CHAPTER V

CASE STUDIES

Validating the Q-system requires an unsupported excavation on the mine which in today's context is rare. The 1st case study, 10 level crosscut on No. 9-Shaft, Impala Platinum, was developed in 1981 and supported only with spot bolting in wider span sections.. The tunnel was also geologically logged in 1981.

The 2nd case study, chosen to provide a more even spread from good to poor rockmass conditions, is a conveyor decline tunnel at No. 14-Shaft Impala Platinum. This tunnel is currently being developed and needs to be supported immediately because of block fallout's soon after the blasting operations. The above is suitable for the study because a detailed log of the rockmass response to tunneling can be kept for the purposes of evaluating the Q-System.

5.1 Methodology

The Q-system is assumed to include enough information obtained from underground observations to provide a realistic assessment of the rock mass strength and hence the stability of any excavations developed in that rock mass. The Q-system parameters is given below :

$$Q = \frac{RQD}{Jn} \times \frac{J_r}{J_a} \times \frac{J_w}{SRF} \qquad (5.1)$$

where, RQD is Rock Quality Designation, Jn is the joint number, Jr is the joint roughness, Ja is the joint alteration, Jw is the joint water and SRF is the stress reduction factor.

The methodology used to estimate each of the above parameters from underground observations will be discussed briefly.

5.1.1 Estimating the RQD from Scan line Measurements

The RQD for a rock mass can be calculated from scan line measurements taken underground. A scan line is defined as a line, usually a tape, set on the surface of the rock mass, and the survey consists of counting the number of joints which intersect this line along its length. The scan lines were chosen to be 10 meters long to avoid the possibility that the joint spacing is greater than the length of the scan line.

In the tunnels the tape was laid to the length of the tunnel to form the scan line. The number of joints or planes of weakness that crossed the tape in those ten metres were counted. The number of joints counted divided by the distance of 10 metres gives the number of joints per metre in that direction. This value is equal to S in equation 5.2.

For the width of the excavation (i.e. 0-5m) jointing was counted and divided by the width of the excavation in the 10m section. This value is equal to D in equation 5.2.

Observations included any falls of ground or areas where portions of the hangingwall have been exposed in the vertical direction, where low angle or horizontal joint and other discontinuities were seen. The number of joints or planes of weakness that occur in the vertical direction was also measured and divided by the height distance. The value obtained is equal to V in equation 5.2. The sum of the joint densities for the three directions are calculated using equation 5.2 and RQD calculated using equation 5.3.

$$J_D = V + S + D \tag{5.2}$$

$$RQD = 115 - 3.3 \times J_D \tag{5.3}$$

If the RQD obtained from equation 5.3 was less than 10%; the value entered into the Q-rating equation was 10. If the value obtained from the equation 5.3 was greater than 100% the value entered into the Q equation was equal 100.

5.1.2 Estimating Jn

This number is a measure of the number of joint sets observed at the site. To select the correct discontinuities as joints it was necessary to work according to the following definitions :

Aperture - is the perpendicular distance between adjacent rock surfaces of a discontinuity tendon.

Joint - is a break in the rock of a geological origin, not man made, along which there has been no visible displacement or movement.

Joint set - is a group of joints, which run parallel to each other and a joint system is made up of two or more intersecting joint sets.

Random joints - are joints which do not have the same orientation as the joint sets observed. They are not visible for long distances, only a couple of centimeters or perhaps meters.

The rock mass rating sheet (Table 4.9) was used as a guide to obtain the values entered into the Q rating equation.

5.1.3 Estimating Jr

The Joint Roughness is defined as the measure of the surface unevenness and waviness of the joint relative to its mean plane. This unevenness and waviness will influence the ability of the two surfaces to slide against each other. This has an interlocking effect that prevents the blocks from sliding. In underground observations it is important to distinguish between the unevenness, which is a small-scale feature and waviness which is a large-scale feature (see Figure 5.1). The in-situ shear test provides us with a value which can be defined as the internal angle of friction.

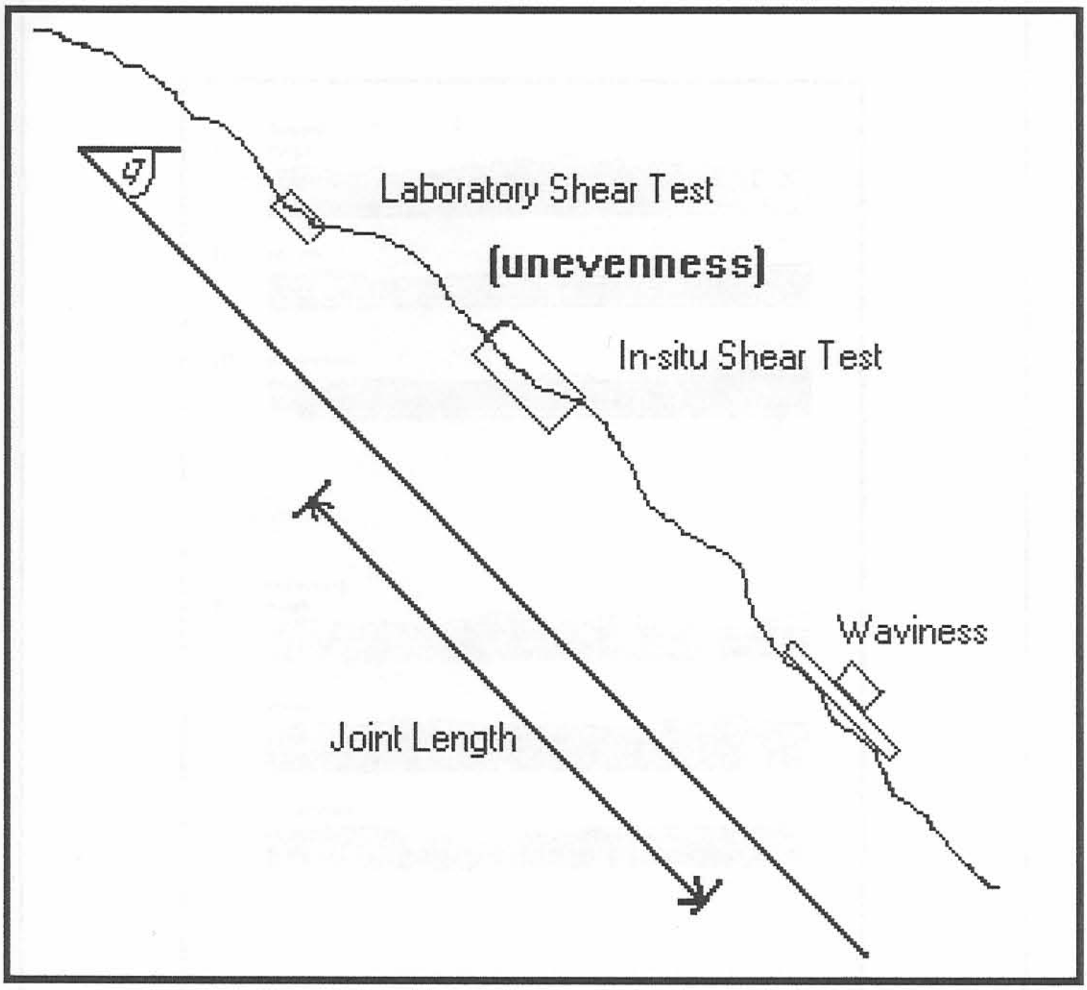

FIG. 5.1 - Different scales of roughness, small scale of laboratory shear test, medium scale of an in-situ shear test and the large scale waviness of the joint (After Barton et al, 1974)

Figure 5.1 shows the difference between small scale and large-scale roughness. This will assist the user to make a final decision about the joint roughness number. The internal angle of friction is used to determine joint alteration number in the absence of mineralogical properties.

The next step was to observe the joint roughness, which in many occasions were tightly closed and were difficult to define in the limited dimensions of the tunnel. Definitions shown in Figure 5.2 and in Table 4.9 were used as a guideline in defining Jr.

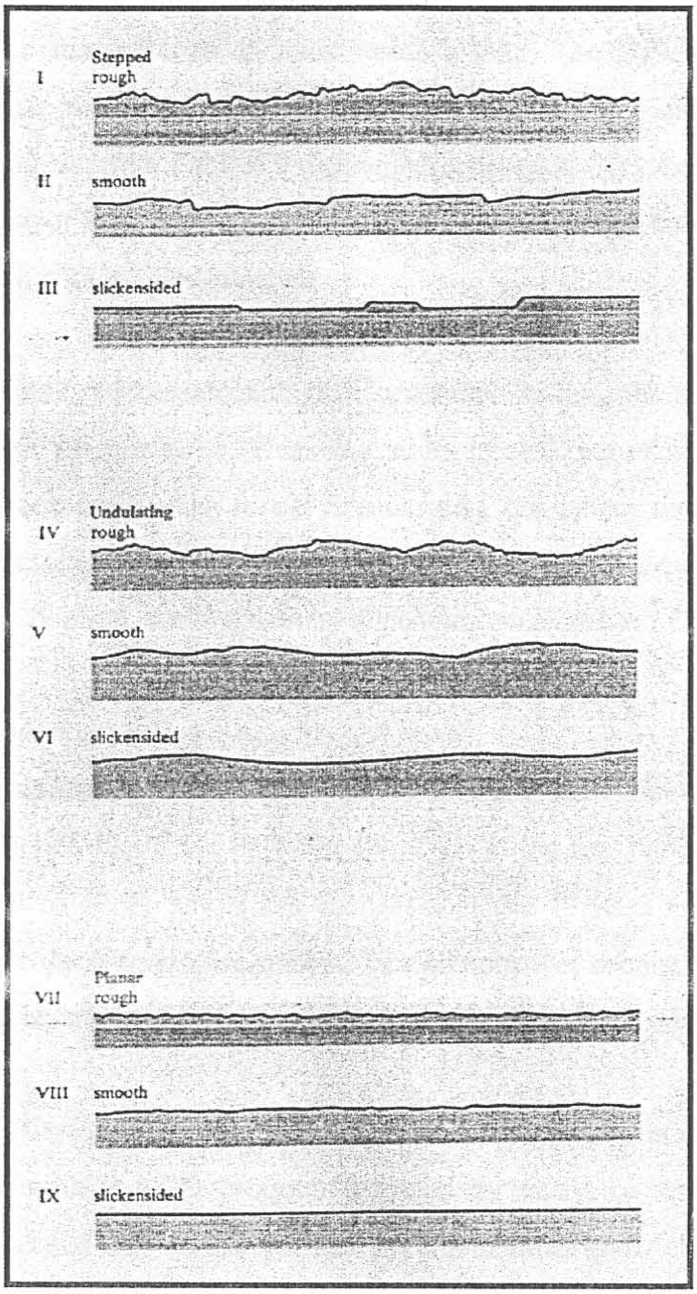

FIG. 5.2 - Profiles of different classes of joint roughness (After Barton et al, 1974)

5.1.4 Estimating Ja, Jw and SRF

Joint Alteration refers to the filling found along the joint plane. The thickness and strength of the filling determines the strength of the joint and its ability to resist slipping. The joint alteration can range from tightly closed joints with no filling to joints with fillings thicker than 3mm or zones of crushed rock. Table 4.9 was used as a guideline. Water is very critical to the stability of excavations and consequently there is an adjustment to de-rate the joint strength due to the presence of water inside a joint. The presence of water will reduce friction or cause the filling in the joint to weather, thus increasing the instability of the hangingwall and sidewall. Table 4.9 distinguishes between a dry excavation or minor inflow, medium inflow or pressure, outwash of joint fillings, large inflow or high pressure in competent rock with unfilled joints etc.

The SRF component includes geological structures in the rock mass. The SRF is divided into three major categories i.e. Weakness zones intersecting excavation, which may cause loosening of rock mass when tunnel is excavated, competent rock, rock stress problems, squeezing rock, plastic flow of incompetent rock under influence of high rock pressure and swelling rock, chemical swelling activity depending on presence of water.

5.1.4.1 Numerical modeling using MINSIM W

It was further necessary to investigate the stress influence to choose the correct category in the stress reduction factor list in Table 4.9. Thus in the first case study the stress analysis was conducted to verify any stress changes that the 10 level crosscut might have been subjected to. In the second case study, 23 Level conveyor decline, which is currently being developed and no stoping had been undertaken, therefore virgin stress conditions prevail.

The program MINSIM W (CSIR, 1997) was used to conduct the stress analysis for the first case study. Minsim is a 3D program designed for analyzing stresses and displacements which are associated with tabular excavations and assumes linear elastic behaviour. It is well suited to the analysis of narrow tabular stopes as those found in the platinum and chrome mines of the Bushveld complex. MINSIM is optimized for deep level mining, but can be used for modeling shallow mines if the finite depth option is selected (Jager and Ryder, 1999).

In its basic form, MINSIM comprises two separate programs. The data input for these programs is in the form of pure ASCII text files, which can be created or edited with any ASCII text editor, such as Microsoft's notepad. However, the input data is fairly complicated, and even slight errors can lead to major problems at a later stage (Jager & Ryder, 1999).

5.1.5 Evaluating the Q-value

The Q-Value was obtained using the guidelines outlined above and in Table 4.9. This was compared to Figure's 4.7, 5.3, 5.4, 5.5 and Table A.1, A.2, A.3, A.4 and A.5 to evaluate the Q-System for underground rockmass classification application.

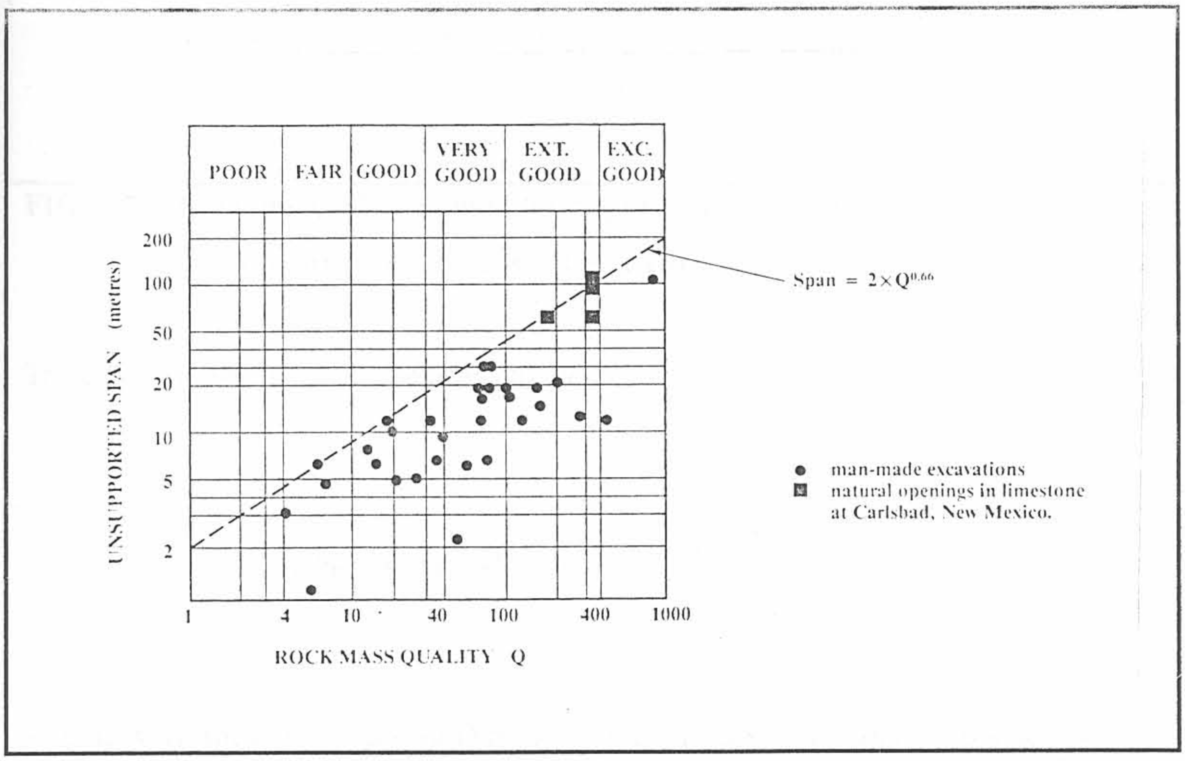

FIG. 5.3 - Man made and natural unsupported excavations in different quality rock masses (After Barton, 1976)

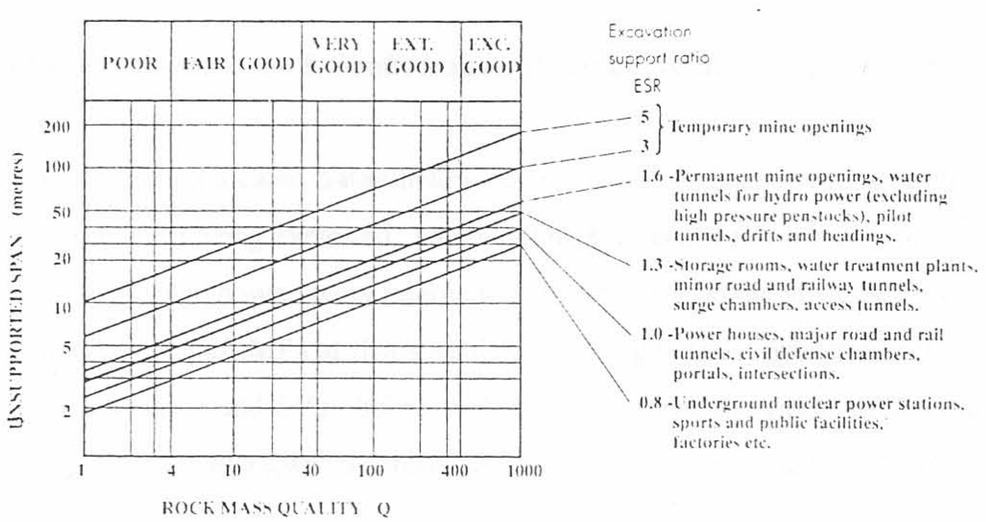

FIG. 5.4 - Recommended maximum unsupported excavation spans for different rock mass quality (Q) and ESR values (After Barton, 1976)

The equation which defines the lines plotted in figure 5.3 can be written as follows :

$$\text{Span of opening} = 2 \times ESR \times Q^{0,4} \tag{5.4}$$

Alternatively, the critical value of Q for a given excavation span can be expressed in the form :

$$Q = (span / (2 \times ESR))^{2,5} \tag{5.5}$$

The support guideline issued by Barton et al (1977) can be viewed in Appendix A. Support for categories 1 to 38 is listed in Table A.1 to Table A.4 . Figure 5.5 is a rough guideline of the placing of the categories used in the Q-system.

5.2 10 Level Crosscut, No. 9-Shaft, Impala Platinum (Plan 1 - Appendix E)

10 Level Crosscut is located 640m below surface at No. 9-Shaft where both the Merensky and UG2 Reefs are being exploited. The middling between the two reefs varies between 90 and 100m, with no known stress interaction to date. The initial stress state at No-9-shaft was estimated to be 20 MPa at this specific level using 9,8m/s^2 (gravitational acceleration), 3200 kg/m^3 (density) and 640m below surface to determine the virgin stress condition. The crosscut intersects hangingwall 1 through to hangingwall 5 (See Figure 2.2, p8).

The crosscut average width is 3,0m with sections that widen out to 5.2 metres in places. The only support installed in this crosscut was at these wider sections and spot bolting was conducted in one area. The tunnel was developed in 1981, over the last 20 year period the excavation has been subjected to water, ventilation and possible stress changes. The Q-system was used to obtain a Q-value for this tunnel along seventy seven 10m intervals representing the total tunnel.

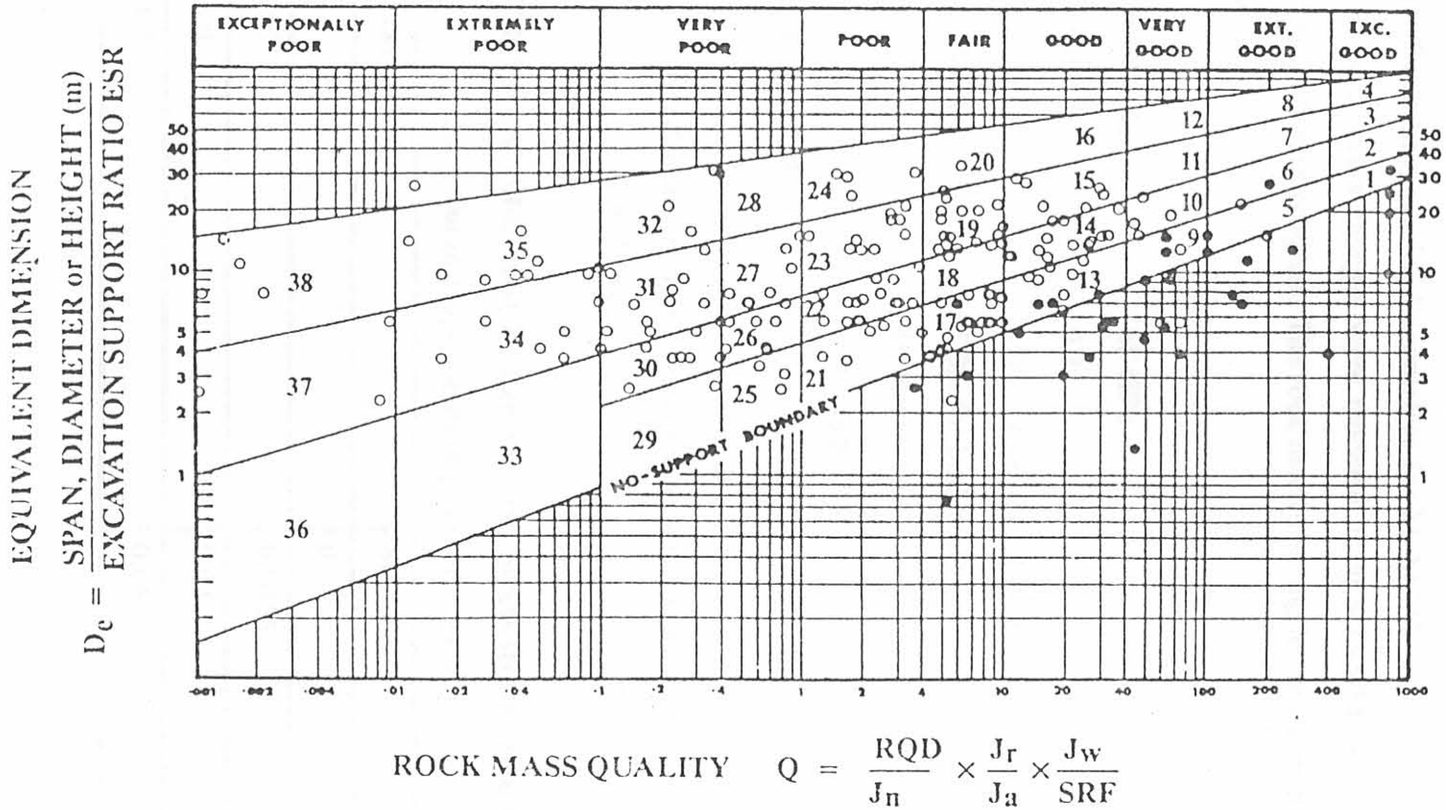

FIG. 5.5 - Recommended support for different rock mass quality (Q) and ESR values (After Barton et al, 1977)

5.2.1 MINSIM W, Computer Modeling

The MINSIM W program was used to determine the excavation stress reduction parameter.

5.2.1.1 Rock Engineering Parameters for MINSIM

The crosscut stress changes were modeled using MINSIM version 3.2 (CSIR, 1997) for Windows software program. The rock mechanics modeling parameters used are stipulated below :

Young's Modulus	-	68 GPa
Poisons Ratio	-	0.2
K-ratio	-	1.0 (Spencer, 1993)
Stoping Width	-	1,0m for both reefs
Coarse size	-	10 metres
Depth option	-	Finite depth
Backfill "Soup" Width	-	1,0m

Backfill stress / strain relationship is used to simulate the behaviour of the 3m x 6m yielding pillars shown below in Table 5.1.

TABLE 5.1 - Backfill "SOUP" Parameters - Stress/strain relationship of the 3m x 6m in stope yielding pillars (After T.J. Kotze, 1997)

Stress (MPa)	Strain
0	0
-5,65	0,0025
-14,0	0,04
-14,3	0,06

5.2.1.2 Mining Steps

Table 5.2 lists the various mining steps involved in conducting a simple stress analysis. The stress analysis we are interested in is the 1st two mining steps. These mining steps provide information regarding any stress changes that the tunnel might have been subjected to.

TABLE 5.2 - Mining steps modeled using MINSIM W

Mining Step	Action
Step 1 – 12/97	Merensky reef mined out – Merensky Crosscut Pillar left intact
Step 2 – 01/98	UG2 reef mined out with 30% Geological losses and a 70m wide crosscut left intact

5.2.1.3 Model

Six on reef windows were placed on the Merensky reef and 7 on-reef windows on the UG2 reef. For the off-reef stress analysis five vertical sheets were used to determine stresses on the various levels. These 5 off-reef windows were not used by choice but rather because the limitation inaccuracy of one window where each line would have represented 40m. A plan view of the UG2 in the model is shown below in Figure 5.6 with the window placing shown in Figure 5.7.

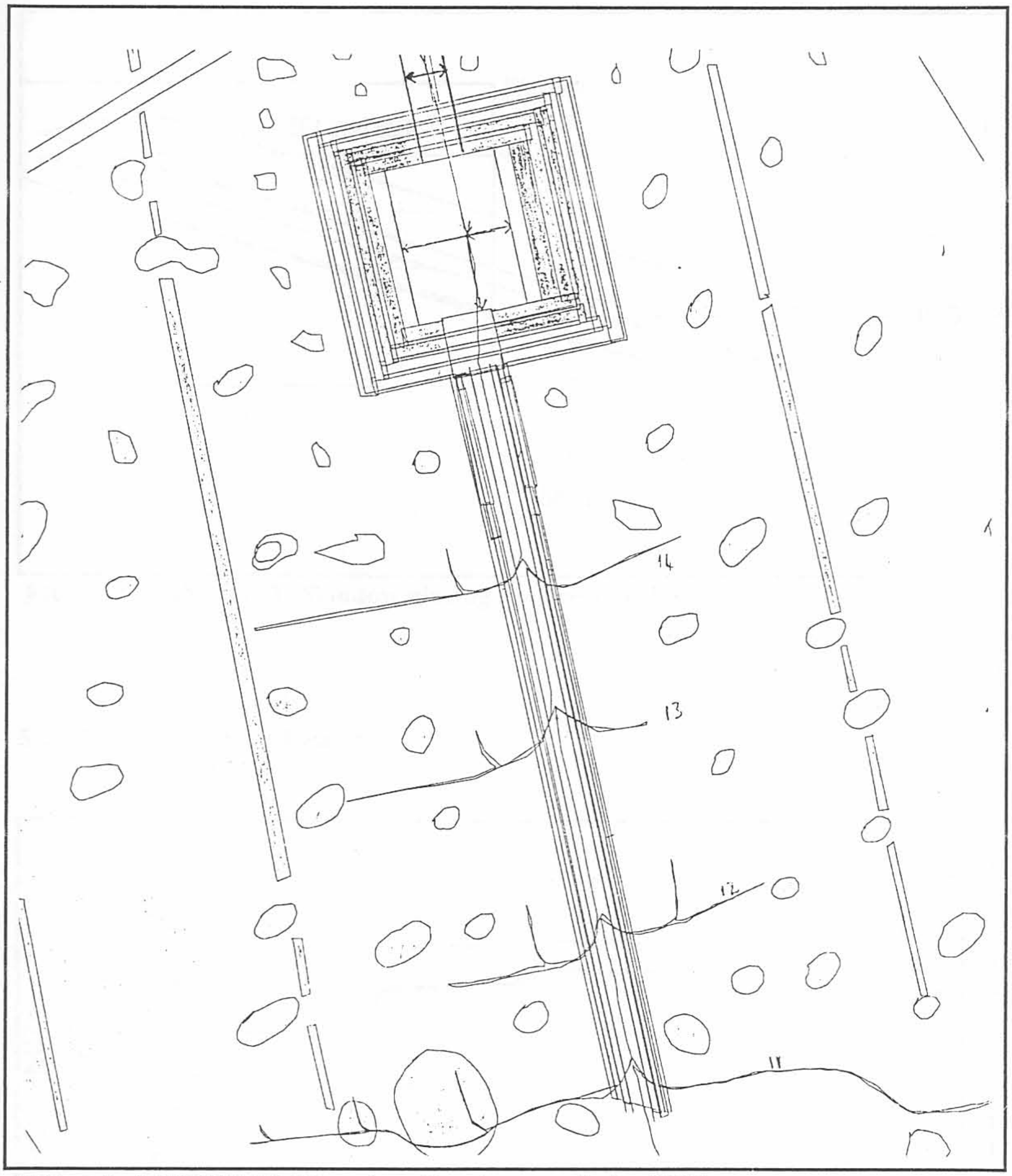

FIG. 5.6 - MINSIM W, Plan view for stress analysis - UG2

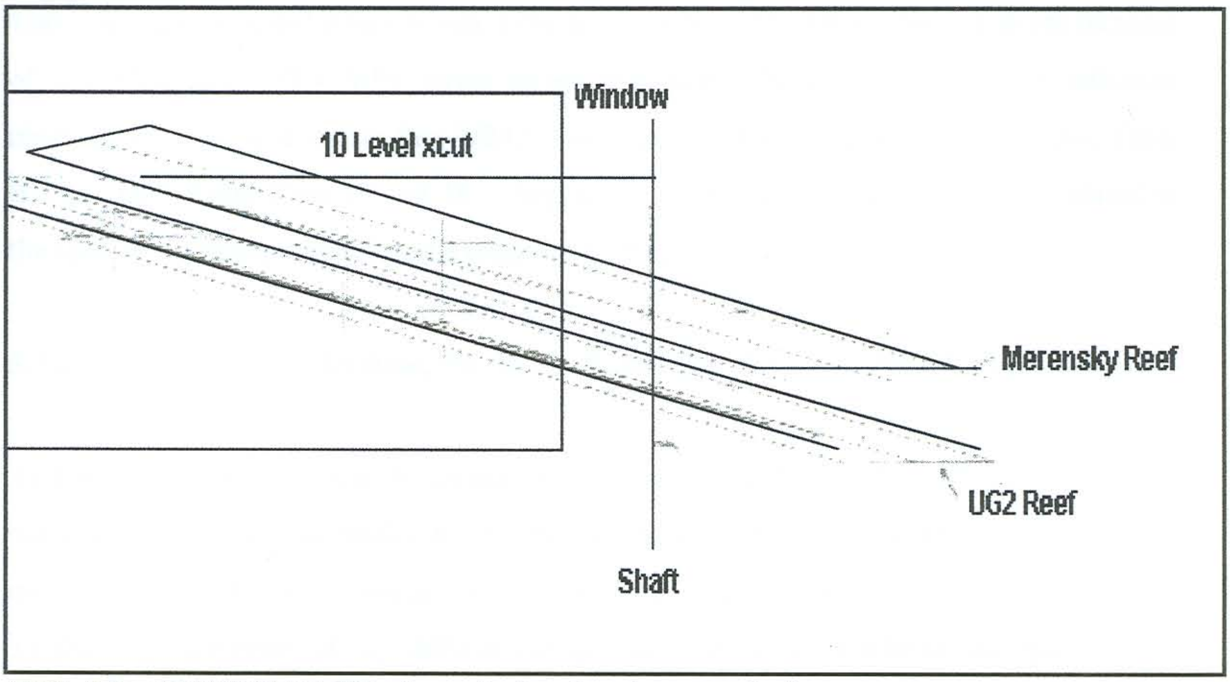

FIG. 5.7 - MINSIM W, Window placing for stress analysis

5.2.1.3 Stress Analysis Results

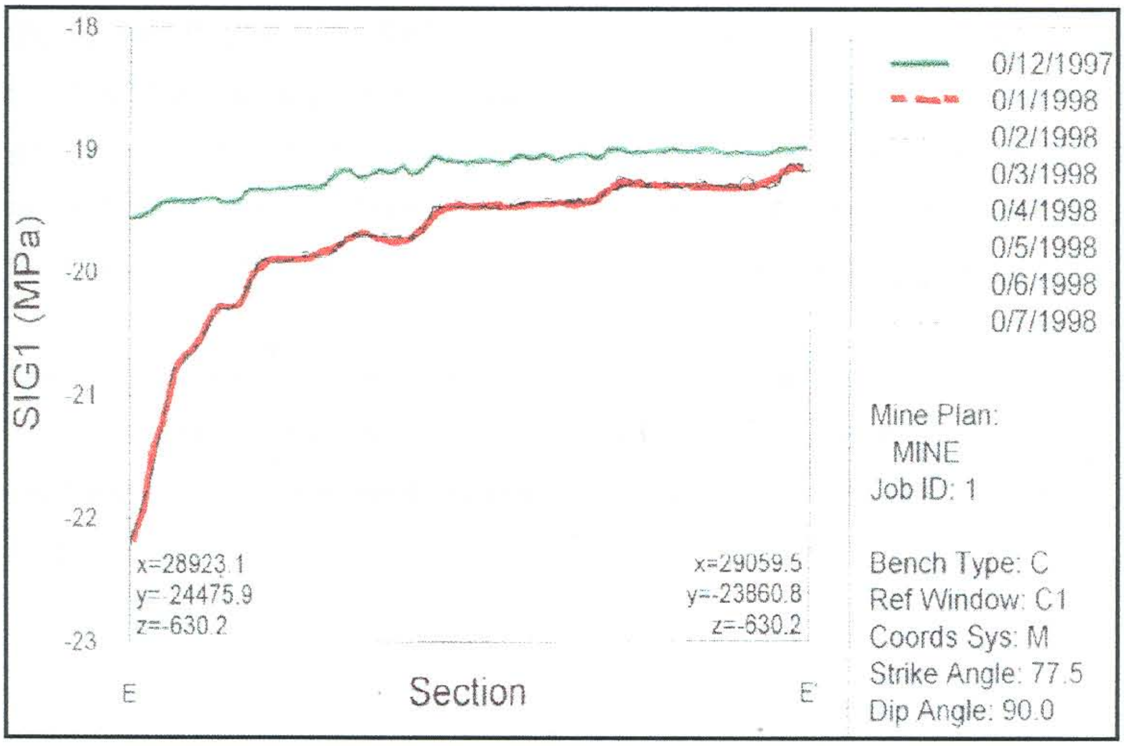

FIG. 5.8 - Section of crosscut showing stress state prior to and after mining of the two reefs

The maximum principal stress Sigma 1 reaches a high of 22,1MPa. This is a stress increase of 2,7 MPa from 19,4 MPa virgin stress condition. The only visible stress influence observed underground was at Peg W24331 where stress fracturing was observed (see Table B.1 site no. 19 and Plate 17 and 18 – Appendix C). The stress fracturing can be related to the spotted anorthosite which has a tendency to fracture when exposed.

5.3 23 Level Conveyor Decline, No. 14-shaft, Impala Platinum (Plan 2 - Appendix E)

23 Level Conveyor Decline is located 1058m below surface at No. 14-Shaft, which is currently mining the Merensky Reef. The conveyor decline extends the depth to which mining can take place, known as the "Deeps". The initial stress (virgin stress) state at No. 14-shaft is in the order of 32,3 MPa at this specific level. There will be no anticipated stress changes which will enhance the instability of the excavation. The conveyor decline is currently being developed in footwall 16 anorthosite, which can be viewed in the generalised geological succession which is shown in figure 2.2. The decline is developed with an average width of 5,6m with 30cm over break in most sections.

The support installed in the decline consists of 3,0m long, 16mm diameter, shepherd crooks which are full column grouted at a spacing of 1,0m on both dip and strike. The tunnel has given rise to a fair amount of concern with hangingwall and sidewall stability with fallout's up to 2m high occurring. Commonly these fallout's occurred almost immediately following the blast. This excavation has been subjected to water and ventilation for two months.

The Q-value rock mass classification was conducted using Table 4.9 as a guideline. 12 Stations were Q-rated at 10m intervals. The highest value was determined to be at 1.3 and the lowest value was 0.6 thereby ranging between poor and very poor as shown in Table B.1).

5.3.1 23 Level Conveyor Decline Stress Analysis

The stress analysis showed that there is little stress change in the conveyor decline. In the future it is anticipated that stress change will not influence the stability of the excavation. The virgin stress state will have to be taken as the ultimate stress state, namely 32,14 MPa. The K-Ratio for this depth on Impala is assumed to be one, which means a hydrostatic stress state (Impala Platinum Ltd., 1999).

5.4 Q-Rating information analysis of 10 level crosscut and 23 level conveyor decline

The ratings obtained (See Table B.1 and B.2) were scrutinized in detail to determine *critical Q-value parameters (*See Table B.3). The Q-Values were plotted and showed a fairly uneven spread throughout the exceptionally good to fair Q-Value classification of 10 level crosscut to a poor very poor classification of 23 level conveyor decline (see Figure 5.9).

The rockmass in the 10 level crosscut was classified as good (45%), very good (25,9%), exceptionally good (12,9%) and fair (14,9%). The one poor ground condition case (1,3%) is mainly due to a multiple shear zone which is not supported. Six of the seven fair ground condition cases are also not supported. One of these sections is a wider section and is supported with 1,8m long shepherd crooks, spaced 1m apart.

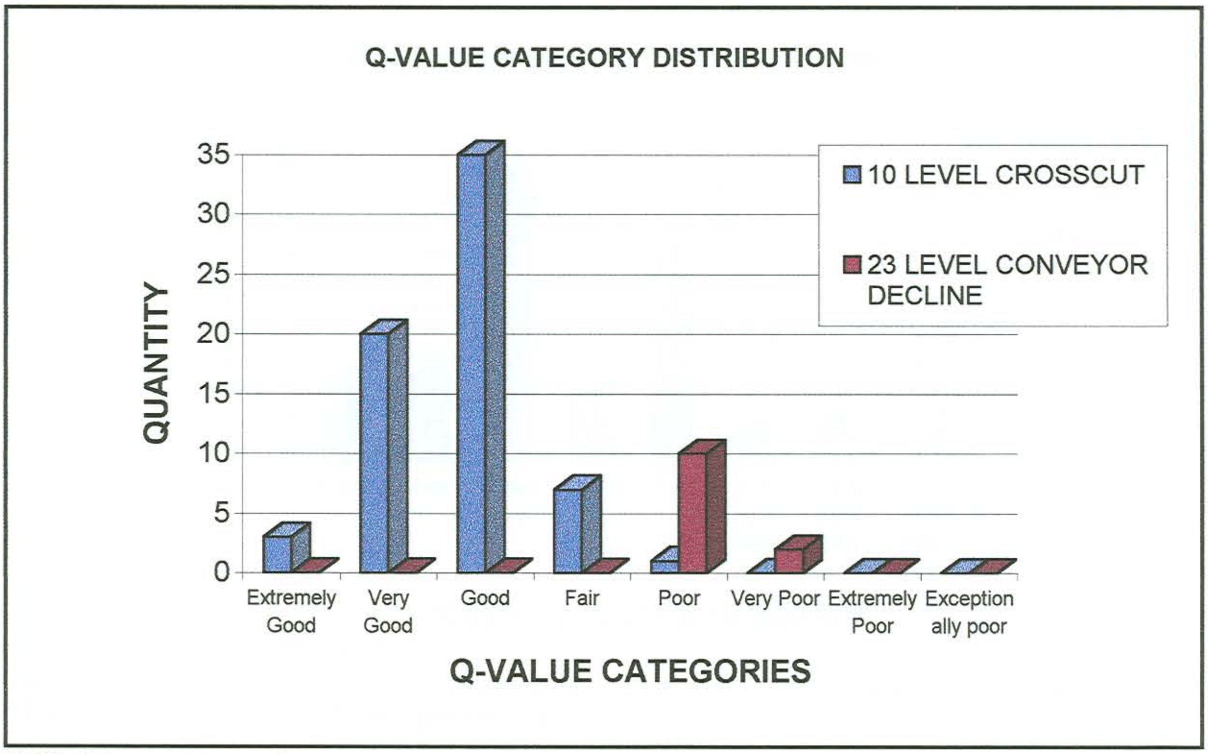

FIG. 5.9 - 10 Level Crosscut and 23 level conveyor decline Q-value distribution

The conveyor decline critical Q-value parameters mainly consist of poor (83%), very poor (17%). For this case the excavation will be compared to supported workings (See Figure 5.4, p 70). The 3,0m long shepherd crooks are spaced 1m apart on dip and strike. These parameters / categories were further broken down in the sub headings describing the typical ground condition characteristic (see Table B.4).

In Figure 5.10 it is shown that the ground condition by means of *joint number category* mainly consists of 1 joint set which represents 35% of the tunnel that was rock mass classified using the Q-system, followed by two joint sets (25,9%) and the rest of the plot tends to the massive rock mass with fewer joints presence.

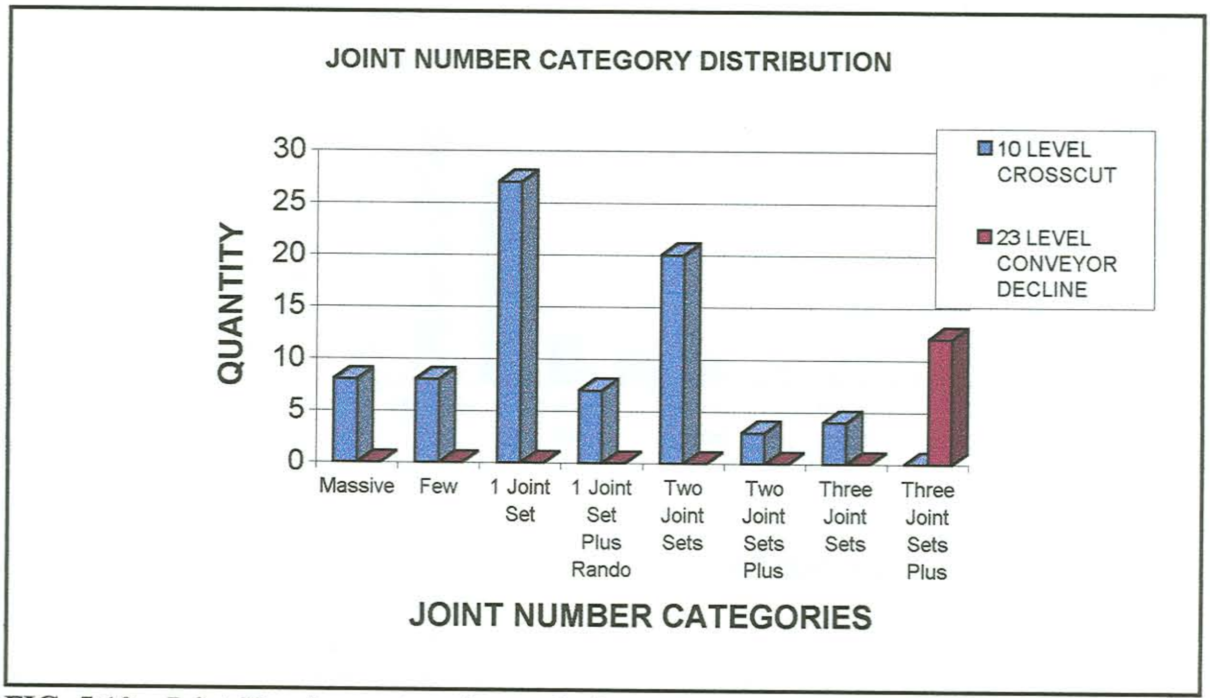

FIG. 5.10 - Joint Number categories analysis

In Figure 5.10 it is shown that the ground condition by means of joint number category in the 23 level conveyor decline mainly consist of 3 joint sets plus random (i.e. 100% of all the stations q-rated). The joint number category was analysed and plotted against the average q-value in Figure 5.11. Thus the average Q-value (1,2) obtained in the 23 level conveyor decline compare quite well with the three joint sets observed underground in the excavation.

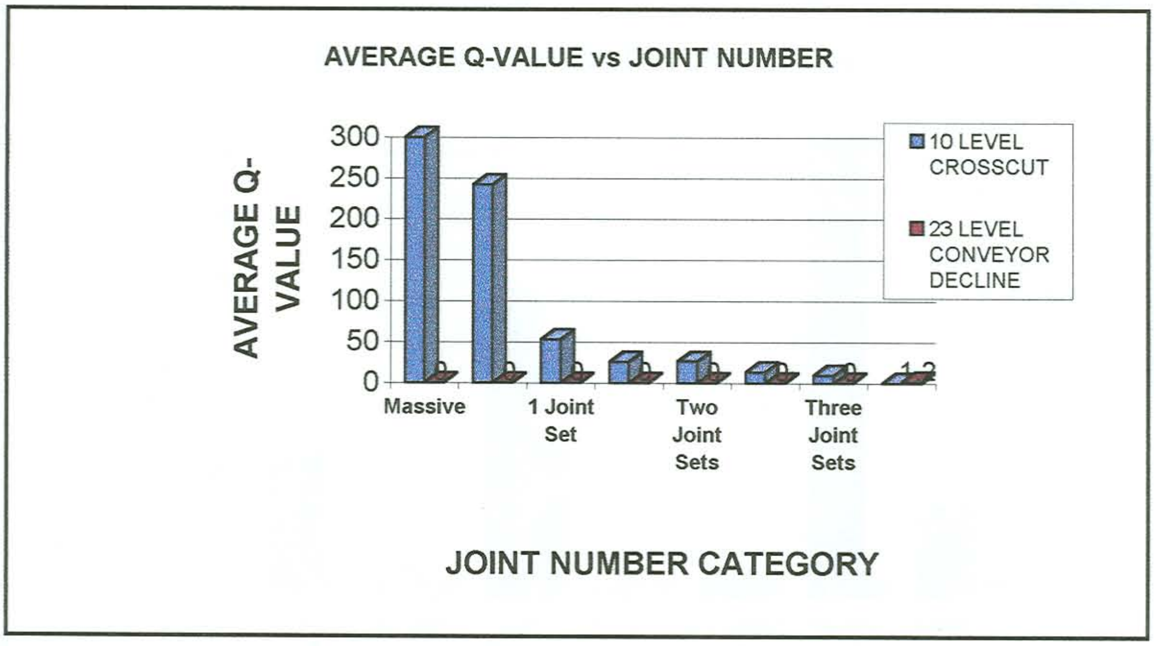

FIG. 5.11 - Joint Number categories vs Average Q-values

It is evident in Figure 5.12 that the 10 level crosscut *jointing roughness* consist mainly of rough / irregular planar (45%), rough / irregular undulating (29,8%) and joint spacing further than 3m (12,9%). This is a further indication of the very good ground conditions that was experienced in the crosscut over the last 20 years.

The spread of the joint roughness categories (Figure 5.12) in the 10 level crosscut is not so even from joint spacing >3m through to smooth planar parameters. This is also revealed in the plot against correlation with the Q-Values (see Figure 5.13). The 23 level conveyor decline jointing roughness however consists mainly of the rough / irregular undulating category (see Figure 5.12). This can be viewed in Plate 26 and 27 (Appendix D). This is slightly contradicting with the very poor ground conditions experienced in the conveyor decline soon after the blasting operation. This joint roughness is supported by the measured internal angle of friction of 35 degrees (see Plate 28 - Appendix D). There can be a slight adjustment due to size affect. The two samples were taken from the sidewall and were hand specimen size.

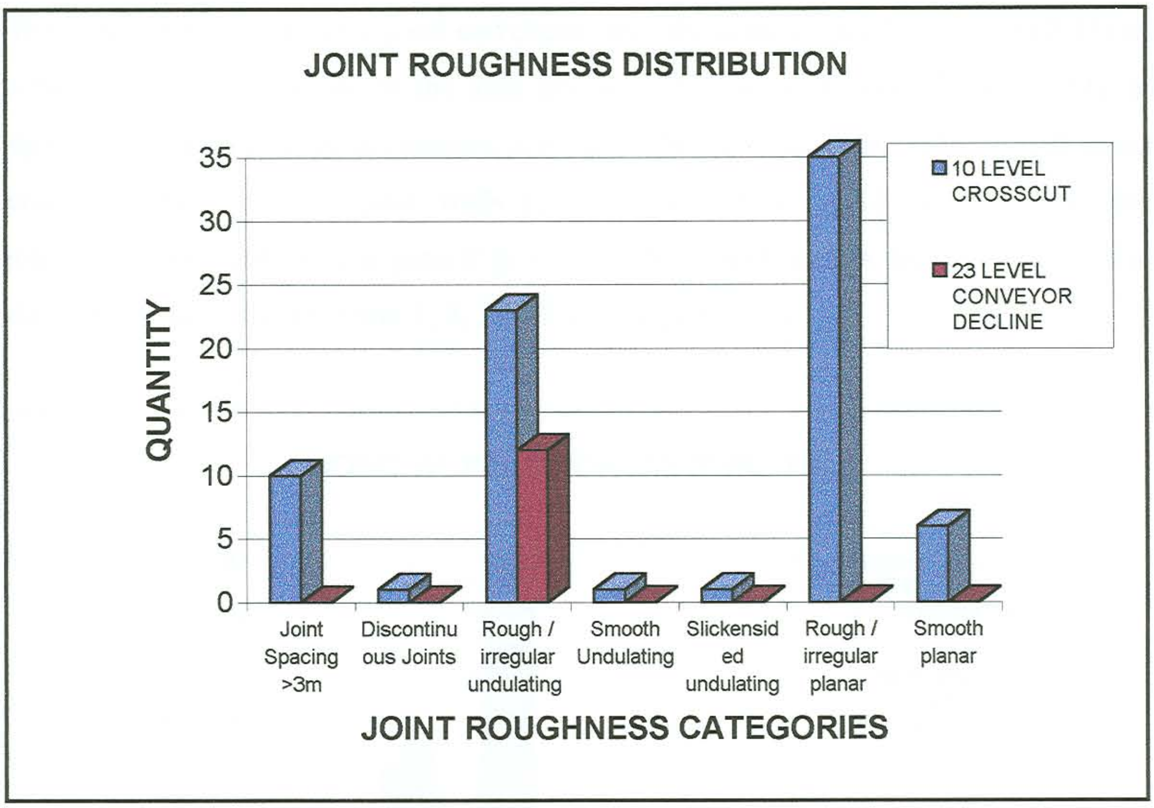

FIG. 5.12 - Joint roughness categories

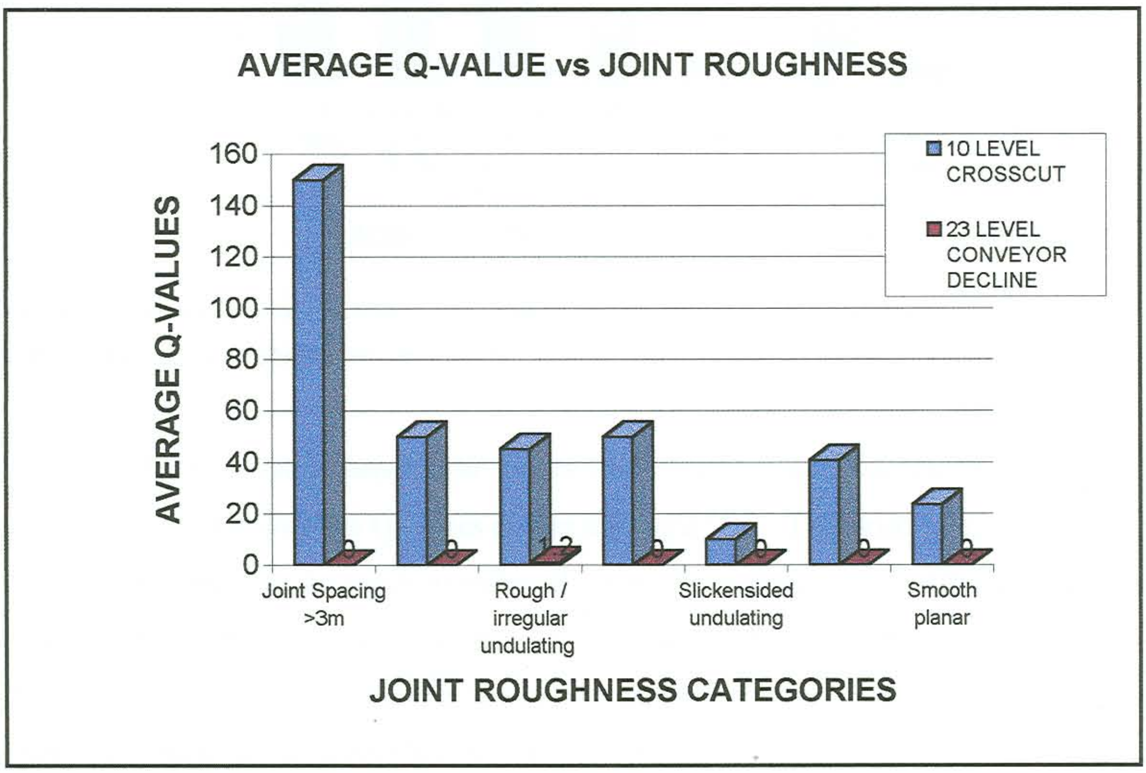

FIG. 5.13 - Joint Roughness categories vs Average Q-value

The *joint alteration* shows a good correlation with the average Q-values (Figure 5.15) with a fairly normal distribution of the joint alteration sub parameters (see Figure 5.14). The significant joint alteration parameters are the slightly altered joint wall-non soft mineral coating (52%), unaltered joint walls (28,5%) and tightly healed (12,9%). The above information substantiates the general ground condition and justifies the excavation stability throughout the years (see Plate 1, 2, 3, 7, 8,13 - Appendix C).

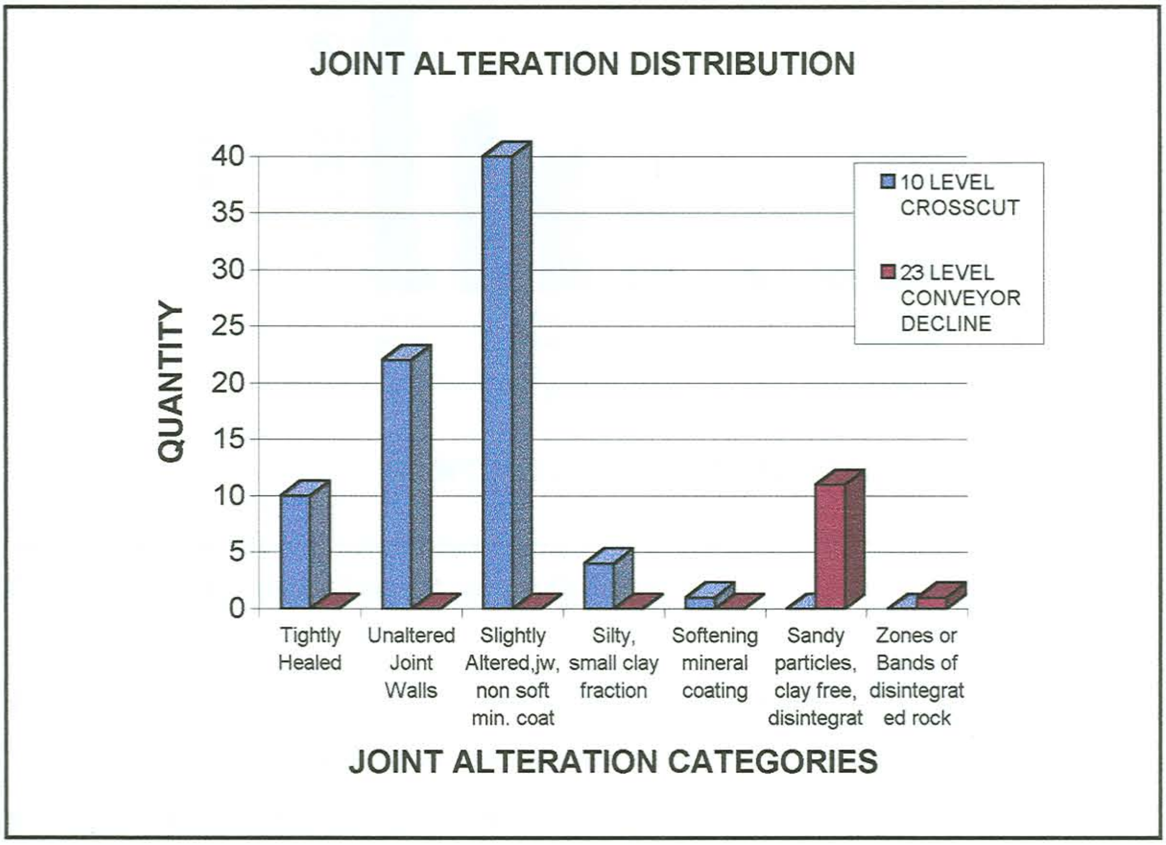

FIG. 5.14 - Joint Alteration categories

The joint alteration sub parameters in Figure 5.14 provide a skew distribution with a good correlation with the average Q-values shown in Figure 5.15. The most significant joint alteration parameter is the sandy particles, clay free, disintegrated rock (83%) and zones or bands of disintegrated rock (17%). This can be viewed in Plate 26 and 27 (Appendix D). The above information substantiates the general ground condition and justifies the excavation instability so soon after the blasting operation .

The plot in Figure 5.14 supports poor ground conditions in the 23 level conveyor decline and can be taken as a general occurrence on Impala when poor ground conditions are found which are structurally controlled and the stability of the excavation is influenced.

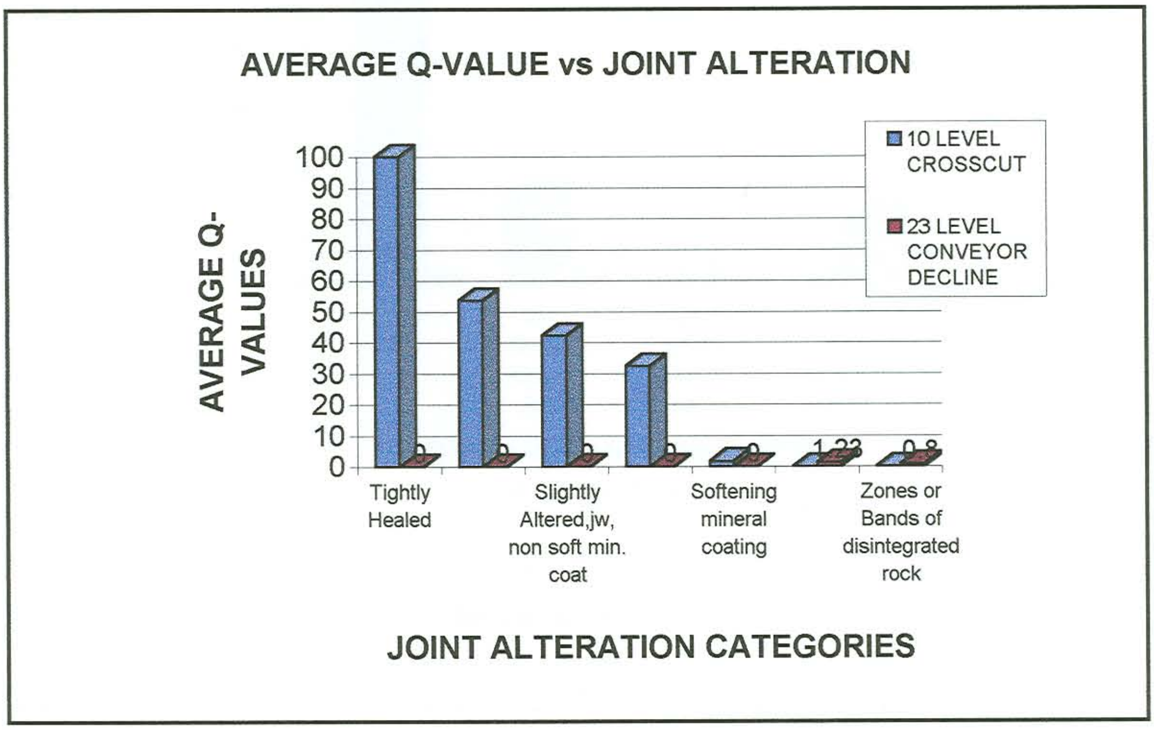

FIG. 5.15 - Joint Alteration vs Average Q-value

The last parameter critically reviewed was the stress reduction factor and is shown in Figure 5.16. A high amount of medium stress (78%) was significant of the crosscut with single shear and multiple shears with 21% and 1% respectively (Plate 7, 8,14, 15 and 16 - Appendix C). There is also a good correlation with the average Q-values (see Figure 5.17). The main and only parameter described in the 23 level conveyor decline is loose open joints, sugar cube, heavy jointed, with the emphasis on loose open joints (see Plate 23, 24, 25, 26, 26 and 27 - Appendix D).

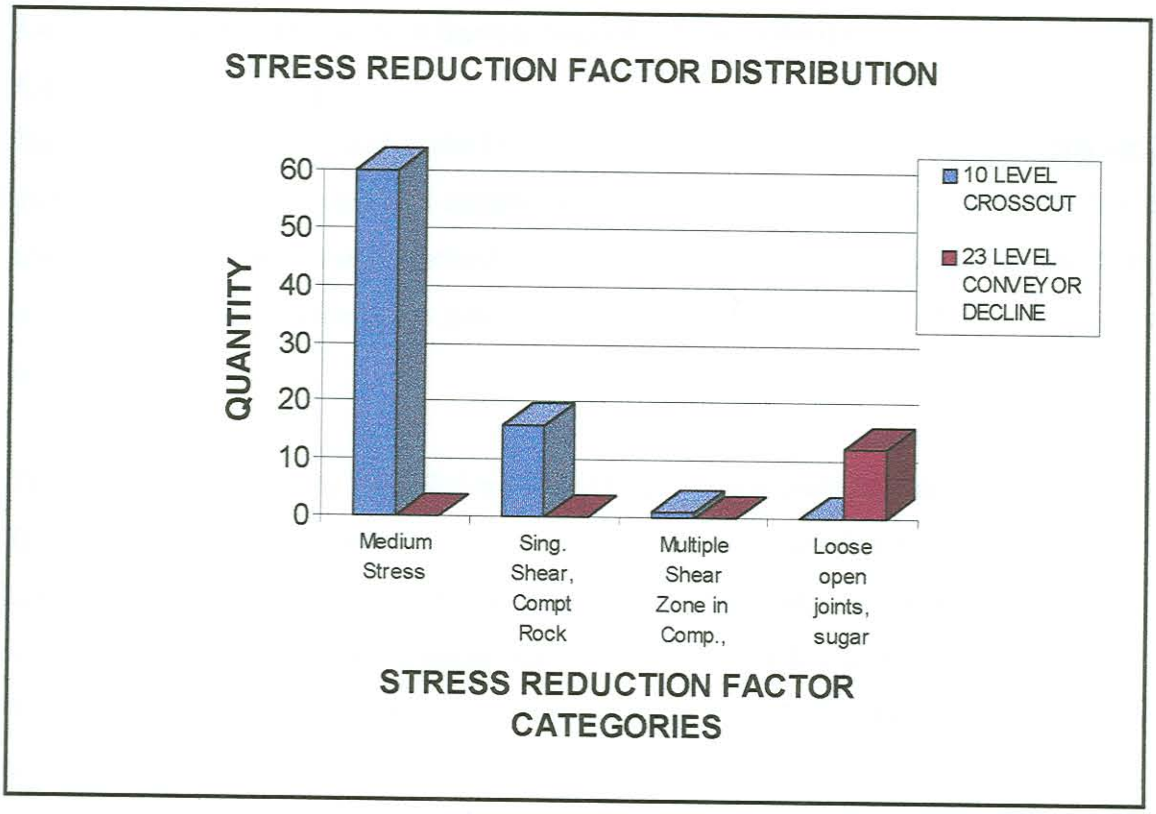

FIG. 5.16 - Stress Reduction Factor categories

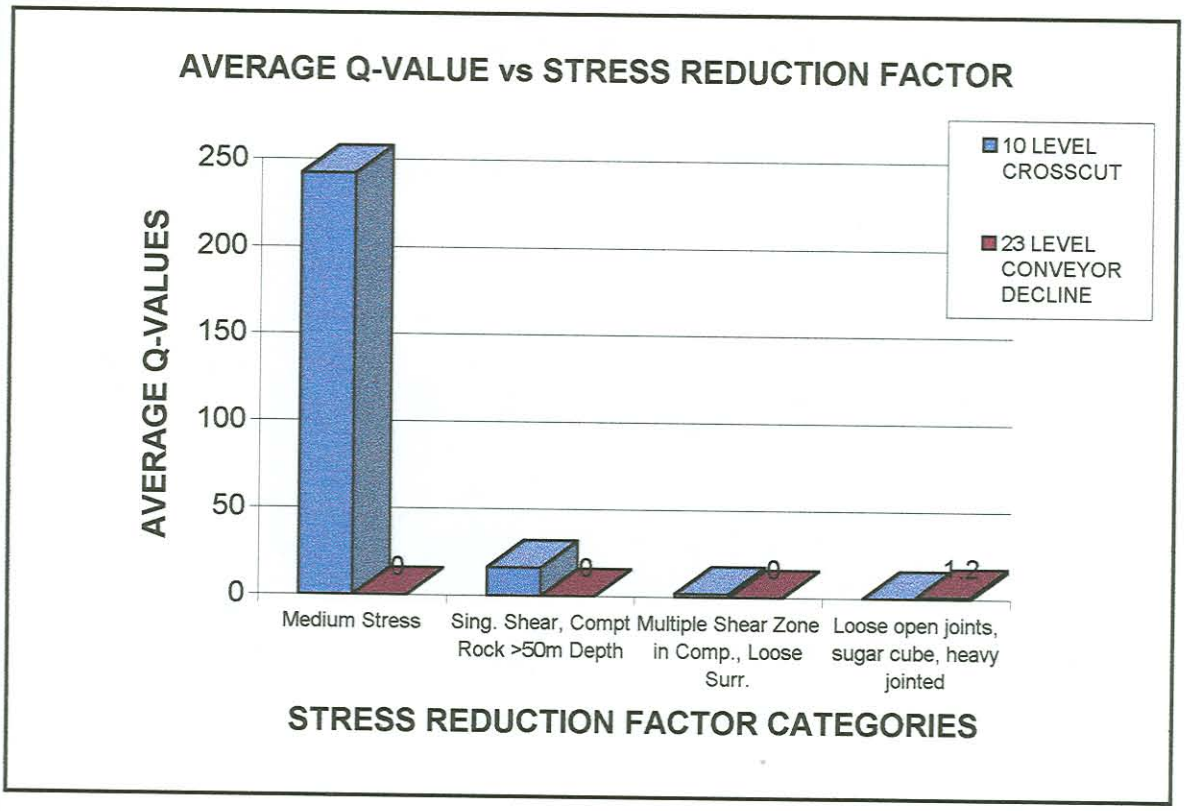

FIG. 5.17 - Stress Reduction Factor vs Average Q-value

5.4.1 Q-Rating Comparison to Barton Support Requirements

5.4.1.1 Unsupported spans

The data (Q-values) obtained in the 10 level crosscut at No. 9-Shaft provided a data set for unsupported spans that could be compare to the Barton graphs (Figure's 4.5, 4.6, 5.3, 5.4 and 5.5). The data is used to check whether the observations made satisfy the Barton criteria. Where correlation is not good, modification to the system for application at Impala would be necessary.

The calculated data set in Table B.1 was used to produce a Barton comparison - Equivalent Dimension vs Q-values Table B.3. A scatter plot was constructed to compare to the Barton unsupported line (see Figure 5.18). It must be noted that only the areas where no support was installed in the 10 level crosscut was used to produce the scatter plot (see Plate 1,2, 3,4,5,7,8, 11,12,17 and 18). An excavation support ratio of 1,6 was used for a permanent mine opening. The data provided some interesting information :

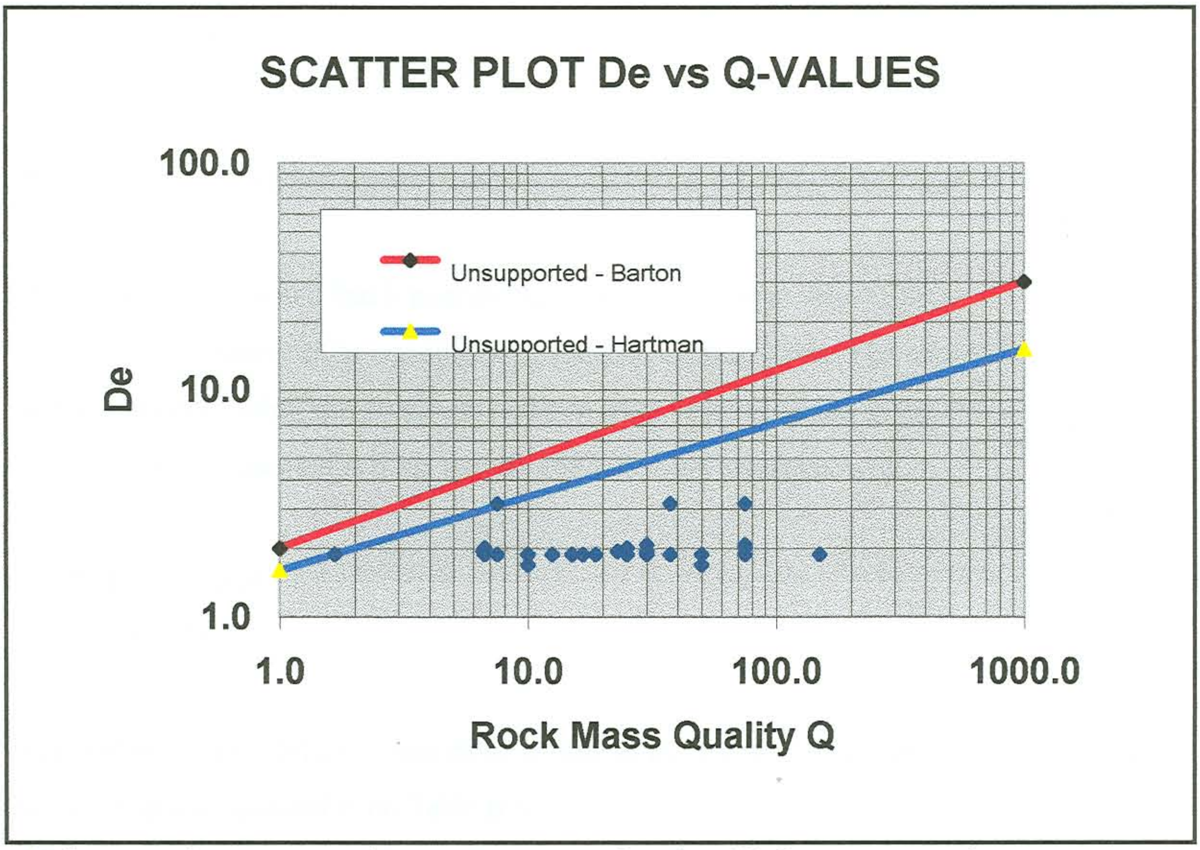

FIG. 5.18 - Scatter plot of De vs Rockmass Quality (Q) for an excavation support ratio of 1,6 (Permanent mine openings) 10 Level Crosscut

a) The scatter plot for equivalent dimension against the Q-value to a log scale was first scrutinized. All the data grouped underneath the unsupported line. According to Barton in this specific case the area should not have been supported to stabilize the excavation. This was just the case. Thus 20 years stability for a section which was not initially rated using the Barton technique as prescribed in Table 4.9 (p56-57).

b) However this recording of data necessitates the need to alter the "no support" line of Barton to an altered, more conservative, Hartman unsupported line for Impala Platinum Limited (see Figure 5.18). The calculations below support the above mentioned statement.

The Barton formula is shown below :

$$Span = 2 * ESR * Q^{0.4} \tag{5.6}$$

Where, ESR (equivalent support ratio) is equal to 1,6 for this specific case.

An altered unsupported line is postulated for equivalent span ratio (ESR) of 1,6 (i.e. permanent mine openings) for Impala Platinum Mine. The altered line was constructed using the maximum and minimum value obtained in the data set. The ESR in the equation is fixed to 1,6. The following are unknowns :

a) Q - power value
b) constant (2)

The following calculations were done to determine the altered unsupported line equation with the values obtained from Table B.8 :

Site No. 31

Q = 1,7 ; Span = 3 ; De = 1,9

and

Site No. 19

Q = 7,5 ; Span = 5 ; De = 3,1

The above values were substituted into the following equation :

$$Span = A * ESR * Q^{Y} \tag{5.7}$$

where,

A and Y are unknowns

Thus

$$A = \frac{1,875}{1,7^{Y}} \tag{5.8}$$

and

$$A = \frac{3,125}{7,5^{Y}} \tag{5.9}$$

thus, 5.8 and 5.9 can be written as,

$$\frac{3,125}{7,5^{Y}} = \frac{1,875}{1,7^{Y}} \tag{5.10}$$

$$3{,}125*1{,}7^{Y} = 1{,}875*7{,}5^{Y} \qquad (5.11)$$

$$1{,}7^{Y} - 0{,}6*7{,}5^{Y} = 0 \qquad (5.12)$$

The equation in 5.10 was resolved using the Gauss Algorithm root finding method (De la Rosa et al, 1984) :

Thus from 5.11,

$y \neq 0$ follows that $y = 0{,}3442$ (5.13)

Substitute (5.13) into (5.8)

$$A = \frac{1{,}875}{1{,}7^{(0{,}3442)}} \qquad (5.14)$$

$$A = 1{,}56 \qquad (5.15)$$

The above is a perfect solution following the plot in Figure 5.18 and Figure 5.19 below. Thus the Hartman modified formula can be written as follows,

$$\text{Unsupported } Span = 1{,}56*ESR*Q^{0{,}3442} \qquad (5.16)$$

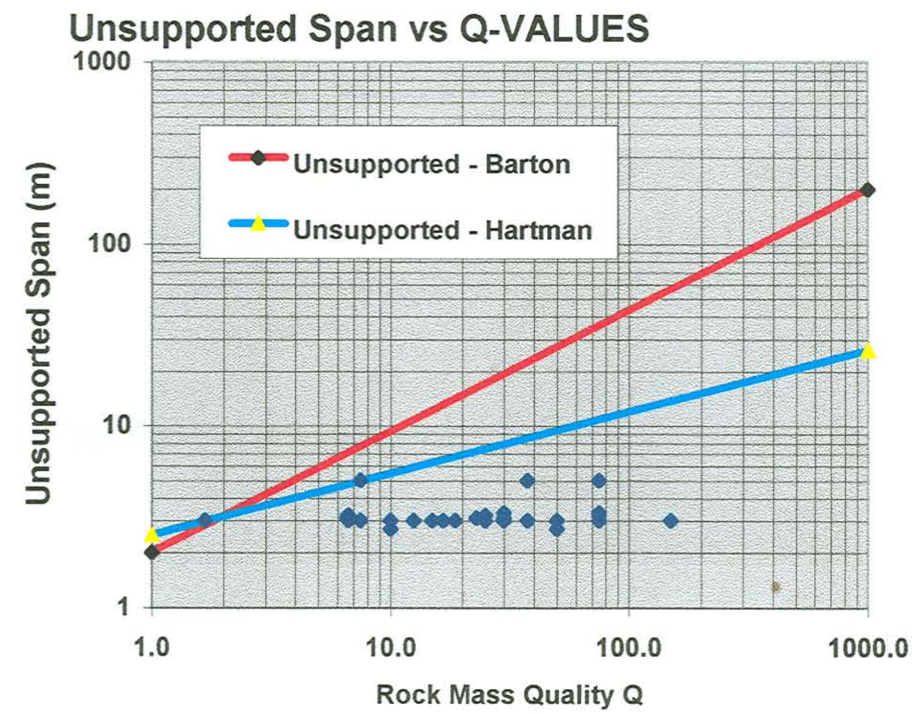

FIG. 5.19 - Scatter plot Unsupported span vs Rockmass Quality (Q)

c) The Data had to be used to compare it with Barton's - Man-made and natural, unsupported excavations see Figure 5.3 (p69). Barton's (1976) unsupported excavations case studies in different quality rock masses created the following formula

$$Span = 2xQ^{0.66} \tag{5.17}$$

The data does not satisfy the unsupported line. Therefore a more conservative approach was adopted and the line had to be altered as with the above unsupported line (see below for calculations) :

Using the following data,

Site No. 31

$Q = 1{,}7$; Span = 3 ; De = 1,9

and

Site No. 19

$Q = 7{,}5$; Span = 5 ; De = 3,1

The above values were substituted into the following equation :

$$3 = 2 * 1{,}7^{Y} \tag{5.18}$$

$$5 = 2 * 7{,}5^{Y} \tag{5.19}$$

Thus following the Gauss algebraic step method (De la Rosa et al, 1984) equation's 5.17 and 5.18 can be written as,

$$0 = 0{,}6667 * 1{,}7^{Y} \tag{5.20}$$

$$0 = 0{,}4 * 7{,}5^{Y} \tag{5.21}$$

The above functions 5.19 and 5.20 can be stepped subtracted according to the Gauss step method,

thus $$0 = 0{,}4 * 7{,}5^{Y} - 0{,}6667 * 1.7^{Y} \tag{5.22}$$

and $$y = 0{,}3441921 \tag{5.23}$$

thus giving the following modified Hartman formula,

$$Span = 2 * Q^{0,3441921} \tag{5.24}$$

5.4.1.2 Supported spans

The Q-ratings obtained in the 23 level conveyor decline at No. 14-Shaft is a data set of 12 points with 10m intervals to compare to the Barton graphs (see Figure 5.3 & 5.4, p68, and Figure 5.5, p69). It was necessary to include the exceptionally poor to the poor range Q-values in the Barton graph. The data set was plotted on a scatter plot graph (see Figure 5.20 and Figure 5.21) using the calculated data set from Table B.9. An excavation support ratio of 1,6 was used as this is a permanent mine opening.

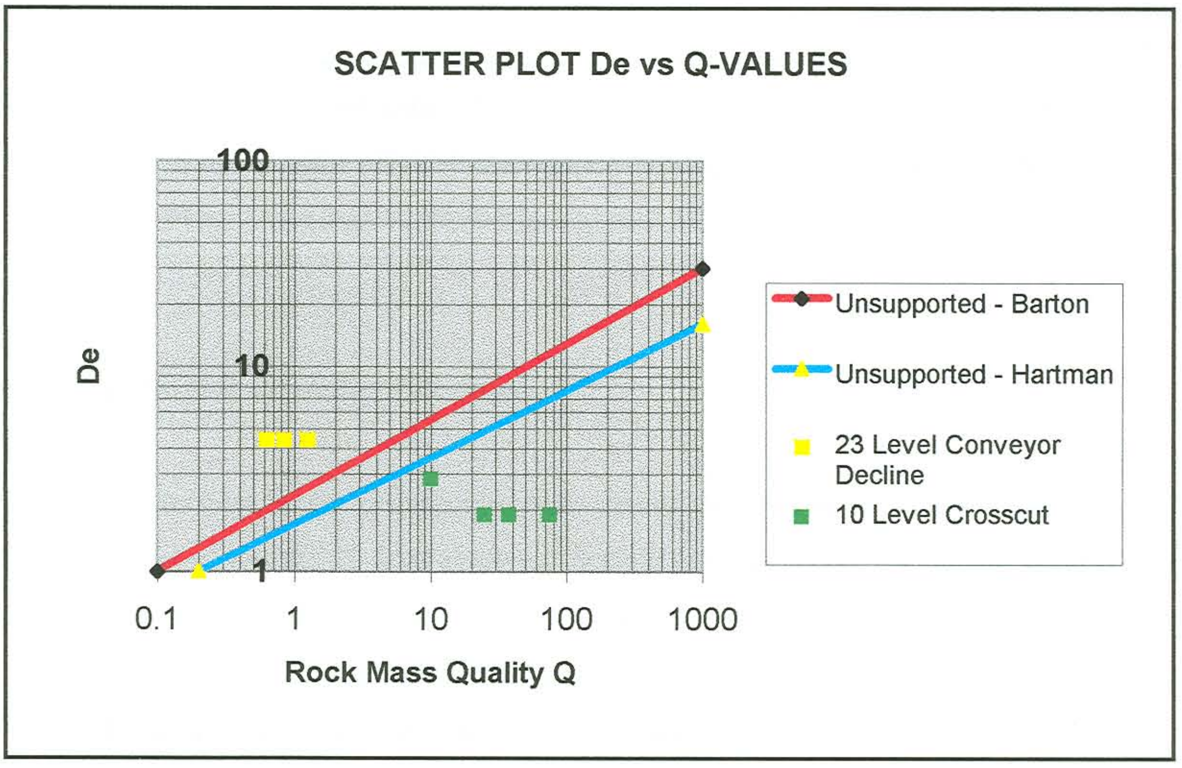

FIG. 5.20 - Scatter plot for supported data points on an Equivalent Dimension vs Rock Mass Quality : Q-Value Excavation Support Ratio of 1,6 (Permanent Mine Opening) 10 level crosscut and 23 Level Conveyor Decline

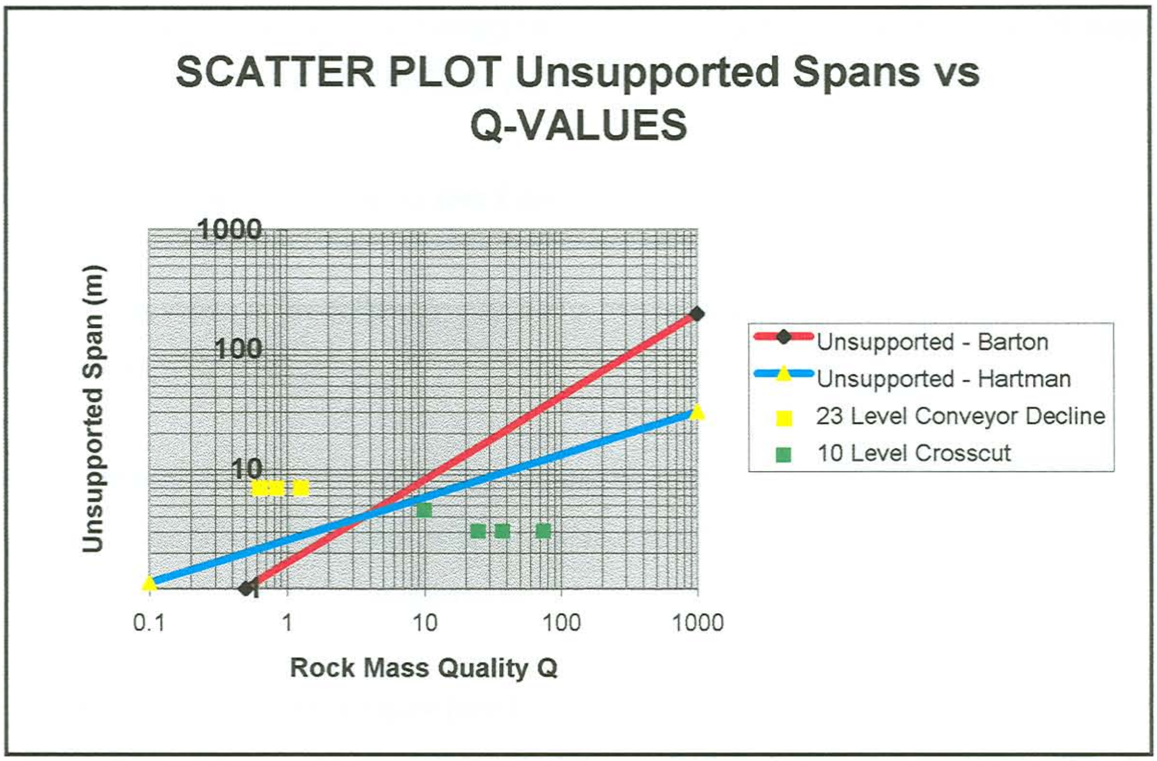

FIG. 5.21 - Scatter plot for supported data points on an unsupported span vs Rock Mass Quality : Q-Value Excavation Support Ratio of 1,6 (Permanent Mine Opening) 10 level crosscut and 23 Level Conveyor Decline

The data provided interesting information :

a) The scatter plot for the unsupported span against the Q-value to a log scale showed that all the data from the 23 level conveyor decline grouped above the Barton unsupported line and the data from 10 level crosscut group below both the Barton and Hartman unsupported line. Thus showing that the 10 level crosscut excavation support, according to Barton and Hartman, was unnecessary. However the 23 level conveyor decline data points should have been supported to stabilize the excavation, as was the case in practice. The support installed is 3m long, 16mm diameter, shepherd crooks on a 1m spacing on strike and 0,75m on dip. 50mm Fibre re-enforced shotcrete will also be added to complete the support installation.

The data set of 10 in the 23 level conveyor decline (see Table B.9) describing the poor Q-value category, according to Barton (1976), must be supported to the listed No. 21

support category and the remaining two must be supported according to No. 26 support category (see Table A.2 and Table A.3).

i) No. 21 Support Category (see Figure 5.5, p71)

The provision is the following : RQD/Jn = 12.5 and Jr/Ja = 0.75 :- Thus the type of support to be used is :

Systematic bolting - un-tensioned, grouted to a 1m spacing.
Shotcrete 25mm to 50mm with Supplementary Note I by Barton et al (1977) as a prescription (Still to be completed in 23 level conveyor decline).

ii) No. 26 Support Category (see Figure 5.5, p71)

No provision required for the following, however the following support is required :
Systematic bolting - tensioned, (expanding shell type for competent rock masses, grouted post-tensioned in very poor quality rock masses), 1m spacing. with Supplementary Note VIII, X (See Table A.5).
Shotcrete 50mm to 75mm with Supplementary Note XI (See Table A.5).

OR

Systematic bolting - un-tensioned, grouted, 1m spacing, with Supplementary Note I, IX (See Table A.5).
Shotcrete 25mm to 50mm.

b) The support bolt length according to Barton, must be 2.3m. The critical bond length for a 16mm diameter shepherd crook using cement capsule grouting with 1hour curing is 23cm. This suggests that the 3m long shepherd crook is long enough but does not have the load bearing capabilities to withstand the theoretical load it will be subjected to.

c) The spacing requirement for the bolts has to change to a 1m by 75cm spacing or alternatively change the steel parameters to a 550 MPa steel to increase the load bearing capabilities of the support tendon.

The above investigation into the Barton rockmass classification has provided the author with the necessary confidence to use the scheme. It also provided me with insight into the rock character, structure and excavation size relationship, as well as the ability of the rockmass, surrounding the excavation, to withstand the force of gravity. The above is enforced by the interpretation of the findings. Meaning that the proposed line below a rock mass quality Q-value of one will have to be critically evaluated and measured against the obtained values.