International Society for Rock Mechanics and Rock Engineering
Norwegian Group for Rock Mechanics

ISRM International Symposium
Eurock 2025 – Expanding the Underground Space
Trondheim, Norway, 16-20 June
ISBN 978-82-8208-079-8

Investigating Roof Bolt Efficacy in Blast-Induced Damage Zone Around the Footwall Drive of an Underground Copper Mine

Satyam Choudhury, Bibhuti Bhushan Mandal & Shantanu Patel
Department of Mining Engineering, Indian Institute of Technology, Kharagpur, India
skc@kgpian.iitkgp.ac.in

Abstract

The drilling and blasting method is commonly used in underground mining for both development and ore extraction. However, this process often results in damage to the surrounding rock mass, as new cracks are formed due to the blast. Understanding the reduced rock mass properties in the blast-damaged zone is critical during mine design. Since in-situ testing for these properties is challenging and time-consuming, empirical methods like the Hoek-Brown criterion have been widely adopted. Initially, the Hoek-Brown criterion did not account for blast damage, but subsequent studies highlighted the significance of such damage. As a result, a disturbance factor (D) was introduced in that ranges from 0 (undamaged rock) to 1 (highly damaged rock), to incorporate blast-induced damage. In practice, a constant disturbance factor is often applied across the entire rock mass, even though the intensity of damage typically decreases with distance from the excavation face. This study first aims to investigate the blast damage zone using a borehole camera so as to define an extent of damage zone and then asses the roof bolt efficacy around the blast-induced damage zone by considering the varying disturbance factors. The maximum axial stress on a roof bolt increases significantly with the level of blast damage: for a disturbance factor (D) of 1, the maximum axial stress is estimated to be 75.44 MPa, while for D = 0, it is only 3.46 MPa. These findings suggest that roof bolts experience much higher axial stress in areas closer to the excavation face, where the rock is more fractured due to blasting. This highlights the importance of optimizing roof bolting design and installation to effectively address the varying damage levels within the excavation damage zone, improving the stability of underground excavations.

Keywords

Blast-induced damage zone, Disturbance Factor, Roof bolts, Axial stress distribution

ISRM

NBG

1 Introduction

In Underground metal mining the ore is generally extracted by drilling and blasting method which often results in the development of the excavation damage zone which refers to irreversible damage to the surrounding rock (Cai et al., 2004).This encompasses the failed zone, where complete detachment of blocks/slabs from the rock mass occurs, and the damaged zone, characterized by irreversible shifts in rock properties such as the formation of micro-cracks, fractures, reduced deformation modulus, and altered permeability. Within the EDZ the physical and mechanical properties of rock mass degrade significantly, resulting in excavation hazards such as spallation, collapse, and even V-shaped notches (Martino & Chandler, 2004). The blast-induced damage zone is a section of the Excavation Damage Zone (EDZ) around which the in-situ rock mass properties and conditions have been altered due to stress redistribution and induced fracturing. Rock engineers need to investigate the rock failure criteria and design a stable stope as well as support system before carrying out any mining operation. However, the decision sometimes results in ore loss and higher support requirements as the underground structure are designed in an overconservative way. This happens due to our lack of understanding about the disturbance zone around the underground opening. Hoek-Brown criterion(Hoek & Carranza-Torres, 2002) is generally used while designing the underground structure and has gained significant recognition worldwide among researchers and rock engineers due to its practical application in rock engineering practice. This is because it incorporates a comprehensive description of the various structural characteristics that influence the deformability and strength of the rock masses. However, the key aspects that need to be accurately determined when using the Hoek–Brown failure criterion is the disturbance factor (D) and geological strength index (GSI). Though there is advancement in quantification of GSI the disturbance factor is purely based on qualitative assessment of rock mass. The scale used for D varies from 0 for undisturbed in situ rock masses to 1 for highly disturbed rock masses. Hoek et al. (2002) gave guidelines on how to select a value for D based on the state of the in-situ rock mass after it has experienced stress relaxation and blasting damage. The same value for D is applied for the entire stope whereas damage intensity should be decreased away from the face. In this study, field investigations were conducted to assess the extent of blast-induced damage zones (BIDZ) surrounding a drive using borehole cameras. Subsequently, numerical models were analysed, varying the BIDZ parameters to understand their implications on support design for underground metal mining.

2 Background

2.1 Blast Induced Damage Zone (BIDZ)

To make an informed assessment of failure processes and their potential impacts, it is essential to understand the characteristics of the Blast-Induced Damage Zone (BIDZ). Blast-induced damage is broadly defined as any form of damage to the rock mass that originates from blasting activities. This damage typically includes the formation of microcracks, fractures, and the degradation of the rock mass properties, significantly influencing the stability and behavior of the surrounding rock (David Saiang David Saiang, 2008). Fig. 1 illustrates the nature of blast-induced cracks around a tunnel boundary, which include both macroscopic and microscopic fractures of varying shapes and sizes. These cracks are typically radial, highly anisotropic, and non-persistent, with their distribution, extent, and intersection influenced by factors such as rock mass conditions, in-situ stresses, and blasting parameters. Unlike naturally formed geological structures, the kinematics of blast-induced blocks differ significantly. Thus, the extent of the blast-induced zone must be examined so as to design an optimal support system for underground structures.

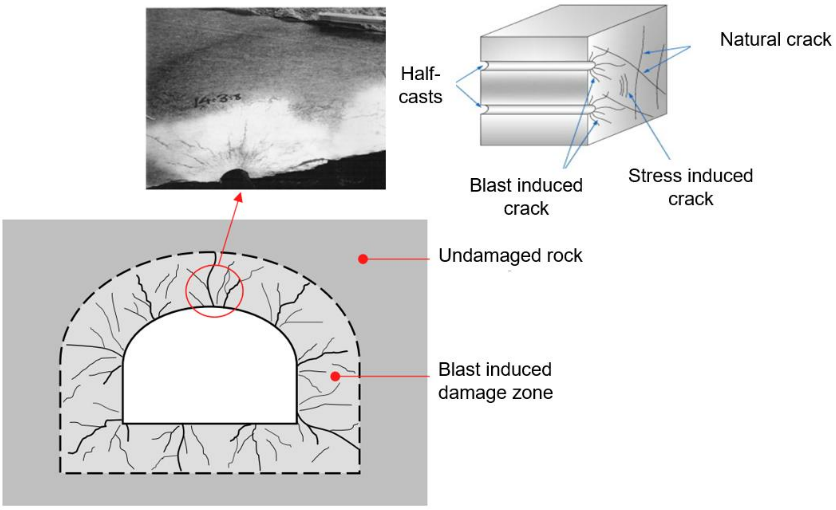


Fig. 1 Rock mass condition around a tunnel boundary excavated by drill and blast. The damaged zone comprises of discontinuous fractures of microscopic to macroscopic sizes with complex fracture patterns due to radial cracks. (Olsson et al., 2008)

2.2 Disturbance Factor (D) in Hoek-Brown failure criteria

Stability and deformation analyses of underground openings require an accurate estimation of rock mass strength. The mechanical properties of rock masses, such as strength and stiffness, are reduced by blast damage and stress relief resulting from excavation activities. These effects often cause the rock mass to undergo relaxation and dilation(Marinos & Hoek, 2018). This degradation is commonly referred to as the damage, blast, or disturbance factor (D). (Sonmez & Ulusay, 1999) proposed a disturbance factor approach, where the rock mass could be continuously defined based on the type of excavation, as demonstrated in five slope failure cases. Finally, (Hoek & Carranza-Torres, 2002) introduced the blast damage factor within the Hoek–Brown criterion to account for these blast-induced changes. The guidelines on how to select a value for D based on the state of the in situ rock mass after it has experienced stress relaxation and blasting damage as shown in the Fig. 2.

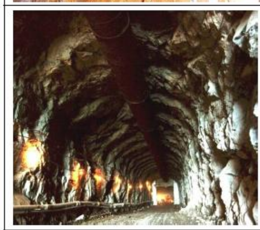
Appearance of rock mass	Description of rock mass	Suggested value of *D*
	Excellent quality controlled blasting or excavation by Tunnel Boring Machine results in minimal disturbance to the confined rock mass surrounding a tunnel.	$D = 0$
	Mechanical or hand excavation in poor quality rock masses (no blasting) results in minimal disturbance to the surrounding rock mass. Where squeezing problems result in significant floor heave, disturbance can be severe unless a temporary invert, as shown in the photograph, is placed.	$D = 0$ $D = 0.5$ No invert
	Very poor quality blasting in a hard rock tunnel results in severe local damage, extending 2 or 3 m, in the surrounding rock mass.	$D = 0.8$

Fig. 2 Guidelines for estimating disturbance factor D (Hoek & Carranza-Torres, 2002)

In practice, it is common to assign a uniform disturbance factor (D) to the entire rock mass. However, this approach often results in overly conservative designs. For instance, the elastic modulus of the rock mass can vary significantly by as much as fivefold between D=1 and D=0, assuming all other factors remain constant (Liu et al., 2023). So it is crucial to first assess the extent of blast-induced damage and then limit the application of reduced rock mass properties specifically to the affected zone. This targeted approach ensures that the rock mass behavior is accurately represented, leading to more efficient and cost-effective support design while maintaining safety and stability.

2.3 Rock bolt support

Rock bolts have been extensively used in underground mines and civil tunnelling projects as competent ground reinforcement elements due to their outstanding performance in mitigating the roof and side-wall failure (Li et al., 2017).They reinforce the weak rock mass at the excavation surface by transferring its load into the stable rock mass far into the country rock. In field, the rock bolt is predominantly subject to axial loading that would lead to the failure in some cases (Li et al., 2021) as such, it becomes essential to have a profound understanding of the load transfer mechanisms, load-displacement behaviour and the axial stress distribution of rock bolts.

3 Case Study

3.1 About the Mine

To study the excavation damage zone an underground copper mine is selected which is located near Ghatsila in East Singhbhum district of Jharkhand, India. The Kendadih Mine employs the Room and Pillar Stoping method for ore extraction, where the width of the ore body ranges between 1.5 to 4.0 meters. Fig. 3 showcases the transverse Section of the mine. A raise is put along the H/W contact from lower level to upper level. A sill pillar of 5m above the lower level and a crown pillar of 5m below the upper level are left as support. Rib pillars, left at a width of 3 meters between consecutive stopes, contribute to ore loss, affecting overall recovery efficiency. Understanding the extent of blast damage and stability in the host rock and ore body due to blasting and stress redistribution is critical to optimizing support design without compromising safety.

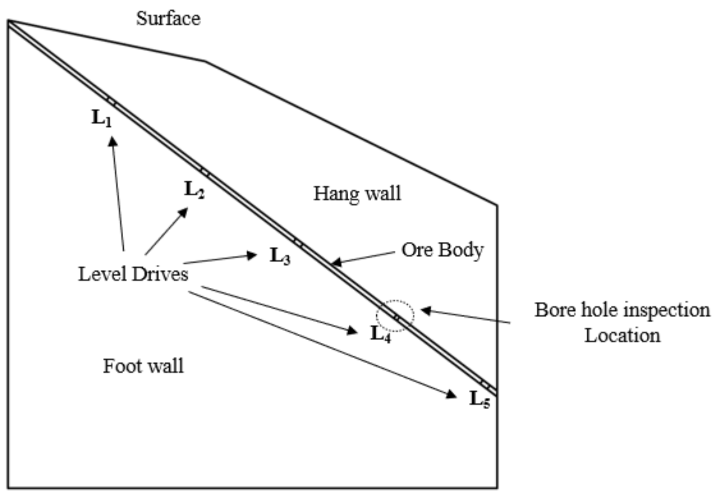


Fig. 3 The transverse Section of the mine

3.2 Analysis of Fracture from Core

Boreholes were drilled mainly at three different angles in the hanging wall, in the footwall & as well as through the ore body to a depth of 10 meters as shown in the Fig. 4. For some boreholes, the initial one meter of rock was found in disced shape. For most of the boreholes, the initial few centimeters of rock are highly damaged and can be termed as highly damage zome. Two different types of fracture patterns were observed from these core boxes as shown in Fig. 5. For different boreholes, the integrity of the rock varied as the RQD from the core varied significantly. The RQD of the cores were calculated for every meter of core drilled. The RQD for different boreholes varied between 0% in the first one meter to 90% at a depth of around 5 meters or more which signifies the existence of the damage zone around the drive i.e., the peripheral rock around the drive is highly damaged whereas the rock beyond 2 meters is less damaged. Fig. 6 showcases the variation of RQD and core recovery for different boreholes.

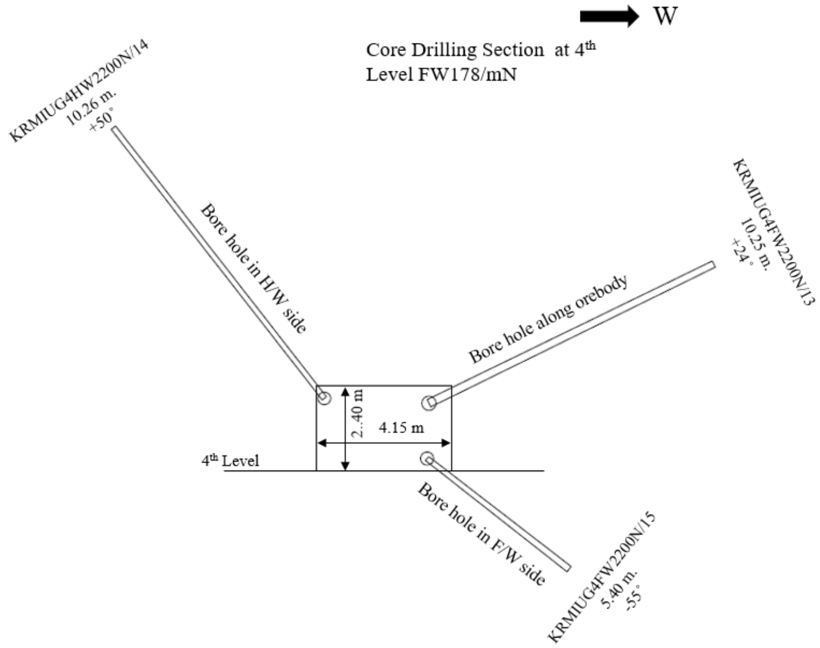


Fig. 4 Core drilling section at 4th Level of the mine

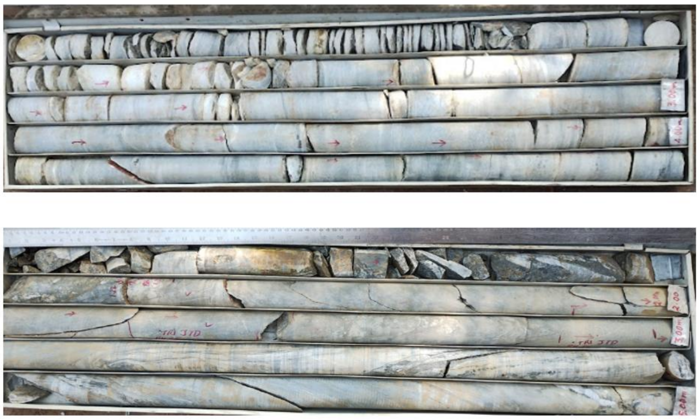


Fig. 5 the borehole log, and the core of the damaged rock

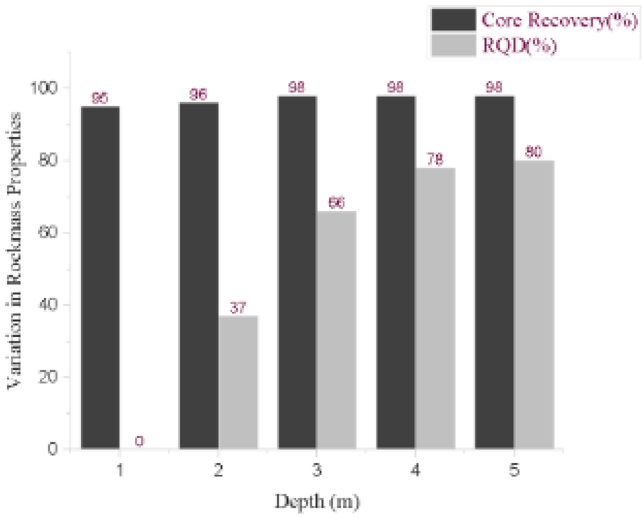


Fig. 6 Variation in RQD & Core Recovery at different depths from excavation surface

3.3 Borehole Camera Inspection

A borehole camera was used to capture photographs of the borehole wall at various depths, as shown in Fig. 7. The images reveal that the rock is highly damaged up to a depth of approximately 60 cm, with a higher concentration of cracks observed in this zone. Two primary types of cracks were identified: geological cracks, characterized by infill material, and blast-induced cracks, which appear relatively fresh. The number of cracks decreases gradually with depth, and beyond 200 cm, the number of cracks reduces significantly. Based on this observation, the damage zone is estimated to extend up to a depth of 2 meters and the finding corroborated by the core samples collected from the same boreholes.

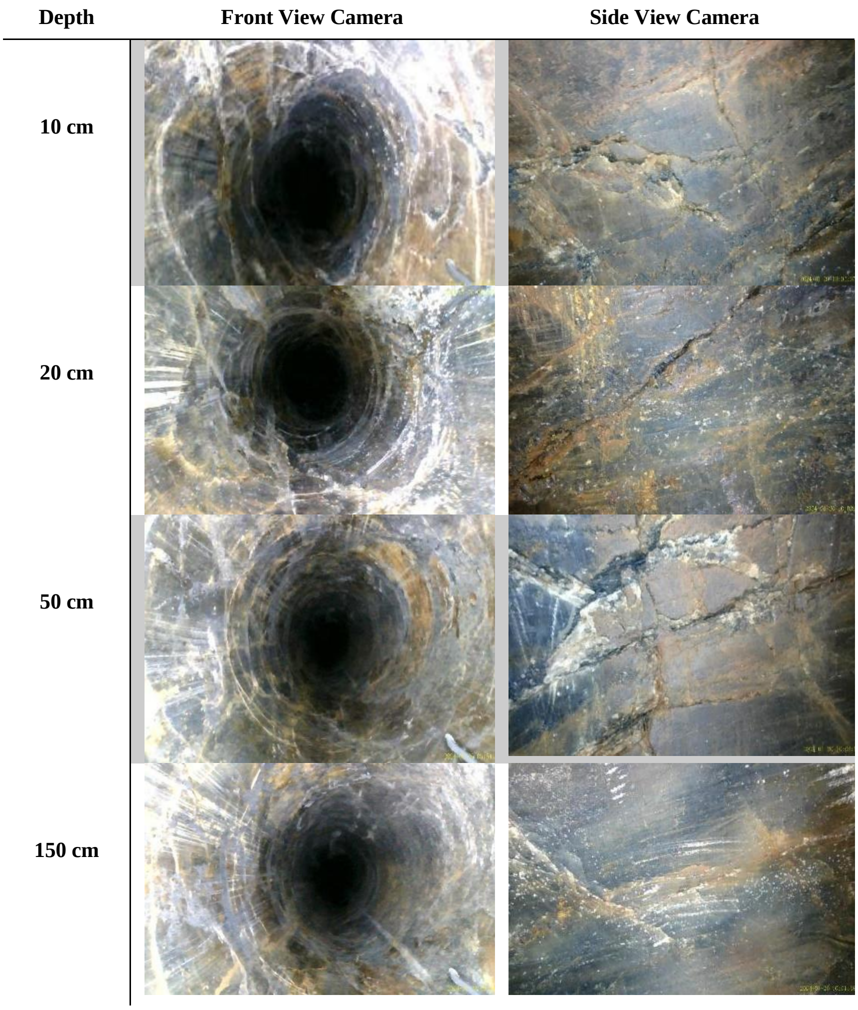


Fig. 7 Front & Side view images of borehole wall at different depth.

3.4 Face Mapping Data for Qualitative assessment of GSI and D

The majority of joints are < 6 m long and semi continuous to continuous, spaced < 2 m apart, smooth to rough and dry to stained with occasional seepage. However, seepage was found to be more in 3rd level in the hanging wall. Here a semi-continuous joint in rocks refers to a partially persistent joint, meaning it extends for a significant distance but does not form a fully continuous plane through the rock mass. Table 1 & Table 2 showcase the filed data collected from face mapping of the hanging wall of the mine and Table 3 showcases the qualitatively assigned value of GSI and disturbance factor at that place.

Table 1 Summary of the field conditions

Conditions	**Field Observations**
Weathering	No
Rock Strength	Strong
Discontinuity type	Joints
Hardness	Hard
Roughness	Sharp
Seepage	Yes

Table 2 Summary of the Joint Orientation

Rock Type		**QBCS (Quartz Biotite Chlorite Schist)**		
Joint Set		J1	J2	J3
Dip	[°]	76	45	58
Dip Direction		N125E	N23E	N78E
Spacing	[cm]	21	16	5

Aperture	[mm]	2	1	2
Persistence	[cm]	142	102	89

Table 3 Qualitative assessment of GSI & D

Location	**GSI**	**D**
4th Level(H/W)	55-65	0.5-0.7
4th Level (Ore)	50-60	0.5-0.7
4th Level(F/W)	50-65	0.5-0.7

4 Numerical Analysis

Finite Element Method (FEM) analysis was conducted using Rocscience 2D software to evaluate the stability of an underground opening. Different cases were tested by varying the disturbance factor (D), while keeping the extent of the Excavation Damage Zone (EDZ) constant at 2 meters, based on borehole camera analysis. The study focused on analyzing the maximum axial stress on the bolts under different scenarios to understand how variations in the disturbance factor affect bolt stress. Additionally, the distribution of axial stress along the bolt length with depth was examined to assess the influence of damage on reinforcement performance. These analyses provide valuable insights into the relationship between disturbance factor, bolt stress behavior, and the overall stability of the underground excavation.

5 Result & Discussion

The relationship between the disturbance factor (D) and the maximum axial stress on roof bolts highlights the impact of blast-induced damage on roof support performance. For undamaged rock (D=0), the maximum axial stress on rock bolts is minimal at 3.46 MPa due to the higher strength and stiffness of the intact rock mass, which limits load transfer to the bolts. However, as the disturbance factor increases, the maximum axial stress rises sharply, reaching 28.33 MPa at D=0.3 and 36.65 MPa at D = 0.5. This trend indicates that even moderate levels of blast damage significantly amplify the stresses experienced by the bolts, as the fractured rock mass relies more heavily on the support system for stability. At D=0.7, maximum axial stress on the bolt increases further to 48.36 MPa, reflecting the highly fractured nature of the rock, and finally peaks at 75.44 MPa for D = 1, where the rock mass is highly damaged and largely incapable of self-support. Fig. 8 showcases the relationship between maximum axial stress on rock bolts with disturbance factor. The sharp increase in axial stress with higher disturbance factors highlights that roof bolts in highly damaged zones experience much greater loading. This places a higher demand on the bolt's tensile capacity, potentially leading to failure if the bolt design is inadequate. Bolts in areas with D ≥ 0.7 must have higher tensile strength and load-bearing capacity to accommodate the elevated stresses.

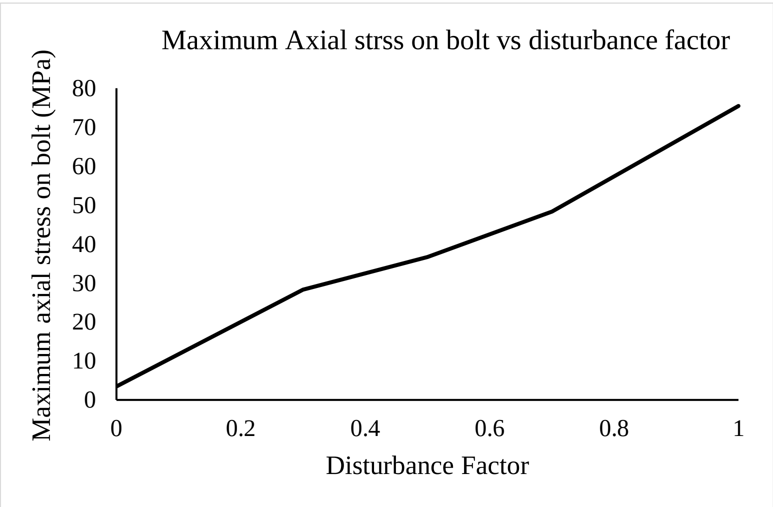


Fig. 8 Relationship between maximum axial stress on rock bolt with disturbance factor

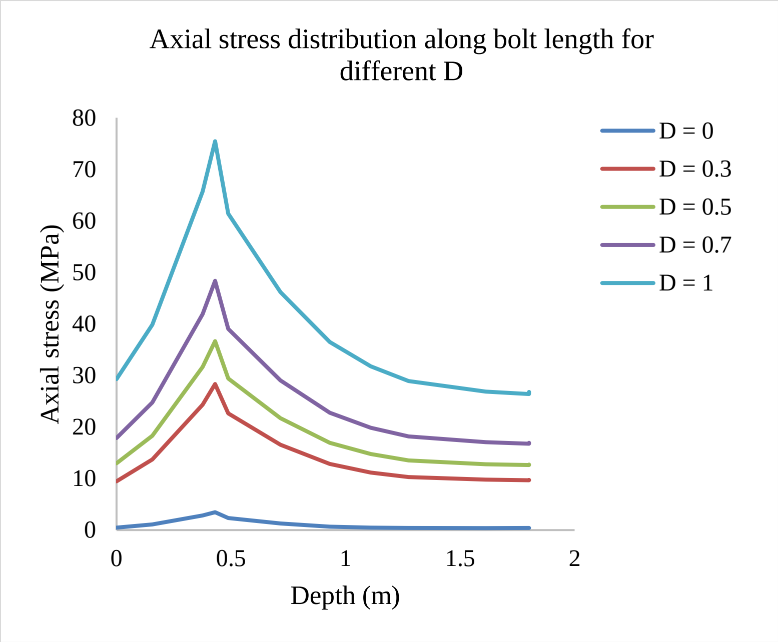


Fig. 9 Axial stress distribution along the bolt length for different disturbance factors

Fig. 9 showcases the distribution of axial stress on roof bolt for different disturbance factors. Beyond the depth, the axial stress decreases to 26.39 at a depth 1.8 meters. A similar trend is observed for different disturbance factors. This suggests that the blast-induced damage is more concentrated closer to the excavation face, resulting in greater loading on the roof bolt in this region, where the rock is more fractured and unstable. The results indicate that the extent of blast-induced damage is not uniform but progressive, with stress intensifying as rock damage increases closer to the excavation face. This reinforces the need for zonal support strategies that account for varying disturbance factors.

References

Cai, M., Kaiser, P. K., Uno, H., Tasaka, Y., & Minami, M. (2004). Estimation of rock mass deformation modulus and strength of jointed hard rock masses using the GSI system. *International Journal of Rock Mechanics and Mining Sciences*, *41*(1), 3–19. https://doi.org/10.1016/S1365-1609(03)00025-X

David Saiang David Saiang, L. (n.d.). *Behaviour of Blast-Induced Damaged Zone Around Underground Excavations in Hard Rock Mass*.

Hoek, E., & Carranza-Torres, C. (2002). *HOEK-BROWN FAILURE CRITERION-2002 EDITION*.

Li, D., Li, Y., Chen, J., & Masoumi, H. (2021). An analytical model for axial performance of rock bolts under constant confining pressure based on continuously yielding criterion. *Tunnelling and Underground Space Technology*, *113*(February 2020), 103955. https://doi.org/10.1016/j.tust.2021.103955

Li, D., Masoumi, H., Saydam, S., & Hagan, P. C. (2017). A constitutive model for load-displacement performance of modified cable bolts. *Tunnelling and Underground Space Technology*, *68*(May), 95–105. https://doi.org/10.1016/j.tust.2017.05.025

Marinos, P., & Hoek, E. (2018). GSI: A geologically friendly tool for rock mass strength estimation. *ISRM International Symposium 2000, IS 2000*.

Olsson, M., Markström, I., Pettersson, A., & Associates, G. (2008). Svensk Kärnbränslehantering AB Methodology study for documentation and 3D modelling of blast induced fractures. *CM Gruppen AB, May*. www.skb.se.

Sonmez, H., & Ulusay, R. (1999). Modifications to the geological strength index (GSI) and their applicability to stability of slopes. *International Journal of Rock Mechanics and Mining Sciences*, *36*(6), 743–760. https://doi.org/10.1016/S0148-9062(99)00043-1