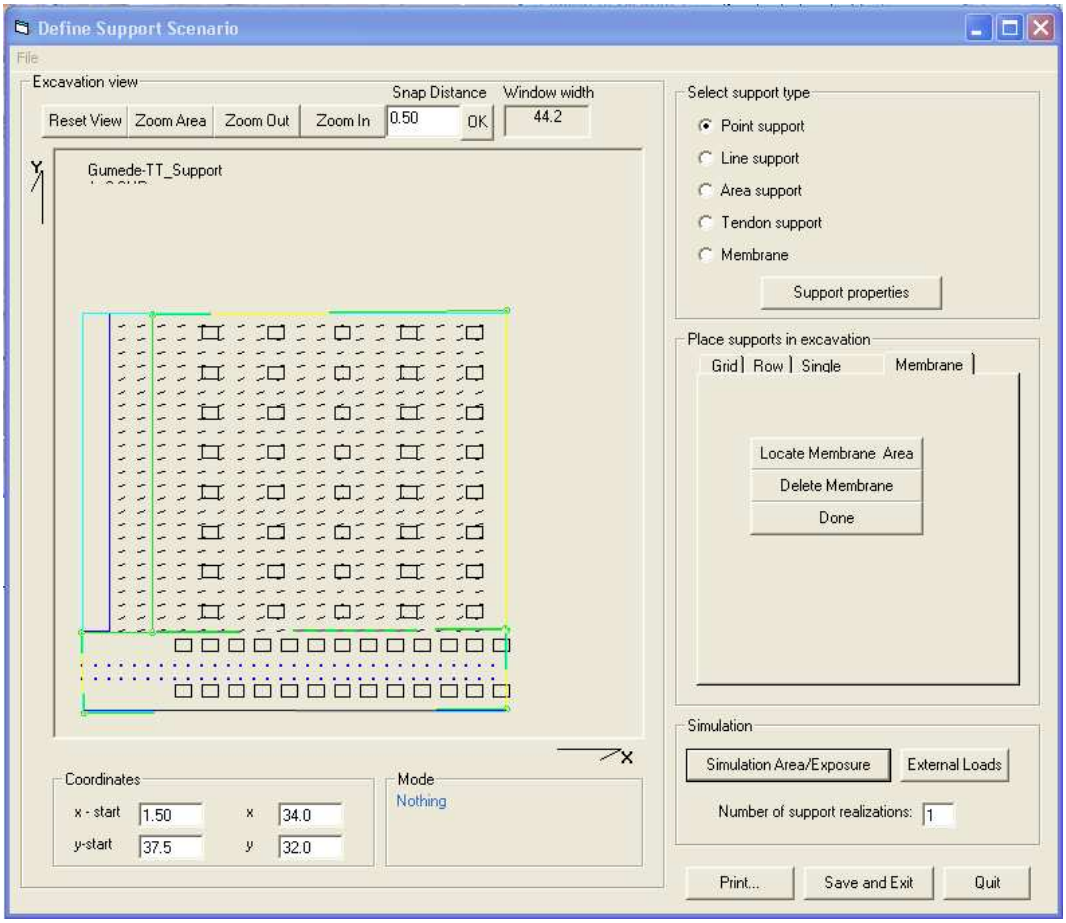


Figure 6-2: Scenario B support

It is necessary to emphasize that the risk cost analysis is carried out to compare the net present cost of support designs and the associated potential rockfall consequences of one scenario against another over a period of mining. A scenario is defined by the support design in a panel and the potential rockfall occurrences in the panel. Hence, each scenario has its own support design, and potential rockfall occurrences. Therefore, the risk cost can be determined for any single panel/scenario.

6.2 Analysis of rockfalls

Only 28 FOG records were available in the mine's rockfall reports for the 2007- 2008 period, this suggesting that the records are incomplete. A complete record of falls of ground would be ideal for a back analysis of risk cost of the support used on the mine in the same period. It however is inapplicable in a study in which the risk cost effectiveness of different proposed support scenarios is being compared, the reason being that each support system stabilises the rock differently, hence different potential rockfall distributions are expected under each scenario. JBlock software was therefore used to statistically simulate potential keyblocks that could occur in an area traversed by joints. The effect of the different support systems was then evaluated by obtaining the number of unstable blocks falling out after support is installed. This provides a potential rockfall distribution for each support type.

`

6.2.1 JBlock input parameters

No joint data was available on the mine hence data from a neighbouring mine with a similar geological environment was used. The joint input parameters are listed in Table 6-1. The specific gravity for the blocks is taken to be 2700 kgm^{-3}. The mean friction angle for all the joints has been set at 30° with a standard deviation of 10°.

Table 6-1: JBlock joint input parameters

Joint Set	Dip	Dip Direction	Spacing*			Length		
			Mean	Min	Max	Mean	Min	Max
J1	77	215	0.9	0.2	3.0	2.2	1.5	3.0
J2	50	266	0.8	0.2	2.0	3.5	1.5	10
J3	60	22	1.0	0.3	10	3.8	1.5	15
J4	62	78	1.0	0.1	10	3.4	1.5	10
J5	89	155	1.0	1.0	10	3.0	1.5	8
J6	63	117	0.3	0.3	10	2.9	1.5	7

Using this joint data, JBlock generates keyblocks in the excavation geometry. The geometry, as shown in Figure 6-3 has the same dimensions in Scenario A as B. After the generation of the key blocks, support elements are introduced into this geometry based on the patterns in Figure 6-1 and Figure 6-2. JBlock then evaluates the effectiveness of each support system in preventing rockfalls.

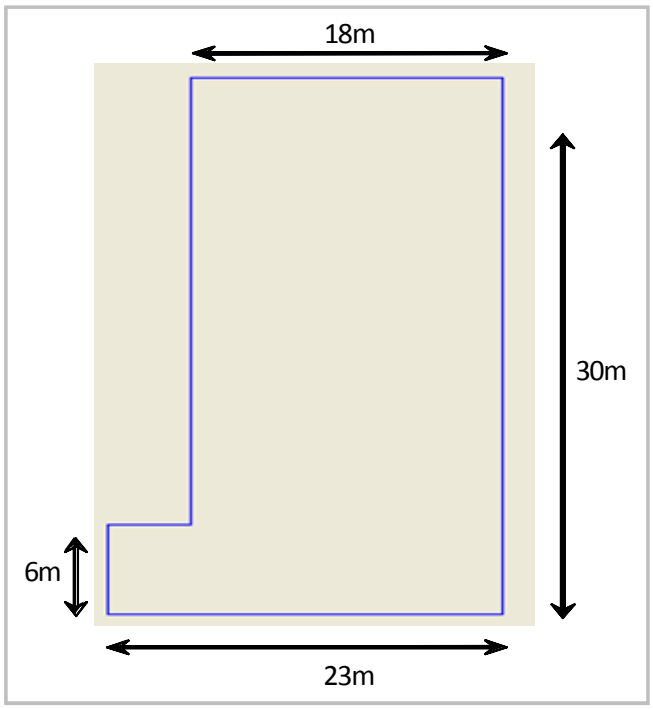


Figure 6-3: Excavation geometry

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6.2.2 JBlock results

In Table 6-2 comparative statistics from the JBlock simulation are presented. Though a larger area was simulated in B, A has a higher percentage of failed key blocks (61%). A has a larger average failure volume, 122m^3, whereas B's average is 53m^3. Figure 6-4 and Figure 6-5 show the distribution of the volumes of the rockfalls generated for A and B respectively. This distribution was applied in the risk cost software.

Table 6-2: JBlock statistics

	Scenario A	Scenario B
Area simulated (m^2)	9120	9420
Percentage of failed keyblocks	61%	50%
Average rockfalls per 1000m^2 exposed	2141	1810
Average failure volume(m^3)	122	53

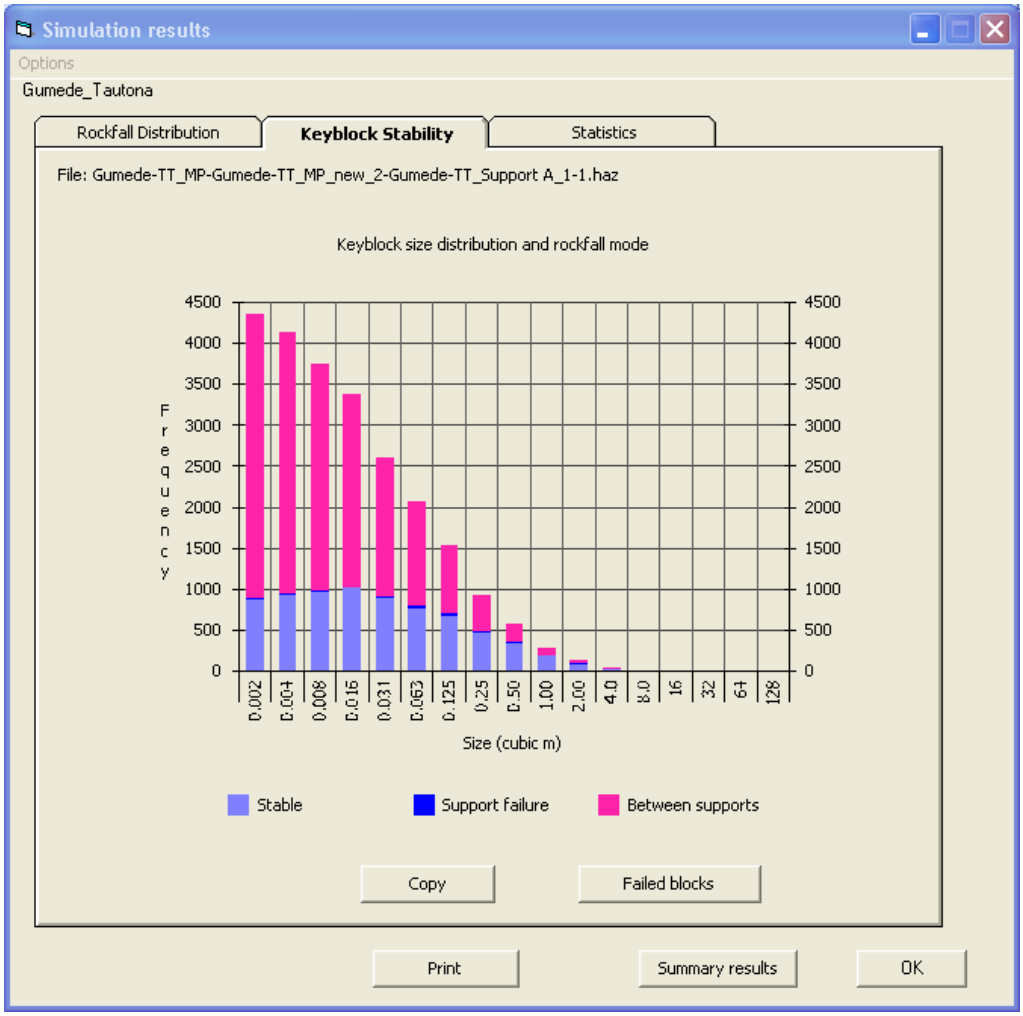


Figure 6-4: Scenario A rockfalls

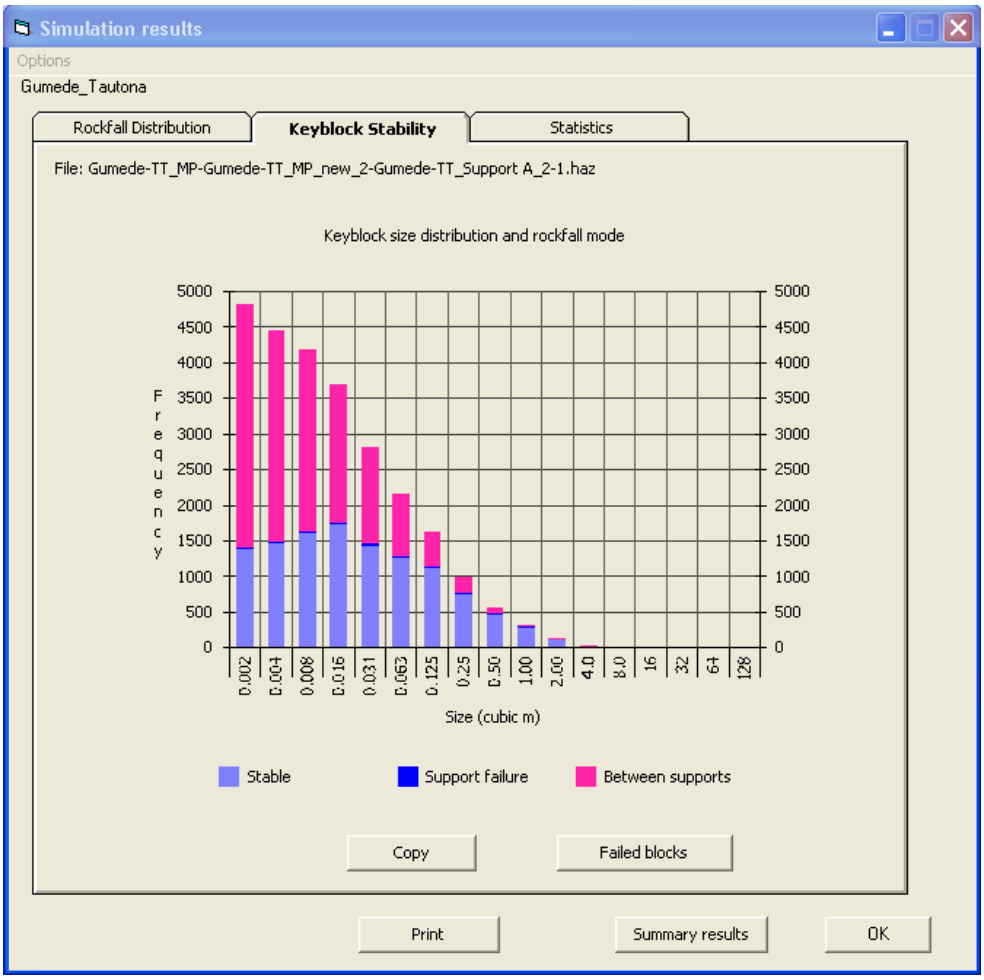


Figure 6-5: Scenario B rockfalls

6.3 Risk-cost software inputs

In the model, the Scenario Comparison form (Figure 6-6) is the main user interface. For each scenario under comparison the support pattern and resulting rockfall data should be defined. The support pattern is input directly from the proposed design or through JBlock. Rockfall data will be either historic rockfall data for the area (or similar areas) or JBlock generated rockfall data (using joint properties of the area). For each scenario, a form (Figure 6-7) issued to input this data.

The parameters, dimensions and strategies input tabs (Figure 6-6) allows input of data that is applicable to all scenarios under comparison. These tabs will be discussed in the sections that follow.

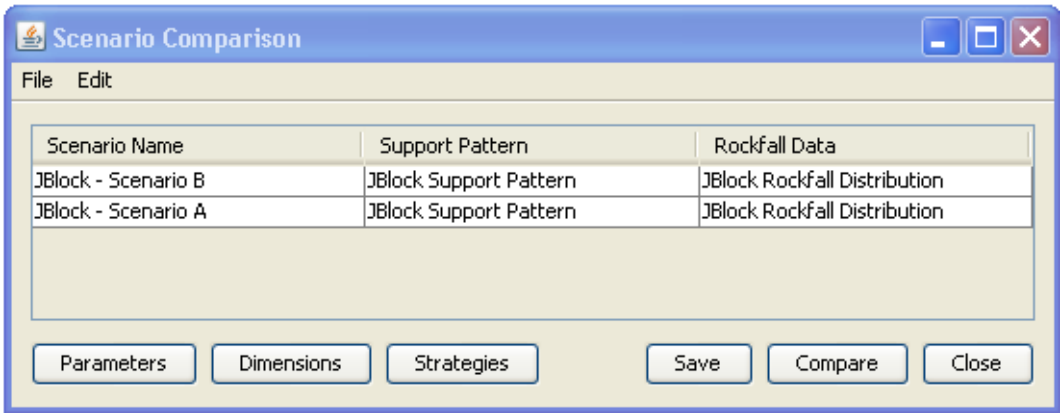


Figure 6-6: Scenario comparison form

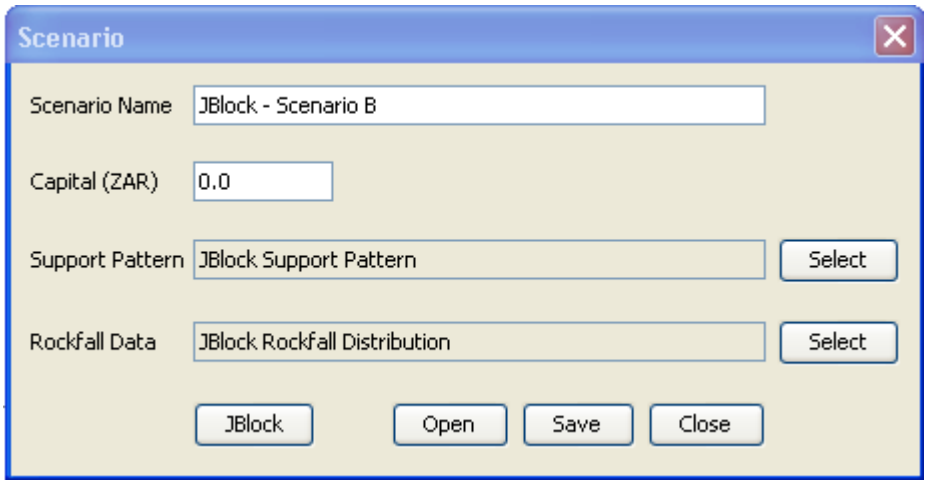


Figure 6-7: Individual scenario form

6.3.1 Support pattern

Figure 6-1 and Figure 6-2 illustrate the support patterns employed in this analysis. The unit cost of each type of support (obtained from Table 5-2) is input into the software database through Figure 6-11. An identity code uniquely identifies each support type in JBlock and the database. When inputting a support pattern in the Support Pattern form (Figure 6-8), selecting the support file under the JBlock tab will give the quantity and cost of the standard support in the excavation geometry. The re-supporting pattern is captured under a separate tab (Figure 6-10). The cost per unit or area is calculated automatically.

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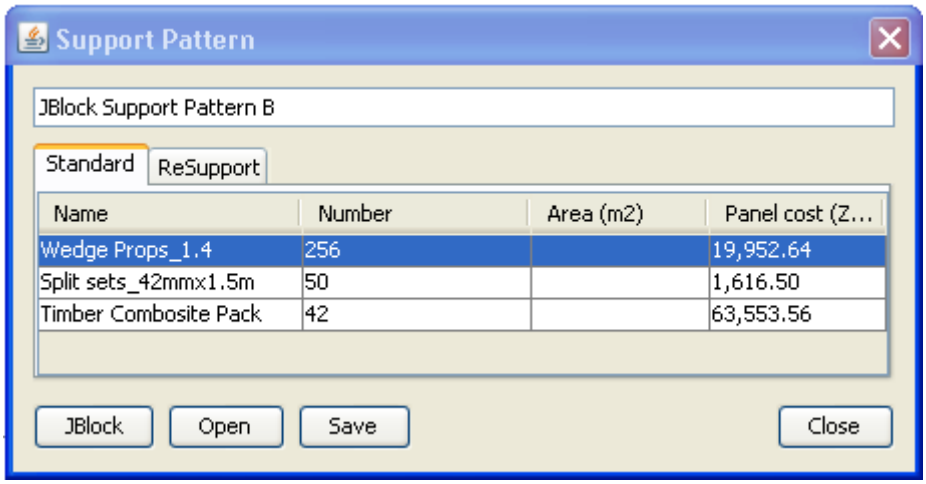


Figure 6-8: Scenario B support

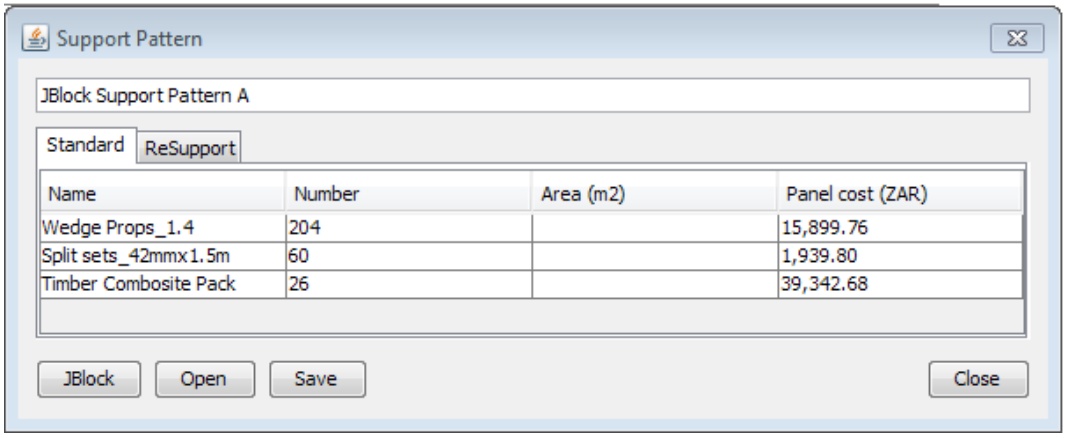


Figure 6-9: Scenario A support

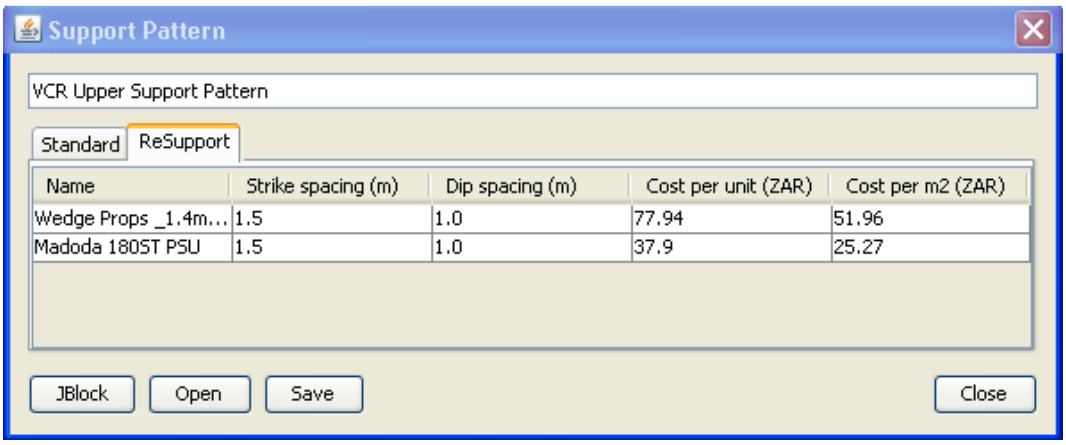


Figure 6-10: Re-support pattern form

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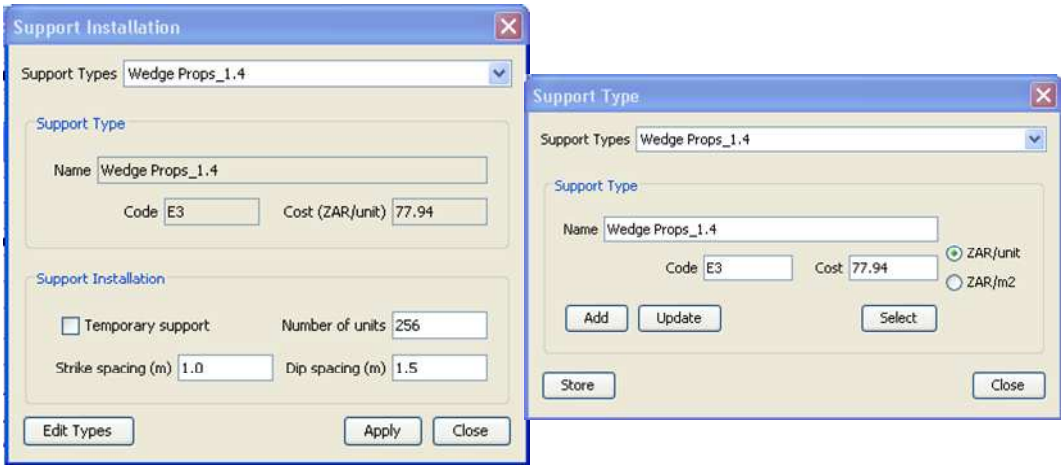


Figure 6-11: Support quantity and cost

6.3.2 Rockfall data

As discussed in section 6.2, JBlock is used to generate rockfall data for each support scenario. The JBlock file is input into the risk-cost software where it is binned according to the length of the rockfalls along the face. Figure 6-12 and Figure 6-13 show the dimension bins and the frequency of rockfalls in each bin for scenarios A and B respectively.

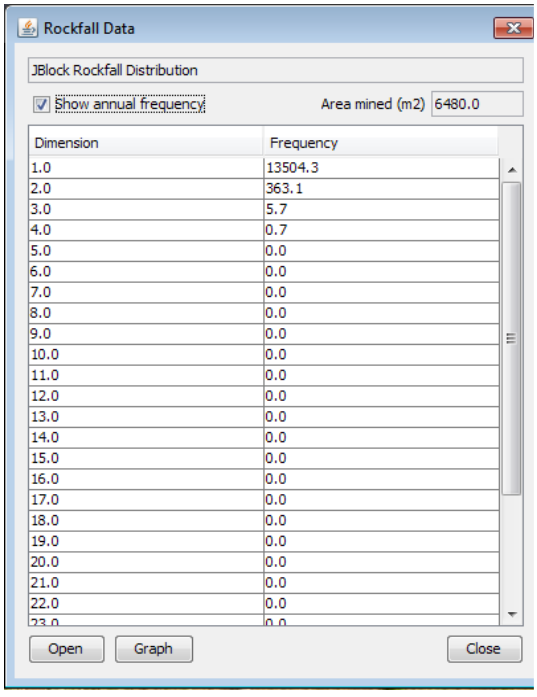


Figure 6-12: **Scenario A bins**

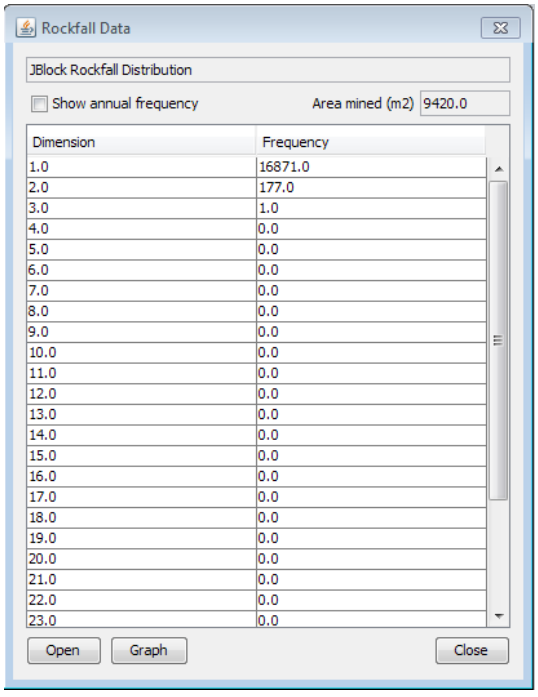


Figure 6-13: Scenario B bins

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6.3.3 Parameters form

The parameters form (Figure 6-14) consists of tabs that define the Mining, Exposure, Injuries and Costing parameters.

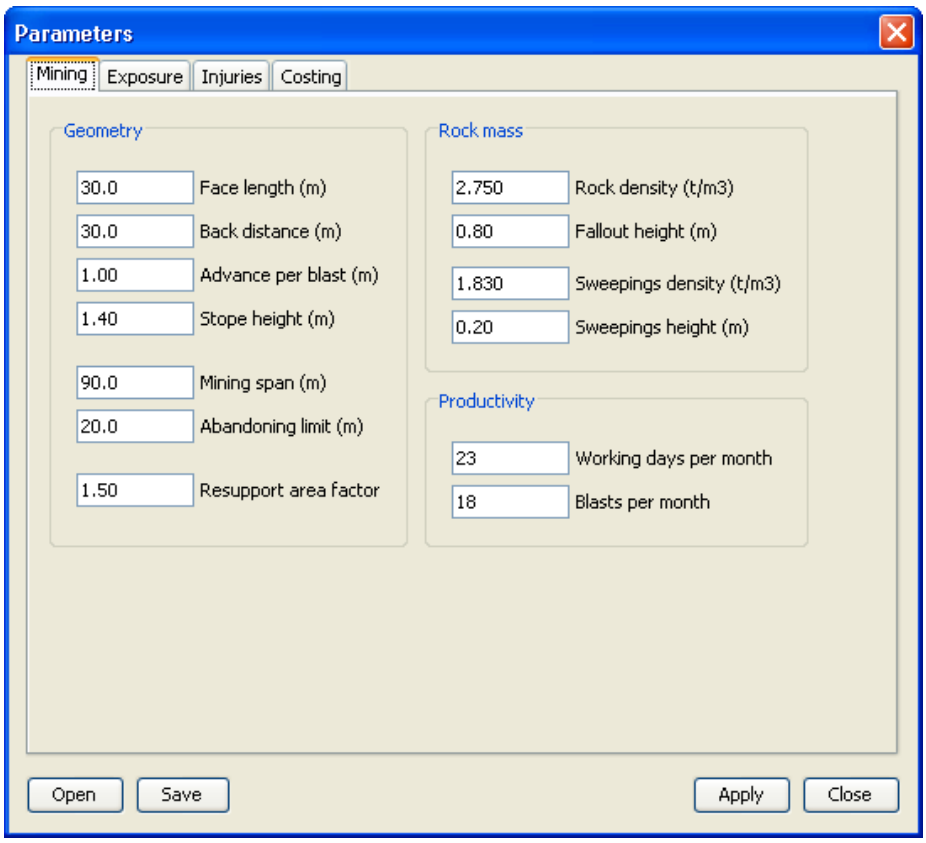


Figure 6-14: Parameters

6.3.3.1. Mining tab

The mining tab comprises of the subsections: Geometry and Rock mass, and Productivity.

Geometry and Rock Mass: The stope geometry and rock mass are defined in this section. The geometry and rock mass parameters are illustrated in Figure 6-15.

Productivity: 18 blasts are taken on average in a 23-day month. These values were obtained from mining personnel on the operation.

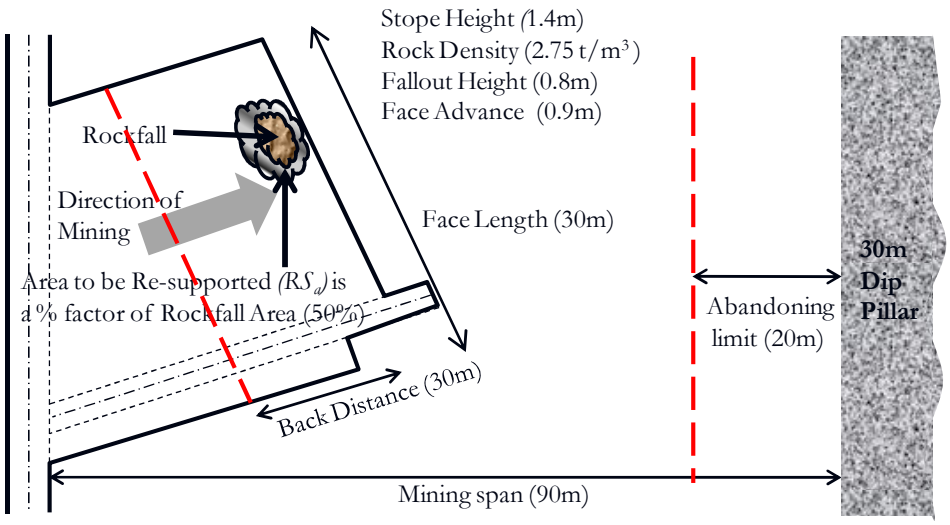


Figure 6-15: Mining parameters

6.3.3.2. Exposure tab

This tab as illustrated in Figure 6-16 comprises of subsections: Area, Personnel and Scaling, and Equipment.

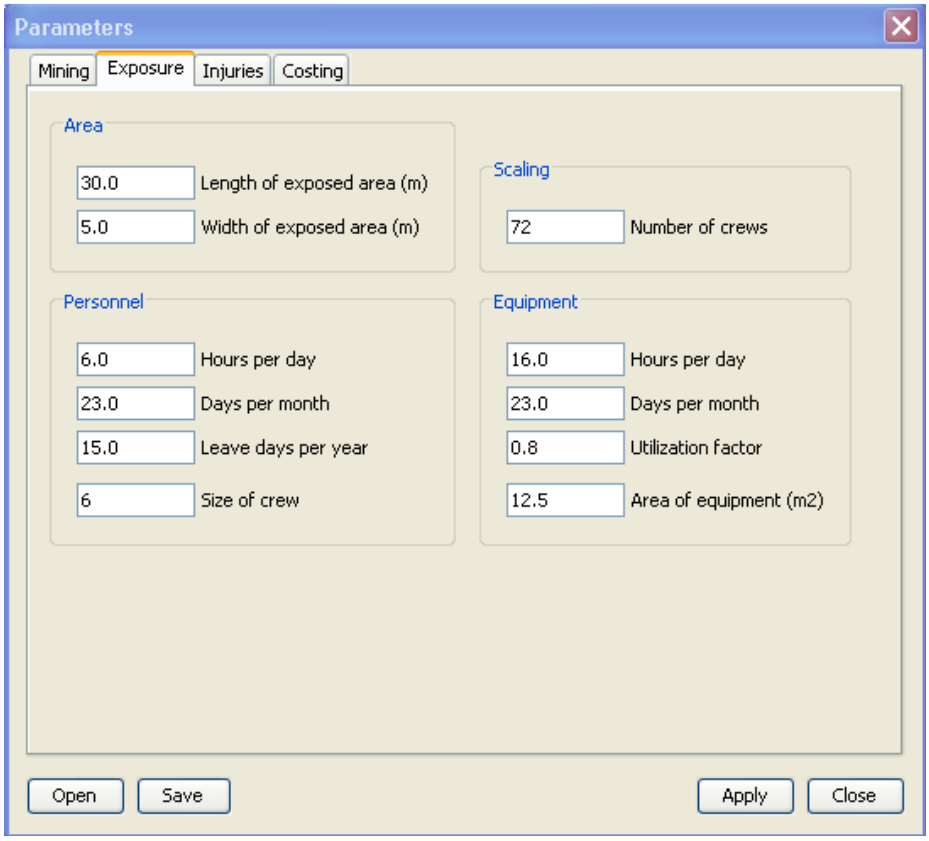


Figure 6-16: Exposure tab

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Area: This is the face area in which personnel are working, hence exposed to FOGs. The Length is equal to the face length, 30m. The width is the maximum distance between the backfill and face after the blast which is 5.0m.

Personnel and Scaling: The panel crew of 6 work in this area an average of 6hrs per day, 23 days a month with an average of 15days leave per annum. A total of 72 crewswork in this Ground Control District.

Equipment: While the values shown in Figure 6-16are the typical availability and utilisation figures, the software has not yet been developed to calculate equipment losses.

6.3.3.3. Injury tab

This tab as illustrated in Figure 6-17 comprises of the following subsections: proportions, days lost, investigation, SIMRAC levies, medical costs, wages and compensation.

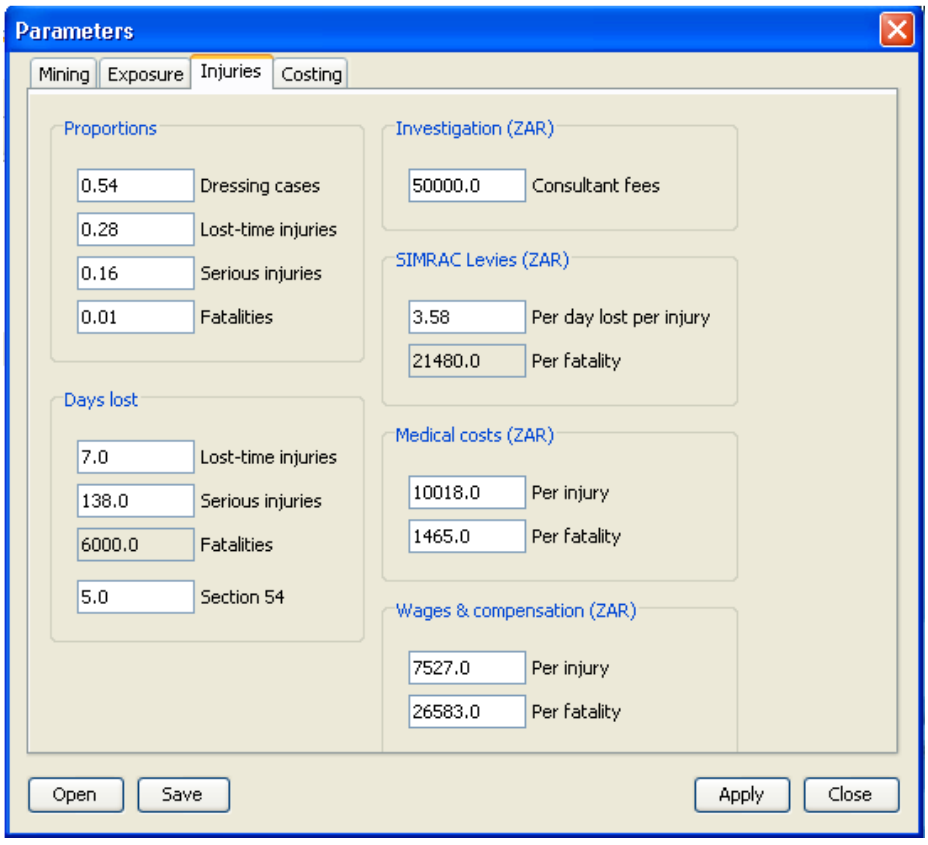


Figure 6-17: Injuries tab

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Proportions: The rockfall related (gravity) accident statistics and classification of the injuries were presented in Chapter 5. Based on this information and data on reportable accidents obtained from Marx (1996), typical average days lost per category were determined to be:

- Dressing case, 0 days
- Lost-time injury , 7 days
- Serious injury , 138 days
- Fatality, 6000 days

For fatalities, the average operations shut down time, Section 54 was 5 days.

Investigation: The costs of hiring external consultants during enquiries were not available on the operation, hence the figure used was that obtained from workshops.

SIMRAC Levies: The levy has a Safety Risk and a Health Risk component. The software however, adopts only Safety Risk as it is the component that influences the changes in the levy penalties payable when rock fall accidents occur. The current value is R 3.58 per day lost (Mine Health and Safety Council). A fatality is 6000 days lost hence the levy is R 21,480.

Medical Costs: No medical costs were available from the operation hence the costs obtained from Impala Hospital were used. The figures in Figure 6-17 are an average for 2008 medical bills.

Wages and Compensation: No costs were available hence results by Marx (1996) were escalated to present day values and averaged to a cost per incident.

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6.3.3.4. Costs tab

This tab as illustrated in Figure 6-18 comprises of cleaning up, sweepings, profit loss, and net present cost inputs. Calculation of these inputs was discussed in section 5.1.2.

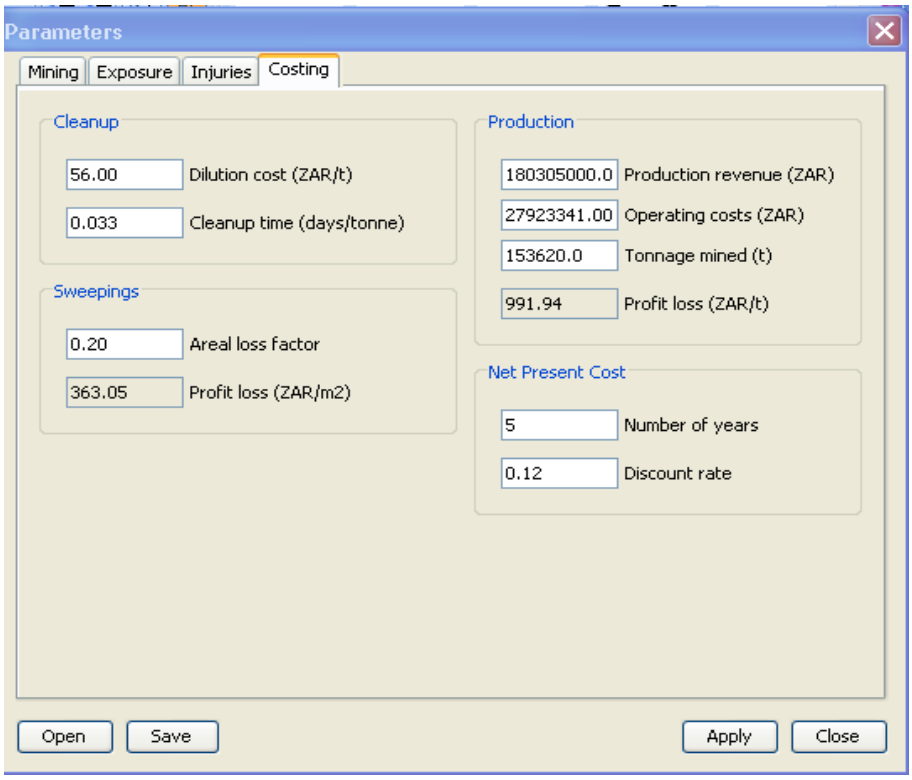


Figure 6-18: Costs tab

6.4 Remedial strategies

The software allows the user to define suitable remediation strategies. Figure 6-19shows the strategies form. The user can define minimum and maximum dimensions that apply to the rockfall classes and select the corresponding remedial action to be employed by checking the appropriate box. Where the user checks re-establishing, more information is required to be input under the re-establishing tab. Figure 6-20 illustrates the re-establishing tab. The information required here has been described in section 4.5.2. Where re-establishing is not applied, null values are applied.

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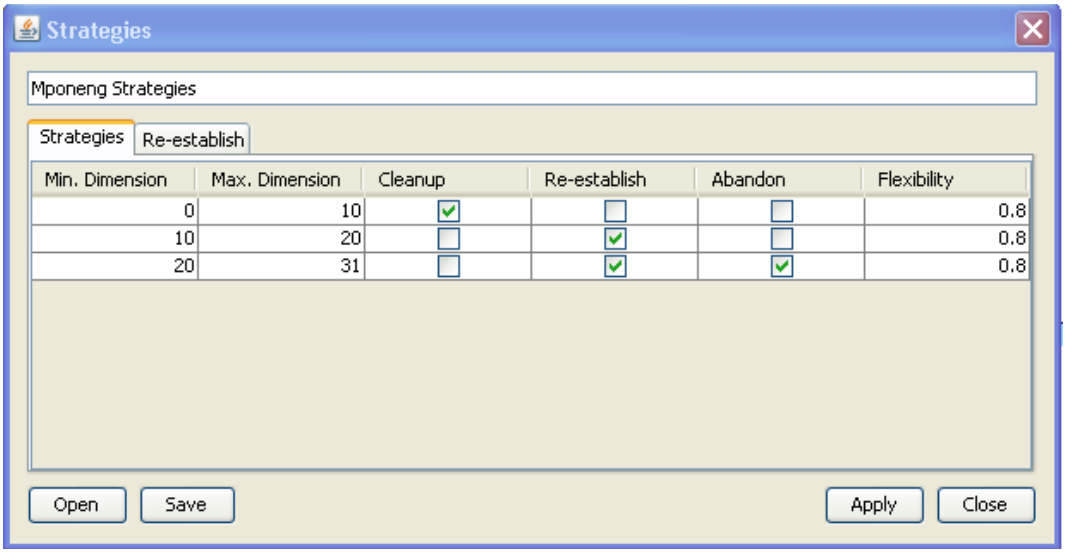


Figure 6-19: **Strategies form**

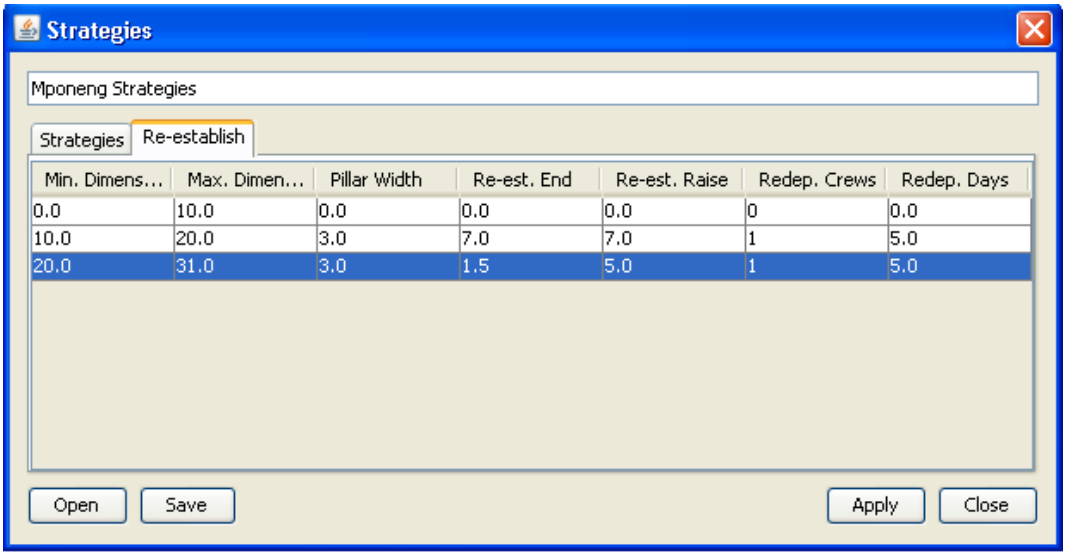


Figure 6-20: Re-establishing

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6.5 Risk-cost results

The costs of risk mitigation, injury, damage to excavation and reserve losses and the NPV for the two scenarios (Scenario A and B) are output. These are determined for a single panel, as shown in Figure 6-21, or for the whole Ground Control District (GCD) as shown in Figure 6-22. Injury costs include the cost of temporary mine closure, medical, wages and compensation, investigation and SIMRAC levies. Damage costs include production loss, re-establishment of panels, re-deployment of personnel, rehabilitation (re-supporting), clean up and dilution costs.

The factors not included in the injury or damage costs were either too complex to quantify or no meaningful data were obtained to quantify them. Due to these limitations or the assumptions made in the methodology, the costs presented could be somewhat understated. Therefore, these figures can be taken to be lower bound estimates of the cost.

6.5.1 Risk mitigation

Scenario A is a design with elongates only in the back area. Scenario B on the other hand is composed of a more dense elongate pattern and timber packs in the back area. Clearly, it is expected that type A support will cost less than type B.

Table 6-3: Comparison of support types

	Support type A	Support type B
Face	1.5m x 1.0m camlock props	1.5m x 1.0m camlock props
Back area	No timber packs. Elongates @ 2.0m x 2.0m spacing.	Timber elongates @ 5m x 3m spacing. Elongates @ 1.5m x 1.0m spacing
Gully	2m on strike timber pack spacing. Split sets @ 1m x 1m grid spacing	2m on strike timber pack spacing. Split sets @ 1m x 1m grid spacing

The risk mitigation cost in scenario A is R 409 199.20per panel (R 40million in GCD) whereas for scenario B it is R 976 369.59 per panel (R 97.6million in GCD). It is therefore apparent that there is a cost saving of R 567 170.39 per panel and R 57.6million in the GCD if scenario A is used.

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6.5.2 Injury cost

The expected injury cost is R 193 383.75per panel (R 19,3million in GCD) for scenario A and R 121 144.72 (R 12,1million in GCD) for scenario B. Installing more support in a panel (Scenario B) therefore reduces the expected injury cost by R 72 239.03. In a GCD, a saving of approximately R 7.2million is realised.

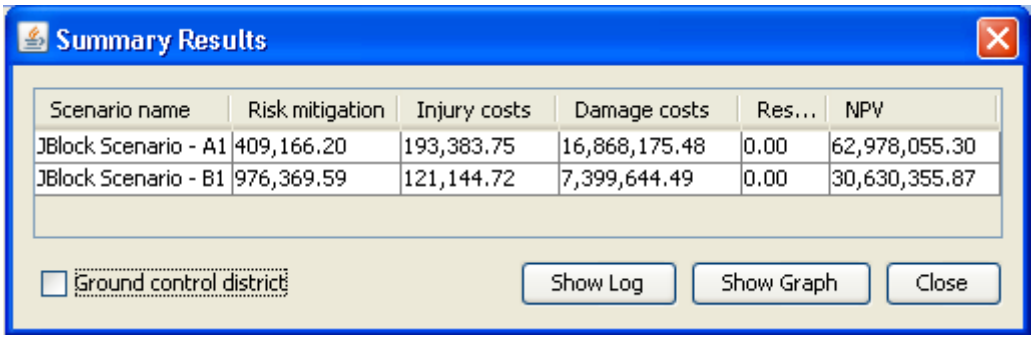


Figure 6-21: **Summary results – single panel**



Figure 6-22: Summary results – ground control district

6.5.3 Damage to excavation cost

The expected damage to the excavation cost is R 16 868 175.48 per panel (R 1,69 billion in GCD) for scenario A and R 7 399 644.49 per panel (R 0,734 billion in GCD) for scenario B. An inadequately supported panel (scenario A) results in damage worth R 9 468 530.99 (R 0.96 billion in GCD) more than a panel with more support (scenario B).

6.5.4 NPV

The NPV of this design over a 5 year period at a discount rate of 12% is R 62,98million per panel (R 6,2 billion in GCD) for scenario A. For scenario B, it is R 30,63million per panel (R 3, 06 billion in GCD). The NPV for the panel and GCD for both scenarios are also plotted against the risk mitigation cost as a Net Present Cost (NPC) as shown in

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Figure 6-23. This chart illustrated that where minimal investment is made in risk mitigation(scenario A), the NPC of the risk mitigation and consequences of rockfalls is very high. Where increased investment is made (scenario B) the resulting NPC is much less.

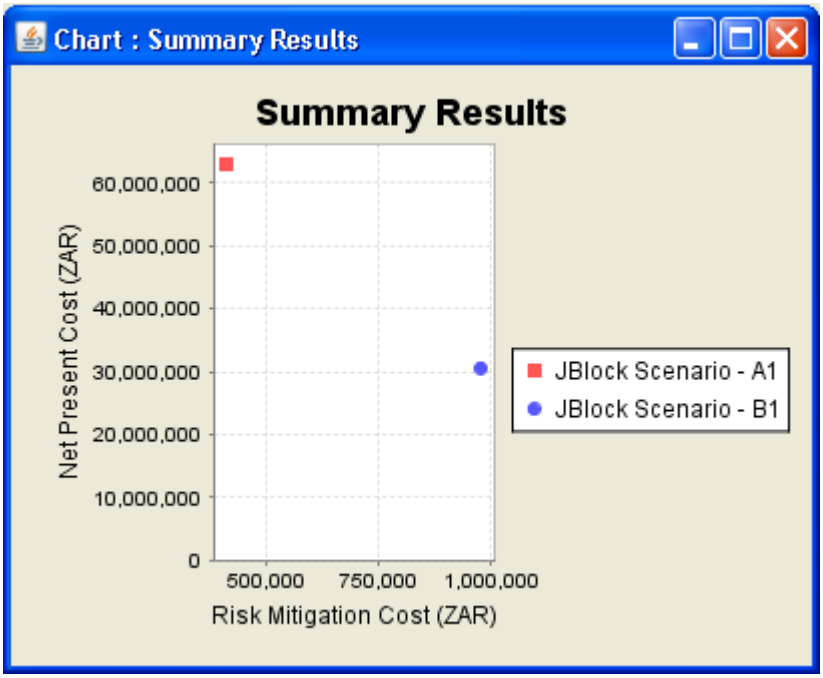


Figure 6-23: Results chart

Where an analysis of several scenarios is carried out, the results can then be plotted on this chart for comparison purposes. The chart provides a simplified means of illustrating the risk-cost associated with each support strategy. It provides a clear and easily understood illustration of risk for management and other stakeholders.

6.5.5 Individual simulation of rockfall costs

For each scenario, individual rockfalls are simulated. The detail of each simulation is shown in the log sheet under the 'Show Log' tab in Figure 6-21 and Figure 6-22. Figure 6-24 shows four rockfalls that fall under the 1m bin. The costs include clean up and dilution at R 123 per rockfall and production loss at R 6509 per rockfall, this gives a total cost of R 6632.7 per rockfall. For the four rockfalls, the total cost is R 26,530.83.

These results are comparable to those obtained in section 5.1.3. Differences will occur mainly because the calculations in the case study are based on measured dimensions of the rockfall and not the bins as applied in the software.

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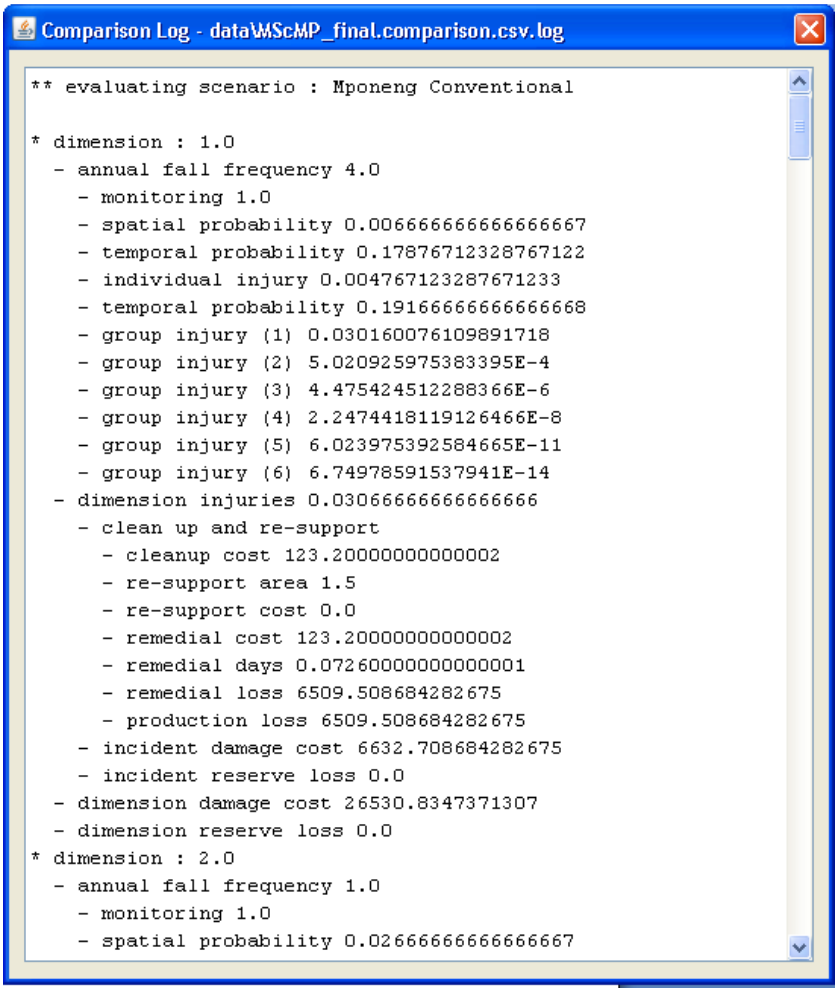


Figure 6-24: Results log

6.5.6 Personnel risk - FN charts

The software also represents personnel fatality risk by plotting an FN chart for the panel and the GCD, Figure 6-25, for each scenario. At this stage the software is still being developed hence the terms showing the level of risk have not yet been perfected.

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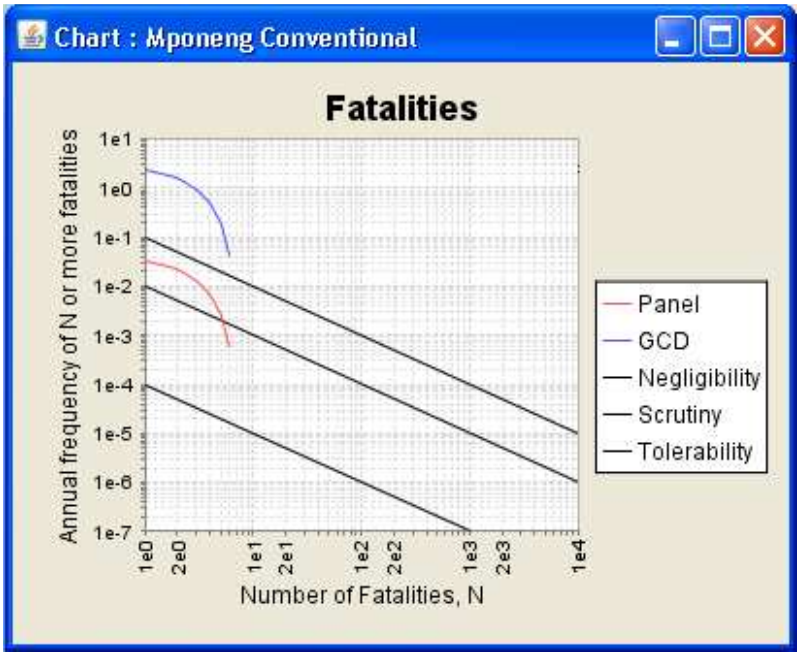


Figure 6-25: FN plot

6.6 Summary

In this chapter, a risk cost evaluation comparing two support designs in the same mining environment was conducted. This evaluation estimated the risk cost associated with each design therefore allowing more informed decision-making. This evaluation further demonstrated how valuable and applicable this methodology is to the South African narrow tabular mining environment. The methodology provides a useful tool for planning and optimisation of support design, which is an essential component for mining operations. This tool has not been available in the mining industry up to this time. The software, which is an interface for this methodology, is still under development and will be refined as more beta testing is conducted.

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7 Conclusion and recommendations

There are few published or documented studies on the quantification of financial consequences associated with rockfalls in South African narrow tabular mining operations. The methodology and results presented in this dissertation are believed to be the most comprehensive work on this subject to date. The development of this methodology into software also means that the results of this research will be beneficial to the South African narrow tabular underground mining operations when the software enters the public domain. From this work, the following conclusions can be drawn.

7.1. Consequences of rockfalls

The direct consequences of rockfalls identified are injuries and fatalities, damage to equipment and machinery, and damage to excavations. However, the research further identified the associated resulting indirect consequences.

- Indirect consequences resulting from injuries and fatalities include temporary mine closure, medical and rescue operation costs, wages and compensation, investigation costs, re-training costs, SIMRAC levies, legal costs, insurance premiums, industrial action and stakeholder resistance.
- Indirect consequences resulting from damage to equipment and machinery include production loss due to equipment downtime, cost of redeploying machine and machinery, repairing or replacement costs.
- Indirect consequences resulting from damage to the excavation include loss of reserves, production, re-establishment of stopes and access ways, cleanup and dilution, re-supporting requirements, redeployment, insurance premiums and possible stakeholder resistance.
- Some rockfalls will result in severe consequences whereas others will result in minor consequences. For example, a small rockfall can cause a minor injury or a fatality.

7.2. Methodology to quantify cost of consequences

A methodology to quantify the cost of consequences was developed. This methodology focuses on the determination of costs associated with personnel injuries and fatalities, and damage to excavations, as summarised in the bullet points below. Costs associated with equipment damage could not be determined due to lack of information on how such costs are handled.

- Personnel injuries and fatalities vary depending on personnel exposure and spatial coincidence of the personnel. Where a FOG occurs and the individual or crew is in

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the same place as the rockfall, there will be an injury or fatality. If the individual is not in the exact same space then it is a near miss.

- Costs including mine closure, SIMRAC levy, and investigations are calculated in the methodology.
- Costs including medicals, wages and compensation are extrapolated from the work done by Marx (1996). Mines affiliated to Rand Mutual Assurance (RMA) incur only insurance premium costs. This insurance covers the medical, wages and compensation costs of the injured.
- Damage to excavation consequences vary in relation to the class of the rockfall. Classification of the rockfalls is in accordance with the lengths and the remedial strategy applied.
- Small rockfalls, class 1, result in cleanup and dilution costs.
- Class 2 rockfalls result in cleanup and dilution, re-supporting and costs resulting from loss of production.
- Class 3 rockfalls result in sweepings loss, reserve losses due to pillars left and production losses due to re-establishing,
- Class 4 rockfalls result in the re-establishment or abandoning of a panel. The consequences include sweepings loss, production loss due to re-establishing or abandoning, redeployment of personnel, and reserve losses due to re-establishing or abandoning.
- Input data required for use in the calculations are obtainable from the Mining, Rock Engineering, Safety, Finance and Survey Departments on an operation.

7.3. Application of methodology

The proposed methodology was further developed and tested in two case studies, AngloGold Ashanti (Mponeng Mine) and Impala Platinum (12 Shaft). These two operations were selected in the platinum and gold mining sectors as they maintain more comprehensive rockfall databases. In the dissertation only the damage to excavation consequences are determined in the application and the following conclusions are drawn:

- In both case studies, loss of production is the major cost driver. It is R 992/t and R 1127/t for Mponeng and Impala operations respectively. This is upwards of R 1.2million for Mponeng and R 0.8million for Impala 12 Shaft for large falls above 10m in length.
- Reserve losses are the next largest cost driver. In both cases these losses were upwards of R 100 000 for a 10m rockfall.

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- Sweepings losses are R 545/m^2andR 727/$m^2$for Mponeng and 12 Shaft respectively.
- Dilution costs are R 56/t and R 97/t for Mponeng and 12 Shaft respectively.
- Re-supporting costs are proportional to the area to be re-supported, the density factor, and the cost of the support installed. The two operations employed similar re-supporting strategies, hence on both sites for example, a 9m rockfall would cost R 6 500 to re-support using pre-stressed elongates at 1m x 1m spacing for both.
- Falls less than 10m in length cost a maximum R 50 000 at Mponeng, and R 90 000 at Impala.
- Rockfalls larger than 10m were in the range of R 1million to R 2.5million in both case studies. This suggests that any substandard supporting practice or inadequate design will cost the mine severely should there be a fall of ground greater than 10m in length.

7.4. Illustration of value creation through safety spending

Some managers appear to believe that cost-cutting measures on a mine should include minimising as much as possible on any expenditure including ground support initiatives. Others are reluctant to opt for new strategies proposed as being more adequate by the Rock Engineer. However, the risk-cost approach described in this dissertation has clearly indicated the value created by investing in safety spending. The following conclusions are drawn from the results of the risk-cost analysis:

- Current industry practice in selecting support is to look only at the risk mitigation cost whereby the cheaper option is usually selected. From the analysis, scenario A had a lower risk mitigation cost, R 409 199.20 per panel, and whereas scenario B was R 976 369.59 per panel. Based on the risk mitigation cost criteria, a potential cost saving of R 567 170.39 per panel could have been realised in selecting scenario A.
- Analysis of potential rockfalls in the two scenarios using JBlock showed that more rockfalls are experienced in scenario A compared to scenario B. Scenario A therefore carries a higher rockfall risk. The consequences of rockfalls are discussed above.
- As there are people working in the panel where the rockfalls occur, injuries are anticipated. In scenario A, the expected injury cost is R 193 383.75 per panel and in scenario B, R 121 144.72 per panel. Investing safety spending, as done in scenario B, therefore reduces the expected injury cost by R 72 239.03 in a panel. In a Ground Control District (GCD), a saving of approximately R 7.2million would be realised.

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- The damage to excavation costs for scenario A and B are R 16 868 175.48 per panel and R 7 399 644.49 per panel respectively. This indicates that in a panel with minimal support, scenario A, R 9 468 530.99 is lost due to excavation damage and remedial strategies in the panel.
- The total costs of risk mitigation and the expected consequences over the period of mining for each panel are summed and a Net Present Cost (NPC) is determined. NPC provides a simplified means of expressing the overall risk-cost associated with each design.
- As the risk-cost is expressed explicitly in financial terms, management and other stakeholders can easily understand the risks involved with each support strategy. Scenario A NPC is R 62,98million per panel and scenario B is R 30,63million per panel. Scenario A therefore carries higher risk than scenario B.
- In light of value generated from safety spending, based on the risk mitigation criteria alone, scenario A would have been the favourable option. However, an analysis of the effect of the expected consequences showed that scenario B carries less risk-cost. It is therefore worthwhile to carry out a risk-cost study, although, in order to do this the cost of the consequences first have to be quantified.
- Management can be involved in setting the acceptable risk criteria as the risk cost is quantified and easily expressed.

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10 Appendices

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Appendix 1:

IMPALA ANNUAL REPORT 2008 (Extract)

Impala Rustenburg key statistics

		FY2008	FY2007	% change
Mining sales	Rm	20 889	17 401	20.0
Platinum		12 087	9 573	26.3
Palladium		1 173	1 068	9.8
Rhodium		5 179	4 061	27.5
Nickel		1 506	1 758	(14.3)
Other		944	941	0.3
Mining cost of sales		(7 345)	(6 643)	(10.6)
On-mine operations		(5 860)	(4 798)	(22.1)
Concentrating and smelting operations		(1 057)	(918)	(15.1)
Refining operations		(476)	(377)	(26.3)
Amortisation		(691)	(594)	(16.3)
Increase in inventory		739	44	1 579.5
Mining gross profit		13 544	10 758	25.9
Profit from metal purchased transactions	Rm	54	39	38.5
Sales of metals purchased		15 638	12 413	26.0
– IRS		14 914	12 229	22.0
– Other		724	184	293.5
Cost of metals purchased		(15 584)	(12 374)	229.0
– IRS		(14 846)	(12 226)	221.4
– Other		(738)	(148)	(398.6)
Gross profit in Implats group		13 598	10 797	25.9
Gross margin ex-mine	%	65	62	4.9
Other operating costs	Rm	(426)	(385)	(10.6)
Royalty expense		(548)	(1 624)	66.3
Sales volumes ex-mine				
Platinum	000oz	1 043.7	1 119.3	(6.8)
Palladium	000oz	426.7	446.9	(4.5)
Rhodium	000oz	101.9	110.6	(7.9)
Nickel	000t	6.7	7.1	(2.8)
Sales volumes metals purchased – IRS				
Platinum	000oz	640.7	677.3	(5.4)
Palladium	000oz	410.9	433.8	(5.3)
Rhodium	000oz	92.7	92.2	0.5
Nickel	000t	3.4	4.7	(27.7)
Prices achieved ex-mine				
Platinum	$/oz	1 588	1 190	33.4
Palladium	$/oz	381	335	13.7
Rhodium	$/oz	6 941	5 104	36.0
Nickel	$/t	30 206	31 645	(4.5)
Exchange rate achieved ex-mine	R/$	7.29	7.20	1.3
Production ex-mine				
Tonnes milled	000t	15 855	16 302	(2.7)
Platinum refined	000oz	1 044.0	1 055.3	(1.0)
Palladium refined	000oz	436.6	472.0	(7.5)
Rhodium refined	000oz	124.9	102.9	21.4
Nickel refined	000t	6.9	7.0	(1.4)
PGM refined production	000oz	1 841.1	1 872.4	(1.7)
Total cost*	Rm	7 819	6 477	(20.7)
per tonne milled	R/t	493	397	(24.2)
	$/t	68	55	(23.6)
per PGM ounce refined	R/oz	4 247	3 459	(22.7)
	$/oz	585	481	(21.6)
per platinum ounce refined	R/oz	7 489	6 138	(22.0)
	$/oz	1 031	853	(20.9)
net of revenue received for other metals	R/oz	(941)	(1 280)	(26.5)
	$/oz	(130)	(178)	(27.0)
Capital expenditure	Rm	3 415	2 098	62.8
	$m	470	292	61.0
* Includes share-based payments	Rm	985	554	(77.8)

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APPENDIX 2:

Compensation for Occupational Injuries and Diseases Act, 1993 (No. 130 of 1993)
Schedule 2: Injury

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Compensation for Occupational Injuries and Diseases Act, 1993 (No. 130 of 1993)

Schedule 2: Injury

Injury		Percentage of permanent disablement
Loss of two limbs		100
Loss of both hands, or of all fingers and both hands		100
Total loss of sight		100
Total paralysis		100
Injuries resulting in employee being permanently bedridden		100
Any injury causing permanent total disablement		100
Loss of arm at shoulder		65
Loss of arm between elbow and shoulder		65
Loss of arm at elbow		55
Loss of arm between wrist and elbow		55
Loss of hand at wrist		50
Loss of four fingers and thumb of one hand		50
Loss of four fingers		40
Loss of thumb--	both phalanges	25
	one phalanx	15
Loss of index finger--	three phalanges	10
	two phalanges	8
	one phalanx	5
Loss of middle finger--	three phalanges	8
	two phalanges	6
	one phalanx	4
Loss of ring finger--	three phalanges	6

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APPENDIX 3:

ROCKFALL MECHANISMS AND PRECURSORS

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No.	FOG TYPE	MINING SECTOR RISK (1 - 5) Most severe = 1					MECHANISM(S)	CAUSE(S)	PRECURSORS	
		TABULAR HARD ROCK	COAL	STEEP VEINS	MASSIVE	SERVICE			MEASURABLE Practicality (1, 2, 3) 1= most practical	OBSERVABLE
	Joint/ fracture release surfaces	Au 1 Bveld 3 Mn (5)	4	(5)	3	2 3 5	Slip as a result of : i) Loss of cohesion ii) Loss of clamping forces	i) Unfavourable geometry ii) Blast/seismic vibrations iii) Water ingress iv) Swelling of infill material v) Deformation breaking asperities i) Adjacent to FOG ii) Increasing distance from face? iii) Bending ?	i) Micro relative block movements (3) ii) Survey (2)	i) Geometry ii) Dripping water iii) Filtering of dust from Hwall iv) Position relative to FOG's and face.
	Joint/joint release surfaces	Au 4 Bveld 1 Mn 1	3	2	2	2				
	As above plus bedding parallel partings	Au 1 Bveld 2 Mn 2	2	5	3	1				
	Backbreak	Au 4 Bveld 4 Mn 5	5	5	4	5				
	Panel/back collapse	Au 3 Bveld 1 Mn 5	4	4	4	5				

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	Wedge failure	Au 3 Bveld 3 Mn 5	4	3	2	2				
	Curved joint failure	Au 4 Bveld 1 Mn5	5	5	3	5				
	Fall from & into weak layer in Hwall(Basal Reef shale)	Au 3 Bveld 3 Mn 5	2	3	2	3				
	Flow of fractured rock mass (WAF) Master bedding plane fault	Au 3 Bveld 4 Mn 4	4	4	2	4				
	Beam failure	Au 2 Bveld 2 Mn 2	1	4	1	3				
	Presence of stress fracturing, fall-out of slabs, sidewall, hangingwall pillars	Au 1 Bveld 3 Mn 5	3	3	2	2				
	Faults/dyke contacts	Au 1 Bveld 1 Mn1	1	1	2	3				
	Toppling from brows or previous FOG	Au 2 Bveld 2 Mn 2	3	5	4	2				
	Collapse of brow	Au 1 Bveld 1 Mn 1	1	4	1	1				
	Uncontrolled Goafing	Au 5 Bveld 5 Mn 5	1	5	5	5				
	Sloughing, steep ore bodies	Au 5	5	4	2	5				

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		Bveld 5 Mn 5								
	Failure of middling	Au 4 Bveld 3 Mn 4	2	4	4	4				
	Sill related falls	Au 4 Bveld 3 Mn 5	3	5	5	5				
	Crown/sill pillar instability	Au 5 Bveld 5 Mn 5	5	3	2	5				
	Failure of stress damaged rock, Carbon Leader	Au 1 Bveld 4 Mn 5	5	5	3	3				
	Collapse of gothic arch and guttering	Au 5 Bveld 3 Mn	3	5	4	3				
	Cross bedding	Au 3 Bveld 5 Mn 5	3	5	5	4				
	Man assisted FOG Blasted support Rigging Scrapper damaged support Decline in support performance with time and closure	Au Bveld Mn								
	Overhanging face and sidewalls high stoping width,	Au 3 Bveld 4 Mn 2	5	2	2	2				

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Sketches of each FOG type, annotated to show mechanisms are required.

What can we measure?

1. Noise
1.1 Single noise
1.2 Change of rate of noise
1.3 Seismic velocity correlated to RMR
2. Movement
2.1 Support load as a proxy for closure
2.2 Differential movement (sensors, laser scanning)
2.3 Absolute closure
2.4 Change of rate of closure
3. Electricity
3.3 Resistivity)
4. Light
5. Gas
6. Heat
7. Geophysical
8. Change in strain = stress

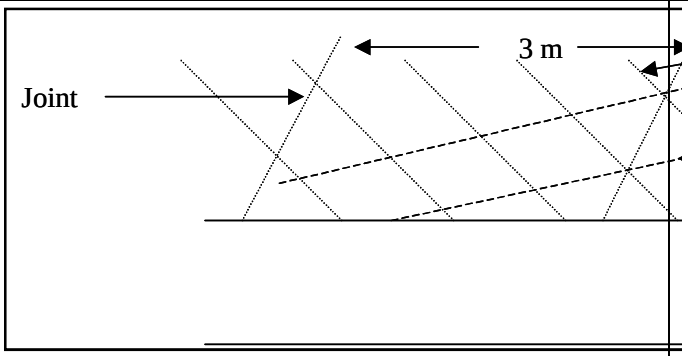
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Mechanisms	Precursors	Causes
Loss of cohesion 1.3. (1, 2, 3, 4, 5, 7, 9, 11. 12,	Noise due to rock or filling rupture. Very small differential movement with l scanner. Change of thermal image and c reflective index. Change in colour.	1. Discontinuities
Reduced friction coefficient 1.3. (1, 2, 3 ,4, 7, 1		1.1 Geology
Change of confining force 1.3. (1, 2, 3, 4, 5, 6, 15)		1.2 Induced discontinuities
Shear forces 1.3. (		1.3 Exacerbating factors
Tensile failure		
Loosening		1.3.1 Water ingress
Buckling		1.3.2 Gas
Toppling		1.3.3 Vibration
Fracturing		1.3.4 Swelling clays
Beam failure		1.3.5Closure
Movement of thrust structures (dome)		1.3.6 Dilation
		1.3.7 Weathering and oxidation
		1.3.8 Stress change
		1.3.9 Impact, application of force (barring)
		1.3.10 Uneven loading by support units
		1.3.11 Blasting
		1.3.12 Expansion and contraction
		1.3.13 Excessive span
		1.3.14 Shearing across surface
		13.1.15 Removing support

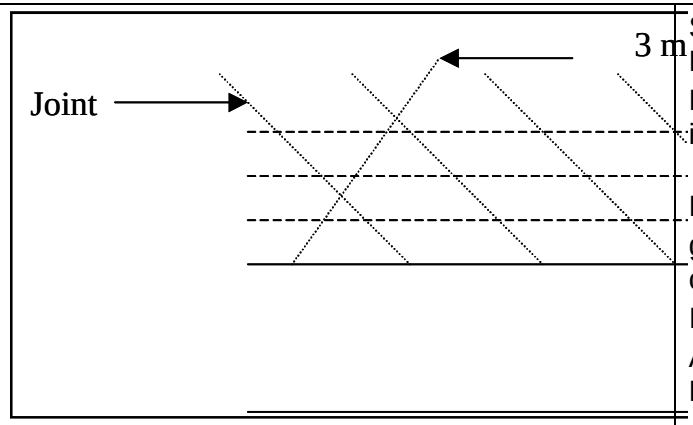
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Statement from Dave

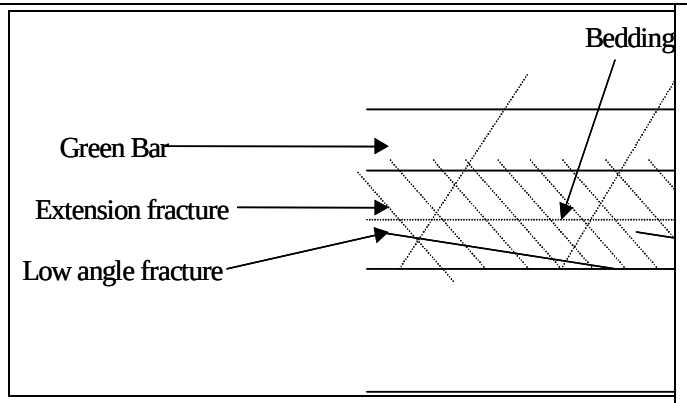
There is an assumption that there is precursive activity before falls of ground occur. This precursive activity is associated with loss of cohesion, loss of stabilising forces and loss of friction.
There is likely to be a huge variation in the type of precursive activity associated with a variety of FOG mechanisms. The trick would be to link this precursive activity to the FOG mechanisms.

	Generic geotechnical rockmass	**Type and mechanism of FOGs**	**Precursors**		**Representative ore bodies**	**Failure mechanisms**
1	The VCR Hard Lava generic is characterised by irregular bedding plane faults with frequent reef rolls. The hangingwall is normally jointed and non-bedded. Fracturing is not always developed, typically 0,05 m spacing, not always dipping towards the face. On a two-day cycle, the closure rate could be as high as 10 mm/day. Stoping width is typically 1 m - 2 m with the reef dipping at between 20º and 30º. Repeated seismicity degrades rockmass rating. **Dominant failure factors** Slope and terrace and bedding plane faulting Fractured quartzite beam, jointing in lave Pillow lavas	***Joint and stress fracture bound blocks.*** Unravelling, Wedge failure from bedding plane fault. Ramp structures (domes) **Mechanisms** Reduced friction coefficient Change of confining force Loss of cohesion	*Differential Movement* **System stiffness** in order to determine how long a precursor would be useful (seismic velocity or RMR) The stiffer the system the less the deformation before failure and v versa VCR generic is a stiff system (rock mass rating) Release of energy (acoustic)	 Schematic of the VCR Hard Lava generic showing blocky structure due to different sets of fractures together with joints.		

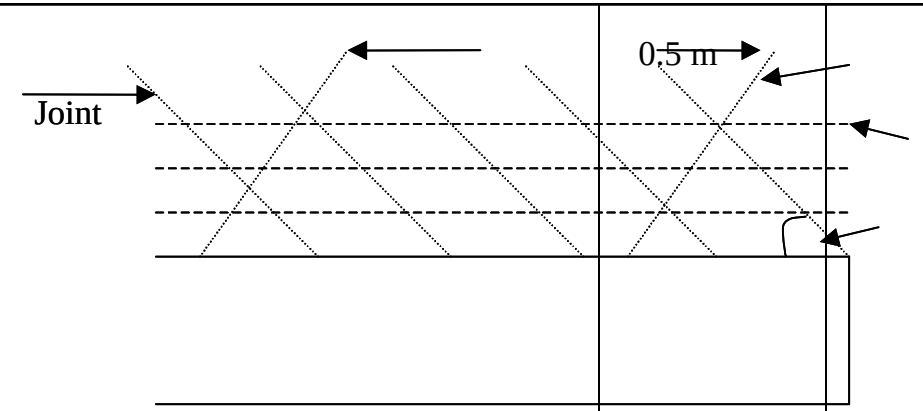
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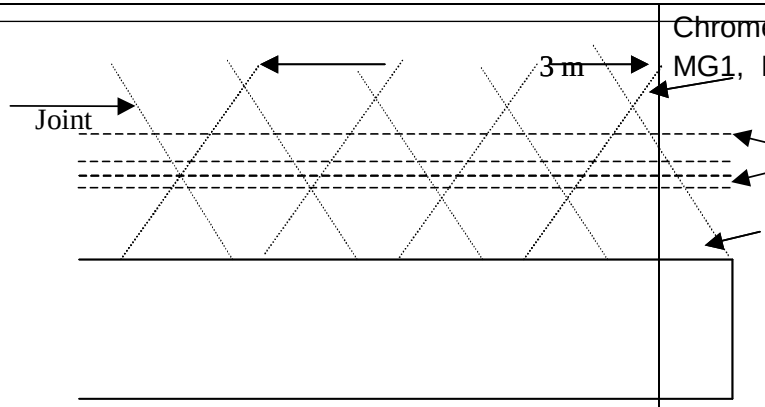
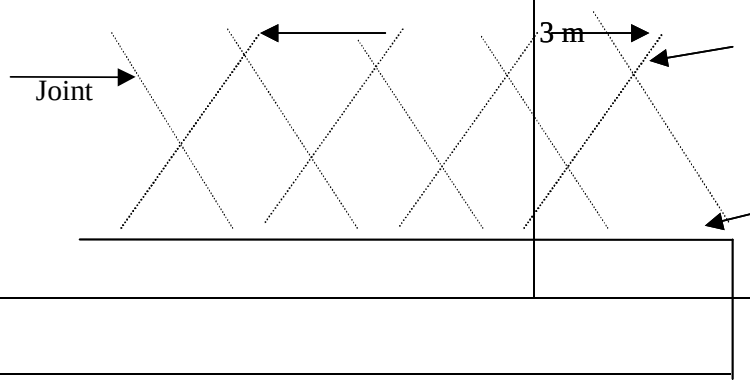
2	In the case of the Vaal Reef generic, sliding and separation on bedding planes, which are approximately 0,3 m apart is not uncommon. There is moderate faulting. Extension fractures are typically, 0,2 m or less apart and orientated at 70º to 80º dipping towards the face. The shear fractures are about 3 m apart and dip in the opposite direction. Blocks have therefore parallelogram or triangular shapes. On a two-day cycle, closure rates could be as high as 20 mm/day to 30 mm/day. Stoping width and reef dip are typically 1 m and 15º respectively. Repeated seismicity degrades rockmass rating. **Dominant failure factors** Bedding (cross bedding, Zandpan marker) Horizontal deformation High frequency faulting	Various forms of beam failure Joint, stress fracture and bedding plane bound blocks. **Mechanisms** Prevention of horizontal deformation can cause buckling failure Horizontal shearing on bedding planes causes vertical deformation	Vaal Reef generic is a soft system (rock mass rating) ***Measure the horizontal deformation on bedding planes (Time domain reflectometry)*** ***Differential Movement (to be developed)*** Crushing of face, crushing of drill holes (observation, acoustic seismic velocity)	 Schematic of the Vaal Reef generic.	Some Kimberley Reefs fall into the Vaal Reef generic category. In F State,, A and B Reefs.	

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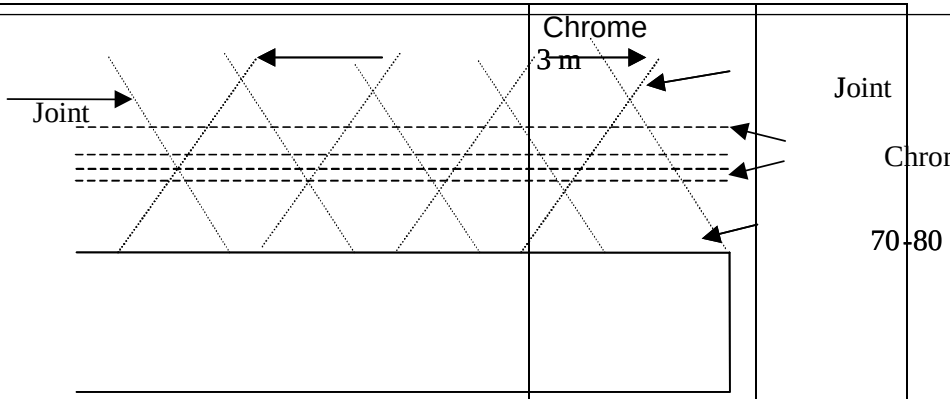
3	The Carbon Leader Reef hangingwall exhibits intense and complex hangingwall fracturing (0,1 m or less apart) including shallow dipping and flat fracturing. It is characterised also by few faults and occasional bedding planes. On a two-day cycle, in backfilled stopes, closure could be as high as 10 mm/day. Non back-filled stopes could experience closure in the region of 50 mm/day. The dip of reef is generally 22º. Repeated seismicity degrades rockmass rating. Falls of ground are generally due to the loss of undercut allowing incompetent (equivalent of Green Bar) material to fail. Dominant failure factors Failure of first or second bedding plane bounded beam which is highly fractured.	Joint, stress fracture and bedding plane bound blocks Factory roof fallouts Mechanisms Reduction of horizontal stress i.e. loss of, hangingwall continuity Unfavourable stress fracture orientation, (toe of panel, gully face intersection)	Measure horizontal stress (seismic velocity, can you link to closure) Planarity of hangingwall (measure) Beam thickness in WAF and Basal also thickness of WAF or shale (measure, GPR)	 Schematic of the Carbon Leader generic.	Other reefs classified under the Carbon Leader generic are Basal Reef, Main reef, certain Elsberg Reefs, Bird Reef and Doornkop Reef. Perhaps Soft VCR	
	Massive siliceous Reefs Partially bedded quartzite hangingwall with well developed jointing. Dominant failure factors Wedges formed by a combination of joints and bedding	Joint, stress fracture and bedding plane bound blocks Stress fractures, faults and bedding	*Differential Movement* System stiffness in order to determine how long a precursor would be useful (rock mass rating) The stiffer the system the less the deformation before failure and v versa Massive siliceous		Elsburg, UE1a Bird Reefs, South Reef, Main Reef, Government Reefs	

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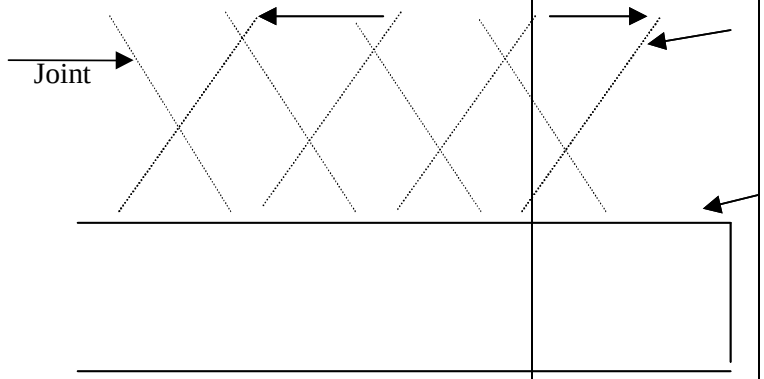
			Reef generic is a stiff system Release of energy (acoustic)	

4	The UG2 generic is characterised by a jointed blocky hangingwall with occasional rolls. At depth, fracturing is not always developed. An important feature is the presence of a variable number of chrome stringers with variable spacings in the hangingwall. If close to the UG2 reef, beam failure can occur. The closure rate is influenced by depth and the presence or absence of large thrust structures (domes) Excessive closure and differential movement means instability. Stoping width is typically 1 m - 2 m with the reef dipping at between 5º and 15º. Dominant failure factors Cantilever and block failure. Beam thickness, parting thickness, cohesion, jointing, Rock mass rating vs. stand up time	Cantilever failure Joint and layer parallel bound blocks Small scal curved Structures Wedge failures Overhanging faces Failure of panel span Premature failure of support Variable rock mass properties	Acoustic Failure of panel span –measure rockmass rating anomalous closure (acoustic, seismic velocity and rock mass rating) Pillars as precursor (measure radial deformation of pillars and observation, photography	 Schematic of the UG2 generic.	Chrome LG6, MG1, MG2 Joint	Chrome stringers 70-80 deg
5	The Merensky Reef is characterised by a jointed blocky hangingwall with occasional rolls. At depth, fracturing is not always developed. The closure rate is influenced by depth and the presence or absence of large thrust structures (domes) Excessive closure and differential movement means instability. Stoping width is typically 1 m - 2 m with the reef dipping at between 5º and 15º. Dominant failure factors	Failure of panel span Premature failure of support Anothosite as Hangingwall Large wedge failure	Failure of panel span –measure rockmass rating Acoustic	Failure of panel span –measure rockmass rating 	Other mines, Joint	70-80

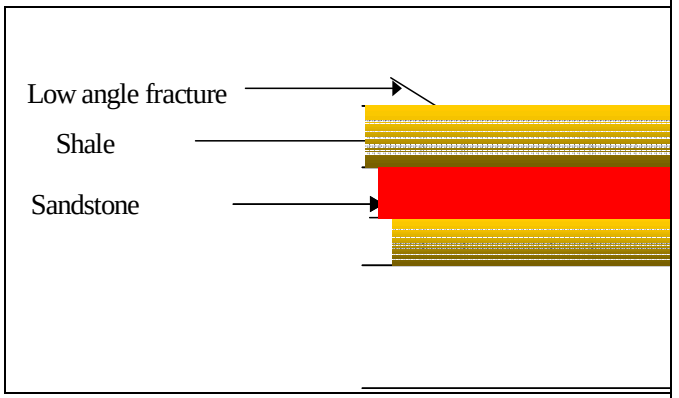
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	Jointed blocky failure, wedge failure. Multiple joint sets make it worse. Joint cohesion. Failures on Faults. Presence of Anothosites			Schematic of the Merensky Reef generic.		
6	Presence of Thrust structures (domes) – high closure and differential closure. The UG2 generic is characterised by a jointed blocky hangingwall with occasional rolls. At depth, fracturing is not always developed. An important feature is the presence of a variable number of chrome stringers with variable spacings in the hangingwall. If close to the UG2 reef, beam failure can occur. The closure rate is influenced by depth and the presence or absence of large thrust structures (domes) Excessive closure and differential movement means instability. Stoping width is typically 1 m - 2 m with the reef dipping at between 5º and 15º. Dominant failure factors Cantilever and block failure. Excessive closure. Beam thickness, parting thickness,	Rotational failurew Cantilever failure Joint and layer parallel bound blocks Small scale curved Structures Wedge failures Overhanging faces Failure of panel span	Widespread closure Measured Identification of small scale Curved jonts –leave pillar GPR Closure on support, noise from support Acoustic Failure of panel span –measure rockmass rating anomalous closure Pillars as precursor (closure , acoustic a GPR)	 Schematic of the UG2generic		

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	cohesion, jointing, Rock mass rating vs. stand up time					
7	Presence of Thrust structures (domes) – high closure and differential closure. The Merensky Reef is characterised by a jointed blocky hangingwall with occasional rolls. At depth, fracturing is not always developed. The closure rate is influenced by depth and the presence or absence of large thrust structures (domes) Excessive closure and differential movement means instability. Stoping width is typically 1 m - 2 m with the reef dipping at between 5º and 15º. Dominant failure factors Excessive closure Jointed blocky failure, wedge failure. Multiple joint sets make it worse. Joint cohesion. Failures on Faults. Presence of Anothosites	Rotational failure Cantilever failure Joint bound blocks, wedge failure Small scale curved Structures Wedge failures Overhanging faces Failure of panel span	Widespread closure measured Identification of small scale curved jonts –leave pillar GPR Closure on support, noise from support Acoustic Failure of panel span –measure rockmass rating anomalous closure (closure , acoustic a GPR)	 Schematic of the Merensky Reef generic.		Joint 70-80

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8	Generic coal measures are characterised by a bedded sandstone – shale roof with bedding thickness of > 0.3m left in the roof. Joint sets are encountered which can lead to instability (slips) Occasional presence of high horizontal stress. Dominant failure factors Soft sedimentary structures, Shale/coal roof. Delamination and buckling. Water pressure. Stooping	Wedge failure one plane is bedding Cantilever failure Buckling Sill snap through Guttering	Cohesion between bedding opening up, delaminating. Sensed by acoustic Feeler gauge Tell tale Acoustic energy meter Drilling penetration rate Extensometers When guttering present beam bending, buckling, hinge point failure			

Ll tale

the parameters considered include the following:

- Orientation and the degree of fracturing;
- Orientation and the degree of jointing;
- Extent of bedding and distance from skin of excavation;
- Extent of fracturing;
- Stoping width; and
- Dip of reef.

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