Evaluation of Pull-out Test to Design Underground Coal Mine Support Systems for Depillaring Operations

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ABSTRACT

Roof bolts are active support systems used extensively in underground coal mines. It keeps on supporting the roof of the driven gallery and does not create any hindrance in the movement of man and machinery. Roof bolts have replaced completely the involvement of props, cogs, chocks, etc. used in earlier days during conventional underground coal mining methods. Mechanized depillaring, also, uses roof bolts as breaker lines while depillaring to control the encroachment of goaf during the caving of the overlying roof inside the goaf. Roof bolts are used in Indian underground coal mines as per usual assumptions based on trial-and-error methods. Generally, the pull-out test is carried out in Indian mining conditions as per the circular given by the DGMS to test the strength of installed bolts in underground coal mines supporting the roof of the gallery. Several studies have been carried out on the design of such support systems in earlier days. However, the design of such support systems based on these studies cannot be directly implemented due to the complexity and uniqueness of Indian geo-mining conditions. In this paper, an effort has been made to study the previously developed design of roof bolts through empirical, analytical, and numerical simulation techniques. This paper also presents a brief introduction about the pull-out test practices in underground coal mines with an attempt to simulate the behavior of bolts in underground coal mines on ANSYS software. Using the ANSYS numerical model for the pull-out test we have conducted various analyses to determine the parameters on which the performance of a roof bolt depends and tried to draw some conclusions based on our findings. It is decided to design the roof bolts soon based on validating the results of this numerical simulation for underground coal mines.

Keywords: *Coal Mines, Roof Bolts; Pull-out Test; Numerical Simulation; ANSYS; Modeling*

1. Introduction

The major source of energy in India is coal. The major energy demand of the country is fulfilled by thermal energy where coal is used as raw material. It has been estimated that the scenario will not change for at least five decades considering the energy demand of India. Currently, in India, there is a huge gap between demand and production of coal. The opencast method of mining is a major coal producer in India and covers the reserve under shallow depth of cover range only. The reserve amenable to the opencast mining method is fast exhausting and the future of the Indian coal mining industry lies with underground mining. In India, the conventional mining methods practiced in most of the coal mines yield less production, productivity as well as safety. Technological development for extraction of coal from deep-seated coal seams and locked-up coal from developed pillars are vital requirements for the industry. Highly mechanized underground mining technologies may play a leading role in meeting the challenges of the Indian underground coal mining industry. However, the adoption of mining technology is site-specific and needs proper R&D support to adapt to Indian geo-mining conditions.

Underground coal mining in India is predominantly carried out by bord and pillar methods. It contributes over 90% of the underground coal working today and is expected to prolong in the future. With the advancement of technology, the mining industry is shifting towards mechanized methods. Various large machines and continuous miners are now used for the development and depillaring of the mines (Mandal et al, 2006). In the Bord and Pillar (B&P) system, after the formation of pillars, consideration must be given to the extraction of coal pillars; the operation is known as pillar extraction and is also referred to as depillaring. In Indian Coalfields during underground mining, various kinds of overlying strata problems are encountered. But for a depillaring operation, the problem became even worse as both highly laminated/weak and massive/strong overlying strata are difficult to handle. Poor efficiency and safety factor of the conventional depillaring operations (Drilling and Blasting) for underground pillar extraction is eliminated by a fully mechanized depillaring operation. Mechanized depillaring (MD) operation provides a faster rate of extraction and improved safety factor along with increased production and productivity of a depillaring operation, which is a blessing for the coal mining industry of the country (Raghavan et al., 2014).

2. LITERATURE REVIEW

When an opening is created in rock, the surrounding strata invariably become unstable but can be strengthened by various methods of support. The supports are mainly of two types, one which resists the fall of the weak roof by external force and the other which enhances the cohesive properties of rock to resist its fall and maintain the load-bearing capacity of the strata. The former consists of timber props, steel arches, timber packs, wire-mesh, and sprayed material such as gunite/concrete. In contrast, the latter consists of reinforcement, tensioned-point anchored bolts, friction bolts (split-sets) & resin-anchored bolts, etc., which modify the internal behavior of the rock mass by the installation of structural elements within it.

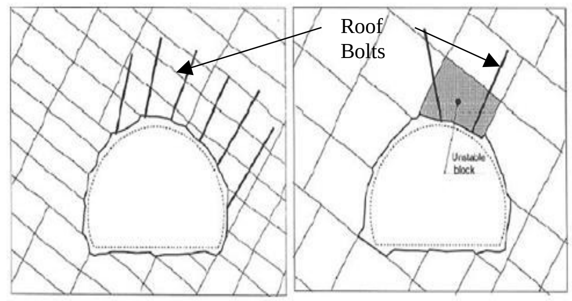


Fig. 1. Rock bolts can be installed both individually and with systematic bolting.

Systematic bolting is a preplanned pattern of bolts based on geological conditions, but individual bolts are also installed to fix single loose blocks (Nilsen and Palmström, 2000).

2.1 Roof Bolt Evolution

The early twentieth century saw many isolated references to the use of rock bolting systems, but it was not until 1947 that the method was developed on an individual scale. In the USA concerns about the rising rate of accidents due to failures of ground prompted the reassessment of underground supports. The USBM introduced roof bolting at this time to help combat adverse statistics. Following the introduction into the mines of the USA and its success not only in combating roof falls but also in making the mining operations more efficient, the practice of roof bolting as a primary support in mines spread throughout the world. By

1952 the use of mechanical point-anchored bolts had spread to the mines of the UK. Unfortunately for several reasons- mainly the ineffectiveness of mechanically anchored bolts in weak strata it had found less utilization in these mines. Synthetic epoxy resins were introduced in the USA in 1956. These were placed into the hole by injection. Cement injection was also tried in France, but the setting time was too short for it to be accepted on a widespread scale. In 1959 SEBV of the Federal Republic of Germany introduced the first resin capsule system. The resin was contained in a glass cartridge, which broke as the bolt was installed. This first resin capsule was called the Klebanker. In early designs using point anchor bolts, the tension in the bolt was required to produce a normal force between layers of strata, thus increasing the frictional resistance. However, the development of fully column-grouted bolts offered a new approach. Beam formation could be achieved with or without tension, the horizontal shear forces being resisted by the lateral stiffness of the grout/bolt combination.

2.2. Types of Bolts

Bolts are categorized into three groups according to their anchoring mechanisms (Li, 2011):

- Mechanically Anchored Bolts
- Friction Anchored Bolts
- Fully Grouted Rock Bolts

2.2.1. Mechanically Anchored Bolts

Mechanically anchored bolts can be divided into two groups: expansion shell anchor and slit and wedge-type rock bolts. The anchoring part is either fixed by a wedge-shaped clamping part or by a threaded clamping (Kilic, Yasar, and Celik, 2002). Mechanically anchored bolts are one of the first rock reinforcements used in underground mining and they are still used around the world, including in Canadian mines. Mechanically anchored rock bolts provide very effective support in many conditions, such as when rock blocks have been loosened by intersecting joints, bedding planes in the rock, or when blocks loosen because of poor-quality blasting.

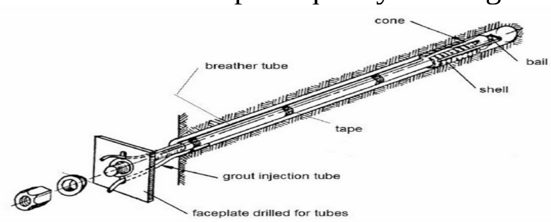


Fig. 2. Mechanically anchored rock bolt.

When the bolt is rotated the wedge is pulled into a conical expansion shell. The shell is forced to expand against and into the rock wall of the hole (Hoek, 2012).

2.2.2. Friction Anchored Bolts

Frictional bolts are bolted in a special way, using frictional resistance to sliding generated by a radial force against the borehole wall over the whole bolt length (Kılıc, Yasar, and Celik, 2002). Friction bolts stabilize the rock mass by friction, without needing any important auxiliaries like mechanical locking devices or grouting to transfer the load to the reinforcing element (Li, 2013). One of the main advantages of friction-anchored rock bolts is that they accommodate large rock formations. Frictional bolts can be divided into two types of bolts available on the market: Swellex Frictional Bolts and Split Set Frictional Bolts.

2.2.2.1. Swellex Frictional Bolts

Swellex bolts were developed by Atlas Copco AB. The Swellex rock bolting system has become standard in mines and tunnels all around the world.

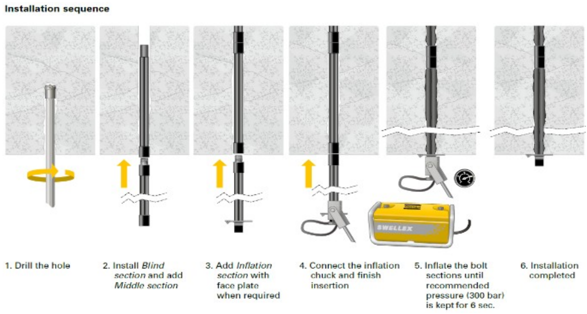


Fig. 3. Friction-anchored rock bolt and injection of Swellex (Atlas Copco, 2012).

2.2.2.2. Split Set Frictional Bolts

Split Set was developed by Scott in collaboration with the Ingersoll-Rand Company in the United States (Hoek and Wood, 1987). It is a bolt for temporary stabilization and consists of a high-strength steel tube slotted along its length and a plate.

2.2.3. Fully Grouted Rock bolts

The main characteristic of fully grouted bolts (dowels) is that they are bolts without any mechanical anchors. Usually, they consist of ribbed rebar installed in a borehole and bonded over its full length to the rock mass. Fully grouted rock bolts are commonly used in mining when stabilizing tunnels, roadways in mines, drifts, and shafts for the reinforcing of their peripheries. Compared to other rock bolts, the fully grouted rebar bolts have benefits such as simplicity in installation, relatively lower cost, and more versatility. It is important to install grouted bolts as soon as possible after excavation. The reason is that they are self-tensioning when the rock starts to move and dilate, so they must be installed before the deformation in the rock occurs and before the bolts lose their interlocking and shear strength. Fully grouted bolts are passive bolts, not activated in the installation phase (Kılıc, Yasar, and Celik, 2002).

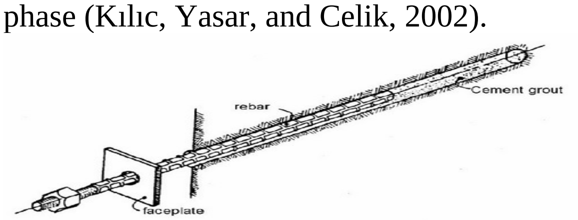


Fig. 4. Fully grouted rock bolt.

The bolt consists of three parts: a rod, a face plate, and a bonding (Hoek, 2012).

They can be divided into two groups:

- Cement-grouted rock bolts
- Resin-grouted rock bolts

2.2.3.1. Fully Cement-Grouted Rock Bolts:

Grouting with cement is the oldest method of full-column anchor bolting. This improved anchorage method works best in weak or fractured strata. The main disadvantage of fully cement-grouted bolts is the uncertainty about cement shrinkage and the longer setting time of the bolts, which limits their use in underground excavations where speed is not required (Peng and Tang, 1984).

2.2.3.2. Resin-grouted rock bolts:

Fully resin-grouted bolts are the most sophisticated rock bolt system currently used. It combines most of the advantages of other bolt systems. Resin and a catalyst are packaged in a plastic tube and separated from each other to prevent chemical interaction. These plastic capsules are then placed in the borehole with a loading stick before the bar is inserted. The bar is rotated into the hole, which breaks the plastic tube and mixes the resin and catalyst together (Hoek and Wood, 1987). The development of resin as a bonding material is a further improvement from the cement bonding agent. The major advantages of the cement agent are better anchorage over a wider range of strata types and shorter setting time than cement. However, the resin's high cost is the main disadvantage (Peng and Tang, 1984).

2.3. Breaker Line Support

In layman's language the breaker line is the line separating the goaf and the working and the supports needed to be provided in the breaker line are known as breaker line supports. More specifically, to prevent Goaf Encroachment/Overriding there is a requirement of applied supports against the immediate roof as a considerable portion of this roof is exposed due to the presence of different openings along the line of extraction. These applied supports act as a fulcrum at the goaf edge between the working area and the goaf to facilitate the breakout of the hanging roof inside the goaf. This is called breaker line support (Ram et al., 2016). In earlier days' wooden supports were used for this purpose, as they were primitive, they didn't provide the facility of quick setting and ease of installation. However, with the introduction of mechanized mining methods which involved not only faster and higher rates of extraction but also increased consideration for roof support, and thus wooden supports had to take the back seat and Mobile BLS came into the picture. Remote-operated mobile breaker lines, (Vuuren, 2002) were more efficient than the props but also had some disadvantages they are comparatively costlier and require more space and power which is a big problem in mines.

Fig. 5. A & B – Roof bolts acting as a caving breaker line in A and no breaker line at all in B, (Madden, 1989).

2.4. Pull-Out Test:

Pull-out testing does not measure the entire roof support system nor includes tests for pre-tensioned bolts or evaluation on the mine roof support system. Pull-out test is used in both laboratory and underground mining to calculate failure stress and capacity of roof bolts. Pull-out tests apply to mechanically, cement- or resin-grouted, or other similar anchor systems. Information gathered from pull-out tests may give a quantitative measure of the relative performance of anchor systems in the same rock type. The rock bolts and dowels tested were installed in percussion-drilled holes using the installation techniques that would be used in a normal underground mining operation. The appropriate use of rock bolts in underground mines represents a cost-benefit problem. Overdesign of ground support systems may lead to inefficiencies in labour and equipment usage, inflating costs, and cycle times. Under-design may result in a decrease in safety factors and production stoppages. An important step in the design and optimization of a ground support system is the selection of appropriate elements and systems for a set of conditions. The performance that may be expected from a particular element given a set of installation parameters and its associated variability should be recognised by those who design the support system. With improvements in the understanding, measurement, and quantification of performance, safer and more cost-effective ground support systems may be developed.

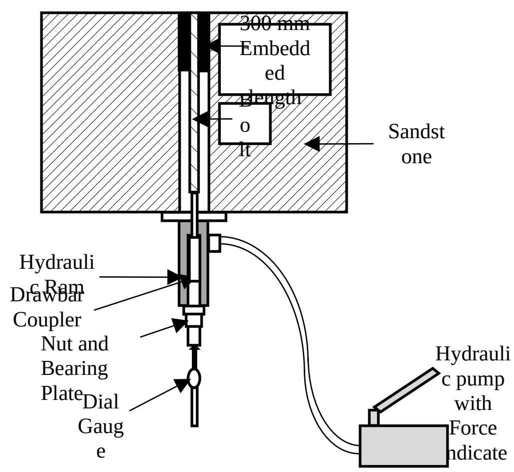


Fig. 6. Pull out test set up (Xiaowei et al, 2007)

To evaluate the rock bolt pull-out test several standards are in common use some of them are as follows (Singh et al, 2013):

Acceptable if anchor displacement of less than 3.18mm occurs at a 7.25-tonne load (Mark et al, 2000).

Acceptable if the bond stress at failure exceeds 5 MPa. Failure is considered to have occurred when the slope of the bond stress versus anchor displacement curve drops below 0.75 MPa per mm (British Coal, 1992).

Acceptable if the bolt can be loaded to the yield strength without sustaining unrecoverable deformation (Cullen, 1989).

In a pull-out test, to establish the bond capacity, it needs to have the bond fails below the steel yield load. For this, the resin encapsulation length was limited to between 0.22m and 0.35m; these tests are known as "short encapsulation" pull tests.

In India, it is common industry practice to install 1.5m long (minimum), 20mm diameter bolts into 27mm diameter holes, with full resin encapsulation (Singh et al, 2013).

Hence after validation of the model, we will use the Indian geo-mining condition to perform the pullout test and by varying important factors of the test, we will try to draw some conclusions that will be useful for the designing of the support system. The factors that we will try to vary will include resin encapsulation length, hole diameter and bolt diameter, and length and diameter of the bolt. We will increase the resin encapsulation length from 200mm to 1500mm with an enhancement of 100mm. We will change the hole diameter and bolt diameter from 20mm and 27mm respectively to 22mm and 35mm. According to Raghavulu et al, 2010 (National Seminar on Underground Coal Mining titled “The Future of Overground Lies Belowground” held on 28th August 2010, Hyderabad) roof bolts of 1.5mx20mm, 1.5mx22mm, 1.8mx20mm, 1.8mx22mm, 2.4mx20mm, 2.4mx22mm are commonly used in India hence we will vary the length & diameter of the bolt in the six stated ways.

3. NUMERICAL SIMULATION

It was an attempt to simulate the pull-out test, thus, this section is devoted to the simulation method that we used. Some important numerical methods are (a) Finite Element Method (FEM) (b) Finite Differential Method (FDM) (c) Boundary Element Method (BEM). Among these, we had selected FEM to do the simulation. The finite element method (FEM)also referred to as finite element analysis (FEA) is a numerical method for solving problems of engineering and mathematical physics. Typical problem areas of interest include structural analysis, heat transfer, fluid flow, mass transport, and electromagnetic potential. The analytical solution of these problems generally requires the solution to boundary value problems for partial differential equations. The finite element method formulation of the problem results in a system of algebraic equations. The method yields approximate values of the unknowns at a discrete number of points over the domain. To solve the problem, it subdivides a large problem into smaller, simpler parts that are called finite elements. The simple equations that model these finite elements are then assembled into a larger system of equations that models the entire problem. FEM then uses various methods from the calculus of variations to approximate a solution by minimizing an associated error function.

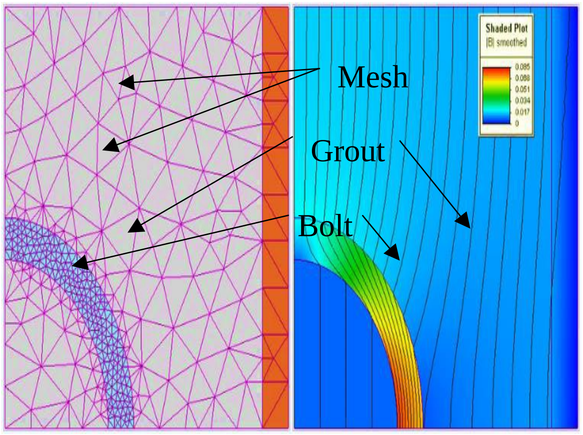


Fig. 7. Example model for numerical analysis using Finite element method.

The finite element method (FEM) involves a series of mathematical computational procedures to calculate the load distribution in each element. Such a structural analysis allows the determination of stress resulting from external force, pressure, thermal change, and other factors (Desai et al, 2011). This method is extremely useful for indicating mechanical aspects of biomaterials and human tissues that can hardly be measured in vivo. The results obtained can then be studied using visualization software within the FEM environment to view a variety of parameters and to fully identify the implications of the analysis. The steps involved in FEA are Pre-processing, Conversion of a geometric model into a finite element model, Assembly/Material Property data representation, Defining the boundary conditions, Loading Configuration, Processing, and Post-processing.

The reason why we selected FEM/FEA for our analysis is because of the advantages it possesses, which include:

1) Modelling of complex geometry and irregular shape is easier as different types of finite elements are available for the discretization domain.
2) Boundary conditions can easily be implemented.
3) Different types of material properties can be easily assigned from element to element with discretization.
4) Problem with heterogeneity, time dependency, anisotropy
5) The systematic generality of FEM makes it a versatile tool for the solution of a variety of problems.
6) Simple, compatible, and result oriented.
7) Easily coupled with a Computer-Aided Design program.
8) Physical interpretation and simulation are easy.
9) Applicable for high-level elements i.e., refined mesh.

Along with these advantages it also had some disadvantages:

1) No advantage of flexibility and generalization.
2) Large amount of data is required as input for perfect interpretation and simulation of the problems.
3) A good understanding of physical problems and experience is required for perfect simulation and analysis.
4) Voluminous output data may be analyzed and interpreted.

As our motive was to analyze only a single bolt, not the whole pillar for designing these disadvantages do not affect us but up to a certain extent. As stated earlier we used FEM to simulate the whole pullout test and for that, we are using ANSYS software. ANSYS Workbench, which is used to perform various types of structural, thermal, fluid, and electromagnetic analyses is used. The entire simulation process is tied together by a project schematic; from which we can interact with applications that are native to ANSYS Workbench or launch applications that are data-integrated with ANSYS Workbench. ANSYS Workbench includes bi-directional CAD connectivity, highly automated meshing, a project-level update mechanism, pervasive parameter management, and integrated optimization tools (ANSYS Finite element software. ANSYS. Inc., Canonsburg (PA, USA).

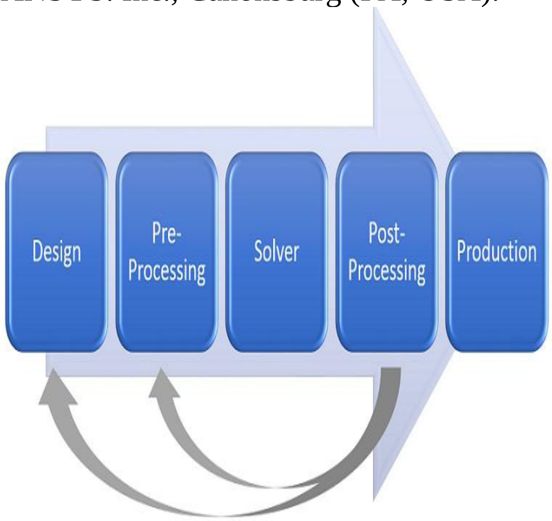


Fig. 8. Workflow algorithm of simulations using ANSYS

3.1. Designing Methodology

Previously, advanced numerical modelling methods of rock bolt performance in underground mines have been presented. Studies show how numerical modelling methods could be successfully used to optimize the load transfer between the bolt and the surrounding strata (Ghadimi et al, 2015). The study indicated that the standard rock bolt reinforcing elements which are commonly used in the numerical simulation of the supported underground excavations cannot be used to optimize the load transfer capabilities of the bolt.

A detailed model of the bolt profile must be constructed, loaded to failure, and compared with other profiles to find the optimum bolt profile with maximum load transfer capabilities between the bolt and host strata. Relations can be established, and equations are used to calculate the pull-out force needed to fail the grout for different bolt profile configurations. The calculations can be applied to any plane of probable failure within the grout.

The important outcome of this study, shown in Fig. 10. is to show that there may be another way to examine grout failure around the bolt for different profile configurations that can be compared with laboratory tests and numerical modeling.

This method could provide a better understanding of the bolt-grout interaction with rock reinforcement.

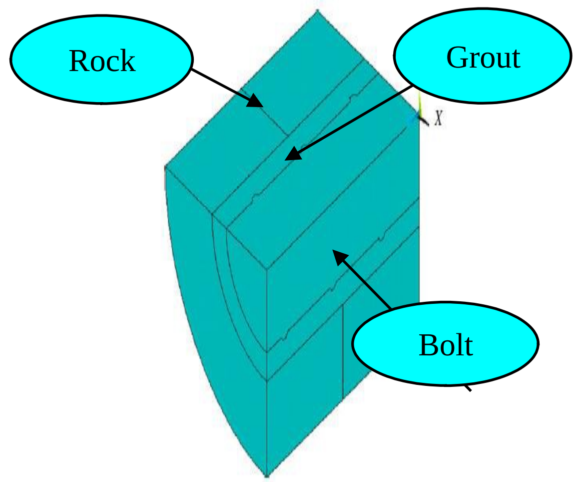
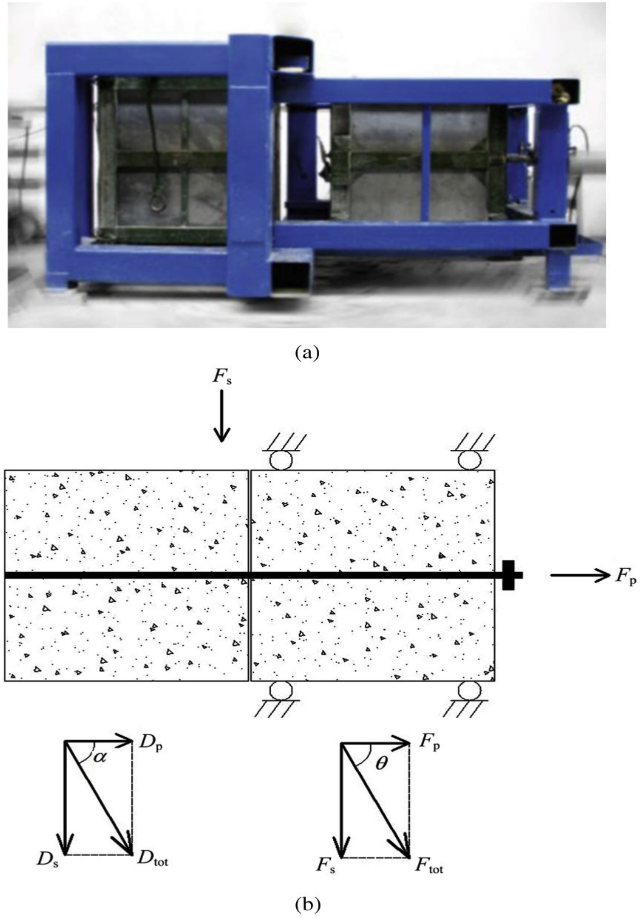


Fig. 9. Three-dimensional image numerical model using ANSYS (Scott et al, 1976).

A three-dimensional finite element model of the reinforced structure subjected to the tension loading can be used to examine the behavior of bolted rock joints and validate results obtained from instrumentation. Two governing materials (steel and resin grout) with an interface (bolt-grout) are to be considered for the numerical simulation.

A general-purpose finite element program ANSYS, specifically for advanced structural analysis, can be used for 3D simulation of elastic-plastic materials and contact interface behavior.

More interactions like grout-rock and joint-joint can be added and analyzed further for a given geo-mechanical condition (Scott et al, 1976).



The whole model can be understood easily if two perpendicular sections are taken, because of symmetry it will reduce the bulkiness of the problem without affecting its property or physical behavior.

The interface behavior of grout concreted as a perfect contact can be determined from the

Fig. 10. SINTEF/NTNU bolt test rig.

(a) Full-scale test rig in the laboratory.

(b) Top-view sketch

test results. However, the low value of cohesion is adopted for grout-steel contact.

The materials tolerate shapes without significant loss in accuracy (Scott et al, 1976). The 3D surface-to-surface contact elements can be used to represent the contact between the 3D target surface (steel grout and rock grout).

This element is applicable to 3D structural contact analysis and is located on the surface of 3D solid elements with midsize nodes.

The numerical modeling can be carried out in several sub-steps and the middle block of the model was gradually loaded in the direction of shear.

3.2. Geometry

To design the geometry and test condition was of our main importance, it should match with the real test conditions. A new test method was developed to simulate the pull-and-shear loading condition on the bolt (Fig 10), (Chen, 2014). Fig. 12. shows the

Fig. 12. Rock Block for test (After Ghadimi)

Block

Fig. 11. Rock Block

constraints on the rock block to put during the pull-out test. With the help of this and Tables 1, 2, and 3 we constructed the geometric model.

The model in ANSYS consists of a block with a hole then the bolt was to be inserted in the hole due to difference in diameter an annulus was created and which is to be filled with the help of grouting material. Fig. 12 shows the geometry that was used for the simulation, and a detailed look at the components and meshing is shown in Fig. 14 and Fig. 15. For simulation the constraints are important and hence four fixed supports are being used for four sides as shown in Fig. 11.

As per standard norms we will start with 10KN and maintaining a difference of 10KN we will increase the force to 200KN.

At each interval, we must record the deformation of the system. Using ANSYS and Ghadimi et al, 2015 we obtained Table 4.

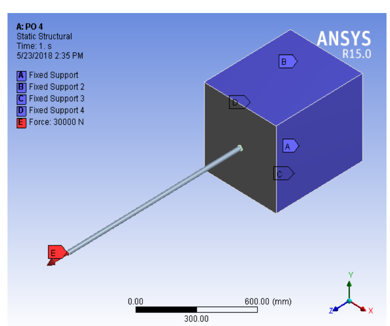


Fig. 12. Constraints with pull-out force

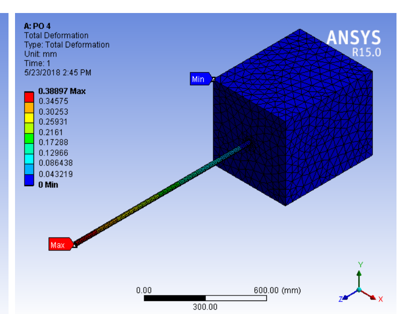


Fig. 13. Deformation with colour map

Grout

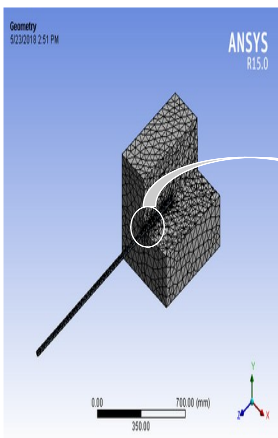


Fig. 14. Cross-sectional View of Geometry

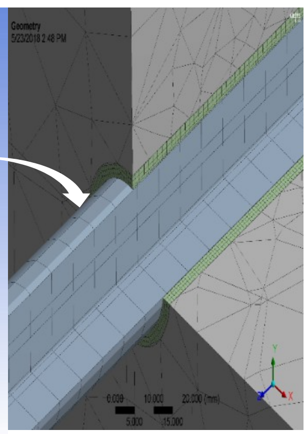


Fig. 15. Zoomed view of cross-section

4. VERIFICATION & ANALYSIS
4.1 Verification

After modeling the geometry and putting the constraints (using Tables 1 and 2) the only task left is analysis for that we will pull the bolt as shown in Fig 16.

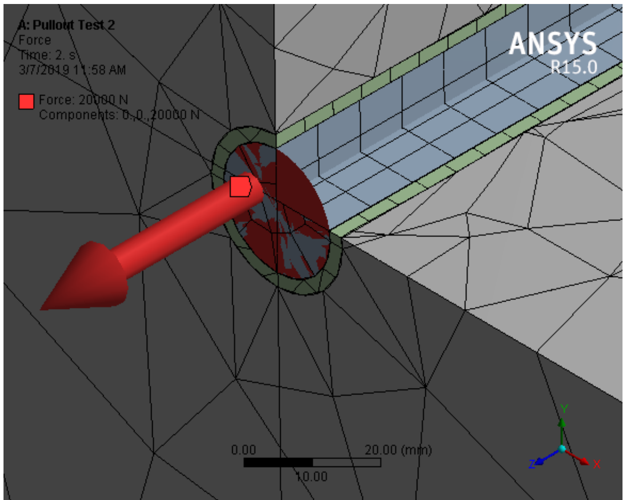


Fig. 16. Application of pull-out force

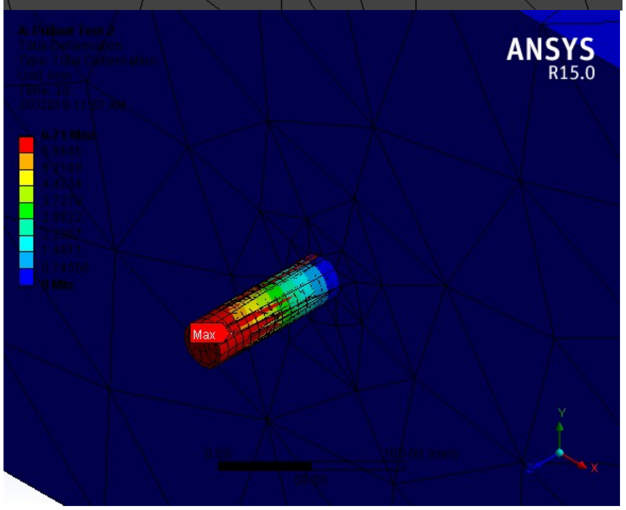


Fig. 17. Deformation

We used the work of Ghadimi for our analysis and included their parameters as well. We found that the ANSYS model yields in a similar way to that of the actual test.

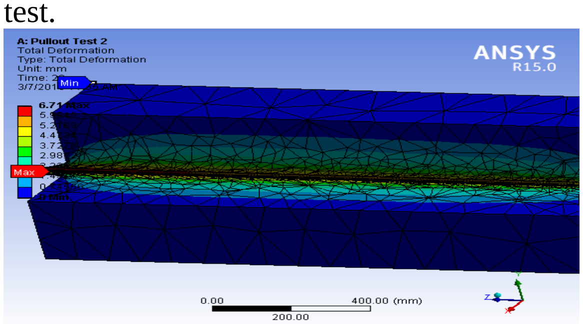


Fig. 18. Total Deformation in the bolt

In Fig. 17 and Fig. 18, we can observe the deformation in the bolt. Here we can figure out that the failure occurs between the grout and the rock.

Table 1. Parameters for Block

Parameters	**Values**
Block type	Sandstone Cube
Cube Dimension	1.5m*0.5m*0.5m
Hole Length	1.5m
Hole Diameter	27mm
Density	2500 kg/m^3
Young's Modulus	20GPa
Po	
Bulk Modulus	1.3333E+10 Pa
Shear Modulus	8E+9 Pa

Table 2. Parameters for Grout

Parameters	**Values**
Grout type	Resign Grout
Grouting Length	1.5m
Thickness	2.5mm
Density	1650 kg/m^3
Young's Modulus	12GPa
Poisson's Ratio	0.2
Bulk Modulus	6.6667E+09 Pa
Shear Modulus	5E+09 Pa

The geometry obtained after putting all the parameters in ANSYS considering Mohr-Coulomb's elasto-plastic rock failure model with a non-associated flow rule. After modeling we performed the same test. A sharp increase in the magnitude of displacement is an indicator of the plastic movement of the grout against the rock. By contrast, if the displacing speed converges toward zero or the vertical displacement settles down to a certain magnitude, the immediate roof of the opening then reaches its stable state.

Using the information in the table the graph is plotted and then we found that the green bold curve represents the numerical model and can be suitably used for studying

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various variations in the parameters and modeling conditions.

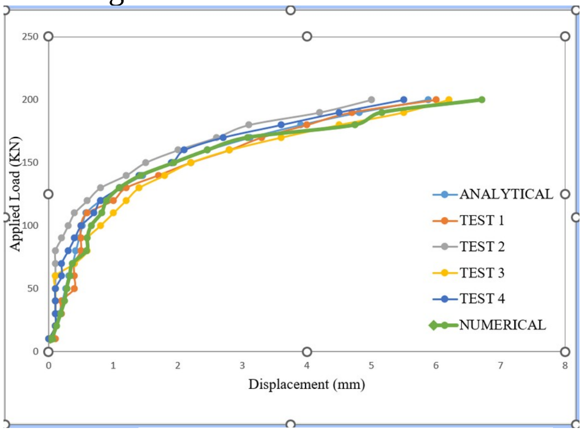

Fig. 19. Graph between applied load and deformation

By merely looking at the graph we can assure that the bond strength would be 190-200kN. After the analysis, we can conclude that the simulated value is close to the field and analytical observation with tolerable errors in precision and accuracy.

4.2 Analysis

The process of learning involves experimentation and experimentation consists of three stages CHECK, EXPERIENCE, and EXPERIMENT. Firstly, we checked that the process of the pull-out test was a time-consuming and laborious process, and then we experienced during the verification stage that the test results could be obtained using numerical methods with tolerable errors in precision and accuracy. Thus, using the above numerical method, we can suitably analyze the performance of the roof bolt with different parameters. Hence our experiment would consist of four different sets of analysis models, which will try to demonstrate the important variations in the performance of the roof bolt. These sets of models will be used to draw suitable graphs and then deduce conclusions. For the numerical modeling of the following models, the fundamental concept of geometry and engineering properties in ANSYS remained the same only the dimensions of the components were changed. The various plots for analysis of the roof bolt are as follows:

Analysis 1: Plot between models by changing grout length from 200mm to 1500mm with an increment of 100mm.

Analysis 2: Plot between models of the bolt diameter and hole diameter 20mm and 27mm respectively compared to 22mm and 35mm respectively at a fixed grout length of 300mm.

Analysis 3: Plot between models of different bolt diameters (i.e., 20mm & 22mm) and fixed bolt lengths i.e., 1.5m or 1.8m or 2.4m.

Analysis 4: Plot between models of different bolt lengths (i.e., 1.5m & 1.8m & 2.4m) and fixed diameters i.e., 20mm or 22mm.

ANALYSIS 1:

Our objective in this analysis is to find the relation between the breaking strength of the roof bolt and the grouting length. In other words, we will observe the trends of deformations by changing the grouting length. In general, for Short Encapsulation Pull-out Test (SEPT) the grouting length of 300mm is being used. Hence, we started with 200mm and increased the grouting length by 100mm till 1500mm (i.e., the fully grouted bolt) and all the other parameters remained constant.

After modeling and performing the test as described earlier the results for deformation at each load stage were noted as in table 10 & 11 and then the plot was drawn.

Table 4: Parameters for Block

Parameters	**Values**
Block type	Sandstone Cube
Cube Dimension	1.5m*0.5m*0.5m
Hole Length	1.5m
Hole Diameter	27mm
Density	2500 kg/m^3
Young's Modulus	4GPa
UCS	22.59MPa
Tensile Strength	3MPa
Poisson's Ratio	0.41
Bulk Modulus	7.41E+09 Pa
Shear Modulus	1.42E+09 Pa

Table 6: Parameters for Bolt

Parameters	**Values**
Bolt type	Mild Steel Rod
Length	1.5m
Diameter	22mm
Density	7570 kg/m^3
Young's Modulus	200GPa
Poisson's Ratio	0.3
Bulk Modulus	1.67E+11 Pa
Shear Modulus	7.69E+10 Pa

Table 3: Parameters for Grout

Parameters	**Values**
Grout type	Resign Grout
Grouting Length	200-1500mm
Thickness	3.5mm
Density	1650 kg/m^3
Young's Modulus	12GPa
Poisson's Ratio	0.2
Bulk Modulus	6.6667E+09 Pa
Shear Modulus	5E+09 Pa

Table 5: Additional Parameters

Parameters	**Values**
μ(rock-grout)	0.6494
μ(grout-bolt)	0.3639
Pretension	29400N

Table 7. Load vs deformation for different grout length (Part 2)

Table 11. Load Vs Deformation for different grout length (Part-2)

Applied force (KN) Grout Length (mm)→	Test 8 (mm) 900	Test 9 (mm) 1000	Test 10 (mm) 1100	Test 11 (mm) 1200	Test 12 (mm) 1300	Test 13 (mm) 1400	Test 14 (mm) 1500
10	0.34	0.34	0.34	0.33	0.3	0.31	0.3
20	0.65	0.65	0.66	0.64	0.59	0.6	0.59
30	0.94	0.94	0.95	0.93	0.86	0.87	0.85
40	1.2	1.2	1.21	1.18	1.1	1.12	1.1
50	2.12	1.43	1.43	1.41	1.32	1.34	1.33
60	2.41	2.52	2.53	2.33	1.51	1.53	1.52
70	2.74	2.78	2.84	2.64	2.6	2.62	1.69
80	3.03	3.12	3.19	2.97	2.87	2.93	2.87
90	3.28	3.39	3.49	3.27	3.19	3.27	3.16
100	3.47	3.63	3.74	3.55	3.48	3.59	3.49
110	3.59	3.81	3.94	3.78	3.74	3.9	3.8
120	5.05	3.9	4.05	3.95	3.97	4.18	4.09
130	4.86	4.42	4.08	4.02	4.16	4.42	4.36
140	5.04	5.92	5.98	4.52	4.28	4.61	4.59
150	6.8	6.77	5.39	5.72	4.32	4.71	4.78
160	6.94	6.99	6.07	5.62	4.35	4.69	4.89
170	7.37	7.38	7.39	5.9	6.28	4.56	4.91
180	8.83	7.69	7.66	6.38	5.91	6.53	4.83
190	9.01	7.9	8.07	7.85	6.48	6.54	6.67
200	10.04	9.01	8.44	8.15	6.42	6.93	6.71

By plotting the load vs displacement curve for various lengths of grout we obtained Fig. 20 from this we can infer that initially when the grouting length was 200mm the breaking strength was about 110KN and when the length was increased to 300mm the breaking strength was found to be about 150KN. But when the length was further increased to 400mm the breaking strength decreased to 130KN and it remained almost constant for grouting length 500mm to 700mm. Now as we had increased the length of the grout to 800mm the breaking strength decreased to 115KN and then it remained constant for the length of 900mm to 1100mm at 120KN but further we observed from the plot that at 1200mm length the strength increased to 175KN but soon at 1300mm the strength decreased to 160KN and again increased at 1400mm to 190KN and remained almost same for the length of 1500mm. Hence now we can infer something about the SEPT and why the length 300mm is preferred.

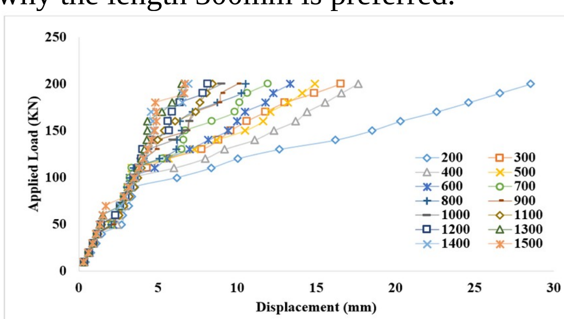

Fig. 20. Graph between applied load and deformation

However, the plot above doesn't directly infer that 300mm is the best length for testing purposes thus to demonstrate it well we tried to plot the deformations of various grouting lengths vs the length of grout at a particular load. For now, we have taken 10KN as the reference load as it is the initial load and plotted the graph as shown in Fig 21.

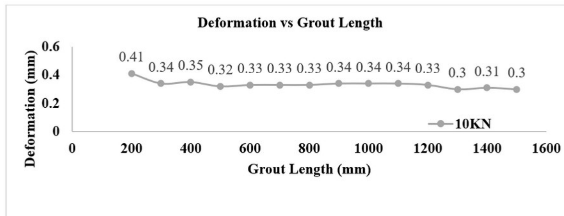

Fig. 21. Graph between deformation and Grout Length at 10KN

From this plot, it is clear that the deformation remained almost the same when we varied the grouting length from 300mm to 1200mm at 10 KN of applied load. After this, we checked the same trend at a higher load of 100KN and plotted the graph as shown in Fig 22. We have again observed the plot of deformation vs grout length giving the same result but this time the results are easily visible.

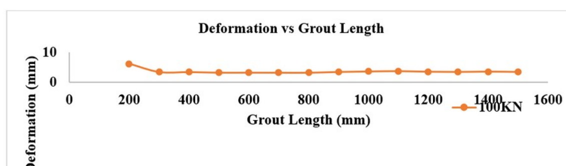

Fig. 22. Graph between Deformation and Grout Length at 100KN

From this plot, it is also made known that the deformation not only remains the same from 300mm to 800mm but also there is a rise in the deformation at 900mm to 1100mm grouting length and again remains almost the same from 1200mm to 1500mm grouting length as it was observed in case of breaking strength. Hence we can conclude that the breaking strength of the roof bolt at a particular grouting length is inversely proportional to the deformation of the bolt at that grouting length for any fixed load less than the breaking strength. Along with this, we can also conclude that the 300mm grouting length is most preferable for SEPT as at this length the deformation is minimal and after this length, the deformation remains almost constant hence can give suitable information about the performance of the bolt in real geo-mining condition.

ANALYSIS 2:

Plot between models of different bolt diameter and hole diameter, (i.e. 22mm & 27mm respectively compared to 20mm & 27mm respectively and 22mm & 35mm respectively) at a fixed grout length of 300mm. The various models used for analysis are mentioned below.

Table 8: Models with various thickness

Table 12. Models with various thickness.

Model	Bolt Diameter (mm)	Hole Diameter (mm)	Thickness (mm)
Model 1	22	27	2.5
Model 2	20	27	3.5
Model 3	22	35	6.5
Model 4*	20	35	7.5
Model 5*	22	25	1.5
Model 6*	22	26	2.0

*These are hypothetical models and are not used commonly, shown only for demonstration.

Model 1 is commonly used for bolting purposes. In model 2 we tried to increase the thickness of the grouting material by decreasing the bolt diameter, further, in model 3, we tried to increase the grouting thickness by increasing the hole thickness and keeping the bolt the same. To verify the trend in model 4 further, we increased the thickness by reducing the bolt diameter. Also to verify the performance of the roof bolt at less thickness of the grouting material we designed models 5 and 6 keeping the thickness 1.5 and 2.0 respectively this has been achieved by changing the hole diameter. After modeling and performing the test as described earlier the results for deformation at each load stage were noted in Table 13 and then the plot was drawn.

Table 9. Load vs Deformation for different models.

Table 13. Load vs Deformation for different models.

Applied force (KN)	Deformation (mm)					
	Model 1	Model 2	Model 3	Model 4	Model 5	Model 6
10	0.34	0.36	0.35	0.39	0.46	0.39
20	0.65	0.68	0.67	0.75	0.84	0.73
30	0.93	0.97	0.96	1.08	1.63	1.02
40	1.19	1.24	1.22	1.37	1.94	2.11
50	2.12	1.48	1.45	1.63	2.26	2.27
60	2.49	2.25	1.66	1.85	3.49	2.61
70	2.78	3.54	1.84	2.02	4.67	2.8
80	3.05	3.65	2	2.14	5.72	3.15
90	3.28	4.02	2.13	2.25	7.1	5.11
100	3.46	4.24	2.25	2.36	8.14	7.1
110	4.31	4.44	2.35	2.45	10.11	8.3
120	5.55	4.57	4.93	2.52	12.39	10.42
130	7.78	4.92	8.35	6.44	13.9	12.52
140	8.83	8.32	9.5	9.16	15.18	14.23
150	9.59	10.46	11.52	10.82	16.53	15.66
160	10.61	11.64	12.07	13.42	17.77	16.87
170	11.8	13.46	12.57	13.92	19.23	18.19
180	13.03	14.28	14.43	14.55	20.57	19.56
190	14.88	15.4	16.4	16.64	22	20.91
200	16.53	17.11	17.52	18.18	23.35	22.4

Hence the order for the thickness is 5<6<1<2<3<4. Now if we observe model 5 its breaking strength is nearly 60KN and by increasing the thickness in model 6 the breaking strength increases to 85KN, which is proportional to the previous model (Refer Fig 23). Now in model 1, (i.e., the most commonly used), we had increased the thickness and the breaking strength increased drastically to 130KN. In model 2 the increased thickness is being achieved by reducing the bolt diameter and we found that the breaking strength increased to 140KN. Model 3 the increase in thickness is large but the increase in breaking strength is not that high, and from the graph, we can observe that the strength is about 135KN. The trend continues in model 4 as well by increasing the thickness to 7.5mm the strength only increased to 140KN.

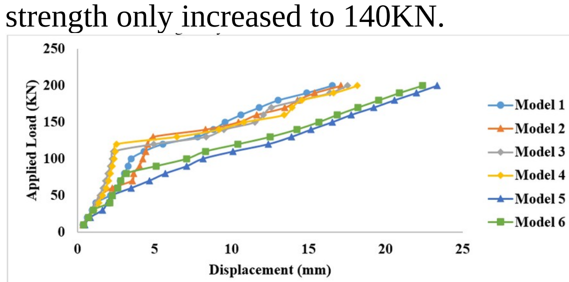

Fig. 23. Graph between Applied Load and Displacement of different models.

To visualize the trend properly we tried to plot the deformations of various models at a particular load. For now, we have taken 50KN and 100KN as reference loads and plotted the graph as shown in Fig 24 and Fig 25.

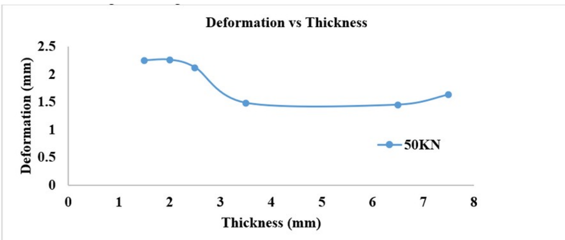

Fig. 24. Graph between Deformation and Thickness at 50KN

Here we can observe that initially, the deformation is high but as the thickness crosses the 3mm mark the deformation becomes almost constant, and again at 7.5mm thickness the deformation increases. Thus at a load of 50KN, the thickness should be 2.5mm to 3mm of the grouting material for minimum deformation and maximum breaking strength. To verify it again we considered the load stage at 100KN as shown in Fig 25. At this stage, it is clear that the minimum deformation occurs at a thickness of 2.5mm and hence it is a preferable choice to select the 22mm diameter bolt with a 27mm hole to get a better performance of the roof bolt. Along with this, the performance of 6.5mm and 7.5mm thickness grouting has also improved at a higher load as compared to a lower load which is genuine.

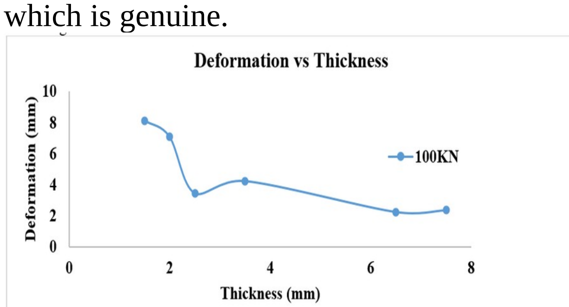

Fig. 25. Graph between Deformation and Thickness at 100KN

Hence we can safely conclude that by merely increasing the thickness the bond strength is not going to increase. Also, the breaking strength is not proportional to the thickness of the grouting material. If we compare the three commonly used models in the mines (as shown in Table 12), we found that model 1 is performing better than 2 and 3, but the performance of model 3 is enhanced at a higher load. We can also observe that model 2 is a poor choice, as in spite of increasing the thickness of the grouting material the breaking strength does not increase as can be visualized from its trends in deformation.

ANALYSIS 3:

Plot between models of different bolt diameters (i.e. 20mm & 22mm) and fixed bolt length i.e. 1.5m or 1.8m or 2.4m and all other parameters are same, with the help of previous model designing technique we had designed and performed the test.

Table 10: Load vs deformation at different bolt diameters for fixed bolt lengths

Table 14. Load vs Deformation at different bolt diameter for fixed bolt lengths.

Applied Force (KN)	**Deformation (mm)**					
Length (mm)→	***1500***		***1800***		***2400***	
Diameter (mm)→	**20**	**22**	**20**	**22**	**20**	**22**
10	0.57	0.51	0.64	0.57	0.78	0.68
20	1.09	0.97	1.24	1.09	1.5	1.3
30	1.59	1.41	1.8	1.58	2.18	1.89
40	2.05	1.82	2.32	2.04	2.81	2.45
50	2.48	3.3	2.8	3.68	3.4	4.37
60	5.06	3.52	5.7	3.93	6.32	4.67
70	5.37	4.03	5.94	4.5	6.96	5.35
80	5.72	4.4	6.45	4.91	7.72	5.84
90	6.07	4.77	6.83	5.31	8.31	6.32
100	6.47	7.41	7.27	8.68	8.74	6.74
110	6.75	7.79	7.57	8.81	9.14	10.76
120	7.04	11.31	7.89	12.51	9.5	14.64
130	21.03	13.29	25.54	14.61	37.33	17.66
140	17.47	16.63	19.29	18.08	45.34	21.21
150	18.62	17.97	19.96	18.96	28.43	21.41
160	21.16	19.34	22.84	20.18	27.71	21.83
170	21.13	20.36	22.21	21.17	25.93	22.81
180	22.47	21.86	23.59	22.74	25.95	24.5
190	23.73	23.22	24.69	24.01	26.74	25.87
200	25.34	24.8	26.49	25.7	28.86	27.2

After performing the test, the data obtained was been noted in Table 14. From Table 14, we can plot the graphs between two different bolt diameters i.e., 20mm and 22mm in three different graphs with different bolt lengths i.e., 1.5m, 1.8m, 2.4m as shown in Fig 26, 27 & 28 respectively.

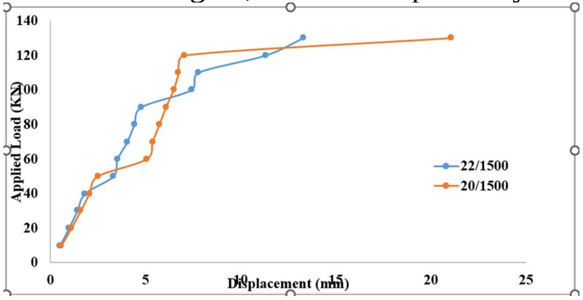

Fig. 26. Graph between Applied Load and Displacement of different bolt diameters with a fixed length of 1500mm

Note: 22/1500 represents the model has a bolt of diameter 22mm and a length of 1500mm. This representation will further be used in all the graphs of ANALYSIS 3 and 4. In all these models the hole diameter is fixed at 27mm and the grouting length is fixed at 300mm, hence the thickness of the grouting material of a 20mm diameter bolt will be more than that of a 22mm diameter bolt (i.e., 3.5mm >.2.5mm).

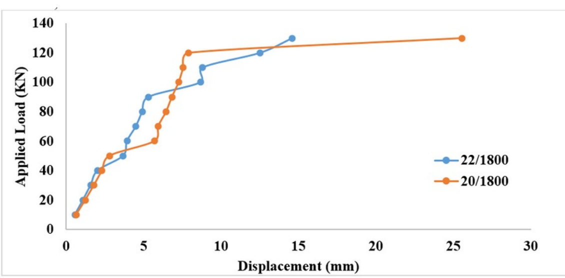

Fig. 27. Graph between Applied Load and Displacement of different bolt diameters with fixed length of 1800mm

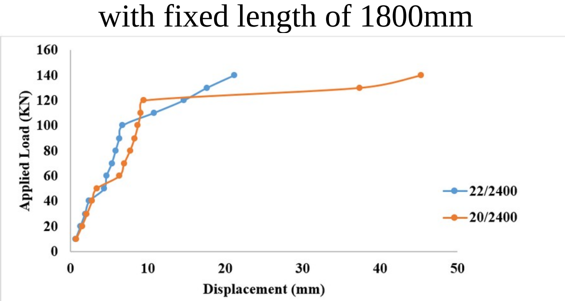

Fig. 28. Graph between Applied Load and Displacement of different bolt diameters with a fixed length of 2400mm

Table 11. Breaking strength of various models

Sl. No.	Parameters of the bolt		Breaking Strength (KN)
	Length (mm)	Diameter (mm)	
1	1500	20	130
2		22	120
3	1800	20	130
4		22	130
5	2400	20	135
6		22	140

As in these models, the hole diameter is fixed at 27mm and the grouting length is fixed at 300mm, hence the thickness of the grouting material of a 20mm diameter bolt will be more than that of a 22mm diameter bolt (i.e., 3.5mm >.2.5mm). Hence the trends in the breaking strength are normal but, if we look at the graph more closely, the deformation trends have a different story to tell, we come to know that the deformation of a 22mm diameter bolt is less than that of 20mm diameter bolt within the range of 100KN load. Hence we can say that the bolt of diameter 22mm has a better performance in spite of having a lower thickness than that of a bolt of 20mm diameter at all lengths. This variation can be understood by considering the contact between the bolt and the rock, as the thickness decreases the distance between the bolt and the rock decreases and thus frictional force arises between them in the initial stage of loading. Later the effect gets reduced as the load increases and trends to the breaking strength of the grout.

ANALYSIS 4:

Here we will try to understand the effect of the bolt length by plotting a graph between models of different bolt lengths (i.e. 1.5m & 1.8m and 2.4m) with fixed diameters i.e. 20mm or 22mm.

Table 12: Load vs deformation at different bolt lengths for fixed bolt diameters

Table 16. Load vs Deformation at different bolt length for fixed bolt diameters

Applied Force (KN)	**Deformation (mm)**					
Diameter (mm)→	***20***			***22***		
Length (mm)→	**1500**	**1800**	**2400**	**1500**	**1800**	**2400**
10	0.57	0.64	0.78	0.51	0.57	0.68
20	1.09	1.24	1.5	0.97	1.09	1.3
30	1.59	1.8	2.18	1.41	1.58	1.89
40	2.05	2.32	2.81	1.82	2.04	2.45
50	2.48	2.8	3.4	3.3	3.68	4.37
60	5.06	5.7	6.32	3.52	3.93	4.67
70	5.37	5.94	6.96	4.03	4.5	5.35
80	5.72	6.45	7.72	4.4	4.91	5.84
90	6.07	6.83	8.31	4.77	5.31	6.32
100	6.47	7.27	8.74	7.41	8.68	6.74
110	6.75	7.57	9.14	7.79	8.81	10.76
120	7.04	7.89	9.5	11.31	12.51	14.64
130	21.03	25.54	37.33	13.29	14.61	17.66
140	17.47	19.29	45.34	16.63	18.08	21.21
150	18.62	19.96	28.43	17.97	18.96	21.41
160	21.16	22.84	27.71	19.34	20.18	21.83
170	21.13	22.21	25.93	20.36	21.17	22.81
180	22.47	23.59	25.95	21.86	22.74	24.5
190	23.73	24.69	26.74	23.22	24.01	25.87
200	25.34	26.49	28.86	24.8	25.7	27.2

It is to be noted that the grouting length is 300mm and all other parameters are the same, as earlier taking help from the previous model designing method we designed and performed the test, and the data obtained was been noted in Table 16 (12). From Table 16, we can plot the graphs between the deformation of three different bolt lengths i.e., 1.5m, 1.8m, and 2.4m in two different graphs with different bolt diameters i.e., 20mm & 22mm as shown in Fig 29 and 30 respectively.

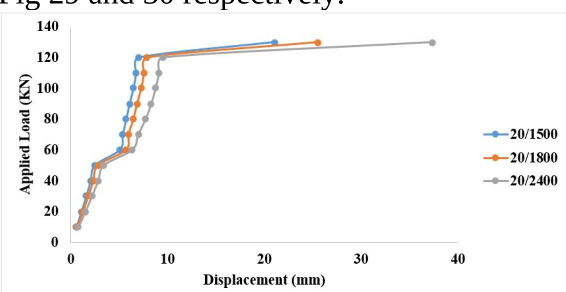

Fig. 29. Graph between Applied Load and Displacement of different bolt lengths with a fixed diameter of 20mm

It is clear from the above graph that the load vs displacement for different lengths are almost similar to each other and shows a breaking strength of 130-135KN. As the thickness is more there is no or very little contact effect of bolt and rock, hence we are getting similar trends.

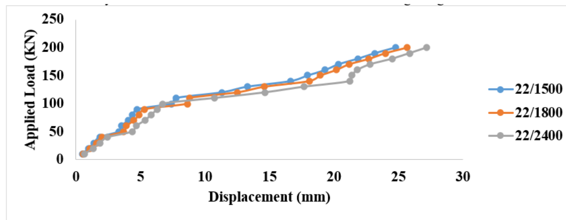

Fig. 30. Graph between Applied Load and Displacement of different bolt lengths with a fixed diameter of 22mm

In Fig 30 we observe that the load vs displacement for different lengths are not similar as it was in the case of 20mm diameter. It is also to be noted that the breaking strength increases as the length increases, but overall it lies in the range of 120- 130KN. In both of the graphs (Fig 29 & 30) it was observed that the deformation was least in the 1.5m bolt moderate in the 1.8m bolt and maximum in the 2.4m bolt, this can be explained with the help of the concept of torque. As the length increases the distance between the grouting length and the point where the force is being applied is increasing and hence the deformation is increasing, but it is also important to note that the effect is very less and shall not affect the choice. Hence the length of the bolt must be selected as suitable for the geo-mining and working condition of the area.

5. CONCLUSION

Generally, the performance of the rock bolts is evaluated using the pull-out tests. Many researchers in the past have performed laboratory pull-out tests to evaluate the various influencing parameters in the design of rock bolts. One cannot always depend on experimental and field studies for the design of the rock bolt system as these studies are often time-consuming, cumbersome, and involve many errors. In order to overcome these drawbacks, many researchers in the past have adopted analytical approaches to study the behavior of rock bolts. However, these analytic formulations are case-specific and are not suitable for generalized applications. Hence there is a need for numerical simulation which will not only save time and effort but also can be generalized to any field data, any bolt or grout parameter. This paper aimed to answer whether the numerical modeling technique (ANSYS) be used for conducting pull-out tests or not. After establishing the validity of the model, various analyses were performed to find out the influencing parameters in the components of roof bolts. The following were deduced from our investigation:

The breaking strength of the roof bolt at a particular grouting length is inversely proportional to the deformation of the bolt at that grouting length for any fixed load less than the breaking strength.

The 300mm grouting length is most preferable for SEPT as at this length the deformation is minimal and even after increasing the grouting length the deformation remains almost constant for a particular load, hence can give suitable information about the performance of the bolt in real geo-mining condition.

Merely increasing the thickness of the grouting material the bond strength is not going to increase. Also, the breaking strength is not proportional to the thickness of the grouting material.

If compared to the three commonly used models in the mines, it was found that model 1 is performing better than 2 & 3, but the performance of model 3 is enhanced at higher load. It can also be observed that model 2 is a poor choice, as in spite of increasing the thickness of the grouting material the breaking strength does not increase as can be visualized from its trends in deformation.

Hence the bolt diameter 22mm and hole diameter 27mm is a good combination that incorporates all the contact, between the grout and the rock, the grout and the bolt, and the bolt and the rock.

The bolt of diameter 22mm has a better performance in spite of having a lower thickness than that of a bolt of 20mm diameter at all the given lengths (i.e., 1.5m; 1.8m; 2.4m).

There is little effect of length on the performance of the roof bolt in general. Hence the length of the bolt must be selected as suitable for the geo-mining and working condition of the area.

In the end, it could be concluded that the model and analysis that had been developed during this study can be further evolved in various Indian geo-mining conditions, and can be used to design an optimum roof bolt system which will not only increase the safety measure or decrease the investment in roof-bolt but will also promote the scientific mining in India.

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