GROUND CONDITIONS AND SUPPORT SYSTEMS AT 1 SHAFT, KONKOLA MINE, CHILILABOMBWE, ZAMBIA.

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Abstract

Ground condition at No1. Shaft varies across the mine. The different ground conditions arise as a result of the differences in the rock strength and structure, seismicity and induced stresses in abutments areas. Apart from being a major cause of Fall of Ground incidents/accidents, adverse ground conditions where encountered has affected the rate of development advance as well as production.

This implies that for support to be cost effective Geotechnical section has to ensure that support is designed to prevailing ground demand to ensure the safety of personnel and minimize loss of production. We have also seen changes in mining methods as a result of change in ground conditions. To avoid incurring unnecessary costs through remedial works resulting from ground deterioration and collapses of development openings due to inadequate support, the principle in engineering design is "Do it right the First Time", and this can only be achieved through good understanding of Geotechnical environment.
The paper discusses the different ground conditions encountered in various development ends at No.1 Shaft. It also discusses the principles of ground support, failure mechanisms, types of ground support systems employed, and the different techniques used to select support systems, and quality control techniques in underground support installation.

1.0 Introduction

Konkola mine is one of the wettest underground mine in the world, with the largest copper resources on the Copperbelt. Konkola pumps an average of 285,000 cubic metres of water per day, generated from both No.1 and 3 Shafts. Total current production stands at about 200,000 tonnes of ore per month. The mine is situated 25 Kilometers north of Nchanga Mine and near the Congo border. No1 Shaft is situated in the southern part of the Konkola Mining License Area, and produces 90,000 tonnes of ore at 3.2 percent Copper. Both development and production, have from time to time, been affected by adverse ground conditions and seismicity.

Support installation forms part of the major mining operations at 1 Shaft contributing about 3% of the total shaft budget, and at an average monthly support cost of $0.34 per tonne of ore mined. Various support systems are used at 1 Shaft to improve stability of different types of excavations in order to prevent accidents resulting from falls of ground and operate economically.

KCM management has placed strong emphasis on ground support compliance at all the mines.

1.1 Geology

The Ore body is stratiform, and strikes generally South East-North West and dipping South West at an angle varying from 30^o in the north to 67^o towards south. The ore body averages 7m in width and is generally regular with minor folding around 300mS position. Geographically 1 Shafts ore body extends from 3000mN to about 860mS on the south. A barren gap in form of a syncline lie above 2400 Level with its lowest point between 1600mN and 2000mN positions

The ore body is contained in a group of sedimentary strata in the lower Roan group of the Katanga System. The ore body is primarily sulphides. The dominant ore minerals are Chalcopyrite, Bornite and Malachite. Above the the ore body are dolomitic shales (which in some places are completely kaolinized and decomposed (1200mN to the South), and bands of quartzites and sandstones.
Structurally, the ore body is divided into five units. At the bottom is the finely banded dolomitic, calcareous sandstone, in places highly weathered to a brown micaceous clay, ‘A’ Unit varying in width from 0.3m to 1.0m and rock mass rating of 21-40. In some places the rock mass rating is less than 20. Above the ‘A’ Unit is the 1.0-1.5m thickly bedded, ‘B’ Unit. The ‘C’ and ‘D’ Units comprising inter-bedded strong siltstone and dolomite bands, lie below the strong siliceous ‘E’ Unit within which the Assay hanging wall usually lies. The hanging wall formation consists of quartzite and dolomitic sandstone bands, which in some places completely kaolinized, resulting into poor hanging wall condition.

1.2 Hydrogeology

The mine pumps an average of two hundred ninety five thousand (295,000) cubic metres of water per day. The water comes from both 1 Shaft and 3Shaft. There are three main aquifers within the sequence, and these are, the Footwall Quartzite/Lower Porous Conglomerate, the Footwall Aquifer and the Hanging wall Aquifer. The cost of pumping water constitutes 20% of the total mining cost at Konkola mine. The presence of water in some parts of the mine has an adverse effect on ground conditions and has an influence on the type of support in development.

1.3 Mining Methods

The ore body is mined by three variations of Sublevel Open Stoping (SLOS) method.

1.3.1 3150 Level, 300Ms SLOS

Up until the middle of 2001, the southern part of the ore body has been mined using conventional sublevel method with in-ore vertical fan drilling from hanging wall drives. The blasted ore was extracted from the stope by gravity method using grizzley arrangement. From 2001, following a series of accidents resulting from fall of ground due seismic events in the area, 75% of the stope developments were relocated in the footwall. Stope drilling is done from the crosscuts below the ore body in horizontal planes to blast the stope down dip. The stopes are 12m wide and 8m pillars with back length of 85m. Broken ground is extracted by mechanized draw point loading. In this section support system is influenced almost entirely by seismicity.

1.3.2 2900, 3150 Level, 1500Mn SLOS

The ore body in the two retreats is mined using conventional sublevel open stoping with gravity loading. Blast holes are drilled vertically from the ore body hanging wall drive mined at 15m vertical intervals. The area experiences insignificant ground instability due to high strength ore Shale and low induced stresses due to presence of the barren gap above 2400 Level. Further to the north at 2475mN, Cut and Fill mining method has been proposed on a trial basis to start in the 2005 FY.

1.3.3 2200/2650 Level NEA Mini Back SLOS

The north extension mining area lies between 2000mN to 3000mN positions and extends on dip from 2200 Level. Current Mining operations are carried out on 2250 and 2350 Levels, using the Mini Back Sublevel Open Stoping mining method with mechanized draw point loading. Only the small 2.0mx2.0m drilling drives and cut out cross cuts are located within the ore body in the competent part of the ore body. The ore body dips at 36-45^{o}, with a stope back length of 45m. Stopes are 20m with 4m pillar. The retreat produces 45% of the total 1 Shaft ore at an average grade of 4% contained Copper.

In 2001 management introduced Longitudinal Room and Pillar (LRP) method to mine the ore body from 2200L to 2650L. The method involved in-ore development of the 3.5mhx3.6mw trough drive, for drilling and mucking of blasted ground. However, because of stability problems with the trough drives the method was abandoned and reverted to mini back SLOS method. All the major development are located within the relatively competent footwall formations. Due to improved rock mass quality of the Ore Shale beyond

2700mN, partial LRP has been introduced with relatively low support costs due to improved ground conditions within the ore body.

1.3.4 3150 Level, Sill Pillar Mining

The sill pillar between 3095L and 3150L is mined by Mechanized Open Stoping (MOS) method. A 3.5mhx3.6mw trough drive is developed within the ore body from where stope drilling and ore mucking is carried out. A 3m thick crown pillar is designed to crush.

1.4 Ground conditions

Falls of ground are a major hazard in underground mining operations at 1 Shaft. For this reason it is necessary to identify the different ground conditions with the potential for falls of ground so that appropriate development support is put in place.
No1 Shaft experiences various inherent geological properties of the rock mass, and stress, producing different types of ground conditions over the entire mine, affecting the stability of different types of excavations including stopes. Depending on the size of excavation and the state of stress, these different kinds of ground conditions produce different magnitudes of deformations around excavations, and hence attracting different support systems. All mine developments at 1 Shaft are mined within five main rock formations. These are, Footwall Quartzite, Argillaceous Sandstone, Porous Conglomerate, Footwall Sandstone, Footwall Conglomerate and the Ore Shale.

In all the formations ground conditions are almost entirely influenced by the presence and intensity of different structural features and the degree of weathering within the rock mass. There are five factors that influence ground conditions at 1 Shaft, that is, jointing, weathering, faulting, stress change and seismicity, and these determine the support conditions for different sizes of developments.
Generally most geological structures at 1 Shaft are considered as potentially hazardous in respect of rock falls, particularly that mining takes place at a relatively shallow to intermediate depth, and that access developments remain positioned in low stress environments. Discontinuities such as weak bedding planes, joints, faults, fissures, singularly or in their combination, create rock fall hazards through formation of unstable blocks.

1.4.1 Jointing

The intensity of jointing in a particular rock mass varies from one place to another, and joints usually occur as sets, each on a particular dip and dip direction. There are three main joint sets plus several other minor sets at 1 Shaft and are usually long every where and only vary in their spacing and joint conditions. Instability in this case is as a result of formation of potentially unstable wedges and blocks in the excavation walls as a result of combination of various types of discontinuities such as joint, bedding plane, weak vein, fissure or stratum material, and fault planes. The prominent joint sets are as shown in table1 below.

Table 1. Prominent Joint Sets

Position	Rock Type	Joint Sets	Dip	Dip Direction
2700mN	Ore Shale	Bed	41	222
		J1	51	060
		J2	77	073
		J3	80	334
300mS	Ore Shale	Bed	53	227
		J1	59	087
		J2	75	032
2700mN	F/W Quartzite	Bed	35	235
		J1	54	050
		J2	80	050
2500mN	F/W Quartzite	Bed	35	213
		J1	50	065
		J2	89	234
2700mN	F/W Sandstone	Bed	35	245
		J1	40	070
		J2	85	300
		J3	76	065

The ore body is the most closely jointed (spaced) rock mass, making it highly blocky in places. In large developments like the quartzite haulages (3.7mhx3.8mw) on 2200L and 2650L instability is normally as a result of two intersecting orthogonal joint sets plus a third near vertical joint set (70-85^o)/300-350^o), which form prisms along the haulage. Since the rock mass at 1 Shaft is, is almost every where well bedded, it is a standard practice to mine the drive as much as possible to conform with the existing bedding planes so as to avoid the formation of unstable blocks or wedges in the back of the development.

Table 2. Range of Joint Spacing for different Rock Formations

Rock type	Range of discontinuity spacing (mm)	Description	Development Type
Footwall Quartzite	600-2000 to >2000	Widely to very widely spaced.	Tramming haulages, main access ramps
Argillaceous Sandstone	60-200 to 200-600	Closely to medium spaced	Access ramps
Porous Conglomerate	200-600 to 600-2000	Medium to widely spaced	Poorly affected by weathering in some places. Gravity and trackless drifts
Footwall Sandstone	60-200 to200-600	Closely to medium spaced	Trackless extraction drives in NEA.
Footwall Conglomerate	200-600	Medium spaced	Can be heavily weathered in places and avoided.
Ore Shale	5-20 to 20-600	Extremely closely spaced to medium spaced	'A' Unit is always avoided. Best ground in Ore Shale is 'E' Unit

1.4.2 Weathering

Weathering has greatly affected the stability of developments in many parts of the mine through reduction in the intact rock strength, rock quality designation and fracture frequency. In areas such as 3150L 300mS and 1500mN, and 2650L NEA, from 2700mN northward, where the Ore Shale is fresh to slightly weathered, thickly bedded and moderately jointed, the ground is competent, with the RMR = 61-70.

Figure 3. (a). Typical weathering profile across
Ore body-3150L, 1500mN

Unit	Soil, EW	HW	MW	SW	FR
A	√			√	√
B				√	√
CD				√	√
E					√

However, the "A" Unit is almost every where weathered in most areas completely decomposed. In the 2200/2650mN NEA, from 1800mN to 2270mN and from 2360mN to 2690mN positions, the ore formation is finely bedded, kaolinized, highly weathered, highly jointed and incompetent, with a rock mass rating of 25-54. Similar ground conditions exist in the 2900/3150L 250mS Fold Area and 2650/2900L 1500mN retreat from 1800mN to 2270mN positions.

Figure 3. (b). Typical weathering profile across Ore body – NEA

Unit	Soil, EW	HW	MW	SW	FR
A	√	√			
B			√		
CD		√	√		
E				√	√

Only the massively bedded, siliceous 'E' Unit at the hanging wall remains slightly weathered to fresh and the drilling drives are positioned in this unit. In the LRP mining method, where the trough drive were mined through this ground, frequent massive falls of ground were experienced along the drives. In the 2650/2900L 1500mN retreat, small 2mx2m drilling drives experienced heavy instability such that addition support in form of sub level timber setts have to be installed in the drive and cut out crosscuts.

1.4.3 Fault Zones

Faults have been identified as major regional structures at 1 Shaft. They occur in groups, one at 2270mN and the other at 27oomN, in the North Extension Area, ranging in width from 40m to 60m. Each group is bounded by two major fault planes with several secondary faults lying parallel in between. The dip averages 75^o-85^o to the north, and trending in the east-west direction . The fault openings contain very weak materials of highly weathered to soft clay material, breccia and angular rock fragments. Looking at the wall rocks they are slickensided with low friction strength. The disturbance and weakening in the adjacent ground, is more pronounced within the ore body than in the waste footwall formations, making ground conditions within the ore formation unbearable. Within the footwall formations, the fault planes are characterized by sharp edges and slickensided, with little or no gouge material. In general, the two fault zones are characterized by extensive fracturing and leaching, resulting into weak ground.

The faults are pervasive in nature as they traverse the entire North Extension Mining Area, from surface to beyond the current mining levels. The presence of faults in the area has influenced the setting of the development layouts, stoping sequence and support conditions within the zones. Where there are numerous displacements, the portion of the ore body is left unmined. The rock mass rating in the fault zones range from 25 to 54. Even mining of the smalll 2.0mx2.0m drilling drives can be extremely difficulty.

1.4.4 Breccias Unit

The unit 0.5m-3m comprises highly fractured loosely cemented sub-angular rock fragments in sand matrix.
It lies immediately above the Porous Conglomerate. It is very prominent in the southern part of the mine from 580mS to 700ms. Stability of draw point cross cuts and brows was heavily affected by the presence of the breccias unit.

1.5 Field Stresses

In conventional SLOS mining method with gravity loading, such as on 2900L and 3150 Levels, main development levels lie within the stress effect of various stoping stages of the ore body. Stress damage is noticed in developments positioned in abutments such as the trough drive, mined within the sill pillar and the footwall extraction haulage mined only 15-20m from the ore body. In both cases the stress effect is the combination of the influence mined out stopes above and the retreating sill pillar stope face on 3150L. In the case of the footwall extraction drive, it experiences a stress cycle of high stress, as the above block is mined out, and low stress as the sill pillar is mined out to 3150L. In both situations, stress induced failures tend to occur in the hanging wall and footwall. Damage due to combined structural and stress conditions is noticed on 2200L in the hanging wall drilling drive. The drive lies very close to the mined out area above (12metres). damage do occur some times right up to the face. To mitigate the problem, the stand up time for the drive is minimized and support is installed close to the face.

Production has been affected by instability of extraction drive and draw point cross cuts on 3095L. Damage to brows and cross cuts was due to over-loading of pillars between cross cuts. The high pillar stresses are due to the high void ratio in the draw level infrastructure coupled with the presence of the extremely weak breccias unit.

1.6 Seismicity

Current seismic activities are confined to the 2900L/3150l, around 500mS area. However, by international standards the seismic events are not frequent and rarely are severe. Events of local magnitude up to 2 had occurred and felt on surface. The seismic activity in this area is strongly related to mining, as observed by an increase in activity levels resulting from high stress build-up in the late stage of stope blasting. Events located in the hanging wall are believed to be as a result of slip on long inclined planar joints around the edges of mined stopes. Damage in developments have been more of a shake-down nature and confined to those areas with inadequate support at the time of the event. The absence of the weak ‘A’ Unit in this area is thought to contribute to seismicity, compared to the north.

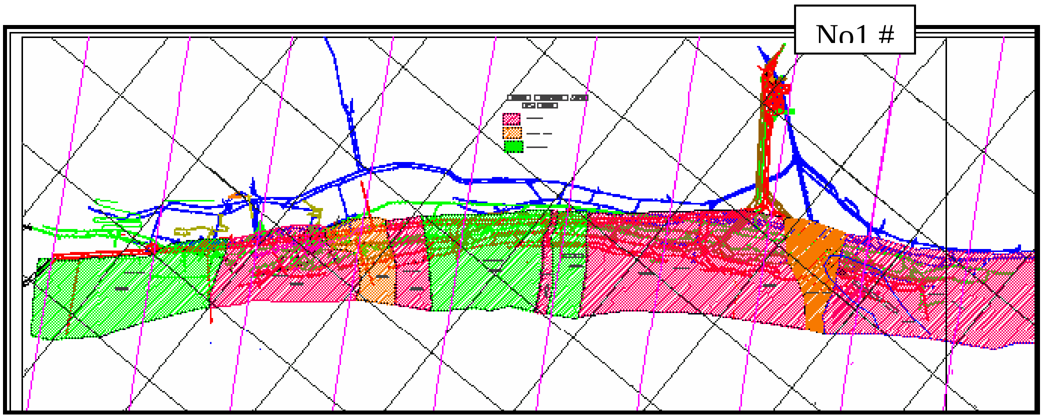
Table 4. Ground conditions in the Footwall

Position	Level	General observations	RMR
300mS	3150	**F/W sst**- hard with numerous rough and smooth joints. ompetent rock,	61-70
		Porous cong- moderate to highly weathered, loosely cemented matrix, few joints	35-50
1500m N	2900	**F/W sst**- hard with numerous rough and smooth joints, competent rock	61-70
		F/W cong- moderately weathered, med hard, few rough joints	61-70
2270m N-S	2250	**F/W sst**- hard with numerous rough and smooth joints, competent rock	61-70
		F/W cong- moderately weathered, med hard, few rough joints	61-70
2700m N-N	2250	F/W sst- hard, moderately jointed, rough joints, competent rock,	61-70
		F/W cong-	61-70
2700m N	2650	F/W Quartzite- strong, moderately jointed, slightly to moderately weathered. RMR 65-80	65-80
200mS	3150	F/W Quartzite- Very strong, few joints, fresh to slightly weathered, competent ground	80-90

Table5. Ground Condition Zones – Ore Shale

Position	Mining Retreat	Zone	Ground condition	RMR
800-300mS	3150L, 300mS	Red	High stress, seismic area.	55-70
300mS-800mN	3150L, 80mS	Red	High stress	55-70,
800-1800mN	3150L, 1500mN	Green	Good ground	55-80
1500-2000mN	2900L, 1500mN	Red	Fair/poor, kaolinised, highly weathered, highly jointed	31-50
2030-2182mN	2250L, 2270mN-S	Red	Fair/poor. Blocky, kaolinised	25-54
2182-2360mN	2250L, 2270mN-N	Red	Poor, fault zone	25-54
2360-2690mN	2250L 2270mN-N	Orange	Fair/poor, blocky ground, high stress,	55-70
2690-2813mN	2250L, 2700mN-S	Red	Poor, fault zone	25-54
2836-3000mN	2250L, 2700mN-N	Green	Good Ore shale	65-85

In general, there are three categories of ground conditions at 1Shaft. The classification is based on the potential for rock falls to occur in a particular rock mass due to stress or rock structure. The classification is done as a way of making workers aware of the different ground conditions in the mine. The **Red Zones** are areas of very weak rocks, Blocky ground and high stress areas, with high potential for rock falls. Orange zones with fair ground conditions, and green zones with good ground conditions. These zones only apply to conditions within the ore body. In the footwall ground conditions are generally fair to good.



3000 2500 2000 1500 1000 500mN 0 500mS 1000 S

N S

Poor-Very Poor Good-Fair Good

Fig 1. Konkola 1 Shaft Ground Condition Plan

2.0 Failure Mechanisms

Different failure modes have been noticed in various areas of the mine. No stress failure has been observed in developments mined in hard, less jointed quartzite formations due to the low field stresses. On 3150L failure in the trough drive mined through strong Ore Shale, with relatively few discontinuities, occurs by crushing and splitting in the up-dip and down-dip sides. The condition of the development deteriorates even more in the vicinity of the open stope. Where the ‘A’ Unity is exposed along the drive, progressive failure of the weak unit occurs creating a deep cavity on the footwall side resulting into excessive span causing unstable conditions in the brow area (See Fig 1).

On 6 June 2003 a seismic event (1.2 magnitude) occurred in the mine causing extensive damage to the 2mx2m drilling drive on 3115L and 3.5mx3.6m trough drive on 3150L with similar failure mode.

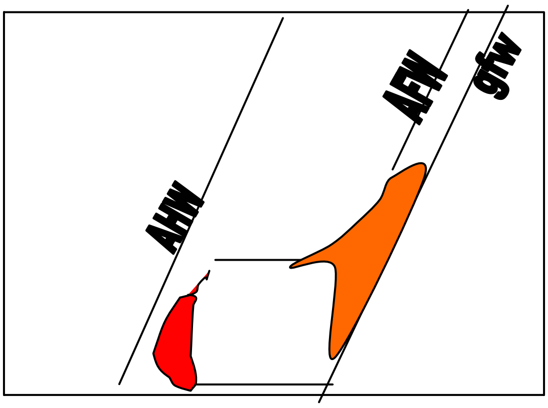


Fig 2. Failure of 'A' Unit in proximity to Brow area of Sill Pillar Stope

The damage in the trough drive was a shake up in nature of the already inadequately supported stress induced failed rock mass (Fig 2).

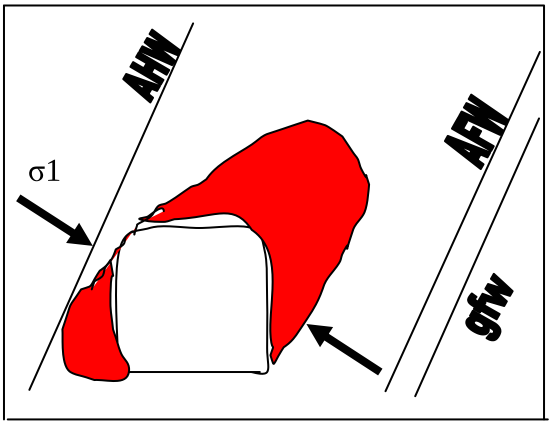


Fig 3. Damage following seismic event on 3150 Level, 770mN Trough drive

Similar type of failure has occurred in sublevel drive in the 2900L/3150L south retreat due to seismic events. No stress failure has been observed in the sublevels of the 2900L/3150L 1500mN retreat due the high strength of the ore Shale in the area, and low induced stress levels due to the presence of a barren gap above 2400L.

Over the years the 3150L footwall haulage mined in the moderately strong Porous Conglomerate, supported with timber, has been heavily damaged leading to total collapse over a distance of 375m (Fig 3). Damage was caused by induced stresses resulting from stoping of the blocks above the 3095Level. The haulage was rehabilitated using 2m roof bolts, mesh wire, tendon straps and 40t mechanically anchored post grouted cable bolts.

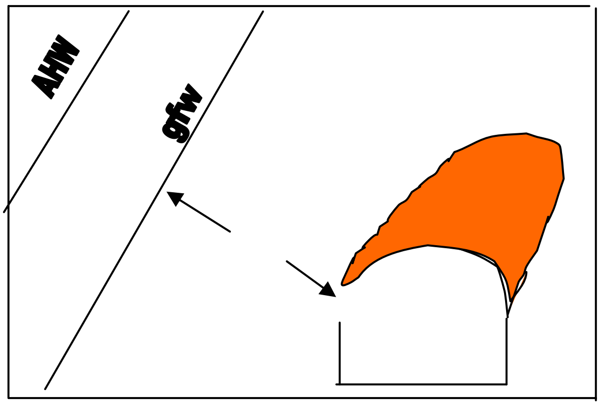


Fig 4. Stress damage of old Footwall Haulage at 800mN

In the heavily jointed, highly weathered, blocky Ore Shale, in the North Extension Area, the rock mass surrounding the old trough drives on 2200L and 2250L, developed in the abutment, failed by pieces of rock sliding on discontinuities and crushing on the updip side.

Due to inadequate support in some places complete collapse of the drive occurred, and could propagate a long way up, resulting into abandonment of sections of the ore body (Fig 6). Similar failure mode occurred in the 2200L 2m x 2m drilling drives, mined in the abutment, 8m below the stoped out area (Fig 5

Structural controlled failures involving falling of prisms/wedges and blocks from the roof and sliding

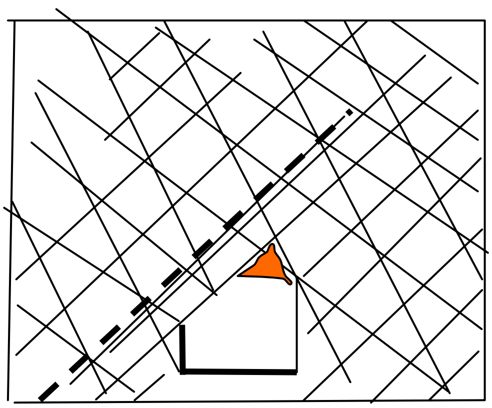


Fig 5. Shanty back mining to reduce Size of potential wedge in the roof

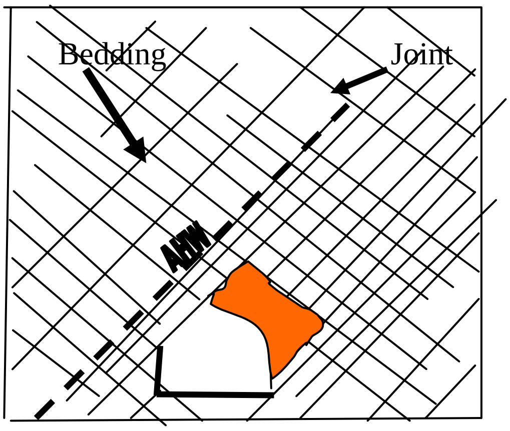


Fig 6.Typical failure in H/W drive mined In jointed kaolinzed Ore Shale

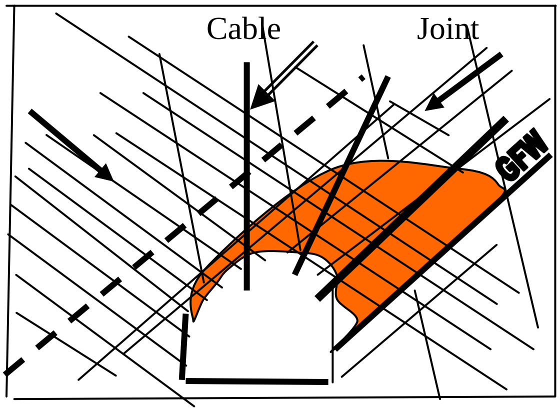


Fig 7. Combine collapse of trough drive on 2250L in blocky weathered Ore Shale.

out of the sidewall is the most common type of failure in some parts of the mine the main tramming haulages, drain drives and extraction drives on 2200L and 2650L.

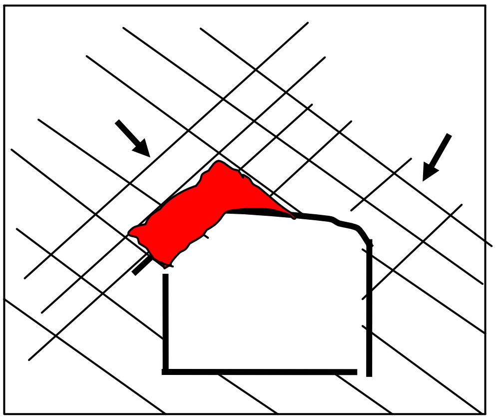


Fig 8 (a) Typical sliding failure in In Quartzite haulage

Failure is controlled by three prominent joint sets, the bedding plane dipping to the west, Joint set 1 dipping to the east and the near-vertical joint set, trending east west.

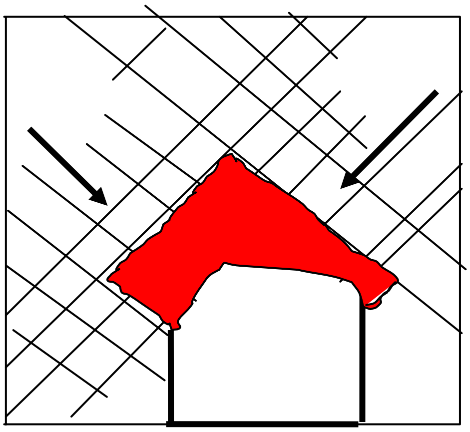


Fig 8 (b) Total roof collapse in Quartzite Haulage

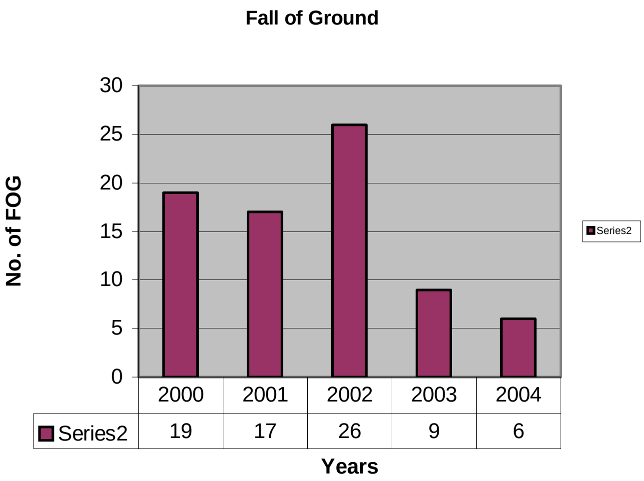
3.0 Falls of Ground

Regardless of the strength and structural properties of a rock mass, loose ground is always present, particularly after blasting. Due to the numerous adverse ground conditions, falls of ground have been one of the major causes of accidents at 1 Shaft. Apart from fall of ground resulting into personal accidents, collapses of developments have been costly in terms of equipment damage, rehabilitation, and loss of production at the mine.

However, to date the number of rock falls and deaths associated with falls of ground has decreased over the years chiefly as a result of improved support designs, better compliance with mine safety and company policies. The figure below shows the trend in the number of falls of ground accidents during the period from 2000 to 2004 (July). Apart from accidents due to falls of ground, Geotechnical section investigates every fall of ground incident in order to find out the route cause, and nature of failures.

During the two years period, the 2200l/2250L North Extension area, accounted for 52% of the total rock fall incidences at 1Shaft. The 80mS and 300mS areas together accounted for 36%, while 12% the rest of the mine. Out of the 25 of the incidents, 48% occurred in developments mined within the Ore Shale, 24% in the Footwall Sandstone and 16% in Porous Sandstone and 12% in Argillaceous Sandstone and Footwall Quartzite. The high number of incidences in the NEA is due to its varied adverse ground conditions compared to other areas, especially within the ore body.

Fig 9. Falls of Ground Accidents 2000-2004 (Aug)



The largest recent seismically induced fall of ground incident occurred on 7 June 2003 on 3150L and 3115L at 775mN position involving a total of 243 tonnes of ground. It caused extensive damage to both developments disrupting production resulting into a loss 4600 tonnes. Mesh wire, straps and cable bolt support had not been install at the time of the incident. Only 2m roof bolts had been installed as the area had just been through between two approaching ends.

A major free fall of ground, in form of prisms, occurred on 2250L 2405mN extraction drive, in December 2003, involving about 500 tonnes of ground. During rehabilitation one and half month's production (15,000 tonnes) was lost from the retreat At the time of the incident the area had been supported with 2m roof bolts, mesh wire and tendon straps. Cable support had not been installed. Majority of falls of ground are in form of free falls of well-defined wedges/prisms and blocks, and occur at the face during barring down and drilling. Few are seismically induced, confined to drives, and are primarily of a shake down in nature.

Table 6. FOG Incidents by rock formation for the period 2003-2004

Year	Ore Shale	F/W Sandstone	Porous Conglomerate	Argillaceous Sandstone	F/W Quartzite	Total	Ave per month.
2003	11	2	0	0	1	17	1.42
2004	2	3	1	1	1	8	1.33
Total	13	5	1	1	2	25	1.39

3.1 Causes of FOG Incidents/Accidents

Free Ground fall incidents are caused by inadequate or lack of support in the areas. The main causes of FOG accidents have been identified as:

- Personnel walking or working in unsupported ground
- Inadequate barring down working places
- Temporary support not installed or poorly used
- Not adhering to standards
- Substandard inspection
- Inadequate hazard recognition
- Poor attitude to safety by some Workers.

Management is committed to ensuring that accidents due to FOG are minimized or completely eliminated. One of the major steps taken by management in an effort to minimize FOG accidents was the introduction of the use of capsulated fast setting cement. This allowed support to be installed right up to the face. Also achieved through safety awareness campaigns, announcements, posters, and safety talks at working places. In most cases disciplinary actions are taken as way to deal with those ignoring safety rules, and these are accompanied by monitory penalties.
To enhance workers awareness, Geotechnical section conducts Rock Hazards Identification and Treatment courses for all underground workers including contractors. Explanations are given why it is important to stay away from unsupported roofs in any working place underground. Fall of ground accident statistics are shown to workers. For the past two and half years about 85% of the intended target group has been achieved every year. The course has now been incorporated into the Underground Training School course for all categories of underground workers. Program to train trainers is under way.

4.0 The Role of Bolting in Rock mass Control as applied at 1 Shaft

4.1 Rock Bolting Theory/Concepts

The principal function of rock bolting is to reinforce and support partially detached (loose), thinly laminated, or other wise incompetent rock mass that would be subjected to failure under the action of gravity. The function of bolts is based on four support theories.

4.1.1 Creation of Compression Zone

Mechanically anchored roof bolts have not been used at 1 Shaft as a support system, largely because its high price. However, the principle of compression zone has been applied in the support of footwall extraction and trough drive developments on 3095 and 3150 Levels, using mechanically anchored 40 Ton post grouted cable bolts in the crown. Cables were pre-tensioned to 10 tonnes before grouting. Conditions have improved tremendously in these areas.

4.1.2 Suspension Theory

The concept of rock bolt suspension theory of grouted or slack bolts considers the bolt as a suspension device transferring the weight of the block or loose zone near the opening to the stronger strata away from the opening. The concept is well applied in jointed and highly stressed ground conditions. The 2m long roof bolts have proved to be adequate in developments such as tramming haulages and extraction drives on 2200L, 2650L and 3150L where they are installed with or without any secondary surface restraints, such as mesh and straps.

4.1.3 Rock Reinforcement

The principle of rock reinforcement applies to un-tensioned grouted bolts anchored tightly together with the rock, using cement grouts, with the bolts offering resistance to rock deformation during active displacement of rock. Application of this principle requires that that the grouted bolts are installed as soon as practical before substantial ground movement takes place, and is achieved through the use of fast setting capsule cement right up to the face.

4.1.4 Beam Effect

Application of Beam Effect is noticed in the stabilization of shanty backs to prevent thin beds in the hanging wall from bulking and breaking, by binding them into large beams.

4.2 Support Design

Design of support is based on maximum loads that would occur on the bolts. Hence, it is critical that both ground demand and bolt capacities are well understood.

4.2.1 Demand and Capacity

Understanding ground demand is the start of support design. Demand is brought about when the state of stress and gravity is disturbed in the rock mass and requires to be restored. At 1Shaft, Rock mass demand or load is assessed using stress analysis, rock mass classification and wedge analysis. Once the potential unstable wedges/blocks or depth of rock failure is established, the length and spacing of bolts are then determined for acceptable safety factor . Lack of understanding demand would lead to poorly designed support system and can be a costly venture to the operation of the mine in terms of safety of employees, safety of equipment and economic extraction of ore. However the basic principle in support is that ground should be made to support itself through correct mine design, proper sequencing and good drilling and blasting practices.

In stress analysis we look at the level of induced stresses around the development and compare that with the strength of the rock mass, and the effect of their interaction with other excavations. Using numerical analysis the extent of ground deformations/failure (loads) can then be estimated. The other methods that are used to determine the load include, empirical support design based on RMR, wedge analysis and through observation in damaged/fallen areas. For some time now instrumentation has not been used at 1 Shaft in determining load for support design. Rock mass rating and wedge analyses are the most widely used techniques at 1 Shaft.

Table 7. Observed Approximate maximum
Depths of Ground falls

Level	Posit	Rock type	Span (m)	Depth (m)	Load per linear mtr (tonnes)
2250	2405 mN	SST	3.8	2.5	24.7
2250	2590 mN	SST	3.8	2.0	19.8
3150	800mN	SST	4.0	3.0	31.2
2900	1450 mN	PC	3.8	1.5	14.8

Visual observations on the different modes and depth of various stress failures in trough drives and footwall haulages in old areas had to some extent helped in confirming the results of numerical analysis. Table 8 shows the observed maximum depth of ground falls in stress-damaged areas of developments located in the footwall on 2250, 2900 and 3150 Levels.

Results of wedge analysis carried out in different developments indicate that most potential wedges along the 3.6mwx3.5mh drives can be stabilized using 2.0m roof bolts on 1.2m x1.0m grid. Example of results from Wedge analysis is shown in table 9, with the data obtained from developments mined within ore body and footwall formations.

Table 8. Example of Results of Wedge Analysis using UNWEDGE Software program.

Area	Rock type (m)	Span (m)	Apex height (m)	Base length (m)	Wedge (tonnes)
2250	F/wsst	3.5	2.6	7.0	24.0
2250	F/wsst	3.5	1.2	3.0	5.6
3150	Shale	3.5	1.9	10.4	15.0
3150	Shale	4.0	2.3	6.7	26.0
3150	Shale	3.5	0.7	3.2	3.1
3150	Shale	3.5	1.5	3.3	4.4
3150	Shale	3.5	0.6	2.7	2.9
2650	FWQ	3.8	1.6	14.9	34
2650	FWQ	3.8	2.0	14.0	44

The capacity of the support element is provided by the strength of the material and the surrounding grout in case of grouted bolts. Once the demand and bolt capacity are known, the support system is designed in terms of length, orientation and spacing of bolts.

4.3 Current Support Systems at 1 Shaft

As mentioned earlier in the introductory remarks, support installation forms part of the major mining operations at 1 Shaft. The different support materials used at 1Shaft include the flexible and super flexible support as primary and secondary support systems. Primary support in form of roof bolts is installed immediately behind the advancing development face using the Minova Fast Setting Capsule Cement. Four Cam-lock Props under mesh are used as temporary support at the face during support and production hole drilling. Secondary support, where it is required, is usually installed at a later stage behind primary support, normally in anticipation of ground deterioration due to changes in stress regime as a result of stoping. However, certain ground conditions of poor rock mass quality would demand immediate installation of secondary support a short distance from the advancing face.

4.3.1 Primary support system:

Primary support comprises the following.

- 1.5m and 2.0m long fully threaded roof bolts with 150mmx150mm domed plate and nut.
- 1.5m and 2.0m, 18mm diameter de-stranded hoist ropes.
- S39 X 1.8m long Split Set Stabilizer (Perma Set).

The Minova Fast Setting Capsule cement is the basic grouting medium used in primary support installation, with Portland cement use in the grouting of cable bolts and all de-stranded hoist ropes. Perma Sets are used only when ground water has been encountered and Cement grout cannot be used.

4.3.2 Secondary support system**:**

- 25 Tones, 15.2mm diameter bulbed strand, 6.5m long (active and passive).
- 15Tonnes (after burning), 18mm diameter de-stranded plane wire ropes.
- Mesh wire 3.1mm gauge, 100mmx100mm, 2.4m wide 15m long.
- Tendon Straps, 6m long by 0.3m wide.
- Shotcrete, 50mm thick.
- Passive, Timber and Yielding Steel Arch Setts.
-

The choice of application of a particular system depends on ground demand and material capacities. Where 50mm shotcrete layer was applied in the trough drives on 2200, 2250 and 3150 Levels, excavations conditions remained good. The tables below show the type of support systems used in different excavations mined in different ground conditions.

4.3.3 Support Standards

Based on the designs support standards are developed as a guide in support installation in different ground conditions of the mine. Face crews and contractors are urged to strictly adhere to support standards. During routine audits, for quality control, Geotechnical staff measure the quality of installed support against the standards.

Table 9. Current Support Systems – Support densities

Table 9 (1) 3150 Level, 300mS, SLOS

Ground conditions	Development Type		Remarks
	Footwall drive / x/c (2mx2m)	F/W Extraction drive x/cuts, and trough drive (3.5mx3.6m)	
Bolt grid	0.8mx0.8m	0.8mx0.8m	
Seismic Area/High stress	6.5 bolts/m 0.17 rolls of mesh/m 0.33 straps/m	10.5 bolts/m 0.33 rolls/m 1.0 strap/m 1.5cables/m	Support redesigned following seismic fatal accident

Table 9 (2) 2200/2650Level, NEA, Mini Back SLOS

Ground conditions	Development Type				Remarks
	Drilling drive (2mx2m)	Trough drive (3mx3m)	Extraction x/cuts (3.5x3.6)	F/W Extraction drive (3.5x3.6)	
Bolt grid	1.0mx1.0m	1.2mx1.0m	1.2mx1.0m	1.2mx1.0m	
'E' Unit	4.5 bolts/m	-		-	Preferred ground for small drives
Fair ground	5.5 bolts/m	6.5 bolts/m 0.2rolls/m 1.0 Straps/m	6.5bolts/m	6.5 bolts/m 0.13rolls/m 0.33 Straps/m 0.5cables/m	At junctions only
Fault zone	5.5 bolts/m 0.17rolls/m 0.17traps/m	-	6.5bolts/m	6.5 bolts/m 0.5 rolls/m 0.33 Straps/m 1.0 Cable bolts/m 50mm Shotcrete	Large ore body drives avoided in this ground. Straps may be installed along x/cut.
Weak Ore Shale, high stress	5.5 bolts/m 0.17rolls/m 0.17traps/m	-		-	Trough drives are avoided in this ground.

Table 9 (3) 2900/ 3150evel, 1500Mn, SLOS

Ground conditions	Development Type				Remarks
	Drilling drive (2mx2m)	Trough drive (3.5mx3.6m)	F/W Extraction drive (3.8mx3.9m)	Quartzite Tramming Haulage/ramps (3.8mx3.9m)	
Bolt grid	1.0mx1.0m	12mx1.0m	1.2mx1.0m	1.5mx1.5m	
Competent Ore Shale Units B-E	3 bolts/m	-	-	-	Now Bamboo or de stranded hoist ropes
Stressed/distressed ground – mostly Porous Conglomerate	-		7.5bolts/metre 0.33 rolls/metre 0.33 straps/metre 1.0 cable bolts/m	-	40tonnes cables @ 1/m 25tonnes @ 1.5/m
Fair Ore Shale stressed in the 800mN	-	6.5 bolts/metre 0.33 rolls/metre 0.33 straps/metre 1.0 cable bolts	-	-	40tonnes cables @ 1/m 25tonnes @ 1.5/m
Footwall Quartzite, few joints, fresh to slightly weathered, Competent ground	-	-	-	5.5 bolts/metre.	Bolts installed in roof to just below the shoulders.

Table 9 (4) 2650/2900Level, 1500mN SLOS

Ground conditions	Development Type		Remarks
	Hanging wall drilling drive and x/cuts(2mx2m)	Quartzite Tramming Haulage/ramps (3.8mx3.9m)	
Bolt grid	1.0mx1.0m	1.2mx1.2m	
'E' Unit	5.5 bolts/m		Preferred ground for drilling drives
Blocky ground, kaolinized, weak Ore Shale, high stress	5.5 bolts/m 0.17 rolls/m Sublevel timber sets	-	Only cut out cross cuts are mined through the weak Ore Shale
Jointed, slightly to moderately weathered F/W Quartzite	-	7.5 bolts/m, Occasional mesh and straps	Common ground conditions on 2200L and 2650L.

4. 29

4.4 Support Costs

Support costs vary according to the mining method and the prevailing Geotechnical conditions in the area. From the table below, the 3150L retreat attracts less support due to the nature of the mining method and the high strength of the Ore Shale in which 70% of the development is located, and low stress. In the NEA, despite the Ore Shale being poor in most areas, major developments are located in the footwall. The high support costs in the three areas, with fairly competent Ore Shale, is mainly due to the effect high stress and seismicity.

From Table 10, support costs per metre for a 2.0mx2.0m drive, range from $ 11.34 in the 3150L to $17.10 in the NEA and $47.68 in the seismic area, Whilst for the 3.5mx3.6m support cost range from $74.99 in the NEA to $180.00 in the seismic area.

Table 10. Support costs by Area and Mining Method

Retreat	Mining method	Stope tonnes	Cost per tonne ($/t)	Support cost distribution (%)
2650L NEA	Mini back SLOS	15000	0.45	6
2900L, 1500mN	Conventional SLOS	32000	0.35	11
3150L, 1500mN	Conventional SLOS	42000	0.15	5
3150L, 300mS	Conventional SLOS D/Dip	50000	0.59	26
3150L, 80mS	MOS -SPR	35000	0.90	27
3150L, 1075mN	MOS -SPR	35000	0.84	25

4.4.1 Recent Measures taken to Reduce Support Costs

Over the past one and half years, efforts have been made in the reduction of support costs in various areas of the mine. The first step in reducing support costs has been through positioning developments in the best possible ground, avoiding locating critical excavations in very weak rock mass. This has paid off quite significantly in the NEA, where mining layouts are dictated largely by ground conditions. Trough drives are mined within the ore body only where the Ore Shale is competent. Constant face inspection and mapping is essential in order to keep developments in best possible ground. Regular support reviews have been carried out based on information obtained from Bore Holes and at the face as mining development progresses.

The table shows substantial savings that have been achieved through support redesign. The largest saving was made in the area of cable bolt support in the North Extension Area, as well as in the 3150Level 1500mN Retreat

Table 11. Support Cost Reduction Measures

	25T Cable bolts @ 0.5 bolt/m	18mm De stranded wire ropes @ 0.5/m	Wedge analysis and support Capacity	70	To date, old areas still standing well
	Fully threaded roof bolts @ 8/m on Capsules	Fully threaded, 4 bolts/m on capsules, plus de stranded wire ropes @ 4/m on Portland cement	Insignificant stress or wedge failure on sidewalls of drives and cross cuts.	24	Wire rope fabrication and installation by contractor
3150L, 1500mN Sublevel Drilling drives	1.5m Fully threaded @ 3/m	1.5m Bamboo/Wire rope @ 3/m	Rock mass rating, Stress Analysis.	70	Wire rope fabrication by contractor, installation by KCM.
2900,3150,1500mN Extraction Haulages and Trough drives.	40T Anchor, grouted Cable bolts @1 bolt/m	25T Multi-bulb, grouted cable, @ 1.5/m	Wedge, Stress Analysis	60	Keeping Factor of Safety constant
All Areas	Heavy Duty Tendon Strap	Light Duty Tendon Strap	Subjected to stress and wedge loading close to stope face	26	Effective in excavations with poor profiles

4.5 Problems affecting support installation

Like at any other mine, support installation is not carried out without any difficulties. Problems of maintaining quality support apply to both primary and secondary support installation by both face and contractor crews. Poor drilling in terms of depth, orientation with respect to discontinuities, dip and spacing of holes, inadequate grout in holes, loose face plates on roof bolts is not uncommon in sections. A number rock blocks have fallen down due to poor hole grouting. These become serious near misses in the mining environment. Fall of ground have occurred between widely spaced bolts and fretting around roof bolts with loose faceplates. In addition, poor roof bolt spacing makes it difficult for the support contractor to install wire mesh and tendon straps. Loss of grout through fractured ground or open fissures can be a serious hazard if not reported to the supervisor. A number of rock fall incidences have occurred as a result of poor support not adhering to standard.

4.5.1 Quality Control and Compliance

Quality control in ground support starts with inspection of support materials and equipment before drilling and installation can commence. Once on site the next stage in quality control is the drilling of the correct size of the hole by getting the right drill bit.

To ensure that standards and compliances are adhered to Geotechnical staff conducts routine support audits in all the sections. Disciplinary action is taken against any face crew found working in an area, which is not in compliance.

One most important quality control measure done by the Geotechnical staff on installed bolts is the pull out resistance tests (Table 12) conducted once a week. The tests are carried out using 18 Ton Pull rig with hydraulic hand pump. Ten bolts are pulled at random in anyone area. Anything less than 10tonnes is considered fail. If one bolt fails the grouting crew is cautioned in order to learn what might have happed. More than one bolt disciplinary action is taken for poor grouting practice.

Table 12. Typical Pull Test results

No	Area	Wall	Test to 10 T		Bond Type	Grout Length	Remarks
			Pass	Fail			
1	2550L 2270mN Extr. dr.	H/W	P(10)	-	Capcem Capsules	2.0m	N/PPulled
2	“	“	-	F(7T)	“	“	P
3	“	“	P(10)	-	“	“	N/PPulled
4	“	“	P(10)	-	“	“	“
5	“	“	P(10)	-	“	“	“
6	“	“	P(10)	-	“	“	“
7	“	East	P(10)	-	“	“	“
8	“	“	P(10)	-	“	“	“
9	“	“	P(10)	-	“	“	“
10	“	“	-	P(4)	“	“	P

Fitting tight the face plates on the roof bolts is achieved by using sockets fitted onto T-Bars. However, in some places, faceplates still remain loose on the roof bolts because sockets are not found everywhere at all times. In order to eliminate excuses of socket unavailability, the use of

Lacing Eye nut was introducedd. The faceplates can then be tightened using steel bars up to 20mm in diameter by any body at any time in their working places

5.0 Conclusion

Whether under design or over design of support, both can be costly in terms of safety risks and material and installation costs. It is important therefore that ground demand is carefully evaluated if the right quantity of support is to be estimated. However, though unavoidable, estimating these loads that will act on the supports is the most difficult part of support design process. Support standards developed are not static and should change from time to time depending on encountered ground conditions. Quality control of support installation is an important part of the support process. It is necessary that the people involved in the installation fully appreciate the need to install as per design through continuous communication with designers and supervisors.

Continuous assessment of ground conditions in a mine is key to economical support system, through estimation of loads in changing ground conditions. To some extent lack of specialist support contractors has contributed to poor quality of support installation in the mines.

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