The Effectiveness of Split Sets as Primary Support System at Mponeng Mine

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This paper evaluates the effectiveness of Split Sets as a primary support system in development ends at Mponeng Mine. Access and reef development are done in virgin stress conditions. Split Sets as primary support must provide adequate support resistance in a static stress environment to support a 95 percentile fall out thickness of 0.93 m, determined from the development fatality statistics of Mponeng Mine. Three experiments were conducted to determine the mechanical properties of Split Sets. The first two experiments were done to calculate pull out forces theoretically. The third experiment was done to determine actual pull out forces and to relate these to the calculated pull out forces. The major contributing factor to the variance between the calculated values being between 50 % and 70 % lower than the actual values obtained is the assumption made to calculate the theoretical pull out force. In practice, holes are not perfectly round or straight and blocks in the fracture zone around an excavation move relative to each other, increasing the frictional properties between the Split Sets and the surrounding rock.

Problems were encountered with the pull tests done underground, where the retaining washers of the Split Sets failed before the Split Sets could pull out. The retaining washers were replaced with retaining rings, which solved the problems with the premature failing of the Split Sets. From the experiments conducted and underground observations it was found that the Split Sets provide adequate support resistance and are effective as primary support in development ends.

Introduction

Split Sets, or friction rock stabilizers, have been used at Mponeng Mine for more than a decade in development ends and stope gullies. Due to the fact that they don't need to be grouted they have been particularly well suited to the high speed mechanised development practiced at Mponeng Mine.

Concerns over the performance of Split Sets at Mponeng Mine prompted the project to evaluate the performance of the Split Set units underground.

Geological Environment.

Mponeng Mine lies on the boundary between Gauteng and the North West Province, approximately 70 km south west of Johannesburg and 8 km south of Carletonville.

Mponeng Mine exploits the Ventersdorp Contact Reef (VCR) to depths of in excess of 3000m below surface. The hangingwall to the reef is a strong, competent, jointed lava of the Ventersdorp Supergroup. The VCR rests on strata of the Witwatersrand Supergroup
The Carbon Leader Reef, currently mined at neighbouring mines, lies at a depth that currently renders its extraction uneconomic at Mponeng Mine.

Pre-VCR tectonics, associated with the formation of the Bank Anticline, caused uplift of the Witwatersrand sediments, centred on the region to the east of the mine. This resulted in gentle up-warping and erosion of strata. Lower Witwatersrand Shales were consequently exposed in the core of the anticline.

The VCR was deposited on the uneven floor created by the uplift, now represented by a shallow angular unconformity. The footwall lithologies to the VCR therefore vary across Mponeng Mine as the unconformity cuts deeper in an eastward direction into older strata.

Mining Method and Development.

General mining layout and mining method.

The mining method used at Mponeng Mine is Sequential Grid mining. This method is a controlled adaptation of scattered mining layouts for a deep mining environment. Planned dip pillars are left systematically, and geological structures bracketed. Access development is done in virgin stress conditions. Main haulages are developed on strike of the reef and crosscuts are developed from the haulages to access the reef. Raises are developed on reef from level to level. The level spacing between the levels above 109 level is 150 m apart. The level spacing of the levels below 109 level was reduced to 120 m apart to reduce the back length of the raises. Slushers are developed approximately 10 m below the raises to avoid the development of excessively long ore passes. The tunnels below 109 level are in the footwall; the tunnels above 109 level are in the competent Ventersdorp lavas except for 99 level, which was also developed in the footwall. The haulages in the hanging wall are some 150 m above the reef elevation and the footwall haulages are some 100 m in the footwall.

Rock Properties.

The development at Mponeng Mine takes place in three different rock types, namely the Ventersdorp lava, quartzite and the Booysen Shale. See Table 1 for a description of the properties of these rock types.

Rock Type	Properties	UCS.
Quartzite.	Quartzite is a metamorphic rock. In general the quartzite is a well-bedded rock with the beds ranging in thickness from 0.3 m up to 1.5 m. The bedding dip at an average of 18-20° from north to south. See Figure 1. Cohesion between the bedding planes is relatively low.	200-250 MPa.
Lava.	Lava is a massive igneous rock. The lava is mainly basalt. The lava is very competent. Various joint sets occur in the rock. The joints are mainly flat and steep dipping. The flat dipping joints are also known as lava flow planes. The joints usually have infilled surfaces consisting of calcite or quartz. The cohesion between the joints is relatively low.	250-300 MPa.
Booysen Shale.	The Booysen Shale is a silt stone The shale is laminated rock with a bedding thickness ranging between 0.5 cm and 10 cm. See photo 2. Cohesion between the beds is very low.	90-140 MPa.

Table 1. W.A. Lenhardt for Technical Development Services, done in May 1991, determined the rock strengths.

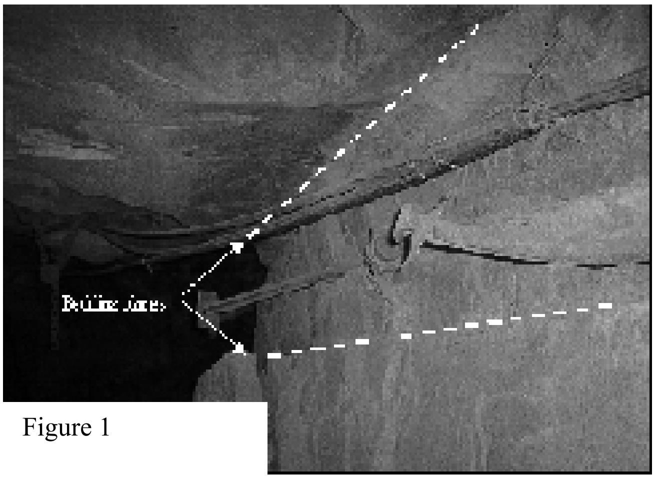

Figure 1

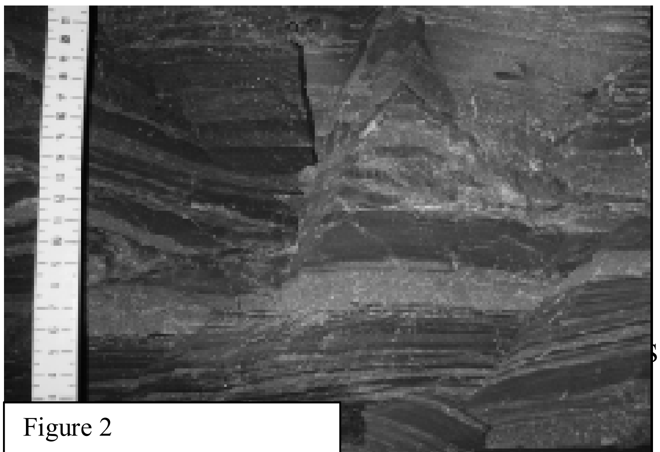

Figure 2

Ground conditions in tunnelling.

The stability of the tunnels in the lava is mainly governed by the presence of joints and faults (minor geology). The joints and faults form blocks when they are intersected and become unstable before it can be supported. These blocks fall out, either with the blast or during examining operations when it is barred down. Stress fracturing around the tunnels, further decreases the stability of the tunnels. Some areas on 84, 89 and 94 level are supported with Split Sets alone (no secondary support) and are very stable.

The stability of the tunnels in the quartzite is governed by bedding thickness and the presences of minor geology. Stress fracturing was observed to increase in intensity where the bedding thickness is smaller. When the fracture intensity increases the block sizes that form become too small to be supported with Split Sets alone. Minor geology further reduces the block sizes. The direction of the tunnel in relation to the dip of the bedding planes is an important factor in the overall stability of the tunnel. Tunnels developed along the dip direction of the strata have less over break problems than tunnels developed against the strata. Tunnels developed along the strike of the strata tend to fall out on the up-dip side of the strata.

The tunnels developed in the shale are generally very stable considering that they are developed in a much weaker rock. The tunnel walls in the shale are extremely fractured. Instability in the hanging wall is mainly gravity driven and time dependant. It was observed that most scaling from the sidewalls mainly occurs within 5 m and 15 m behind the face. The scaling is mainly due to the shale deforming in a ductile manner where the small blocks are pushed out due to dilation. A layer of shotcrete close to the face is used to control the scaling.

Support Requirements.

Primary support.

The fundamental purpose of tunnel support is to maintain the integrity and stability under static states of stress. The support system must control slow stress driven deformation of the rock mass. The support system must not only support the dead weight of the potentially unstable blocks, but by increasing inter-block frictional forces and resisting the buckling of rock slabs and constraining movement of key blocks in the rock wall, prevent the unravelling and differential movement of the rock mass.

To enable rapid development progress, the primary support system needs to be designed to cater for the current short term rock mass conditions. This section is aimed at determining what is required from the Split Sets as primary support.

Function of Split Sets.

Split Sets are installed close to the face before blast. Split Sets are a friction based support system therefore the holes in which the Split Sets are installed are drilled slightly smaller than the Split Set diameter. The Split Sets are driven in these holes with a pushing device attached to the drill.

The Split Sets must provide adequate support resistance to cater for the short-term rock mass conditions and it must enable rapid progress of development ends.

The following sections are aimed at determining the required support resistance under static stress conditions and to verify whether the Split Sets comply with the requirements. The mechanics behind the functioning of the Split Sets are also investigated.

Support resistance required.

An investigation was done on the fatal accident statistics in development ends in order to determine the rockfall thickness, to calculate the required support resistance. See Figure 3.

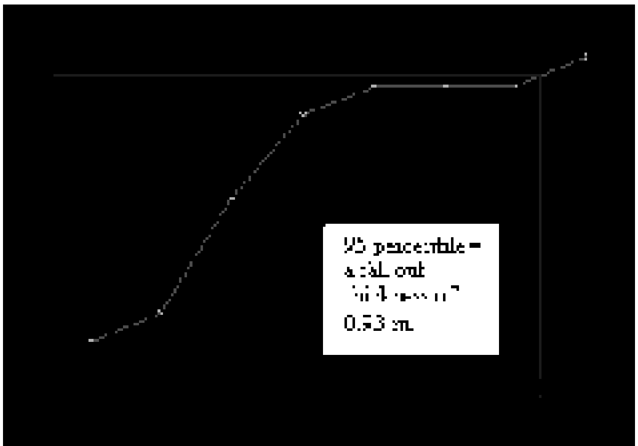

Figure 3. Cumulative fall out thickness for development ends at Mponeng Mine.

One problem with this method is that only a limited amount of data was available to do this analysis since fall out thickness was only recorded with the fatal accidents.

Since the start of Mponeng Mine, 12 fatal accidents in development ends occurred where a fall out thickness was recorded. The fall out thickness of 0.93 m, determined from the Mponeng Mine analyses seen in Figure 3, will be used for the support resistance calculations.

In order to determine what is required from the primary support in the tunnels it is necessary to know what must be supported. As mentioned above the primary support must support a certain amount of dead weight (2), it must resist the movement between relative blocks (1) and it must prevent the stress fracturing from opening up (3). See Figure 4 for the various support functions.

Figure 4.

According to the tributary area theory the maximum support resistance that will be required per development end before the stoping operations are started is as follows: Tunnels are not exposed to major seismicity early in its life.

Tunnel Width = 3.7 m

Length of tunnel = 10 m

Fall out thickness = 0.93 m

Density of Rock = 2.78 t/m^3

Total dead weight to be supported = 3.7 m x 10 m x 0.93 m x 2.78 t/m^3 x 9.81 m/s^2 = 938 kN.

Support resistance required = 938 kN ÷ 37 m^2

Total support resistance required = 26 kN/ m^2

Experiments to determine the physical properties of Split Sets.

The mechanical properties of Split Sets

Mponeng Mine uses the 1.5 m and 2.1 m long, 42 mm diameter Split Sets.

The Split Sets are manufactured from ASTM A607 450 MPa (Yield stress) 550 MPa (Ultimate tensile stress) Steel, with a thickness of 2.3 mm. The nominal yield load of the material is 90 kN and the nominal ultimate load is 110 kN. It is recommended that the Split Sets be installed in a hole with a diameter of 1 - 3 mm smaller than the Split Sets diameter. The Split Sets ranges in diameter between 42 and 43.6 mm. The slot width range is between 14.5 – 16.5 mm. The Split Sets is supplied with a 125 mm x 125 mm bearing plate.

This section aims at describing the tests conducted on the Split Sets as well as to provide the results from these tests. Firstly it was necessary to establish what the Split Set does inside the drilled hole and how it functions. First of all there must be a resisting force in the hole to resist the Split Set from pulling out of the hole. This resisting force is the product of the normal force due to the reduction in diameter of the Split Set when it is inside the hole and friction properties between the Split Set material and the rock. This resisting force must be large enough to provide

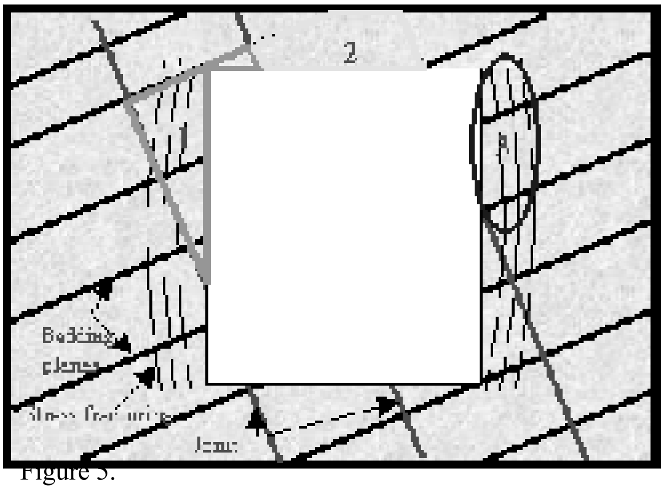

Figure 5.

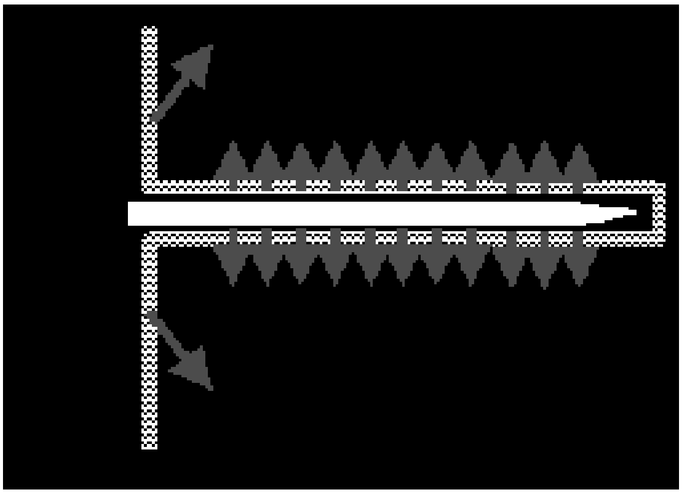

Figure 5. The various elements of a Split Set.

Three experiments were conducted on Split Sets in order to determine the relevant technical information. The following sub-sections describe the experiments conducted and the results obtained from the tests. The first two experiments were conducted in order to calculate the pull out loads theoretically. These two experiments are the uniaxial compression test using a point load tester and the slide test in order to determine coefficient of friction between the Split Set material and the various rock types.

The third experiment is underground pull tests conducted on the Split Sets and was done to determine a relationship between the theoretically calculated pull out loads and the actual values.

Experiment 1: Uniaxial compression of the Split Sets.

This experiment was conducted using a point load tester in order to determine material properties of the Split Sets. Load spreader plates were used to avoid point loading the material. The spreader plates were 65 mm long and 5 mm wide and were taped to the Split Set. The load was applied in a direction at right angles to the slit of the Split Set (See figure 6). See Figure 7 for the results obtained from these tests. The Manufacturer did shear tests at the CSIR where the Split Sets were tested with the gap facing upward (0°), the gap facing to the side (90°) and where the gap faced down (180°). A summary of the results of these tests is given in Table 2.

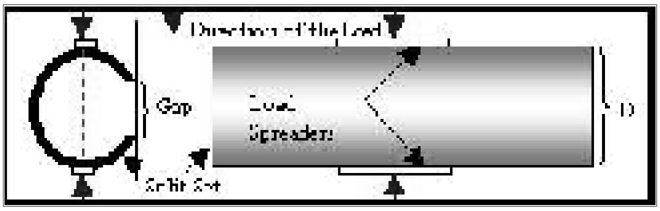

Figure 6.

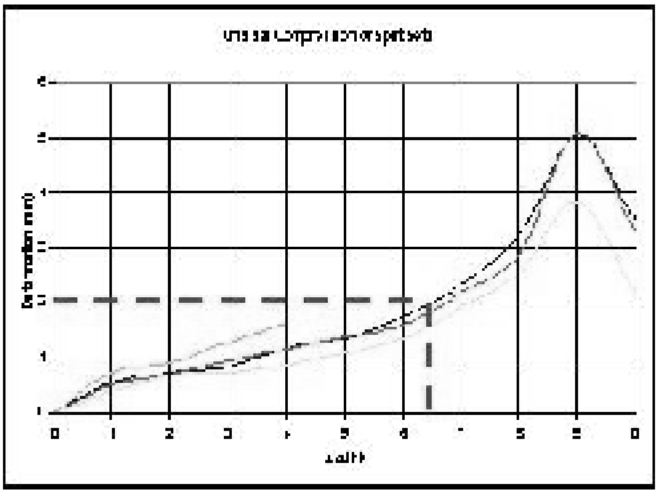

Figure 7. Summary of the point load tests on the Split Set.

The first point load test was done on a 200 mm long piece of Split Set to obtain a result that would not be influenced by the ends of the Split Set. This gave roughly the same results as the other tests where the point load was applied at two different points along a Split Set

From Figure 7, it can be seen that from the supplier's recommendations that the Split Sets must be installed in holes that are 1 – 3 mm smaller in diameter than the diameter of the Split Sets, a peripheral force in the range of 4 – 9 kN is exerted on to the sides of the holes. The starting diameter of the drill bits is 41 mm in diameter. The support holes are drilled first in all the development ends and there is an approximate change in diameter of 1 mm before the face is drilled.

It can be assumed that total amount by which the hole is smaller than the Split Set is 2 mm and that the average load of 6.5 kN per 200 mm length of Split Set will be exerted on the side of the hole in one direction. The main shortcoming of this test is that the deformation of the Split Set does not only exert a large load in one direction along the hole, but rather a series of smaller loads in various directions along the entire diameter of the hole.

Experiment 2: Slide Test.

The reason for this experiment was to get a coefficient of friction in order to calculate the peripheral forces of the Split Sets when it is in the hole. Since the required force to pull the Split Set of the hole will be just greater than the product of μ (coefficient of friction) and N (the radial peripheral force). From this data it will be possible to calculate a critical bond length for the Split Sets to be effective.

This test was done to determine the coefficient of friction between the Split Set material and the various rock types. This is a simple test where a flat piece of Split Set steel was used and rock samples were slid on the metal. The Split Set material was smooth on the one side and had small patches of rust on the other end. The samples were tested on both sides of the Split Set material.

A total of 10 samples were tested for each rock type. The samples were from diamond drill core and were flattened with a diamond saw. The samples had a relatively smooth surface, since the coefficient of friction is dependant on the surface in contact random samples were roughened to give more or less the same texture as the surface of a drilled hole in the underground environment. The roughened samples yielded more or less the same results as the smooth samples.
The samples were placed on the Split Sets steel and were tilted until the samples started to slip, the angle of slip was recorded and the coefficient of friction was determined from using the following equation.

Equation 1:

$\mu = \tan\theta$

Where: μ = Coefficient of friction

Peripheral Load/ 200 mm	Theoretically calculated Pull out loads. (kN) Quartzite (Smooth)	Quartzite (Rusted)	Lava (Smooth)	Lava (Rusted)	Shale (Smooth)	Shale (Rusted)
3 kN	9.6	13.2	7.8	11.1	9	12.3
4 kN	12.8	17.6	10.4	14.8	12	16.4
5 kN	16	22	13	18.5	15	20.5
6 kN	19.2	**26.4**	15.6	22.2	18	24.6
6.5 kN	20.8	**28.6**	16.9	24.1	19.5	**26.6**
7 kN	22.4	**30.8**	18.2	**25.9**	21	**28.7**
8 kN	25.6	**35.2**	20.8	**29.6**	24	**32.8**

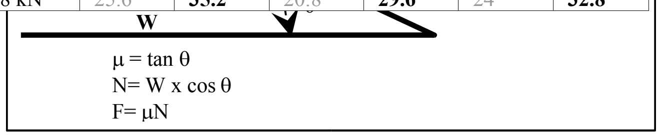

Figure 8.

The results given can now be used to determine the peripheral forces acting on the perimeter of the drilled holes, using the formula given in Figure 8.

Theoretical calculations.

The results obtained in experiments 1 and 2 will now be used in this section to calculate theoretically whether the Split Sets will supply enough support resistance in order for them to comply with the calculated support resistance obtained from the fall out thickness obtained previously.

Table 2.

But first it must be noted that these calculations will be based on various assumptions:

- The holes are perfectly round and straight.
- The calculations do not consider movement between blocks in the fractured rock mass.
- The calculations do not consider that the rock properties can change along the length of the hole.
- It is assumed that the direction of the pull out force is parallel to the direction of installation.

Theoretical calculations:

Pull out force (F) = Coefficient friction (μ) x Peripheral force (N)

F = 0.32 x (6.5 kN per 200mm)

F = 0.32 x (6.5 kN x [2000mm/200mm])

F = 0.32 x (6.5 kN x 10)

F = 20.8 kN.

The results from the theoretical calculations based on experiments 1 and 2 are given in Table 3. Note that only the figures in black bold will have a pullout force large enough when installed on a 1 m square pattern to provide adequate support resistance to support the dead weight.

Table 3.

Experiment 3. Underground pull-tests.

These tests were done in order to determine pull out loads for the Split Sets at various time intervals and for a range of lengths of Split Sets. The initial plan was to install 2.1 m Split Sets in various development ends over a period of a month and on the last day of installation to pull all the Split Sets to determine the increase in pull out loads over a period of time.

The biggest problem with this method was that the retaining washers on which the Spit Sets was, pulled either broke off on installation or failed before the Split Set pulled out. The suppliers were called in and it was decided to change the retaining washers to 8 mm retaining rings. See Figure 9 for photos on the failure of the retaining washers

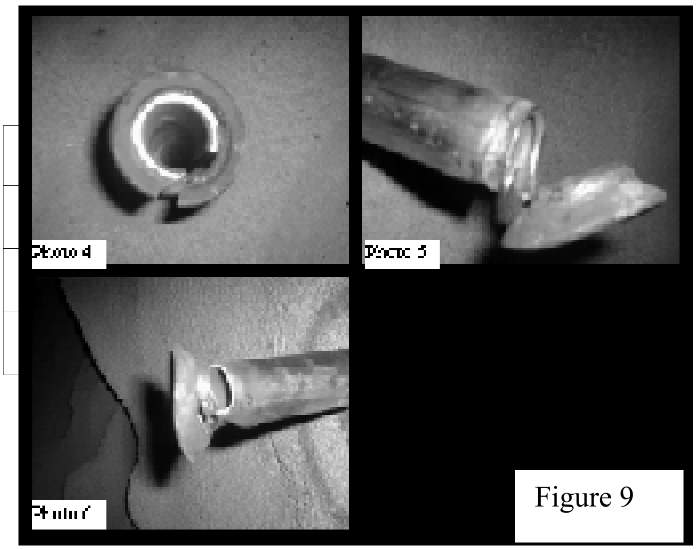

Figure 9

The following equipment was used to conduct the pull test:

- A hydraulic hand pump with fitted pressure gauge calibrated in tons.
- A custom-made splitset puller head and rod capable of loads up to 15 t.
- A hydraulic ram.
- A steel tower.

The following results were obtained from the pull tests. See Figure 10.

From the 65 pull tests that were done, it can be seen that the pull out force required for the Split Sets increase with time. This is due to movement of the stress fracturing relative to each other clamping the Split Sets in the holes.

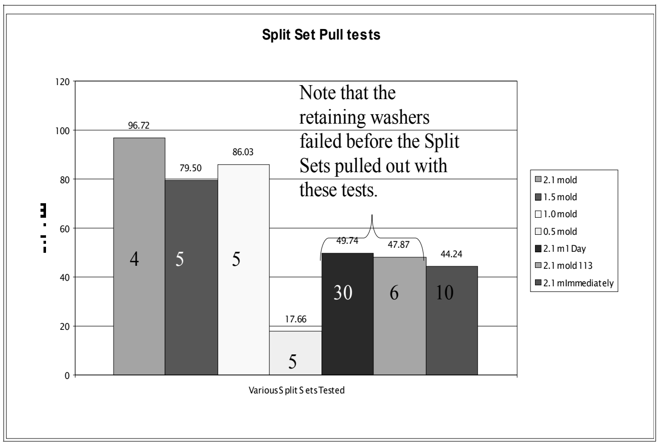

Figure 10. The numbers on the different bars indicate the number of samples that was pulled in that range.

As mentioned above the pull tests did not go according to the initial plan. It was then decided to install ten 2.1 m Split Sets with the 8 mm retaining rings in a development end and then to pull them immediately after installation. In order to determine whether there would be an increase in pull out force with time, it was decided to pull Split Sets on random basis. It is not possible to give an exact date of installation but the Split Sets classified, as old would have been pulled between 1 and 2 years after installation.

Pull tests were done on the Split Sets ranging in lengths of 0.5 m, 1.0m 1.5 m and 2.1 m. These Split Sets was installed some 2 years before it were pulled. These tests indicate that there is a definite increase in pull out force required with increase in length. It also shows that a length of 1 m must be anchored in the solid to provide sufficient anchorage for the Split Sets. Refer to Figure 10.

The Split Sets that were pulled 1 day after installation did not pull out; the retaining washers broke before the Split Sets could pull out. The fact that the retaining washers broke off indicates that the Split Set load did increase significantly over a period of one day.

The pull out force required pulling the Split Sets out immediately after installation is 44 kN.

	Theoretically calculated peripheral forces from actual pull out loads. (kN/ 200 mm)					
Pull out loads (2.1m)	Quartzite Smooth	Quartzite Rusted	Lava Smooth	Lava Rusted	Shale Smooth	Shale Rusted
44 kN Immediately	13.75	10.0	16.9	11.8	14.6	11.9
97 kN > 2 years	30.3	22.1	37.3	26.2	32.3	23.7

Comparison between theoretical calculation and practical pull out forces obtained.

This section will compare the theoretical calculations with the actual pull test results obtained. A back calculation will be done using the actual pull out forces and the coefficient of friction obtained from Experiment 2 will be used to calculate the peripheral forces required to keep the Split Sets in the hole.

Once again with this back calculation it is necessary to make assumptions:

- The holes are perfectly round and straight.
- There is no additional deformation in the rock after the hole is drilled.
- The direction of the pull out force is parallel to the direction of installation.

Pull out force = 44 kN

		Smooth	Rusted
Coefficient of friction =	Quartzite	0.32	0.44
	Shale	0.3	0.41
	Lava	0.26	0.37

$F = \mu \times N'$

$N' = F \div \mu$

$N' = 44\ kN \div 0.32$

$N' = 137.5\ kN/ 2000mm$

$\mathbf{N' = 13.75\ kN/ 200mm}$

The theoretically calculated peripheral forces from the actual pull out loads are given in Table 4.

The peripheral forces that were calculated above are 50 % to 70 % higher than what was measured in Experiment 1. This is due to the fact that the load in the experiment was applied in a single direction. In practice this peripheral load will be applied in various directions along the circumference. The hole deflects, which increase the peripheral forces inside the hole.

Another reason for the variance, may be that the coefficient of friction between the Split Sets and the holes are higher, due to the different textures between the hole surface and the Split Set surface in contact.

Table 4

Conclusion.

From underground investigations and task observations it was found that Split Sets used as primary support in development ends are adequate.

Even though problems were encountered with pull tests, where the retaining washers failed prematurely, a average pull out value of 44 kN immediately after installation was determined. When the Split Sets are installed as per mine standard on a 1.5 m square pattern the Split Sets provides a support resistance of 29 kN/m^2. This is above the required support resistance of 26 kN/m^2.

The support resistance of the Split Sets does increase with time as a maximum pull out load of 96 kN was measured on Split Sets that was installed more than 2 years before this evaluation started.

The experiments done on the Split Sets to enable the theoretical calculation of the pull out loads proved that the calculated N' values are significantly lower, between 50% and 70 %, lower than calculated N' values obtained from the actual pull out loads. This variance is due to the assumption required for the theoretical calculations.

From this study the best method to determine whether the Split Sets would be adequate as primary support system is to conduct physical underground pull tests.

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