International Journal of Advance Research in Science and Engineering
Volume No.07, Special Issue No. (02), January 2018
www.ijarse.com
IJARSE
ISSN: 2319-8354

DESIGN PRINCIPLE OF ROCKBOLTING

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ABSTRACT

This article presents the standards of underground rockbolting. This paper discussed about underground loading conditions, natural pressure zone around an underground opening, selection of rockbolt types, determination of bolt length and spacing, factor of safety, and compatibility between support elements. Different types of rockbolting used in engineering practice are also mentioned. The general principle of selecting strong rockbolts is valid only in conditions of low bearing capacity of rocks. A natural pressure arch is formed in the rock at a certain distance behind the tunnel wall. Rockbolts should be long enough to reach the natural pressure arch when the failure zone is small. The bolt length should be at least 1to1.5m beyond the failure zone. Finally, rockbolts should be compatible with other support elements in the same support system in terms of bolt length and spacing, maximum allowable displacement, factor of safety and energy absorption capacities.

Keywords: bolt length and spacing, bearing capacity, factor of safety, pressure arch, rockbolting.

I. INTRODUCTION

Rockbolt is the most widely used support element of the underground mines and civil engineering. Rockbolting design is based on experience, field practices and the types of rock bolt along with the length and spacing of rockbolt. Attempts are made in this article to summarise the design principles and methodologies hidden in rockbolting practise, which include the relationship between the in situ stress state and rockbolt types, the concept of pressure zone, determination of bolt length and spacing, factor of safety, compatibility between support elements and different types of rockbolts. Since rockbolts were first used for ground support in underground excavations (e.g. Panek, 1964; Coates and Cochrane, 1970; Lang, 1972; Barton et al., 1974; Schach et al., 1979; Farmer and Shelton, 1980; Crawford et al., 1985; Stillborg, 1994). Choquet and Hadjigeorgiou (1993) provided a review on this topic in their paper on the design of ground support.

II. UNDERGROUND LOADING CONDITION

2.1, LOW IN SITU STRESS CONDITIONS

Rock blocks in the roof of an underground opening are prevented to fall as far as a high longitudinal stress exists in the rock. However, they would fall under gravity in low in situ stress conditions. In locations of ground surface, the rock often contains well-developed rock joint sets. The rock joints sometimes are open, which is an indication that the in situ rock bearing capacity is low in the rock. The rock support in low stress rock is to prevent rock blocks from falling. To do so, the maximum load exerted on the support elements, such as rockbolts, is the deadweight of the potentially falling block. This is a load-controlled condition. The rockbolts must be strong to bear the dead load of the loosened rock block. Therefore, use of a factor of safety, defined by the strength of the support system and the dead load of the rock is appropriate for rock support design in a load-controlled condition.

This is essentially the design principle in structure mechanics, which states that the load applied to a structure should not be higher than the strength of the structure, i.e. the strength-to-load ratio that is called the factor of safety, should be larger than 2. This design principle is valid for underground constructions where the total load on the construction structures is usually known dead load. In shallow underground openings, this principle is also valid since the maximum load on the rock support system is the deadweight force of loosened rock.

2.2, HIGH IN SITU STRESS CONDITIONS

It observed in a deep depth mine that the number of geological discontinuities in the rock mass became less and the discontinuities were less opened in depth. For instance, at a depth of 1200 m, it was observed that all of the few discontinuities exposed on an excavation face were completely closed. Therefore, it can be said that the rock mass quality is improved at depth because of the reduction in the number of geological discontinuities. However, the in situ rock stresses increase with depth increase. At depth, the major instability is no longer fall of loosened rock blocks but rock failure caused by stress. High stresses could lead to two consequences in underground openings: large failure in soft and weak rock and rock burst in hard and strong rock. It was observed in some mines strain burst usually occurred below a depth of 500 m and became intensive below 1200 m. Rock failure is unavoidable in high stress conditions. The rock support at depth is not to equalise the dead load force of loosened blocks but to prevent the failed rock from disintegration. In high stress rock masses, the support system must be not only strong but also deformable to deal with either stress-induced rock squeezing in soft and weak rock or rock burst in hard and strong rock.

III. TYPES OF ROCK BOLT

The suitable types of rockbolts for a given rocks are associated with the loading condition in the rock mass. In the case of a load-controlled condition. The strength of the rockbolts is the most important parameter for the selection of types of bolts. The basic requirement is that the strength of the rockbolts must be higher than the load on the bolts. The appropriate types of rockbolts under load-controlled conditions are fully encapsulated rebar bolts, thread bar bolts and cable bolts. In high stressed weak and soft rock, the excessive deflection needs to be accommodated. The general approach to deal with rock squeezing is to use ductile rockbolts in conjunction with other types of ductile surface retaining elements such as mesh. Split set is the typical rockbolt for this purpose in the mining. Split set can displace significantly, but it cannot much restrain the rock deformation because of its low load-bearing capacity. Its main function is to avoid disintegration of the jointed rock mass. An active measure to stabilize squeezing rock is to provide a high support resistance to restrain the rock deformation on the one hand, while the support elements in the support system must be deformable on the other hand. Rock burst is an instability issue in overstressed hard and strong rock. The goal of rock support in

International Journal of Advance Research in Science and Engineering
Volume No.07, Special Issue No. (02), January 2018
www.ijarse.com
IJARSE
ISSN: 2319-8354

such conditions is to absorb the kinetic energy of the ejected rock. Energy-absorbing rockbolts should be used in burst-prone rock masses. The higher the loadbearing capacity of the energy-absorbing rockbolt is the less the ejected rock displaces.

3.1, Friction Type Rock Bolts

Roll formed steel tubing with a welded ring at the driving end. Installed in pre-drilled holes with the same rock drill. Ease of installation and low cost makes this a popular rock bolt.

3.2 Deformed Grouted Bar Bolts

Installed with either cement or polyester resin. Ease of installation makes this a popular rock bolt. Cost of bolts and resins make these bolts more expensive than most.

3.3 Shepherds Crook Bolt

Installed in pre-drilled hole. This bolt provides effective support in areas prone to seismic events or high stress changes. Often used to hang pipe or cables.

3.4 Sling Type Eyebolt

Bolt installed in a pre-drilled hole using a wedge. Used for the suspension of air and water columns, ventilation columns, electrical cables.

3.5 Mechanical Anchor Bolts or Expansion Shell Bolts

Used in most Canadian mines. Installed in a pre-drilled hole using the same rock drill to torque up the nut on the bolt to proper tension. Low cost makes this a popular bolt.

IV. DESIGN PRINCIPLE

Determination of bolt length and spacing has been a topic for discussion probably since rockbolts were first used for ground support in underground excavations (e.g. Panek, 1964; Coates and Cochrane, 1970; Lang, 1972; Barton et al., 1974; Schach et al., 1979; Farmer and Shelton, 1980; Crawford et al., 1985; Stillborg, 1994). Choquet and Hadjigeorgiou (1993) provided a review on this topic in their paper on the design of ground support. The following presented are the principles for the determination of bolt length and spacing that are used in the practice of rockbolting to date. From the point of view of operation, the bolt length should be less than half of the opening height for roof bolts and half of the span for wall bolts in order to avoid installation difficulties:

$$L_b \leq 0.5H \text{ (for roof bolts)} \tag{1a}$$

$$L_b \leq 0.5B \text{ (for wall bolts)} \tag{1b}$$

Where L_b represents the bolt length, H is the opening height, and B is the opening span. The bolt length is also associated with the bolting principle. In the case that the failure zone is limited to a relatively small depth. The bolt length should be at least 1 m longer than the depth of the failure zone, i.e.

$$L_b \geq d_f + 1 \tag{2}$$

Where df is the depth of the failure zone. In the case of a vast failure zone, the bolt length is short, varying from 2 m to 3 m, but its upper limit is governed by Eq. (1). For tunnels excavated in moderately jointed hard rock masses, the Norwegian Road Authority proposed the following formula to determine the length of un-tensioned bolts in the central section of the tunnel for the purpose of suspending the failure zone (Statens vegvesen, 2000):

$$L_b = 1.4 + 0.184B \quad (3)$$

In practice, the bolt pattern in systematic bolting is such that the in-row spacing and the distance between rows are equal. The bolt spacing, s, is recommended to be in the range from 1 m to 2.5 m. However, rock joint spacing should be also taken into account when the bolt spacing is determined. A rule of thumb is to set the bolt spacing equal to 3-4 times the mean joint spacing in the case of a mean joint spacing in the range of 0.3-1 m, i.e.

$$S = (3\text{-}4)e \quad (4)$$

Where, **"e"** represents the mean joint spacing.

In the case of a vast failure zone, the Norwegian Road Authority recommends the use of relatively short tensioned rockbolts to establish an artificial pressure zone in the failure zone. The bolt length is still estimated using Eq. (3), but the bolt spacing is recommended to be smaller than 3 times the mean joint spacing, i.e.

$$\mathbf{S \leq 3e} \quad (5)$$

it is required that the rockbolts interact with each other and an interaction⁰ zone is formed in the bolt-reinforced rock party. Assuming that the reinforcement angle of a single rockbolt is 90 , the thickness of the interaction zone, t, is related to the bolt length (Lb) and spacing (S) as follows:

$$t = L_b - S \quad (6)$$

The bolt length is usually short, 2-3 m, in this type of rockbolting. The thickness of the interaction zone is required to be at least $0.5L_b$ in order that a strong enough artificial arch can be established in the broken rock. This requirement leads to a bolt spacing that should be less than half of the bolt length, i.e.

$$S \leq L_b/2 \quad (7)$$

In the design stage of an underground rock excavation, bolt length and spacing are often determined with the help of empirical methods recommended in various rock mass classification systems. In the Q rock mass classification system(Bartonetal.,1974), the bolt length and spacing can be found in a chart based on the Q-value of the rock mass and a geometrical parameter called the equivalent dimension (Barton and Grimstad, 2014). The equivalent dimension is defined by the span of the excavation and a coefficient describing the intended use of the excavation (road tunnel, underground station, etc.). In the rock mass rating (RMR) system by Bieniawski (1989), bolt length and spacing, as well as other types of support measures, are empirically recommended in a table for five classes of rock mass quality.

V. FACTOR OF SAFETY

Rock blocks in the roof may become loosened in shallow tunnels wherein situ rock stresses are low. The loosened blocks tend to fall under gravity. The load exerted on the rockbolts is equal to the dead load of the falling blocks. In this load-controlled condition, the factor of safety (FS) for rockbolting is defined as

$$FS = \frac{\text{Load capacity of the bolts}}{\text{Total load on the bolts}} \quad (8)$$

In this case, a safe rock support requires that the load on the bolt is less than the strength of the bolt, i.e. FS > 1. It is required that the factor of safety is in the range of 1.5-3 in rockbolting design.

5.1, FACTOR OF SAFETY IN SQUEEZING ROCK

Rock deformation can be significantly large in tunnels excavated in highly stressed soft and weak rock because of major rock failure. The essential driving power for the rock deformation is the strain energy released from the rock mass after excavation. The greater part of the released strain energy is dissipated in rock fracturing, which in turn brings about rock deformation. In extremely poor rock conditions, the large rock deformation may lead to rock collapse.. In squeezing rock, it is more relevant to define the factor of safety with displacements rather than load and strength. It is required, from the point of view of stability, that the displacement of the tunnel wall at equilibrium, Ueq, has to be smaller than the critical displacement, Uc, beyond which uncontrollable rock collapse would occur. The factor of safety for the rock support, FS, is thus defined as

$$FS = Uc / Ueq \tag{9}$$

It must be pointed out that the critical displacement Uc is difficult to be quantified even with the help of numerical modelling. Beyond displacement Uc, the rock mass becomes unstable and calculation iterations would become non-convergent in numerical modelling.

VI. CONCLUSIONS

The strength of rockbolts is the key parameter for rockbolting design in low stress rocks. Rockbolts should be deformable in addition to the requirement of high strength in high stress rocks. Rockbolts are energy absorbent in squeezing and jointed rock conditions. There exists a natural failure zone immediately outside of the actual failure zone in the rock surrounding an underground excavation. In the case of a shallow failure zone, the rockbolts should be long enough to reach the failure zone. In the case of major failure zone, short rockbolts are tightly installed to establish within the failure zone and long cables are anchored into the natural failure zone. Determination of the bolt length and spacing is associated with the methodology of rockbolting. In the case of the anchorage of rockbolts in the natural pressure zone, the bolt length should be at least 1.5 m beyond the failure zone. In the case of establishing an artificial pressure zone, appropriate bolt lengths are approximately 3m in mine drifts and upto 7m in large -scale hydropower caverns. Bolt spacing is more important than bolt length in this case. The principle is that the bolt spacing guarantees that the rockbolts interact with each other. The appropriate bolt spacing is 1 m for 3 m long bolts and less than 1.5 m for 7-m long bolts. The rockbolting design is based on the dead load of falling blocks and the strength of the rockbolt in low rock stress locations. The maximum allowable displacement and the ultimate displacement capacity of the rockbolt should also be taken into account. The rockbolts in a rock support system should be compatible with other support elements with respect to displacement and high strength capacities.

VII. ACKNOWLEDGEMENTS

This is to acknowledge our indebtedness to under guidance of **MANIK BADHWAR**, HOD of in CIVIL ENGINEERING DEPARTMENT OF VEDANT COLLEGE OF ENGINEERING AND TECHNOLOGY for

IJARSE
ISSN: 2319-8354

his guidance and suggestion for preparing this article. His towering presence installed in us the carving to work harder and complete this daunting task timely with a sufficient degree of independent study. We are highly thankful for his edifying guidance and encouragement proved to us throughout the completion of this report.

REFERENCES

[1.] Aksoy CO, Onargan T. The role of umbrella arch and face bolt as deformation preventing support system in preventing building damages. Tunnelling and Underground Space Technology 2010;25(5):553e9.

[2.] Bang S. Limit analysis of spiling reinforcement system in soft ground tunneling. Tunnel 1984;3:140e6.

[3.] Barton N, Grimstad E. Tunnel and cavern support selection in Norway, based on rock mass classification with the Q-system. Publication No. 23. Norwegian Tunnelling Society; 2014. p. 45e77.

[4.] Barton N, Lien R, Lunde J. Engineering classification of rock masses for the design of tunnel support. Rock Mechanics 1974;6(4):189e239.

[5.] Bieniawski ZT. Engineering rock mass classifications. New York: Wiley; 1989.

[6.] Choquet P, Hadjigeorgiou J. The design of support for underground excavations. In: Excavation, support and monitoring. Comprehensive rock engineering-principles, practice and projects, vol. 4. Pergamon Press; 1993. p. 313e48.

[7.] Coates DF, Cochrane TS. Development of design specifications for rock bolting from research in Canadian mines. Research Report R224. Mining Research Centre, Energy, Mines and Resources Canada; 1970.

[8.] Crawford AM, Bray JW. Influence of the in situ stress field and joint stiffness on rock wedge stability in underground openings. Canadian Geotechnical Journal 1983;20(2):276e87.

[9.] Crawford AM, Ng L, Lau KC. The spacing and length of rock bolts for underground openings in jointed rock.In: Einsenstein Z, editor. Proceedings of the 5th international conference on numerical methods in geomechanics. A.A. Balkema; 1985. p. 1293e300.

[10.] DSI. Spiles and forepoling boards. Product catalog. DWIDAG-System International; 2015.

[11.] Farmer IW, Shelton PD. Factors that affect underground rockbolt reinforcement systems design. Transactions of the Institutions of Mining and Metallurgy 1980;89:68e83.

[12.] Harrison P, Hudson J. Engineering rock mechanics. Part 2: illustrative worked examples. Pergamon; 2000.

[13.] Hoek E, Brown ET. Underground excavations in rock. London: Institution of Mining and Metallurgy; 1980.

[14.] Hoek E. Model todemonstrate how bolts work. In: Practical rock engineering; 2007. Hoek E. Numerical modelling for shallow tunnels in weak rock. 2015. https://www. rocscience.com/documents/pdfs/rocnews/Spring2003/ShallowTunnels.pdf.

[15.] Kaiser PK, Tannant DD, McCreath DR. Canadian rock burst support handbook. Sudbury, Canada: Geomechanics Research Center; 1996.

[16.] Krauland N. Rockbolting and economy. In: Stephansson O, editor. Rockbolting - theory and applications in mining and underground construction. Rotterdam: A.A. Balkema; 1983. p. 499e507.