CHAPTER 3
BASIS FOR DESIGN METHODOLOGY

3.1 OBJECTIVES OF THE DESIGN METHODOLOGY

Stope support design is one of the more complex design issues in the mining of tabular orebodies. The objective of the research described in the thesis is to quantify stope support and rockmass interaction and through this, develop a methodology that can be used by the rock engineering fraternity for the evaluation of a stope support system.

Figure 3.1 shows the fracturing around a stope in a deep level mining environment according to Roberts and Brummer (1998). It indicates the shear– and extension fractures as well as bed separation that develop around the excavation in a rockmass as a result of the mining process. As mining progresses the stress around the excavation will continue to change in both magnitude and direction, causing a progressive deformation and possible fracturing of the rockmass. Deformation of the rockmass results in the elastic and inelastic closure of the stope excavation.

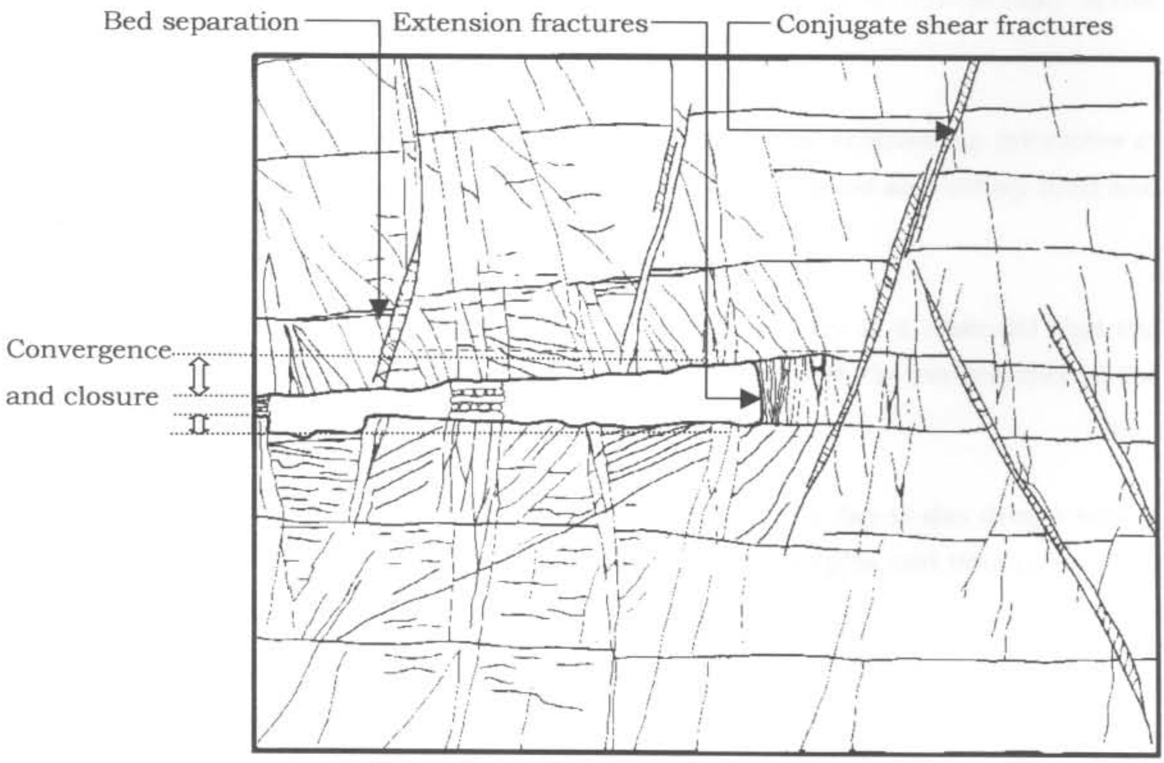

Figure 3.1: Fracturing around a stope (after Roberts and Brummer (1998))

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The rockmass surrounding the stope places a certain demand on the support elements to ensure the stability of the excavation. The support elements in turn have a certain capacity to react to that. The inter-relationship between the capacity of the stope support elements on the one hand, and demand from the surrounding rockmass on the other, forms the basis of this study.

A stope support design methodology should relate realistically to the underground environment to be of practical use for the rock engineer. To accomplish that the mining geometry must be taken into consideration and provision must be made for aspects like face shape, panel leads and lags as well as presence and position of regional support.

The support design methodology should preferably be based on observations and measurements that were taken from underground instrumentation and observations. A model that is developed from this will be more reliable in reflecting the physical underground environment. Underground measurements for the purpose of developing a model, must be taken at regular intervals to fully capture the in-situ rockmass deformation at any particular site for that given mining geometry. The design procedure that is developed from this must be consistent and mathematically stable throughout the total mining process.

The design procedure must further comply with fundamental engineering principles at all times and be described in terms of the laws of physics. It must also satisfy fixed and defined design criteria.

To be of practical use and of relevance to the mining industry is it essential that the methodology takes into consideration factors that influence both the performance of the support element(s) as well as the demand of the rockmass.

For any design procedure to be used in a meaningful way as a day-to day design tool, it must be simple to use and the process should not involve complex and time-consuming computer modelling.

3.2 DESIGN METHODOLOGY

The stope support and rockmass interaction that are described in the thesis can be compared, and is based on the interaction that exists between a rock testing machine and a rock specimen when performing a uniaxial compressive test. The behaviour of

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the rockmass relates to that of the testing machine, while the reaction of the stope support can be compared to that of the rock specimen.

The testing apparatus could either be stiff or soft depending on the design and properties of the equipment. In a similar way the rockmass surrounding the stope will react as either a stiff- or a soft environment. The behaviour of the rockmass depends on the factors governing its behaviour such as mining depth, mining span, presence and position of regional support and rockmass properties.

The area underneath the force-deformation curve for the test machine represents the energy released by the equipment for the specific interval of deformation. Similarly the area underneath its load-deformation curve will represent the energy that is absorbed by the rock specimen for that deformation interval.

When the stiffness of the machine is less than the post failure stiffness of the rock specimen, in other words the energy contained in the testing machine due to deformation of the testing machine itself, for a given interval of deformation, is more than required by the specimen, the specimen will fail in an uncontrolled way. This is referred to as unstable failure and is illustrated diagrammatically in Figure 3.2(a). The opposite is also true where the stiffness of the testing machine is stiffer than that of the rock specimen, the specimen will fail in a controlled fashion or there will be stable failure as illustrated in Figure 3.2(b). The reason for this is that further deformation of the specimen can only occur by the external provision of more energy.

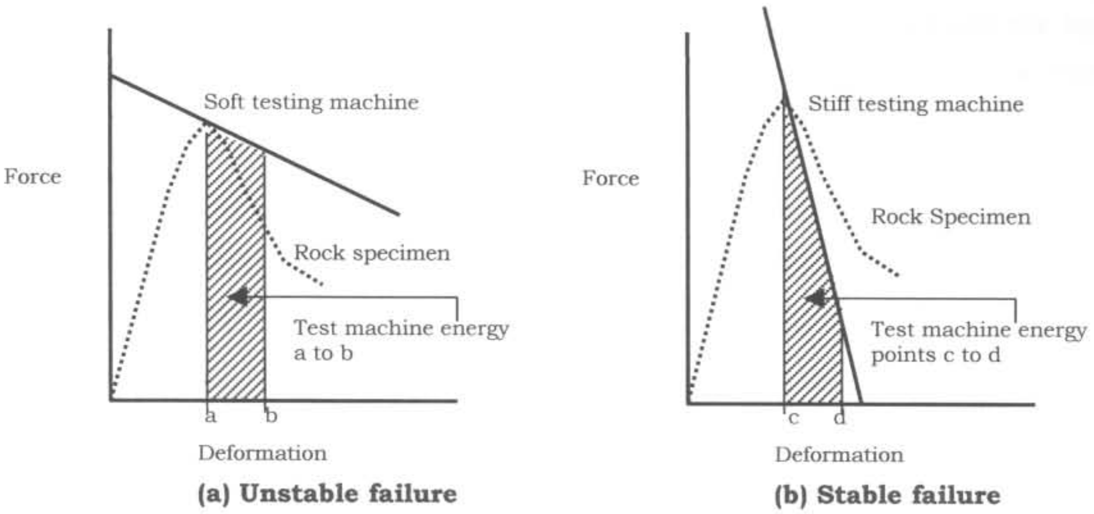

Figure 3.2: Unstable- and stable failure of a rock specimen for soft- and stiff testing machines

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The use of this analogy makes it possible to compare and represent both the demand of the rockmass and the capacity of a support element in a similar fashion. The fact that both of these can be compared and presented on a common force-deformation axis makes this a very attractive approach.

Similar to that of the testing machine the stiffness of the rockmass can be presented on a force-deformation diagram. The rockmass behaviour is represented by the mathematical function:

$$g(x) = mx + F_o \quad (3.1)$$

where:

$g(x)$ = Force (kN) exerted by the rockmass at any deformation x (mm);

m = Rockmass stiffness (kN/mm); and

F_o = Rockmass force at zero deformation, i.e. where x = 0.

The rockmass stiffness at any point for that particular mining stage and excavation geometry is represented by the slope (m) of the function g(x).

It is assumed that the rockmass stiffness is linear. This approach may be criticised, but further work is required to determine it more accurately. The assumption is based on the following:

- Salamon and Oravecz (1976) use the loading machine and rock specimen interaction to describe controlled- and uncontrolled failure of pillars and also illustrate this principle by means of linear load lines. This can be interpreted in mining terms where the rock specimen corresponds to the pillars and the loading machine is analogous to the rockmass surrounding the workings. They stated that the slope of the loading line is determined by the properties of the strata and the mining geometry, and shows that the relative stiffness of the strata decreases with an increase in the number of pillars. The practical implication of this is that as the number of pillars in a panel and, consequently the width of the panel, are increased, the surrounding strata behave as a progressively softer loading machine, so that the likelihood of an uncontrolled collapse becomes higher. The strata stiffness is represented as a single figure by them namely the slope of the loading line as illustrated on pp39 of this reference. The illustration by Salamon and Oravecz (1976) showing a decrease in the strata stiffness with an increasing panel width is confirmed in this study.

- Mauracher (1998) in his Master Thesis studied the behaviour of the hangingwall strata on Beatrix Mine. He concluded that local strata stiffness is a way to describe the hangingwall behaviour and describes the local strata stiffness as being linear.
- No literature could be found that describes the load line of the hangingwall rockmass as being non-linear for a given condition. Research is still required in this field of rockmass behaviour to confirm the assumption made in the thesis.
- This approach may also apply to the UE1a Reef at Randfontein in the Witwatersrand Basin and perhaps the UG2 Reef of the Bushveld Complex. (Personal communication, Roberts (2004)). Applicability to the Carbon Leader and Vaal Reef is less certain and needs to be researched.

The stope support load-deformation curve can be related to that of the rock specimen, and presented on the same force-deformation diagram than that of the rockmass. The mathematical function describing the behaviour (or capacity) of a support element is given by the following n^{th} order polynomial:

$$y = f(x) = c_1x^6 + c_2x^5 + c_3x^4 + c_4x^3 + c_5x^2 + c_6x + c_7 \quad (3.2)$$

where:

y = $f(x)$ = Load generated by support element (kN);

x = Deformation (mm);

c_n = Constants with n = 1,2 .....6; and

c_7 = Pre-stressing during installation. (kN)

The behaviour and hence failure of a support unit is in a way analogous to the pillar stress-strain behaviour shown by Ryder and Jager (2002) where different pillar behaviour, and hence different pillar mode of failure, may be expected with different ratios of pillar width to pillar height. The behaviour of the pillar changes from crush- to elastic- to yield- to squat pillar with an increased width to height ratio. Here the slope of the post peak portion of the curve becomes flatter with an increased pillar width to pillar height ratio. The failure of a timber mine pole can be compared to that of a crush pillar while the behaviour of a mat pack with a low height to width ratio can be related to that of a squat pillar.

According to Daehnke et al. (2001) that if a prop is compressed for a distance exceeding its yielding range, rapid and unpredictable failure can occur. The research described in this thesis attempts to describe this failure as either stable- or unstable failure.

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The support element will start to *shed load* where the tangent to the performance function is negative, i.e. where:

$$\partial/\partial x[f_a(x)] < 0 \quad (3.3)$$

with:

$f_a(x)$ = $f(x).Y.H_f$ - for timber- and cementitious packs (3.4)

$f_a(x)$ = Adjusted support function;

$f(x)$ = Function describing the performance of a specific pack;

Y = Loading rate factor; and

H_f = Support height factor.

The adjusted support performance function for a timber elongate is given as:

$$f_a(x) = f(x).Y.B_f \quad (3.5)$$

where:

$f(x)$ = Mathematical function describing the performance of an elongate; and

B_f = Buckling failure factor for elongates.

Failure of support, that can either be stable- or unstable failure, will depend on the magnitude of the tangent to the support function in relation to that of the slope of the line representing rockmass behaviour at that stage of mining and for the given mining geometry. This is where $|\partial/\partial x.[f_a(x)]| > |\partial/\partial x.[g(x)]|$ for both pack - and elongate support at a given deformation.

The stable and unstable failure of support is demonstrated in the two case studies in Chapter 7.

The capacity to absorb energy is another way of comparing the demand placed on the support by the rockmass to its capacity. The energy for both the rockmass and the stope support element(s) is determined by the integration of the mathematical functions that describe its behaviour. This is done for the given deformation interval that is examined.

Load shedding of the support will take place where the rockmass energy demand exceeds the capacity of the support system. This is true where $\int g(x)\partial x > \int f_a(x)\partial x$ for a given deformation interval x.

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The behaviour and performance of stope support are described in detail in Chapter 4.

3.2.1 Development of the models

The research that is described in the thesis consists of two models that were developed and described in a mathematical format. The first model represents the performance or capacity of the support, while the second represents the support demand from the rockmass.

The objective of combining these into a single model is to represent a combined and comprehensive technique that can be used as a design tool. The model can either be used as a retrospective tool by means of which the past performance of stope support can be studied and quantified where in-situ closure measurements were taken. More importantly it can also be used as a pro-active tool to quantify and compare the behaviour of different supports for a planned mining geometry. The design engineer must have a reliable estimate of stope closure for different stages of mining for the latter purpose. This estimate can either be from underground measurements and or accurate non-elastic numerical modelling of the closure at any point for a given mining geometry.

(a) Stope support model - Capacity

The first model describes the behaviour of the stope support elements with the objective to quantify its capacity. For this purpose load-deformation curves were used from the series of laboratory tests conducted and published by the CSIR Division of Mining Technology (1995) as part of a SIMRAC GAP032 Research project. Laboratory test results were also obtained from manufacturers of stope support products namely Scholtz (1997) and Smit, Erasmus and Grobler (1998).

The laboratory load-deformation curves of the support elements were reproduced mathematically by fitting an n^{th} order polynomial function to the laboratory test data.

Factors that influence support performance must be taken into consideration in order to reproduce and represent the in-situ performance of a support element. Factors such as the effect of the rate of deformation, height of pack installed as opposed to the one tested, buckling potential (in the case of timber elongates) and pre-stressing during installation are taken into consideration. All these factors are described mathematically as separate variables and combined with the support performance function into a final function ($f_a(x)$) to represent the in-situ performance of a support element.

The final sets of equations are written into spreadsheets that were developed to graphically display the performance of a support unit. Factors influencing the support performance can be varied and the effect(s) that it has on the performance of the support element displayed. The spreadsheets were compiled in a way that the in-situ support performance can be compared directly to the laboratory results. The effect(s) and impact of these factors in comparison to the laboratory results can be observed visually when changed or varied. The major advantage of the visual representation of the support performance this way is that it quantifies these influences in a way that the end-user develops an appreciation for the influence that these factors have on the performance of support in direct comparison to laboratory results.

The spreadsheet representation has the advantage that the factors influencing support performance can be changed quite easily and the following calculated for any given in-situ condition:

1. Load (kN) generated by the support unit for a specified deformation;
2. Support resistance (kN/m^2) generated by the support element for a given dip- and strike support spacing;
3. Stiffness (kN/mm) of a support element for any specified deformation;
4. Energy (kJ) generated by the support element for a given deformation interval; and
5. Stress (MPa) exerted onto the stope hanging- and footwall during quasi-static and dynamic loading of the support.

An offshoot of the theory and mathematical models representing support is the application of this approach into a three-dimensional numerical computer model.

It is possible that an estimate of the support resistance generated by a support type can be calculated and contoured. This way the support resistance for a given mining geometry with a given type and support density can be quantified. A program like Minsim W could be utilised for this. The Modulus of Elasticity (E) is modified to a value where the elastic convergence calculated by the numerical analysis approximates underground measurements. It is important to note that the *stresses* shown by this model will not be reliable in such cases and may be incorrect.

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To determine values for the Modulus of Elasticity, an area was modelled and the convergence at the centre of the panel calculated and compared to various closure rates (expressed in mm/m face advance) that were measured underground.

The following mathematical function is introduced in the so-called OWN variable where the engineer defines and describes his own variable to be calculated as part of the numerical output. Support resistance can be calculated and contoured for an on-reef sheet of benchmarks. The variable function that is imported into the program as follows:

$$OWN = f_a(s_z - s_n)/(D_s.S_s) \tag{3.6}$$

where:

$f_a(x)$ = Adjusted support function for a specific type of support;

s_z = Elastic convergence (mm) calculated by Minsim W at the point of interest;

s_n = Convergence that has taken place prior to the installation of the support at a distance n-meters from the stope face;

D_s = Dip spacing (m) of support elements; and

S_s = Strike spacing (m) of support elements.

This application will not be explored any further in this thesis.

(b) Rockmass model - Demand

The second model describes and represents the behaviour of the rockmass. The concept of rockmass stiffness was introduced to describe and quantify rockmass behaviour.

The objective of this part of the research is to develop a methodology that will describe and quantify the behaviour of the rockmass surrounding a stope. The rockmass stiffness is experimentally determined and a wide range of elements that influence its behaviour such as the horizontal stress and the inherent rockmass properties are inclusive of the underground measurements. The behaviour of the rockmass and its response to mining is determined here by means of underground closure profiles.

This model describes the support demand of the rockmass on the stope support. The rockmass behaviour must preferably be described in a way that it can be linked directly

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to the performance or capacity of stope support while giving a true reflection of underground environment.

The rockmass model is developed in such a way that it quantifies the rockmass stiffness at any specific point of interest in the stope for the given mining geometry. This is described by the function $g(x) = mx + F_o$, where the rockmass stiffness is represented by the slope (m) of the line at that particular point for the given geometry.

The methodology to develop this technique has undergone different stages of development. For the very first attempt a correlation between the cumulative square metres mined in a specific area of interest and the force calculated numerically by means of an elastic numerical model, was established. This method yielded a constant ratio between the cumulative square meters mined and the force calculated. The major shortcoming of this approach was establishing the initial area that was mined to be used as the so-called "starting area" that is described in Chapter 5.

The second attempt in quantifying the rockmass stiffness was to evaluate the combined deflection of two orthogonal beams with its centre positioned at the underground point of interest in the stope. This was accomplished by fitting two orthogonal beams at the point of interest to the closest solid abutments (or regional pillars) for the given mining geometry.

The forces that were calculated using numerical modelling at that point were compared to different variables of the ellipse described by the short- and long axis of the two orthogonal beams. This approach initially seemed to give a reasonable correlation between the force calculated numerically and the "effective side" of the ellipse. (The effective side is defined as the square root of the area of the ellipse.) The major shortcoming of this approach is that mining could take place outside the area covered by the ellipse but still in close enough proximity to the measuring station to have an influence on the force calculated by the numerical analysis. This methodology is not able to detect this and calculations or force predictions done under such circumstances can be grossly inaccurate. Figure 3.3 illustrates the elliptical fit to a specific mining geometry at the point of interest as shown.

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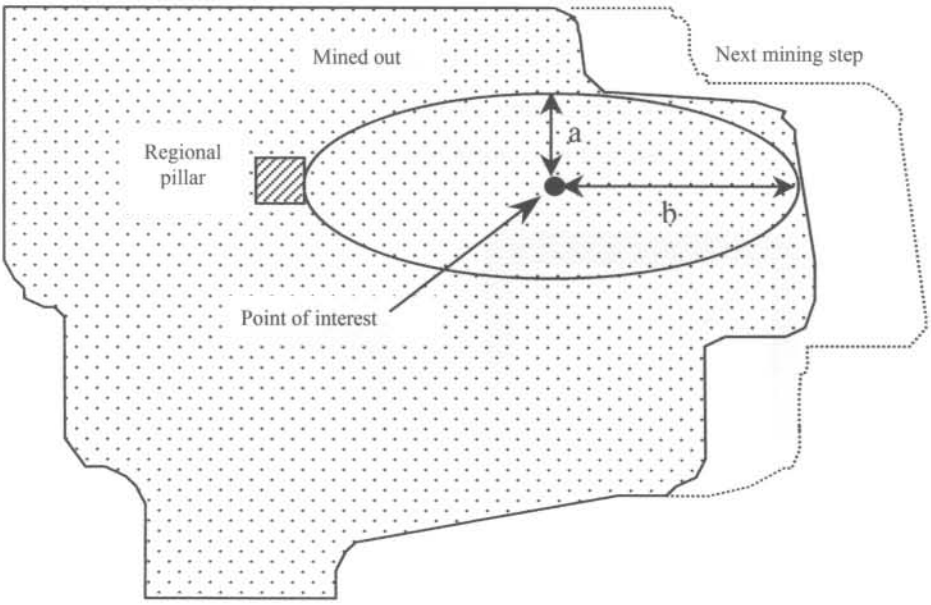

Figure 3.3: Illustration of the elliptical fit to a specific mining geometry

The third attempt that was adopted was the evaluation and study of the applicability of the yield line concept. The reason for reverting to this concept was to arrive at some methodology to determine the rockmass stiffness where it would be possible to take into consideration the face shape, mining geometry and presence of regional support.

The magnitude for the force component on the y-axis of the force-deformation representation is determined through the application of yield lines as described by Johansen (1962). Yield lines apply to plastic plates, such as reinforced concrete slabs and steel plates that are so lightly reinforced that failure begins when the reinforcement yields. The assumption was made that the hangingwall of the stope will react in the same manner when considering its closure profile and the fact that the hangingwall beam remains intact during stope closure similar to a reinforced concrete slab.

The positive ultimate moment corresponds to yielding of the bottom reinforcement and the negative ultimate moment to yielding of the top reinforcement. At failure, plastic deformations occur along yield lines where the reinforcement has yielded and the parts into which the slab is divided by the yield lines are only deformed elastically. Since the elastic deformations can be ignored in comparison with the plastic ones, the individual parts of the slab can be regarded as a plane, and their intersections, that is the yield lines, regarded as straight lines with a good degree of approximation.

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It is thus assumed that deformation occurs only in the yield lines, consisting of relative rotation of the two adjoining parts of the slab about axes whose location depends upon the supports. Each part may be regarded as plane, and will be evaluated as that. Yield lines for square and rectangular slabs are shown in Figure 3.4.

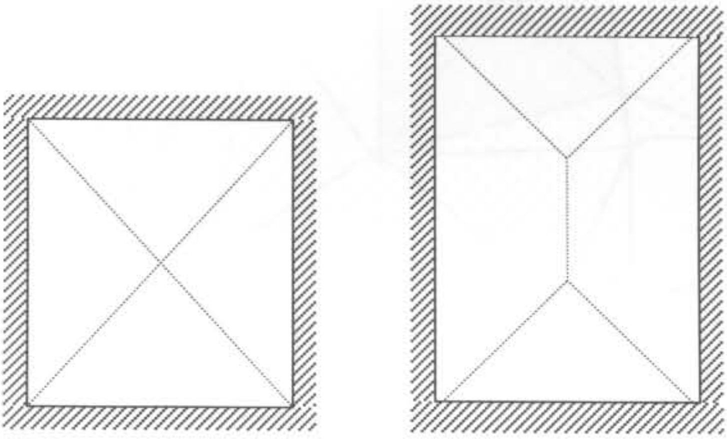

Figure 3.4: Yield lines shown as dotted lines for square and rectangular plates supported at the edges

The following basic rules are applied in establishing the yield lines. These rules are applied to the underground workings as illustrated and described in Chapter 5.

1. The yield lines between two parts of a slab must pass through the point of intersection of their axes of rotation.
2. For a part of a slab supported along its edge, the axis of rotation must lie along the edge, and for a part supported on a column the axis will pass over the column.

Yield lines for the same underground geometry as shown in Figure 3.3 are illustrated in Figure 3.5. The process applied during the construction of the yield lines is discussed in more detail in Chapter 5.

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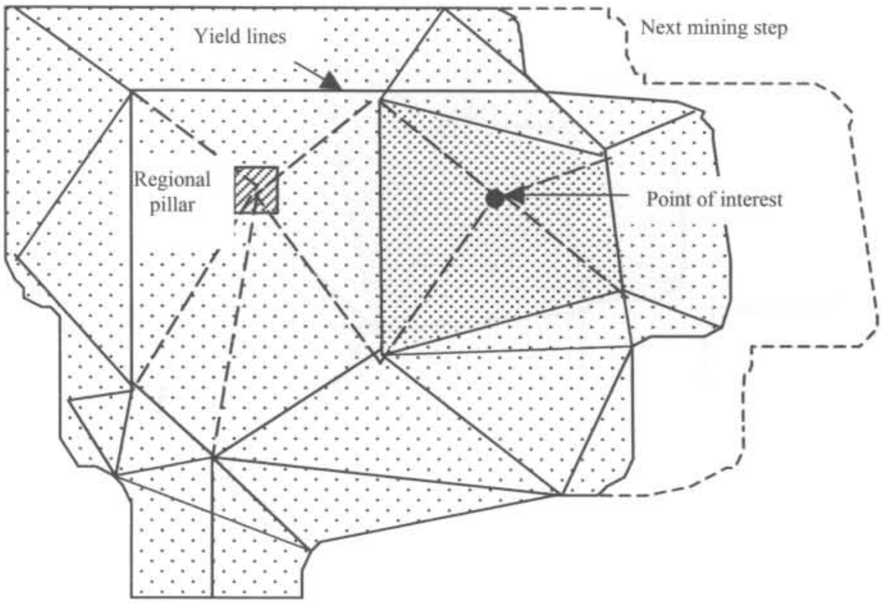

Figure 3.5: Yield lines for the same underground geometry as shown in Figure 3.3

The area represented by the yield lines around the point of interest was compared to the numerical value of the force determined numerically at that point for the given mining geometry. A good correlation was found to exist between the shaded area shown in Figure 3.5 and the force calculated at the point of interest for a given mining geometry.

This yield line principle was developed further to what is referred to as the attributed area concept. An area denoted as A_a is attributed to a point of interest for a given mining geometry. A representative force is calculated for this point of interest at a given mining stage and stope face shape from the attributed area A_a.

The local rockmass stiffness for that specific point underground is calculated using the force component from the attributed area. A certain thickness of hangingwall that is to be supported is taken into account while the closure component is taken as that measured underground. This process is described in detail in Chapter 5.

Figure 3.6 shows the mining layout previously used with the area attributed to the point of interest for the same mining geometry.

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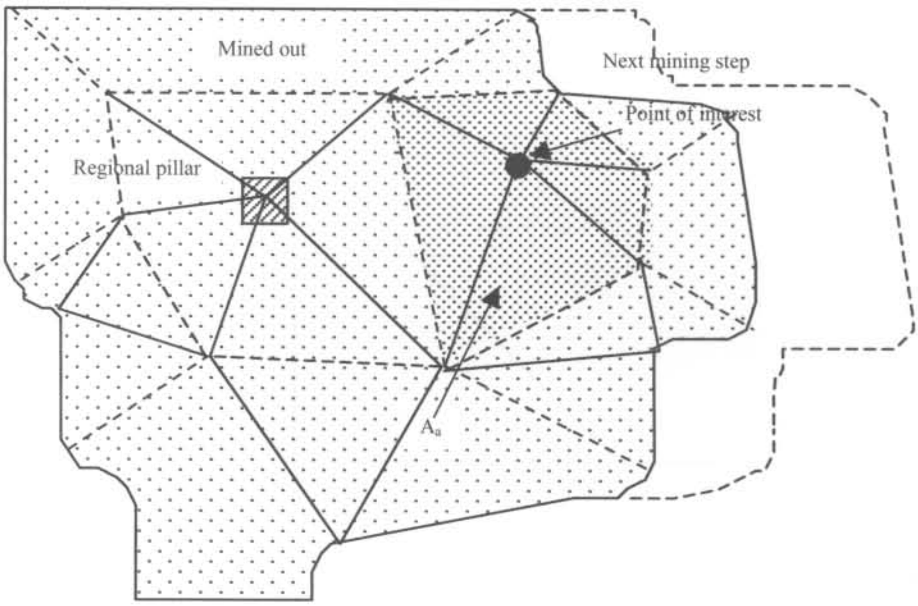

Figure 3.6: Mining geometry showing the attributed area A_a for the given point of interest

An area A_a is attributed to the point of interest and the force component for the rockmass stiffness diagram calculated. The force is a function of attributed area A_a and the thickness of the immediate hangingwall to be supported. The attributed area force is given by:

$$F_o = \rho.h.A_a.g \text{ (N)} \quad (3.7)$$

where:

F_o = Attributed area force (N);

ρ = Rockmass density (kg/m^3);

h = Thickness of hangingwall beam (m);

A_a = Attributed area (m^2); and

g = Gravitational acceleration. (m/s^2)

The rockmass load line is described by the following equation for varying deformation x:

$$g(x) = mx + F_o \quad (3.8)$$

where:

m = Rockmass stiffness (N/m);

x = Deformation of the stope (m); and

F_o = Attributed area force (N) where x = 0.

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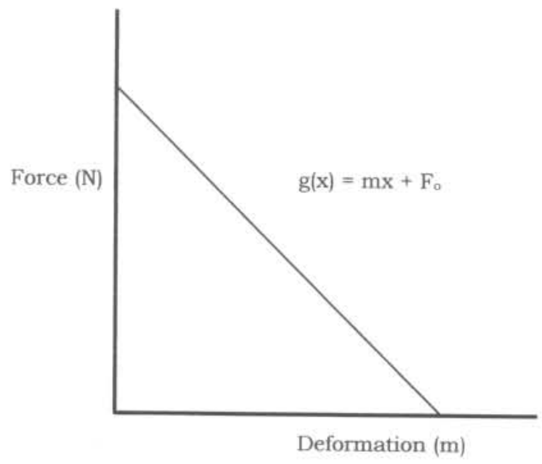

Figure 3.7: Graphical representation of a load line describing rockmass behaviour

(c) Combined model

The stope support and the rockmass models described in 3.2.1 (a) and in (b) are combined and compared during the third phase of the procedure and presented on a common force-deformation axis.

Stope support element performance is described by:

$f_a(x)$ = $f(x).Y.H_f$ - for timber and lightweight concrete packs; and

$f_a(x)$ = $f(x).Y.B_f$ - for timber elongates.

The rockmass is behaviour is described by the rockmass load line as $g(x) = mx + F_o$ with all symbols as defined previously.

To evaluate the stability of the total attributed area that could be supported by a combination of different support types, the function of the support element(s) is multiplied by a factor n_n as:

$$F(x) = \sum n_n . f_a(x) \qquad (3.9)$$

where:

n_n = Number of support elements of say type n,

= $A_a/(d.s)$ for the specific support type;

A_a = Attributed area (m^2);

d = Dip spacing of support units (m); and

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s = Strike spacing of support units (m);

$f_a(x)$ = Adjusted support performance function or the specific support type; and

x = Deformation (mm)

This principle of superimposing the rockmass and support models is illustrated graphically in Figure 3.8.

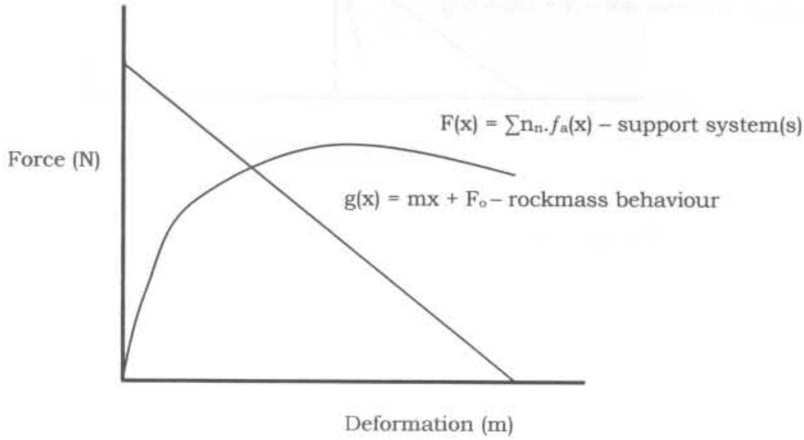

Figure 3.8: Graphical representations of the rockmass load line and stope support models on a common force-deformation axis

Unstable failure of the support will take place where the energy generated by the rockmass for a given interval of deformation exceeds the energy capacity that can be handled by the support system(s). The following criteria are used to test for stability/instability for a given mining geometry and support:

(i) Stiffness comparison analysis

The underground environment will potentially become unstable if the absolute value of the slope of the strata stiffness is less than that of the *post* failure stiffness of stope support system.
This is where:

$|m| < |\partial/\partial x[F(x)]|$ as illustrated in Figure 3.9 where the performance of both the stope support and the rockmass load line are superimposed on a common force-deformation axis.

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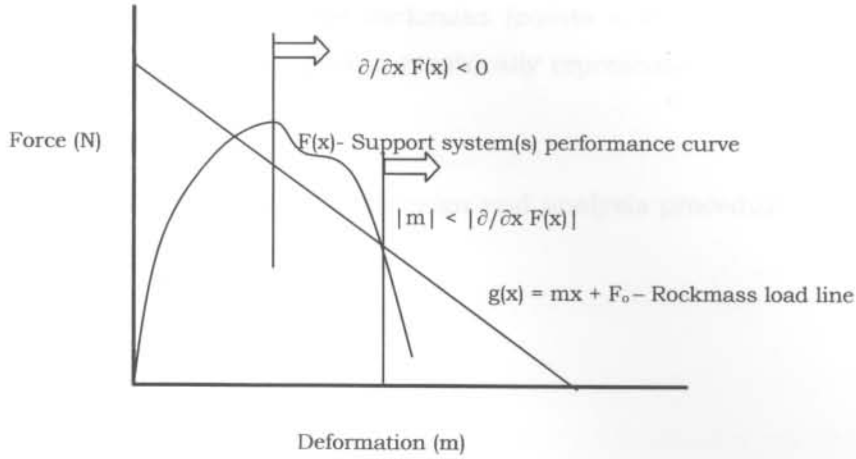

Figure 3.9: Combined models illustrating failure of the supports

(ii) Energy comparison analysis

The condition where the energy generated by the rockmass exceeds that which can be absorbed by the support system(s) may result in an unstable situation.

Mathematically where $\int_a^b g(x)\partial x > \int_a^b F(x)\partial x$ for a given deformation interval x, say a to b, will this result in unstable failure of the support system(s). This principle is illustrated in Figure 3.10.

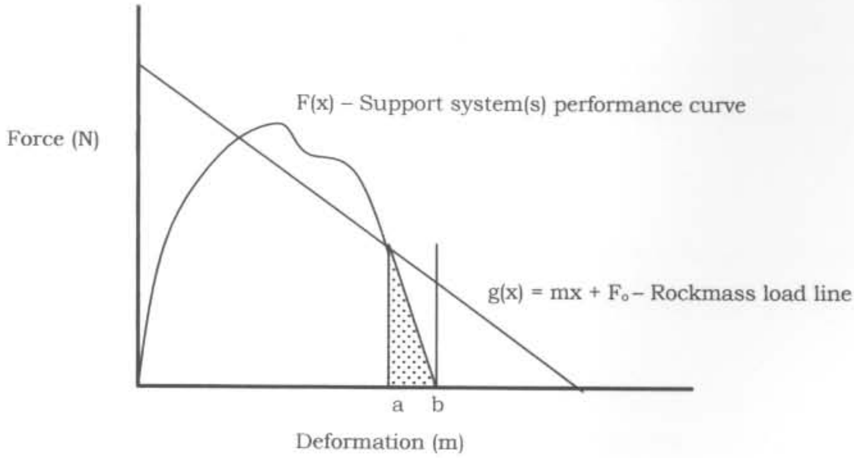

Figure 3.10: Graphical representation of unstable failure of a support system

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The excessive energy generated by the rockmass (points a to b) that will result in unstable failure of the support system(s) is graphically represented by the shaded area in Figure 3.10.

Case studies where the application of this design and analysis procedure is illustrated are shown in Chapter 7.

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