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Performance behaviour of fully grouted rock bolt as support system

Rock bolts are one of the most prominent and economical techniques of rock stabilization used widely in various applications such as mining, tunneling, surface slope stabilization. Various researchers have found that efficacy of rock bolt system depends on the many factors such as annulus space, type of bolt, grouting media, installation mechanism and positioning. Since its inception, many researchers have studied the behaviour of bolts experimentally and analytically. At present in Indian mining industry, the use of fully grouted rock bolts are increasing rapidly owing to the number of accidents in underground mainly due to roof fall. This paper reviews the performance of fully grouted rock bolts under axial loading conditions and presents a case study of an Indian underground metal mine. The review provides an overview of the need for rock bolts to support the roof rock and the bolting system used in Indian mines along with characteristic parameters such as ultimate load, displacement and failure mode under axial load. A study was undertaken to understand the behaviour of fully grouted un-tensioned rebar bolts in an Indian metal mine having competent host rock. The case study describes the performance of fully grouted un-tensioned rock bolt supporting the hangingwall at a depth of 150 m from the surface. The anchorage strength of the bolts were determined in field conditions by conducting pull out test at different durations from the time of installation of the bolt. The generated load-displacement curve shows the de-bonding nature of the installed bolts under axial loading conditions. The safety factor was calculated for the designing of support system of the mine and the rock load was found using empirical equations using rock mass rating. The safety factor calculation resemble successful application of un-tensioned roof bolt system in supporting an underground metal mine.

Keywords: *Rock bolt; tensile load; mining; rock reinforcement; pull-out test.*

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1.0 Introduction

Roof or rock control is a major problem distressing safety, productivity and mechanization in underground mines in India. Since the first use of primitive slot-and-wedge rock bolts in 1927 and the proposed use of rock bolting as a systematic method for roof support by Weigel in 1943, rock bolting has become the most important support system in mining industry. Rock bolts and dowels have been used for many years for the support of underground excavations and a wide variety of bolt and dowels have been developed to encounter different necessities which arise in mining and civil engineering (Snyder, 1983). Rock bolts generally consist of plain steel rods with a mechanical or chemical anchor at one end and a face plate and a nut at the other (Windsor and Thompson, 1996). They are generally tensioned after installation so as to create an “active” support system. When bolts are not tensioned or tensioning is not possible then they act like a “passive” support and are commonly termed “dowels”. The rock bolting is the only support system being used in most of the mines. The performance of bolts depend on the behaviour of rock mass and status of stress developed around an opening (Fig.1). Such system is very efficient in applications such as stabilization of blocky rock masses, rock confinement and improvement of the mechanical properties of the rock (Chappell, 1989; Fine, 1998). The study of performance of bolts has attracted many researchers globally and many researchers have reported the application of bolting system for ground control in mines (Lang et al., 1979; Schach et al., 1979; Bolstad and Hill, 1983; Gardner, 1971). Littlejohn, 1993 underlined that the failure of rock bolt under axial loading is influenced by the material characteristics of the bolting system or may depend on the individual components (bolt, the interface between bolt-grout and grout-rock, grout media itself). Based on the type of anchorage system, Hoek and Wood, and Franklin and Dussealt in 1988 & 1989 respectively classified the rock bolts in three categories; point anchored by wedge mechanism, friction anchored bolts, and fully grouted rock bolts anchored by the grout media between rock and bolt. Windsor, 1997 extensively studied the various bolting system and mechanism after which he classified rock bolt system based on their coupling mechanism into three coupling system viz. continuously mechanically coupled (CMC), continuously frictionally coupled (CFC) and discretely mechanically or frictionally coupled (DMFC).The mechanical behaviour of the fully grouted rock bolt is governed by the type of loading; it is subjected to and the load at which it fails after the grouting. In actual field conditions, bolt behaviour relies on the rock deformation, rock fractures due to tangential stress or may be due to the combination of both. Hence, in the field of rock mechanics the tensile test and shear test are documented as the true measures to determine the rock bolts performance. Over the past decade series of static pull-out tests have been conducted in-situ condition (Singh et al., 2016 a and b) as well as in laboratory with different test bench simulating field loading conditions to apprehend the performance of fully grouted rock bolts by the various researchers around the globe.

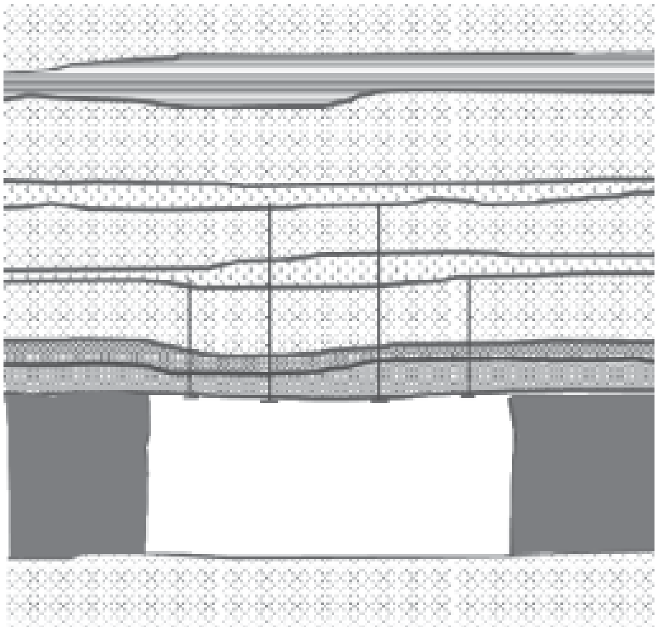

Fig.1 Rock bolt support system in an underground mine opening

1.1 Stress distribution and reinforcement mechanism

Whenever any underground opening is excavated stress equilibrium is disturbed and the portion of strata directly above the opening loses its original support. As a result, the load of the immediate roof is transferred towards the sides of the opening, commonly known as abutments. The roof starts to sag under the gravitational force. If the immediate roof strata are competent, the sag will stop before the roof collapses and the stresses around the opening will eventually reach a new equilibrium. However, in many mines, the immediate roofs are not competent enough to sustain the changes of the stress distribution and the interaction induced by mining. These may finally collapse into the opening if they are not sufficiently supported by rock bolting system (Freeman, 1978). The bolting system enhances the possibility of early stabilization following excavation (Karabin and Debevec, 1976; Serbouski and Signer, 1987; Signer, 1990; Gupta, 1997). The stress disturbance creates five influential zones around a rectangular opening (Peng, 1986), as is shown in Fig.2. In zone 1 and zone 2, the strata are released from the super incumbent pressure and downward vertical displacements takes place due to the gravitational force within the zones. In zone 1, inter laminar separation occurs and the magnitude diminishes gradually upwards due to the clamping action of abutment pressure and frictional resistance between the layers. There is no bed separation in zone 2, but displacement is still noticeable. In zone 3, both vertical and horizontal stresses build up, forming an arch-shapedhigh-stress zone. Floor heave occurs in zone 4. In zone 5, the rib sides expand towards the opening. Both movements in zones 4 and 5 cause vertical movement of the abutment and strata above, releasing the built-up stress in the arch area significantly. Thus a certain amount of floor heaves and rib expansion is beneficial to maintaining the opening's stability.

The main purpose behind the bolting is to make the opening stable and the roof rock intact after an excavation has been made underground. The bolt increases the inherent strength of the rock mass to support itself in the induced stress field (Bolstad and Hill, 1983; Beniawaski, 1984). The reinforcement is greatly subjective to the type of rock bolt and anchorage system installed as the rock type, and lithology of the area may also vary. To maintain the stability of an underground opening, it is essential to keep zone 1 stable. Roof bolts in this zone force all the bolted layers to sag with the same magnitude; the layers within the bolting range thus act like a solid beam. Ideally, the beam must be strong enough to carry all the weight of strata in zones 1 and 2 plus the extra load transferred to the zones by mining activities nearby. Building such a beam is actually the ultimate goal of roof bolting where beam building effect is the prevalent mechanism.

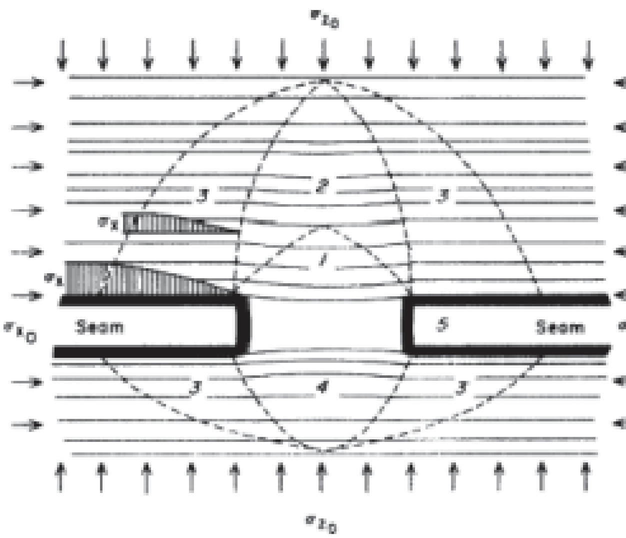

Fig.2 Stress distribution around an opening (Peng, 1986)

2.0 Fully grouted rock bolts and failure mechanism under axial load

The use of fully grouted rock bolts is not a new technology and its benefits over other bolting system have been advocated by several researchers over the years. The fully grouted roof bolts are very successful in supporting various roof strata in underground mines as compared to point anchored bolting system (Carr, 1971; Parker et al., 1973; Reed, 1974).Till now various studies have been taken up by various researchers to understand the load transfer and failure mechanism of fully grouted rock bolts under tensile loading conditions. Some studies were conducted in simulated laboratory conditions while some researchers performed the pull-out test in actual field conditions. These studies include the significant work of researchers such as Pells (1974), Littlejohn and Bruce (1975), Farmer (1975), Serbouski and Signer, (1987), Aydan (1989), Signer (1990), Holmberge (1991), Skybey (1992), Ebisu et al (1993), Benmokrane et al. (1995), Stjern (1995), Mark et al. (2002), Kilic et al. (2002, 2003), Aziz (2003, 2006), Hagan (2003, 2004), Aziz (2004), Compton and Oyler (2005), Aziz and Jalalifer (2005), Karanam and Dasyapu (2005), Jalalifer (2006), Li and Doucet (2012), Li (2010, 2012), Hyett et al. (2013), Martin et al. (2013), Ma (2014), Chen (2014), Chen and Li (2015), Ghadimi et al. (2015), Zhao et al. (2016) and Zhou et al., (2016). To understand the load transfer mechanism pull and push test are being carried out and are acceptable for investigating the bolt-grout-rock interaction and to know about the interface stress distribution in the fully grouted system of bolting. Gardeen et al. in 1977, through his studies, concluded that fully grouted rock bolts are five times more effective than mechanical bolts in beam process, as well as the stiffness in tensile load is 10-20 times greater than that of mechanical bolts. Snyder (1983) through his studies concluded that where the strata layer is relatively thin, fully grouted rock bolts are a safe option for the ground control than another bolting system. Fully-grouted rebar bolts are bound to the grout/rock via ribs on the bolt surface, with the main anchoring mechanism of the mechanical interlocking between the ribs and hardened grout. Bond failure will commence at the loading point when the applied load is beyond a certain level, propagating toward the far end of the bolt with an increase in the applied load (Fig.3). When the grout media is resin than axial deformation up to 100 mm has been reported by Harrison in 1987, as compared to the sudden failure in another bolting system under axial loading. Stjern and Myrvang, 1998 installed the bolt near to face in tunnel blasting and observed no damage to the bolt system due to blasting. The bolt rod of same grade enhances the strength of rock mass with grouting across the whole length of the bolt in comparison with point anchored system (Gray et al., 1998), and the degree of load transfer is also high (Whitaker, 1998). In the fully grouted system, the load transfer depends on the bolt-grout interface and grout-rock interface. Fig.4 depicts the forces which help in determining the degree of load transfer. The load transfer in the fully grouted bolt is determined by measuring the peak shear stress and stiffness of the system.

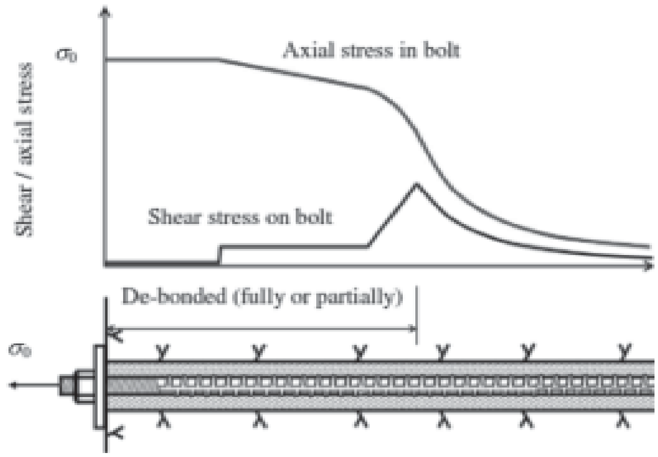

Fig.3 Stress distribution along the bolt length in pull-out test (Li et al., 2014)

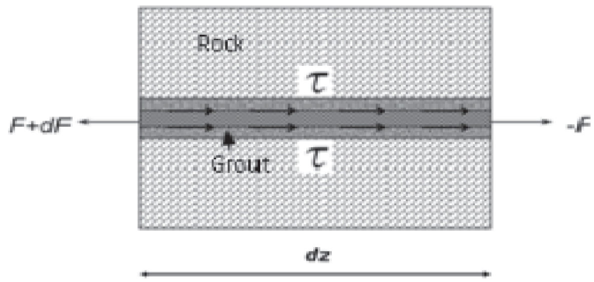

Fig.4 Load transfer and stress distribution in fully grouted rock bolt (Singh et al., 2016a)

Estimating the rate of load transfer Fabjanczyk and Tarrant (1992), conducted several pull-out tests to conclude that the utilization of full load capacity was dependent on the displacement of the bolt system. To understand more about the axial displacement of the bolt with different grout system, a series of simulated pull-out tests were carried out by Stillborg in 1994 using two concrete blocks of 60 MPa in the laboratory. The results of the tests are shown in Fig.5, which highlights the rate of load transfer in resin as grout is higher than other system. However, the effectiveness of resin as grout media after considerable time interval is still under review for long term stability of bolting system.Serbousk and Signer (1987) conducted many pull-out tests in anattempt to know the influence of hole size and grout type on the performance. They conducted tests on the 1.2 m and 0.3 m bolts in 25.4 mm and 44.4 mm diameter holes, with the nature of the applied load was limited to an elastic response and the test were not destructive in nature. When the load is applied and movement start to take place due to interlocking mechanism shear transfer from one media to another takes place until the maximum shear strength is reached. Serbousk and Signer observed that the hole size and grout type does

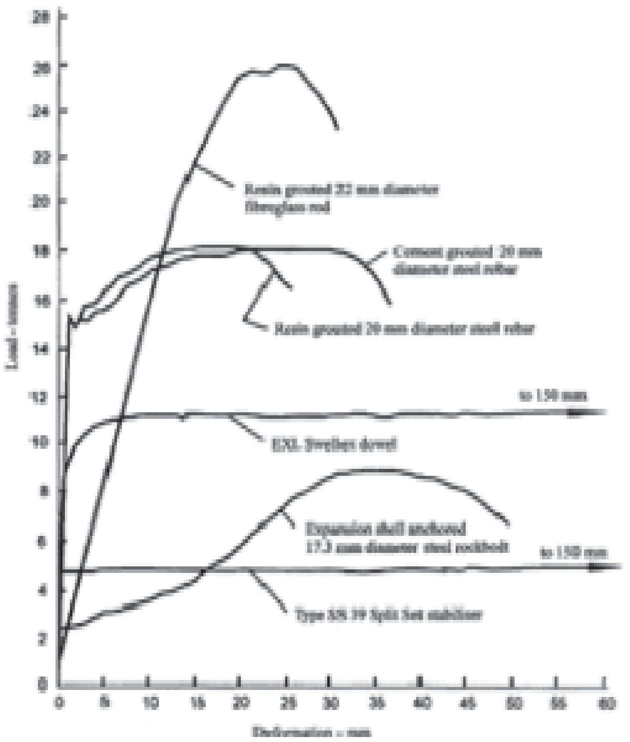

Fig.5 Load-deformation results obtained by Stillborg (1994) in tests carried out at Lulea University in Sweden

not have much influence on the load transfer rates in the elastic limit. However Fabyznchic et al (1998) and Aziz (2004), reported results which differ from the results of Serbousk and Signer. Serbousk and Signer also proposed ananalytical model based on the assumption that there is no rock deformation or slippage between the interfaces whereas Jalalifer shows that the grout got crushed in the elastic limit and experience a non-linear relation.

The design and shape of theboltare also an important parameter while studying about bolt performance as it is the basic and integral part of the bolting which Kilic et al. (2002, 2003) acknowledged and performed series of pull-out test with different types of bolts. The researchers used single, double and tripled conical lugged bolts (Fig.6). The results reported by the researchers indicate that the shape has an influence on the performance of bolt under axial loading conditions with triple conical lugged bolt had the best performance with high deformation and load bearing capacity.

Fig.6 Different type of bolt used in the test

In fully grouted bolting system different interface interact such as bolt-grout-rock interface which may alter the performance of the installed bolt. In 1998, Aydan carried out push and pull test using two steel bar of 13 mm ad 19 mm and investigated the anchorage mechanism of grouted rock bolts and also the effect of parameters i.e. ratio of the bolt to borehole diameter and behaviour of the bolt to grout interface under tri-axial stress. The results of the push and pull tests were different with each other as load bearing capacity of the bolt was 25% higher in push out test which may be due to the poisons ratio effect. The Poison effect was also studied by Jalalifer and Aziz in his set of push and pull test and concluded that the pull-out test is a better measure to know the load bearing behaviour of the bolts based on the axial and lateral strain reading in push and pull test. Aydan based on his outcome and observation suggested that the failure or shearing might occur along one of the interfaces in the bolting system. He further classified failure mode in push and pull test in three failure categories

1 Failure along the bolt-grout interface occurred in bars with smooth surface and bars installed in the large borehole.

2 Failure along the grout-rock interface occurred in bars installed in smaller holes.

3 Failure by splitting of grout and rock annulus.

The previous researchers pointed out the load transfer mechanism but it was in 1990 that Signer through his pull-out tests investigated the transfer of applied load from bolt head to the rock. The hydraulic testing arrangement was used by Signer with strain gauged bolt and dial gauge for the deflection reading. When the load is applied the bolt head would deflect and the amount was measured with the dial gauge assembly (Fig.7a). When the load applied is more, the stiffness of the system was also increased indicating the stiffness of the system due to mechanical interlock among bolt to grout and the rock. Signer in his tests used polyester resin and gypsum as a grout in 19 mm diameter hole and 25.4 mm diameter hole. The results of his tests indicated that 0.56 m of length was required to transfer 90% of the load from bolt to the rock. Fig.7b shows the load distribution along the length of the bolt. Furthermore the results showed that if there is sufficient length of bolt past the yield zone, then the load will transfer from the bolt to the rock. This means that the grouted bolt can still be an effective support past the yield point of the steel.

3.0 Case study

Singh et al., 2016 conducted series of in-situ pull-out tests on the cement grouted rock bolt/dowel to understand the performance of the bolt supporting hangwall of an underground metal mine in India. The mine was explored up to 175 m level and the test was carried out at 150 m level stope. The samples were drawn carefully to determine the geo-technical properties (Table 1) of the roof rock and the rock

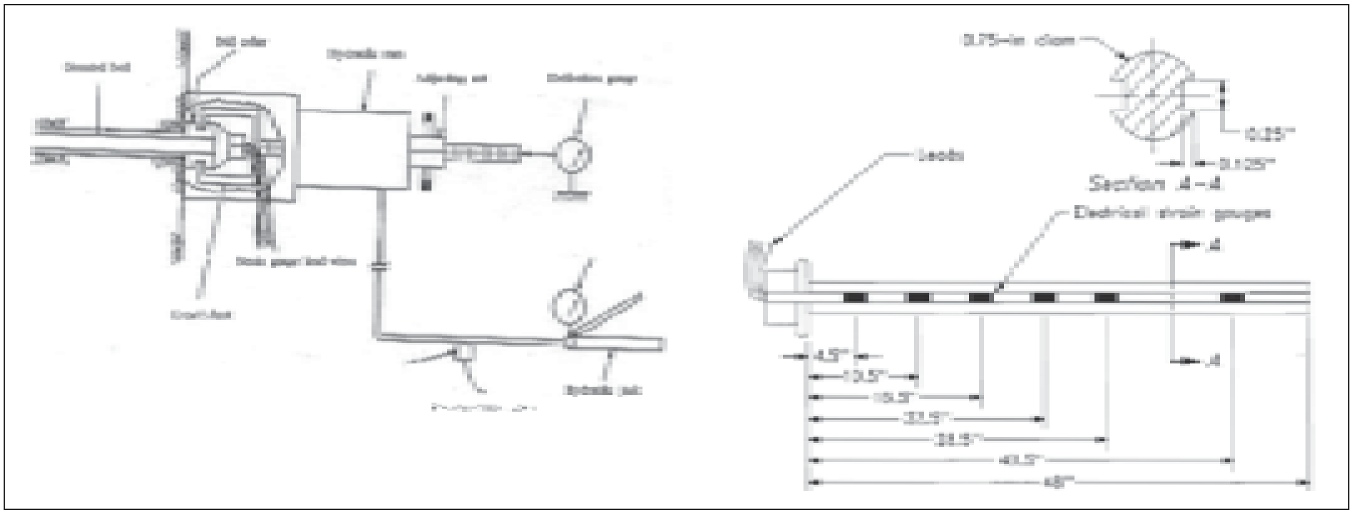

Fig.7a Pull-test arrangement and strain gauged bolt used by Signer (1990)

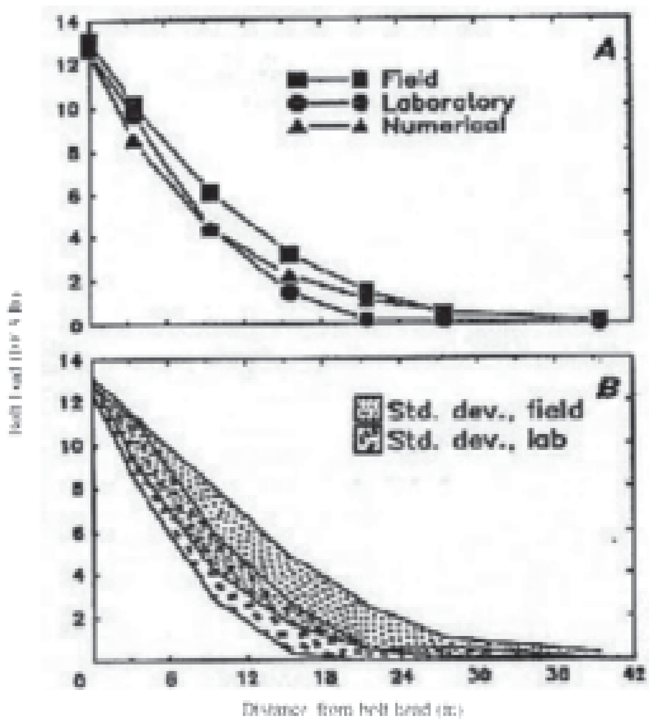

Fig.7b Load distribution along the bolt length

mass rating of the level. The host rock of the mine is hard and competent as the calculated RMR as per CMRI-ISM method is 65.

There were three sets of joints observed in the roof rocks and their strike, dip direction, and dip amount were measured using Brunton Compass. The joints were tightly held with no gauge fillings. The Hanging wall of the mine is supported with TMT bar rock bolt of 22 mm (with rib) diameter and 2.4 meters long at 1.2m x 1.2m intervals in a grid pattern. No other support system being used by the mine. An optimum thick grout (0.35 water cement ratio grout) is pumped into the hole

TABLE 1. SUMMARY OF STRENGTH PROPERTIES OF ROCK MATERIAL

Properties	Values
Uniaxial compressive strength, σ_c (MPa)	65
Tensile strength, σ_t (MPa)	10
Young's modulus, E (GPa)	4.95
Poisson ratio, ν	0.25
Density (g/cc)	2.82
Cohesion, C (MPa)	22.06
Friction angle, (°)	44.5

pneumatically by inserting the grout tube to the end of the hole and slowly withdrawing the tube as the grout is pumped in. Special attention is given that there is no air cavity formation in the hole. Singh et al. conducted a pull-out test on the installed bolts at different time after installing of the bolt i.e. 30 minutes, 2 hours, 24 hours, 7 days and 28 days. The researcher used strain gauges to record the deformation induced due to axial loading of the bolt through hydraulic jack (Fig.8). Table 2 contains the summary of the test for different time interval. A series of 15 bolts were tested for 28 days, 7

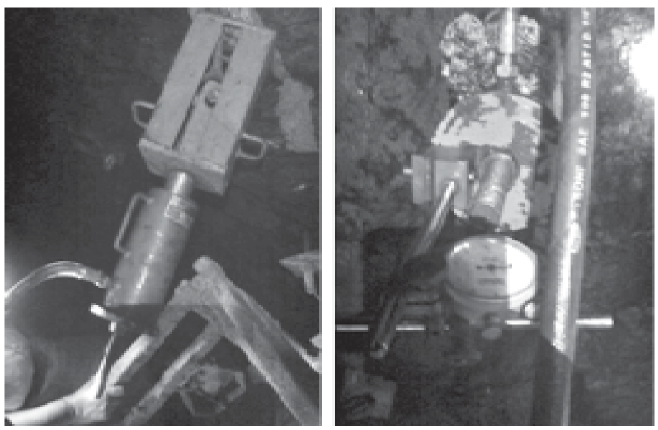

Fig.8 Rock bolt anchorage testing assembly

TABLE 2: SUMMARY OF TEST RESULT AT DIFFERENT TIME INTERVAL

Test time interval	Total test	Hole diameter (mm)	Bolt diameter (mm)	w/c ratio	Mean maximum load (T)	Standard deviation (T)
24 hrs	15	36	22	0.36	5.7	1.6
7 days	15	36	22	0.36	10.8	1.2
28 days	15	36	22	0.36	13.2	1.4

days and 24 hour time interval. The bolt diameter, hole diameter and grouting material were kept same for all the test to understand the in-situ bolt performance with variation in time after the installation of the bolt. Two pull-out tests were conducted for 30 minutes while three tests were conducted at 2 hours duration but the results indicated that the grout was not cured and the bolts pulled-out out very easily. The bonding between the bolt and the grout was not significant with no shear resistance or strength between the interfaces.

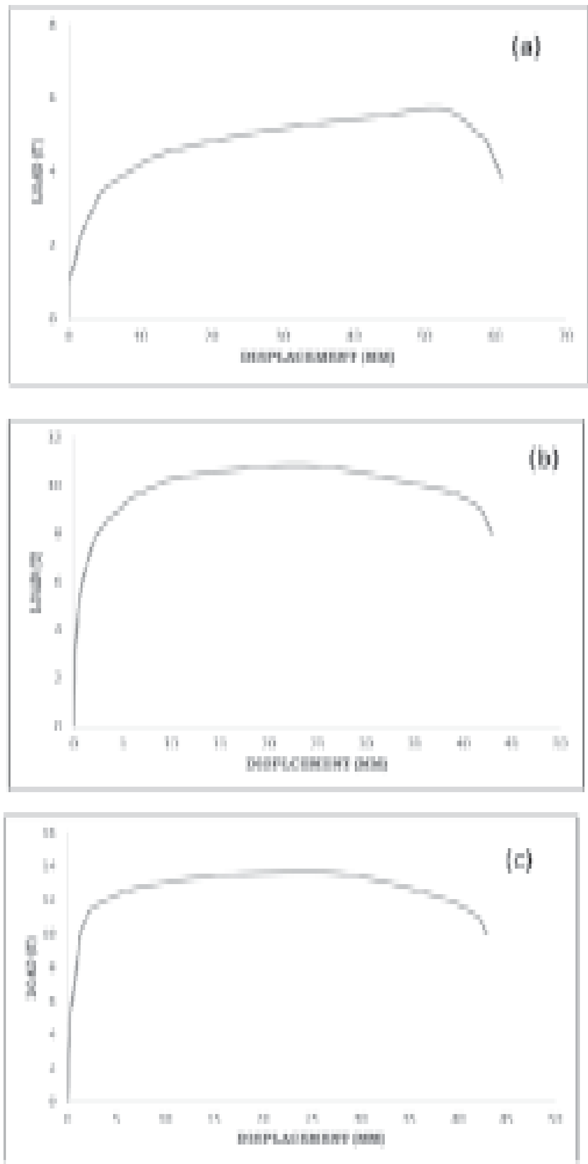

Fig.9 Load deformation curve for axial pull-out test in hard rock mines at different time intervals; a: 24 hours; b: 7 days; c: 28 days (Singh et al., 2016b)

Total 50 tests were conducted and load-displacement curve was obtained in 45 tests while 5 test failed as described above. During the study, the same drill rod was used in drill jumbo to ensure the accuracy of the hole diameter drilled.

The pull-out test results obtained by the authors are presented in Fig.9, which shows that the load-displacement curve at different time intervals of 28 days, 7 days and 24 hours. It was observed that the deformation curve of 24 hours shows low peak load and high deformation for the low axial loads. This may be attributed to the fact that the curing time is not sufficient enough and de-bonding of bolt and grout interface near the collar occurred with slip at lower axial load than 28 days and 7 days tests. In 28 time duration tests after attaining peak load the load goes on to 85% of the peak load with deformation of 5-7 mm only. The bonding between the bolt and the grout was not significant with no shear resistance or strength between the interfaces after 24 hours test duration. The anchorage strength of the rock bolt/dowel after 30 minutes and 2 hours were very marginal and the bolt was pulled out with the weight of the hydraulic testing equipment itself. Further it may be confirmed on the basis of the results that the anchorage strength of cement grouted dowel is achieved after 24 hours and was significantly high for the cement grouted bolt.

3.1 SUPPORT DESIGN

The hanging wall is supported with TMT bar rock bolt at 1.2m x 1.2m intervals in a grid pattern. The back is supported with rock bolts at 1.5m x 1.5m intervals in a grid pattern. Full column cement grout type rock bolts, TMT bar 22 mm (with rib) diameter and 2.4 meters long for hang wall and 1.6 meters long for back with an 'eye' at one end were used. It has been suggested that the draw points, which are mined before the overlying stopes are blasted, are good examples of excavations where un-tensioned grouted rock bolt will work well (Hoek et al., 2000). The essential difference between these systems is that tensioned rock bolts apply a positive force to the rock, while un-tensioned rock bolts/dowels depend upon movement in the rock to activate the reinforcing action.

Designing of the support, the rock load, support resistance, and safety factor were computed using the following equations (suggested by Hoek and Brown, 1980):

$$P_{roof} = B \times \gamma \times \left(1 - \frac{RMR}{100}\right) \quad \dots \quad (1)$$

At, junctions:

$$P_{roof} = \left\{B \times \gamma \times \left(1 - \frac{RMR}{100}\right)\right\} \times 1.2 \quad \ldots \quad (2)$$

where,

'P_{roof}' is the rock load

'B' is the tunnel/opening width in meters

'RMR' is the rock mass rating

'γ' is the density of the rock, kg/m^3

$$SR = {(N \times P_{axial})}/{A} \quad \ldots \quad (3)$$

where,

'SR' is the support resistance

'N' is the number of bolts

'P_{axial}' is the average anchorage strength of the bolt

'A' is the area supported

$$SF = \frac{SR}{P_{roof}} \quad \ldots \quad (4)$$

where,

'SF' is the factor of safety

The values required for the above equations are taken from Table 1. The RMR has been computed using CMRI-ISM method. The mine was visited to observe the strata conditions and the method of working to arrive at the values along with the laboratory results. Samples were drawn carefully from the mine that represents the actual conditions of the mine. Based on the geo-technical parameters of the mine and the values of anchorage strength of the bolt after 28 days, the safety factor was determined for span of 3m and 5m using the equations (1-4). For a span of 3 meters, the safety factor comes to be 1.97 after 28 days of bolt installation. For 5 m span the factor of safety was 1.78 after 28 days of bolt installation which is more than 1.5. Hence the roof rock is stable. For junction, the safety of factor should be more than 2.0 to be on safer side. The safety factor values for junction was 2.51 and 2.22 for 3m and 5m span respectively after 28 days of bolt installation. The safety factor of the support design shows the successful application of the bolt after 28 days of installation i.e. the long term stability of the support system.

4.0 Conclusions

The performance of the fully grouted rock bolts under axial loading conditions has been reviewed in this paper based on the results of various researchers and conducted experiments. With the review, it can be concluded that the bolt behaviour under axial loading and shear loading are the two measures that can be used to evaluate the performance of the bolts. Anchoring of the bolts are desirable in case of fully encapsulated bolts and should have high anchorage values as well. The use of resin as grout makes the bolts capable of supporting the strata immediately after their installation, thus making it safe for the men and machine to work in underground operations where roof is weak and chances of bed separation are relatively high. Although fully grouted rock bolt can carry high loads, however, its displacement capacity is small. One of the important parameters is the surface profile of the bolt which plays an important role in failure mechanism of the bolt. The load transfer in the fully grouted bolts depends on the bolt-grout-rock interface, hence it is imperative to understand the interaction of the three interfaces.The case study presented is to evaluate the performance of un-tensioned fully grouted rock bolt in an Indian hard rock mine and the results of the displacement under applied axial load are presented. The results of the in-situ pull-out test indicate the efficacy of un-tensioned rock bolts in hard rock metal mine. The bolt after 7 days and 28 days of installation have significantly high anchorage strength with deformation of 42-46 mm. Although the un-tensioned rock bolts are not popular now but they are found to be effective in long-term stability and can be used as per site specific requirements. The factor of safety calculations for 3m and 5m span for the bolts after 28 days of installation is more than 1.5 and more than 2 for junctions. Hence fully grouted untensioned eye bolts can be used against pre-tensioned rock bolts in a stope for supporting the hangwall or back.

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