Measurement of typical joint characteristics in South African gold mines and the use of these characteristics in the prediction of rock falls

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Synopsis

The occurrence of fracturing due to high stress levels is a major factor in hangingwall stability in deep level gold mine stopes. However, rock falls cannot be the result of these fractures alone. Rock falls can result only from the occurrence of unstable rock blocks defined by the interaction of the stress induced fractures and naturally occurring geological planes of weakness. These planes include bedding planes and joint set planes. However, there is a general lack of information on the characteristics of these planes (orientations, spacings and lengths)—it does not appear that any such systematic and quantified information, if collected, has been published. To remedy this situation to some extent, joint mapping exercises have been carried out in several geological environments in two gold mines. The data collected on joint geometry included orientation, spacing and length, and this information is included in the paper. In addition, the paper describes the use of the data collected to evaluate the potential for rock falls and the probability of failure of stope support

Introduction

The design of stope support in South African gold mines has traditionally been carried out by rock mechanics personnel on mines as part of their responsibility in terms of the Code of Practice to Combat Rock burst and Rock Fall Accidents on Mines (COP). In the guideline for the compilation of a mandatory COP to combat rock fall and rock burst accidents in tabular metalliferous mines (DME, 2007), the requirement is clearly stated, 'Support design methodologies used must be properly motivated and documented.' This is also a responsibility expected of rock mechanics personnel by their employers. The design of stope support expected by the COP commonly takes into account a mass of rock corresponding with 95% of the expected height of rock fall, determined from documented records of rock falls on the mine (Jager and Ryder, 1999). Static and, if appropriate, dynamic loads, and the capacities of support elements, are taken into account in determining the required spacings of the support elements. This design procedure is commonly applied in the tabular stoping environment in gold mines (Jager and Ryder, 1999).

In the design process outlined above, no account is taken of the actual sizes of rock blocks, slabs and wedges that might be present in the stope hangingwall (the empirical rock fall data required in the COP do take account of observed fall thickness on a statistical basis, but not the lateral dimensions of the blocks). Rock falls can occur only if blocks are defined by natural joints (including bedding or parting planes) or a combination of such joints and stress induced fractures. Although the gold mining industry acknowledges the existence of natural joints and bedding planes in the rock mass, their effect on stability of excavations is not routinely and specifically taken into account in traditional approaches. It was indicated by Stacey (1989) that there were no published records of systematic mapping of joint characteristics in gold mine stopes, and this appears still to be the case as far as South African literature is concerned. Stacey (1989) showed that there is a significant probability of occurrence of rock falls defined by stress-induced fractures and natural joints. Significantly, he showed that, if a single elongate in a 1 m x 1 m pattern was to be lost, the probability of occurrence of potential rock falls increased from 5% to 26%. As a result of this work, the recommendation was made that investigations should be instigated to gather information on the characteristics of joints. No such information has been collected until recently (Gumede, 2006). It is to be noted that the presence of faults will exacerbate the stability situation. They need to be taken into account on an individual basis and are therefore not considered here.

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In this paper, the systematic measurement of jointing geometries (orientations, spacings and lengths) in two gold mines is described, and the interpreted data are presented (Gumede, 2006). These data are used to predict the probability of occurrence of rock falls, and the probability of failure in supported stopes.

Characteristics of rock joints

Since this paper deals with the collection and use of joint data, it is considered appropriate to reiterate briefly some of the terms that are relevant to this topic.

Rock joints

Joints are planar discontinuities of geological origin in a rock mass, along which there is no discernible or visible lateral displacement. Joints are planes of weakness in a rock mass and hence they directly or indirectly influence the stability of the rock mass. Bedding planes are considered as joints in this paper.

Joint geometrical properties

Joint geometrical properties considered important are joint orientation, joint length and joint spacing. Although joint surface properties, which determine joint shear strength, are important to stability, only joint geometry is dealt with in this paper.

Joint orientation

Joint orientation describes the attitude of the joint in space. Orientation is the most important joint property since joints that are favourably orientated with regard to stability effectively neutralize the effects of other properties. A joint's orientation is uniquely described by its dip and dip direction angles. In most cases, joints are found to be clustered in statistically preferred directions, and an individual cluster of these joints defines a joint set. Joints that do not fall within the defined sets are known as random joints. Joint orientations are typically normally distributed (Robertson, 1977; Barton, 1976; Baecher *et al.*, 1977; and Kulatilake *et al.*, 1995).

Joint length

The length of a joint describes the length of the potential failure plane. Discontinuous joints result in a relatively stronger rock mass since, for failure to develop along this plane, rupture must occur through intact rock.

It is not easy to measure the absolute length of joints since the entire joint surface is usually not exposed. Hence joint trace lengths are measured in 'exposures' and the overall length can be statistically predicted thereafter. Accurate measurement of joint lengths is more challenging than measurement of other joint properties mainly because of size bias and truncation or censoring. Size bias occurs because larger joints have a higher probability of being sampled than smaller joints. The sampling error introduced by the size bias can be as much as double the true trace length of the population (Einstein *et al.*, 1983). Joint lengths may be censored above, or truncated below, certain lengths that cannot be measured practically. In field mapping it is common to measure joints from a given minimum length (truncation) and this truncation error results in the overestimation of joint lengths. Some joints tend to extend into or out of the excavation face, resulting in the impossibility of measuring their actual full lengths. It is therefore common not to measure joint lengths that are above a given length (censoring), and this has the effect of underestimating the actual joint length.

Evaluation of extensive data has shown that the frequency distribution of discontinuity trace lengths may be described by either a log-normal distribution or a negative exponential distribution (Robertson, 1977; Barton, 1976; Baecher *et al.*, 1977; and Kulatilake *et al.*, 1995).

Joint spacing

Joint spacing is a measure of jointing intensity in a rock mass, that is, the number of joints per unit distance normal to the orientation of the set. It is taken as the perpendicular distance between adjacent joints. In general, joint spacing values are positively skewed and can be approximated by negative exponential or lognormal functions—most researchers (Call *et al.*, 1976; Priest and Hudson, 1976; Wallis and King, 1981; and Kulatilake *et al.*, 1995) have concluded that a negative exponential distribution is applicable for joint spacings, while others indicate a lognormal distribution (Steffen *et al.*, 1975; and Bridges, 1975). In practice there is little difference between the assumption of a negative exponential or log normal distribution.

Joint surveys

Joint surveying techniques that can be used for underground mapping are cell or area mapping, scan line mapping, and interpretation of joint data from orientated drill core. These methods are all well established (for example, Nicholas and Sims, 2000). Cell mapping involves systematically dividing the face to be mapped into zones of equal lengths called cells. Structural data are then mapped in areas called cell windows (hence the name window mapping). The actual mapping involves visually identifying joints within the cell window and recording their orientations, lengths, spacings, and surface characteristics.

The scan-line mapping technique involves measuring all the joints that intersect a scan-line along its length. A measuring tape is usually used as a scan-line and the properties of only those joints that cross the tape are recorded.

Both cell and scan-line mapping techniques have the disadvantage of mapping only exposed surfaces, thus they cannot be used in determining the structural behaviour behind the exposed surface. In scan-line mapping, less judgement is required during the actual data collection, hence not much geological mapping experience is required. Although more data are collected over larger areas in cell mapping, data from scan-line mapping represent more detailed information per specific location. The mapping data reported in this paper were obtained from the scan-line mapping technique only.

The latest developments in joint mapping include the use of the photogrammetric principles (Feng *et al.*, 2001; Beer *et al.*, 1999; Harrison, 1993; GIS Du *et al.*, 2001), and common

photographs (Hadjigeorgiou *et al.*, 2003). Sirovision (CSIRO, 2004) is a structural mapping technique used for a complete rock structure analysis using imaging, laser scanning and photogrammetric techniques.

Joint mapping in underground gold mines

Most of the work relevant to joint mapping data for underground gold mines was documented and published in the late 1970s and early 1980s. In most cases joint mapping was done as a secondary procedure, the primary aim and emphasis being to map fractures ahead of the mining face. As such, most mapping was done from drill cores, and joint properties such as lengths were not recorded. In fact, all published papers indicate that the mapping was limited to joint orientations (Kersten, 1969; Van Proctor, 1978; Adams and Jager, 1980; Hagan, 1980; Brummer, 1987; and Quaye and Guler, 1998). Hagan (1980) gave a relatively detailed description of joint properties, referring to the vertical orientation of quartz filled joints and estimating their spacing to be between 0.2 m and 2 m. However, there was no mention of other properties, but he did acknowledge that the interactions of joints (including bedding planes and minor faults) and stress induced fractures were responsible for the hangingwall instability at the then Western Deep Levels gold mine. Van Proctor (1978) mentioned the tendency for stress induced fractures in the quartzite hangingwall to follow quartzite filled joints. Besides this information, it appears that no other published information on joint properties in South African gold mines is available, in spite of their important influence on underground stability. It also appears that there is no systematic measurement of jointing carried out on gold mines as input to stability evaluation and support design.

In the work described in this paper, joint mapping was carried out in stopes and development tunnels for two gold mines at depths ranging from 2 500 m to 3 300 m below surface. Figures 1a, 1b and 1c show examples of the types of joints and bedding planes found in these stopes and development tunnels.

For stope scan-line mapping, a 30 m measuring tape was held straight and tight between two strike gulleys. In development tunnel mapping, the tape was aligned with the grade line so that the scan-line had the same orientation as the tunnel. Every discontinuity intersecting the tape was measured and its properties recorded. It is noted that these scan line orientations will lead to bias in the joint data since, ideally, scan lines in three mutually perpendicular directions are required. However, owing to the geometry of the gold mine environment, such ideal conditions are not attainable.

Results of joint mapping

Results of the joint mapping carried out in the two mines will be dealt with in turn below.

Mine 1

Results from the underground mapping include orientations, spacings and lengths.

Joint orientations

Two joints sets were delineated. The dominant set is termed set 1 and the other set 2. The remaining joints mapped are random joints. Average orientations of sets 1 and 2 are 20°/167° and 19°/322°, implying that they are shallow dipping. Most of the joints in stopes were unfilled. However, the few filled joints observed were steeply dipping and had

Figure 1a—Bedding planes in the roof of a development end (note the benching)

Figure 1b—Joint planes in a stope hangingwall

Figure 1c—Stress induced fractures in stopes

rough quartzite as the filling material, with a thickness ranging between 2 cm and 5 cm. Stress fractures were observed to follow the direction of quartz filled joints. This alignment of stress induced fractures parallel to quartz filled joints was also observed by Van Proctor (1978) during fracture mapping in Doornfontein Gold Mine.

Statistical distributions of joint orientations were observed to be normal, as shown in Figure 2 for joint set 1 (a comparison between means and medians indicated a small difference between the two, which is a characteristic feature of a normal distribution). The results correspond with the conclusions of most researchers about joint orientation distributions.

A single shallow dipping dominant joint set occurs in development tunnels, which represents a cluster of bedding planes. Quite a few of these bedding planes are filled, usually with a phyllonite or quartzite infill approximately 2 to 3 cm in thickness. The phyllonite infill is soapy and very smooth. Failure observations in tunnels indicated that failures most commonly result from a combination of these bedding planes and stress induced fractures.

Joint spacings and lengths

An example of the results of analyses of the statistical distributions of measured joint spacings is shown for set 1 in Figure 3.

Discussion of results

A statistical analysis of hangingwall and footwall joint properties confirms a normal distribution for orientation, and a negative exponential distribution for trace lengths. Both negative exponential and lognormal distributions can be fitted for spacings. Only a single joint set is delineated in footwall development tunnels and these joints are the bedding planes. Most of them are not filled and the few that are generally have a soapy phyllonite filling material. The bedding planes showed consistent statistical characteristics, with their orientation distributions being nearly ideal.

Two joint sets are found in stope hangingwalls. Both sets are shallow dipping, but they have opposing dip directions.

Stress induced fractures in the hangingwall were more closely spaced (5–15 cm) than footwall fractures (30–40 cm) observed in the development tunnel. Combinations of steeply dipping stress induced fractures and the shallow dipping hangingwall joints are the main cause of rock falls in stopes. The hangingwall joints have quartzite as their filling material.

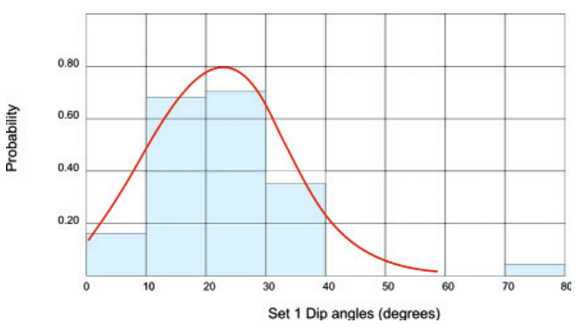

Figure 2—Distribution of hangingwall set 1 dip angles

Hangingwall joints do not show the same level of statistical uniformity as the joints in development ends (footwall). This can be partly attributed to the fact that bedding planes generally show more conformity and persistence than other joints. Also, in development mapping there is a longer mapping span (continuous) available, while in stope hangingwall mapping, the mapping length is limited to a stope face length of 30 m. Table I below gives an overall summary of results.

Mine 2

Joint orientations, spacings and lengths

The analysis of results followed the same procedure as above for Mine 1.

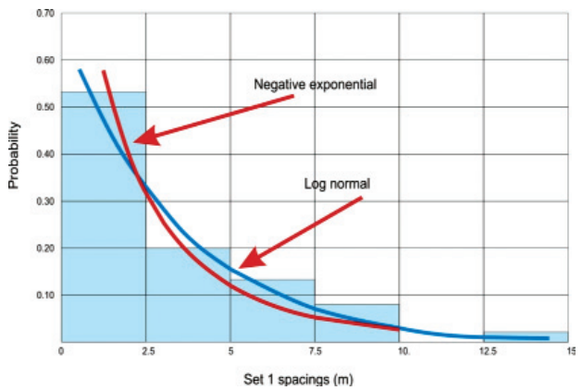

Figure 3—Set 1 spacing in the hangingwall lava showing both the negative exponential and lognormal probability distributions

Table I

Overall joint set data for Mine 1

Location	Set	Orientation (dip/dip direction)	Spacing	Semi trace length
Hangingwall lava				
A	1	23/170	2.5	2
	2	12/05	3.4	2
B	1	20/325	6.2	3.5
C	1	27/156	6.2	5.3
	2	50/154	2.5	0.6
	3	77/336	2.4	1.3
D	1	51/218	5.2	3.9
	2	10/253		
	3	69/56		
	4	88/136		
Overall	1	20/170	3.6	2
	2	19/322	5	2.5
	Random*	65–85/160–200	1.9	1.6
Footwall				
E	1	25/131	5	2
F	1	28/151		
	2	28/315		
G	1	3/123		
Overall	1	28/130	4.3	3.3
	Random*	60–80/140–200	5	2.6

*Random representation of random joints

The dominant joint set (set 1) is a steeply dipping (88°/292°) quartz filled joint set. The quartzite infill thickness ranges between 1 cm and 5 cm. Set 2 joints are shallow dipping (26°/186°). Minor sets (joint sets 3 and 4) are probably an extension of set 1. However, in the analysis of other joint properties such as spacing and length, set 1 and the minor sets (3 and 4) are combined.

Jointing in footwall tunnels for Mine 2 is similar in many aspects to that of Mine 1. It is characterized by a dominant shallow dipping bedding plane set, and the random joints make up about 45% of the joints (compared with 37% for Mine 1).

Statistical distributions for spacings and lengths demonstrated similar trends to those of Mine 1.

Discussion of results

Two sets are delineated in the quartzite hangingwall. Of the two, a steeply dipping and quartzite filled joint set is the most dominant, while the less dominant one is shallow dipping. The interaction of these two sets with random joints and stress induced fractures results in the formation of unstable blocks in stope hangingwalls. A negative exponential distribution was fitted to joint lengths and both exponential and lognormal distributions could be fitted to joint spacings. 60% of hangingwall joints were random compared with about 45% encountered in the footwall. Footwall joints (bedding planes) are usually unfilled and their properties demonstrate a close statistical uniformity. Table II below is a summary of joint properties for Mine 2.

Comparison of data from the two mines

A comparison of the joint mapping results from the two mines reveals the following:

- From the numbers of random joints mapped in the two hangingwalls it can be concluded that in thin reef gold operations, about 60% of the joints are random
- Both mines have shallow dipping joint sets in the hangingwall
- Footwall joints in both mines were consistent in their characteristics and gave nearly ideal statistical distributions. These footwall joints/bedding planes are usually unfilled, although in Mine 1 some bedding planes were observed to contain a soapy phyllonite filling material. Those in Mine 2 are not filled.
- Random joints in the footwall make up about 40% of the joints. In both the hangingwall and footwall, random joints are usually steeply dipping. With this orientation, random joints increase the likelihood of unstable block formation, hence enhancing the probability of occurrence of rock falls.

Table II

Overall joint set data for Mine 2

Location	Set	Orientation (dip/dip direction)	Spacing	Semi-trace length
Hangingwall quartzite				
H	1	87/291		
J	2	22/175		
K	2	19/216		
L	1	86/287		
Overall	1 2	88/292 26/186	5. 3.9	2.3 3.6
	Random*	40–80/150–180	2.1	2.4
Quartzite footwall				
M	1	28/130		
N	1	30/174		
P	1 2	28/151 28/315		
Q	Random			
Overall	1 Random*	23/139 30–50/170–190	3.9 3.9	4 2.6

*Random representation of random joints

Use of the joint set statistical data to determine probabilities of occurrence of rock falls

The availability of measured joint properties in the mines provides the opportunity to use these data for engineering purposes, namely, the quantification of the probabilities of occurrence of potentially unstable rock blocks of various sizes. Two such approaches will be dealt with.

Haines (1984) described a two-dimensional technique for the generation of joint traces using statistical distributions of joint properties obtained from field mapping data. The superimposition of an excavation geometry onto the joint traces allows potentially unstable block geometries to be identified. This interpretation is a manual process which, though time consuming, provides a very good 'feel' for the rock mass. Repetition of this process many times then allows the probability of occurrence of potentially unstable blocks to be determined. Similarly, the probabilities of occurrence of potentially unstable blocks of certain sizes (both area and volume) and, of particular relevance here, of the height of rock fall, can be determined. The application of this procedure has been described by Stacey and Haines (1984), Butcher (2000) and Stacey *et al*. (2005) and has been shown to provide satisfactory results. It therefore provides a powerful tool for the analysis of potential stability in gold mine stopes using measured joint data, and this application is described below.

Another example of probabilistic three-dimensional modelling tools in jointed rocks is JBlock, developed by Esterhuizen (2003). JBlock was used simulate the occurrence of keyblocks in excavation hangingwalls. Simulated blocks were then used to determine the likelihood of failure for different support layouts in hangingwall excavations.

Figure 4 illustrates a simulated joint trace model of the rock mass using measured joint data for the Mine 2 hangingwall. The different colours in the trace model represent different joint sets.

Excavation geometries (in plan and section) are superimposed on simulated trace models and potentially unstable wedges and blocks identified. The identification of keyblocks involves visual analysis of the interaction between the excavation geometry and the trace model and then identification of joints that intersect, in a critical manner, to form potential keyblocks. In Figures 5a 'vertical' sections through stopes are shown superimposed on the joint trace plot. Many such sections can be superimposed on a single joint trace

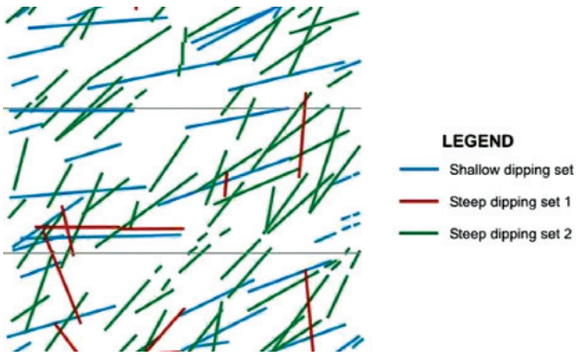

Figure 4—Simulated rock mass using joint data from Mine 2

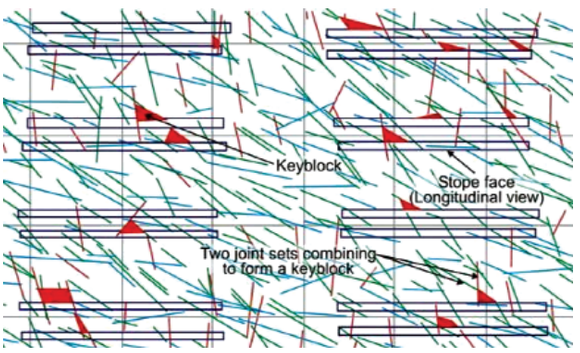

Figure 5a—Potential keyblocks in longitudinal views of a stope in Mine 2

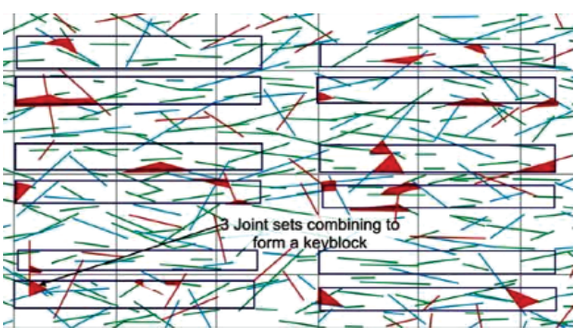

Figure 5b—Potential keyblocks in plan view of a stope in Mine 2

plot. Each section is a statistically independent location. The process involves the generation of numerous joint trace plots and the superimposition of stope sections on the plots such that the number of output data is sufficient to provide satisfactory statistical distributions. Figure 5b shows joint traces in a stope plane section, with stope face areas superimposed. Since the joint traces are generated randomly, the blocks identified from different sectional views are completely unrelated to one another.

The interpretation is systematically repeated for a large number of excavation superimpositions to obtain distributions of the following potentially unstable block parameters:

- number of blocks along the excavation length
- volumes of blocks
- total failure volume per stope
- spacings between unstable blocks
- the range of block sizes for which support must be designed.

Discussion of results

Figures 6a and 6b are probability density distributions, illustrating the distribution of block areas A, from analyses of plan views (the areas of potentially unstable blocks that would be defined on the stope hangingwall), and heights of blocks into the hangingwall z, from longitudinal sections, respectively. The product of the distribution of A and the distribution of z gives the expected block volume. To take into consideration the effect of different block shapes, a factor is introduced from random sampling of numbers between 0.333, for a triangular pyramid, to a value of 1.0 for a rectangular prism. An example of a probability density distribution of potential block volumes resulting from this process is shown in Figure 6c.

The heights of potential rock falls for Ventersdorp Contact Reef (VCR) and Carbon Leader Reef (CLR) stopes, predicted using this process from the measured joint data, and heights

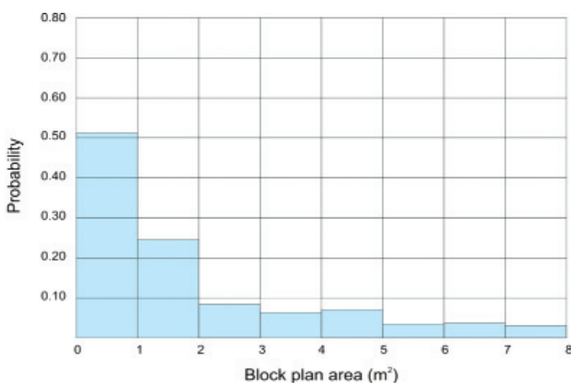

Figure 6a—Distribution of block plan areas in a 30 m stope from a plan section, Mine 2

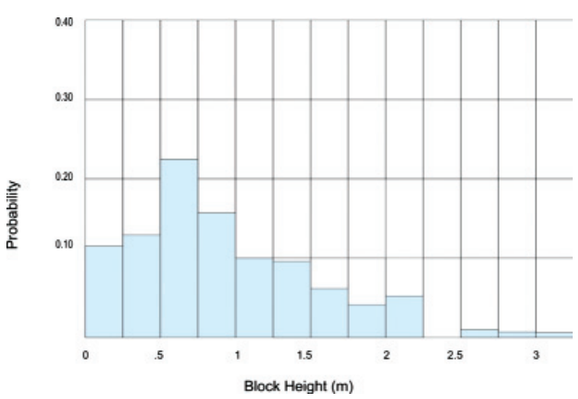

Figure 6b—Distribution of block heights (depth z) in a 30 m stope from a longitudinal section, Mine 2

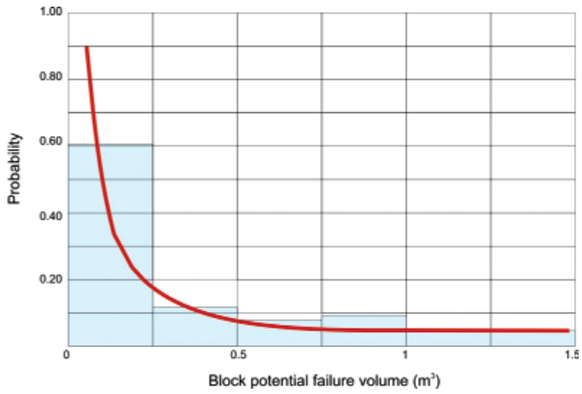

Figure 6c—Distribution of block potential failure volumes in the stope, Mine 2

from empirical data, are shown in Figures 7a and 7b. It can be seen that the agreement between the predicted thickness at the 95% probability and the corresponding published data on empirically determined thicknesses is good for the VCR stopes (1.8 m predicted, 1.4 m empirically determined by Roberts, 1999). In this case there is a substantial set of empirical data (50 rock falls). The summarized updated empirical data presented by Daenhke *et al*. (2001) indicate a height for rock falls of 1.2 m and a height for rockbursts of 1.8 m. The joint trace model interpretation above does not differentiate between static and dynamic conditions, simply predicting potentially unstable blocks. Therefore, the agreement between observed and predicted heights of falls is excellent. For the CLR stopes, a 1.0 m thickness is empirically determined (Roberts, 1999) and 2.2 m is predicted. In this case, however, the empirical data set is limited, containing only 23 rockfalls, and its validity is therefore somewhat doubtful. Updated empirical data from Daehnke *et al*. (2001) indicate a height of fall of 2.2 m for rockburst conditions. Again therefore, there is excellent agreement between the empirical data and the joint trace model prediction.

Although the method described above may appear to be cumbersome, it provides a 'feeling' for the rock mass that would not be obtained were the method more of a 'black box' approach. The design parameters obtained can be used for support design in new areas and in situations in which there are inadequate records of actual rock falls, particularly considering the comparable results obtained. This would require real joint data to be obtained in advance for such new areas as input to the joint trace analysis. In conclusion, the method can be used together with existing support design criteria to improve the support of underground excavations.

The next section demonstrates the use of statistical data on joint characteristics to obtain some of the above parameters and other additional parameters that could not be obtained from the joint trace modelling tool.

Excavation stability analysis using the JBlock program

JBlock (Esterhuizen, 2003) is used both in the probabilistic assessment of gravity driven rockfalls and the evaluation of support effectiveness. The analyses use joint set statistical data to generate potential keyblocks in the hangingwall and these are then randomly 'placed' in an excavation with a known support element layout. The program then determines whether the identified keyblock will cause failure of the support elements and the corresponding failure mode, or fall between them (Esterhuizen, 2003).

The process is repeated many times and the output gives a plot of the probability of failure versus the unstable block volume. Using this method, simulations were carried out with different support elements commonly used in gold mines ranging from point supports (elongates) to line support (point supports with headboards). The simulation runs were carried out for a predefined area within 15 m from the stope face and five thousand blocks were generated in each simulation. The generated blocks were saved and used in the reanalysis of excavation stability with each support system and support layout. The variation of support elements and layouts was applicable for the first three rows of support elements.

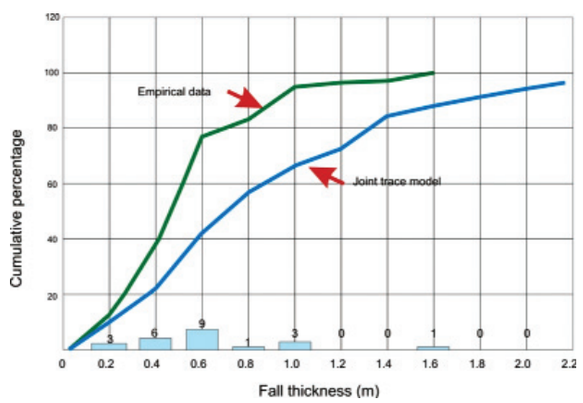

Figure 7a—Comparison of rock fall thicknesses calculated using the joint trace procedure with empirically determined fall thicknesses from Roberts (1999) and Daehnke *et al.* (2001) for VCR stopes

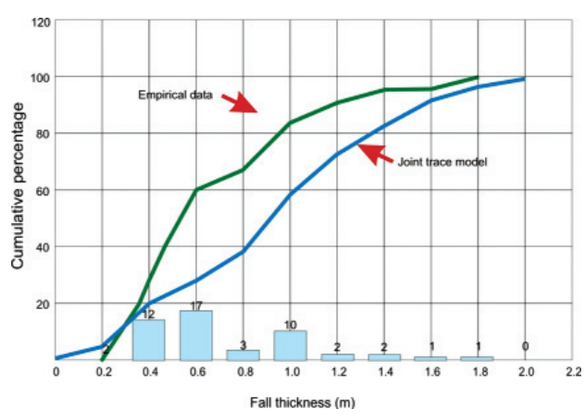

Figure 7b—Comparison of rock fall thicknesses calculated using the joint trace procedure with empirically determined fall thicknesses from Roberts (1999) for CLR stopes

Discussion of JBlock results

Simulation of a large number of keyblocks for different excavation orientations in Mine 1 stopes indicated that 60–80% of generated keyblocks in stope hangingwalls are less than 1 m^3 in size (Figure 8a). The probability of these blocks falling between support elements is between 10% and 15%. Large blocks (>2 m^3) have less than a 3% failure probability, but, unlike the small blocks (1 m^3), they are more likely to fail support elements than to fall between supports (Figure 8b).

For Mine 2, the distribution of sizes of potential failure blocks in the two mines is similar to that of Mine 1. However, failure probabilities in Mine 2 stopes are significantly less than those in Mine 1 (less than 5%). This is because, in Mine 1, there are two shallow dipping joint sets with opposite dip directions and this increases the likelihood of block formation in the hangingwall. In Mine 2, one joint set is shallow dipping and the other is steeply dipping. Furthermore, the two sets have similar dip directions. The likelihood of keyblock formation in the hangingwall with this joint geometry is smaller, hence the difference in failure probabilities.

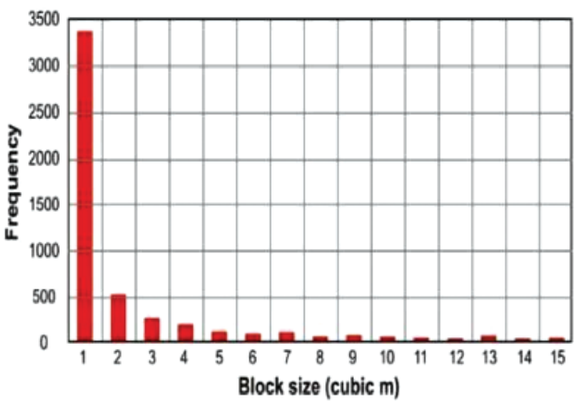

Figure 8a—Potential failure block size distribution in Mine 1

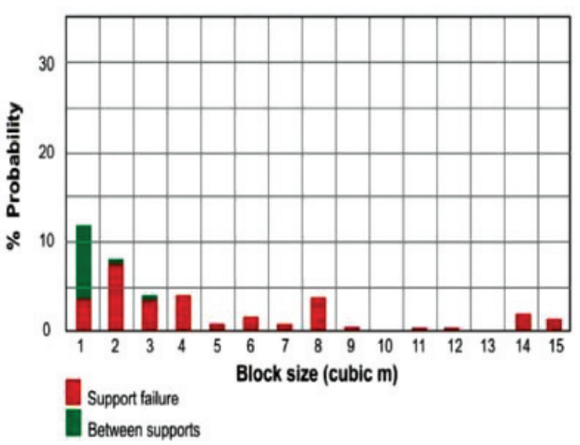

Figure 8b—Distribution of failure probabilities in Mine 1

The two shallow dipping sets in Mine 1 are responsible for the lower fall thickness in stopes while the opposite is true for Mine 2 stopes.

Failure and stability modes

Predicted block failure modes are provided by the JBlock output. Examples of these, for Mine 1, are shown in Figures 9a to 9c. The results show that about 45% of the small blocks (<1 m^3) in Mine 1 are likely to fail by single plane sliding. About 45% of small blocks fail as 'drop-outs' between support elements. For Mine 2, this number is 80%. 'Drop-out' of larger blocks (i.e. greater than 2 m^3) is unlikely. The most common failure modes for larger blocks are single plane sliding and rotation (e.g. Figure 9a). Rotational failure occurs when block weight is less than support capacity, but the block is still able to fail support by the turning moment (leverage) provided during block rotation (Esterhuizen, 2003). Stability of small blocks is mainly mobilized by friction (75%), while stability of the remaining 25% is provided by the support elements. Larger blocks are generally stabilized by support elements (Figure 9b).

Support layout and stability

The main purpose of support is to ensure safety and stability. Ideally, a support system must stabilize all possible

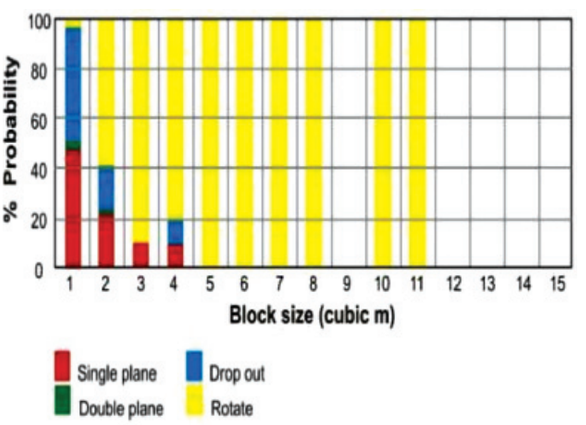

Figure 9a—Block failure modes in Mine 1

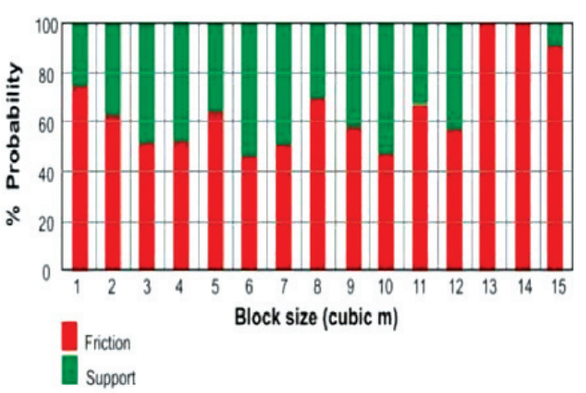

Figure 9b—Stability modes in Mine 1

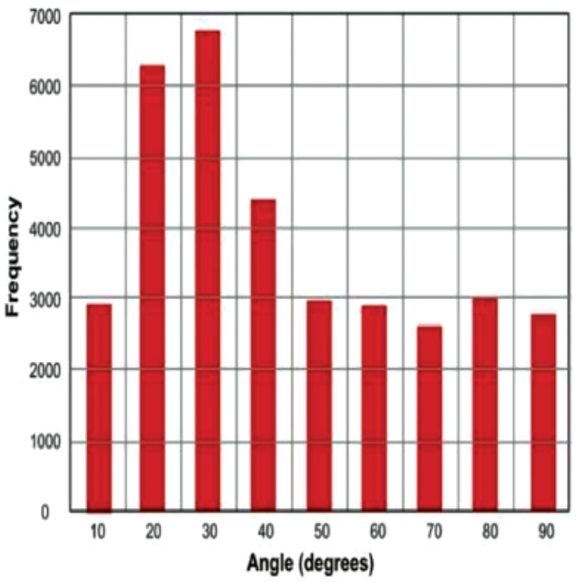

Figure 9c—Distribution of sliding angles of keyblocks in Mine 1

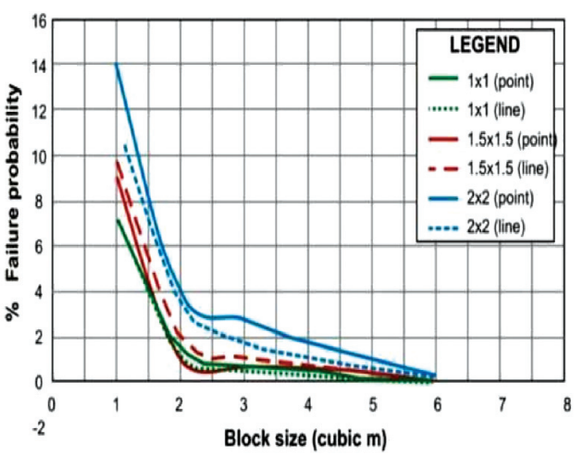

Figure 10—Effect of alternative support geometries on stability

keyblocks, but a pragmatic support system is one that reduces the probability of keyblock failure to acceptable safety levels. Support spacing has a significant influence on stability. Figure 10 shows the results of analyses carried out for different support spacings and types in Mine 1 stopes. Changing from a 2 m x 2 m point support spacing to a 1 m x 1 m point support spacing reduces the failure probability of small blocks from 14% to 7%, while a similar reduction in line support spacing (point with headboards) decreases the failure likelihood from 11% to 7%.

Line support increases the areal coverage of the support, thereby decreasing the likelihood of 'drop-outs' of small blocks. A comparison of failure probabilities between the two support systems reveals that, for a 2 m x 2 m support spacing, headboards reduce the failure probability of small blocks from 14% to 11%.

Aligning support elements in an offset or alternating manner can also improve the areal coverage of the support. For example, offsetting a 2 m x 2 m point support pattern reduces failure probability of a 1m^3 block from 14% to 10%. Any improvement in areal coverage decreases failure probability by reducing unsupported spans in the hangingwall.

Discussion

The research work described in this paper has demonstrated that data on jointing characteristics in gold mine rock masses can be collected practically, and that they can be successfully used to determine the probability of occurrence of potential instability, and the probability of failure of blocks of different volumes. The output from the analyses has shown that probabilities of failure can be significant, and that the smaller the block volume, the greater the probability of failure. The analysis results show that the probability of rock falls between supports is significant. The use of headboards increases the effectiveness of the point supports and decreases the probability of occurrence of rockfalls somewhat, but not significantly. As stated by Jager and Ryder (1999), the majority of falls of ground occur between supports. This represents a failure of the support system and therefore, as stated by Stacey (2003), '... either the support is inadequate, or the support design is inadequate, or both are inadequate. With the fatality data, and the long history of high stress and seismicity, it is prudent to question whether an ethical design process has and is being followed in the design of rock support for these conditions.'

From the JBlock results, it can be concluded that most keyblocks (70% to 80%) are less than 1 m^3 in size and this matches the results obtained from the joint trace model. It is interesting to note that Australian experience is that most injuries are caused by rock falls weighing less than 2 tonnes (Nedin and Potvin, 2005), which implies a volume of less than 1 m^3. Failure probabilities are heavily influenced by joint geometry. For the stope support systems typically installed, these are 10% to 15% for Mine 1 stopes and less than 5% for Mine 2 stopes. The shallow dipping joint planes usually act as critical sliding planes for blocks. Common failure modes are single plane sliding and 'drop-outs' for blocks less than a cubic metre, while larger blocks are more likely to fail by rotation. Small blocks are mainly stabilized by friction, while blocks greater than 1 m^3 in size are usually stabilized by support elements. Areal support coverage, be it headboards or offset support alignment, reduces failure probabilities by decreasing unsupported hangingwall spans.

The results presented in this paper do not take into account the beneficial effect of a stable, continuous hangingwall beam. In this regard, however, it is considered that such beneficial effect cannot be relied upon, particularly as far as the stability of small blocks and wedges are concerned.

Conclusions

Rock falls in South African gold mines usually result when unstable rock blocks are formed as a result of the interaction of stress induced fractures and natural jointing in rocks.

Bedding planes will usually provide release planes, allowing these blocks to fall. The importance of the natural joints and bedding planes in defining the instability has not been given the attention that it deserves, to the extent that there are no documented, published data available on joint set characteristics. This is perhaps an indication that such data do not exist on the mines. This is surprising since, to evaluate realistically the probable dimensions of potentially unstable blocks and the probability of occurrence of rock falls, and to be able to carry out a satisfactory design of support to cater for the identified potentially unstable blocks, data on joint characteristics are essential. The work described in this paper and the joint set data that have resulted, are a contribution towards this goal.

It has been demonstrated that measured joint data can be used successfully to determine the probability of occurrence of rock falls and the probability of failure of stope support due to unstable blocks. It is considered that measured joint data are an essential input for satisfactory design of support for underground stope excavations in the gold mines. It is again recommended that such joint data should be measured systematically on mines so that they can be used for satisfactory support design. The application of joint data in the evaluation of risk is described in a companion paper, also published in this volume (Stacey and Gumede, 2007).

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