Editorial

After a gap we are back into the main stream. During this period Indian Mining sector had faced several challenges of improving on coal stock at thermal power plants. Coal ministry, Indian Railways and our coal producing subsidiaries of CIL did a wonderful job and helped the nation from blackouts. Coal Warriers we call them did wonders.

NCL which is having a brand image to produce coal by handling largest volumes of over burden had augmented it's HEMM capacity by introducing new units. The NCL management did a great job in improving productivity and safety in blasting. To bring to the light of their achievements NCL had organised a very thought provoking Seminar.

This National Seminar on Latest Advancement in Drilling and Blasting & Rock Fragmentation Technology in Opencast Mines", was a successful event with active participation of several leading experts and a good number of papers were discussed. The organisers have brought out a beautiful proceedings volume and we are reproducing some of the selected papers in our Journal.

With the gracious presence of senior officials of DGMS the message was very clear to attain zero accidents in all our operations.

Northern Coalfields Limited had a gigantic task of consuming safely over 24% of bulk SME explosives in their dragline and shovel benches. produced in India.

We have one paper by a young researcher of CIMFR Sri Amar Prakash on Longwall Mining. A student from AKS University he had made this paper rich by dealing on some of the critical issues of Longwall Mining.

Prof S.Jayanthu
Editor-in-Chief

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Persons in the News

Shri Uday A Kaole presently Director (Technical/P&P), Bharat Coking Coal Ltd has been selected for the post of Chairman & Managing Director, Mahanadi Coalfields Ltd. Shri Kaole graduate in Mining Engineering from Nagpur University in the year 1987 and thereafter he obtained 1st Class Mine Manager Certification of Competencies. He also acquired MBA and LLB from Nagpur University. He joined Coal India Limited in the year 1987 at Western Coalfields Limited. He rendered his services in WCL at various capacities before joining SECL in 2007. In the year 2007 he was promoted to M1 and joined SECL thereafter in the year 2012 he was again promoted to M2. Before joining BCCL as Director (Technical) he was shouldering the responsibility as Area General Manager in Wani Area at WCL. Shri Uday A Kaole went to South Africa to attend Mining INDABA in the year 2022. He has a long experience in Underground Mining and especially in Wardha Valley fire in Underground and Opencast.

COAL NEWS

SEVENTH ROUND OF COMMERCIAL COAL MINE AUCTION: CENTRE SIGN PACTS WITH 6 BIDDERS

The Centre entered into pacts with six bidders, including Hindalco Industries and NLC India Ltd, which bagged coal blocks under the seventh round of coal mines auction. The government had launched the seventh round of auctions for commercial mining of coal blocks, putting 106 mines under the hammer. In a statement, the coal ministry said that successful bidders encompass entities such Hindalco Industries Ltd, NTPC Mining Ltd, NLC India Ltd, Shri Bajrang Power & Ispat Limited and Nilkanth Mining Ltd. The mines for which these agreements have been executed are Meenakshi West, North Dhadu (Eastern Part), North Dhadu (Western Part), Pathora East, Pathora West and Sherband. Of the said blocks, four have undergone partial exploration and the other two are fully explored.

The estimated total revenue generation annually from these blocks stands at approximately Rs 787.59 crore, based on production at an aggregated peak rate capacity (PRC) level of seven metric tonne per annum (MTPA). An investment of Rs 1,050 crore has been earmarked for the operationalisation of these coal mines. The revenue-sharing percentages for the successful bidders in these six coal mines range from six per cent to 43.75 per cent, with an average revenue share of 23.71 per cent.

STEAM COAL IMPORTS DIP 6% TO 61 MT IN APRIL-JULY

India's steam coal imports dipped by 6% year-on-year in the April-July period, as less pressure on power plants' stocks and drop in domestic e-auctioned fuel prices reduced the demand for imported fuel. Shipments from Indonesia dipped by almost a fourth from a year ago. Total steam coal imports for the period stood at 60.7 million tonnes, data from the commerce ministry showed. Industry experts and officials attributed the drop to lower imports by power units, cheaper domestic supplies for the non-power sector, and better fuel planning.

Coal stock at power plants, both domestic and imported, did not deplete as much as anticipated from March till July, owing to favourable weather conditions and more opening coal stocks. However, imports were still needed to meet the total requirement, an official said. While February was hotter than usual, March to May was cooler on average. The government had directed electricity generators to import 6% of their fuel requirement in anticipation of higher demand in the summer months. However, the import by power generators for blending was less than 6%, the official said.

While domestic coal supplies and increased captive coal output helped in maintaining sufficient stocks at power plants, blending of imported coal was still much needed. Earlier this month, the power ministry said that despite an increase in domestic coal supply in the first quarter of FY24, it wasn't sufficient to meet the total requirement.

MINING NEWS

HCL TO RAMP UP COPPER ORE PRODUCTION CAPACITY TO 12.2 MTPA BY FY'29

Hindustan Copper Ltd (HCL) said it is implementing expansion projects to increase its mine production capacity from the current level to 12.2 million tonnes per annum (MTPA) by FY 2028-29. HCL Chairman and Managing Director Ghanshyam Sharma at the AGM told shareholders that the company has access to around 55 per cent of the copper ore reserves and resources in India with an average grade of 0.96 per cent. "In FY 2022-23, the copper ore production in India was around 3.35 million tonnes. HCL is implementing a plan to increase its mining capacity from its current level of ore production to 12.2 MTPA in Phase-I in the next 6 to 7 years," Sharma said. He added that the company has achieved ore production of 33.47 lakh tonnes during FY 2022-23 as against 35.70 lakh tonnes produced in FY 2021-22. Sharma said the company is also focusing on exploration and has enhanced its budget for this head by five times as compared to the last 10 years.

"The efforts of exploration have led to an increase in copper ore reserve & resources by 66.59 million tonnes of ore. The geological drilling and associated work completed in FY 2022-23 is around four times theexploration activities undertaken by the Company in the last 10 years," he said. Sharma said the expansion of MCP (in Madhya Pradesh) will augment the ore production capacity from 2.5 to 5.0 MTPA by developing an underground mine below the existing open pit. The proposed expansion of mines in the western sector - Khetri and Kolihan in Rajasthan - will increase ore production capacity fromexisting 1.0 to 3.0 MTPA. The proposed expansion and reopening of Surda Mine in Jharkhand will increase its production capacity from 0.4 MTPA to 0.9 MTPA. Sharma said the expansion projects will boost domestic production of copper metal to reduce India's dependence on imports.

MINERAL PRODUCTION RECORDS 10.7 PER CENT SURGE IN JULY

The mineral production sector in India witnessed a significant upswing in July 2023, with a remarkable 10.7 per cent increase compared to the same month in the previous year. These findings are based on the latest data released by the Indian Bureau of Mines (IBM), indicating a positive trajectory in the country's mineral production, read the Ministry of Mines press release. The Index of Mineral Production for the mining and quarrying sector, using the base year 2011-12 as a reference, reached a level of 111.9 in July 2023. This robust growth demonstrates the sector's resilience and ability to contribute substantially to the nation's economic development.

The cumulative growth for the April-July period in 2023 compared to the same period in the previous year stands at an impressive 7.3 per cent. This consistent performance underscores the sector's role as a vital component of India's economic landscape, read the press release. In July 2023, production figures for several critical minerals revealed promising numbers. Coal production stood at an impressive 693 lakh tonnes, while lignite production reached 32 lakh tonnes. Natural gas utilization recorded 3062 million cubic meters, and crude petroleum production amounted to 25 lakh tonnes.

Additionally, essential minerals such as bauxite, chromite, copper concentrate, gold, iron ore, lead concentrate, manganese ore, zinc concentrate, limestone, phosphorite, and magnesite all contributed positively to the sector's growth read the press release. Chromite, with a remarkable growth rate of 45.9 per cent, led the pack of minerals showing positive trends in July 2023 compared to the same month in 2022. It was followed closely by manganese ore (41.7 per cent), coal (14.9 per cent), limestone (12.7 per cent), iron ore (11.2 per cent), and gold (9.7 per cent). These significant increases indicate the sector's capacity for expansion and diversification.

Furthermore, the mineral sector demonstrated its resilience with notable growth in copper concentrate (9 per cent), natural gas utilization (8.9 per cent), lead concentrate (4.7 per cent), zinc concentrate (3.6 per cent), magnesite (3.4 per cent), and crude petroleum (2.1 per cent), read the press release. While the overall outlook is promising, some minerals showed a decline in production during July 2023. Lignite production declined marginally by -0.7 per cent, bauxite by -3.2 per cent, phosphorite by -24.7 per cent, and diamond by -27.3 per cent, read the release. These fluctuations reflect the dynamic nature of the mineral production sector, which relies on various factors, including market demand and resource availability. The growth in mineral production is not only significant for the sector itself but also holds broader implications for India's industrial and economic development. It highlights the sector's role in contributing to the nation's self-reliance and economic growth.

COKING COAL RATES PUSHING UP STEEL PRICES IN INDIA: JSP MD BIMLENDRA JHA

Steel prices in India are registering an upward trend due to "rapidly" increasing rates of key input material coking coal, industry executive Bimlendra Jha said. Coking coal and iron ore are the two main raw materials used to manufacture steel. While iron ore is available in substantial quantity in India, steel players are bound to meet 90 per cent of their coking coal requirement through imports from countries like Australia and South Africa. "Coking coal prices have increased rapidly (which are) currently trading at USD 341 per tonne CFR (cost and freight) India, from USD 230 a tonne in June-July 2023," Jha, Managing Director of Jindal Steel and Power (JSP), told.

The steel industry is facing an upward movement in prices because there has been a dramatic shift in coking coal prices, so the industry has no option but to pass on the cost to consumers, he said in reply to a question on increasing rates of steel in India. Speaking further, Jha said the market is also witnessing an uptick in steel demand, which is 7-8 per cent. After witnessing a downward trend, particularly in monsoon, there is an uptick in demand, he added. Steel is among the top three most widely used metals and any movement in its prices impacts the entire value chain.

JSW STEEL FIRM CARETTA MINERALS TO SELL ASSETS TO WEST VIRGINIA PROPERTIES FOR USD 24 MN

Caretta Minerals LLC, an asset of JSW Steel in the US, has entered into an agreement to sell its plant and equipment for a consideration of USD 24 million to West Virginia Properties. The deal also includes the sale of property of Caretta Minerals LLC and mineral rights to West Virginia Properties, JSW Steel said in a regulatory filing. "Caretta Minerals LLC has entered into an agreement on 20 September 2023 for selling its property, plant and equipment and mineral rights to West Virginia Properties for a consideration of USD 24 million as operating the mines is not economically viable in the absence of coal mining lease and plant lease," the filing said.

According to the filing, JSW Steel through its wholly-owned step-down subsidiary Periama Holding LLC owns coal mining assets in the state of West Virginia, USA. Caretta Minerals LLC, which is a step-down subsidiary of Periama Holding LLC, was operating coal mines and a preparation plant to beneficiate the coal mined through a leasing arrangement from Alawest Inc and West Virginia Properties (the lessors). However, pursuant to the termination of the coal mining and plant lease by the lessor, the company had recognised an impairment provision for all the investments (including loans extended) relating to these coal assets in the earlier years, JSW Steel said.

SHREE CEMENT'S RS 550-CRORE WEST BENGAL UNIT INAUGURATED

A Rs 550-crore grinding unit of Shree Cement Ltd was inaugurated by Chief Minister Mamata Banerjee in West Bengal's Purulia district on Monday. This is the first such unit of the company in the state, and has a capacity of 3 million metric tonne per annum (MTPA). It will create direct employment opportunities for 1,000 people. The state-of-the-art facility is strategically located to meet the surging demand for cement in West Bengal and neighbouring Jharkhand, and bolster the company's market share in eastern India, a Shree Cement Ltd official said. It is entirely funded through the group's internal accruals, and has already commenced commercial production, he said. Congratulating the company, the chief minister said that this substantial investment would strengthen the state's industrial prowess and underscore its growth potential in the cement sector. "We look forward to witnessing the positive impact it will have on our region's economy and employment opportunities," she said, virtually inaugurating the project from a real estate meet in Kolkata.

Shree Cement Ltd chairman HM Bangur said the company will set up another plant in West Bengal. "This venture not only generates employment opportunities for the region but also empowers us to enhance our manufacturing capabilities, efficiently meeting the burgeoning needs of the West Bengal market," he said. With the commencement of the production at the Purulia unit, Shree Cement's capacity rose to 50 MTPA, and the company said that it is on track to achieve a capacity of 80 MTPA in the coming years.

NHPC, APGENCO TO SET UP 2 HYDEL, RENEWABLE ENERGY PROJECTS IN ANDHRA PRADESH

NHPC has inked an initial pact with the Andhra Pradesh Power Generation Corporation to set up pumped storage hydropower and other renewable energy projects in Andhra Pradesh. In the first phase, the Memorandum of Understanding (MoU) envisages building of two pumped hydro storage projects having a total capacity of 1,950 MW, an official statement said. These projects will be established at Kamalapadu (950 MW) and Yaganti (1,000 MW). Once ready, these projects are expected to create significant employment opportunities and boost the local economy. The projects will be implemented as a joint venture.

NHPC has signed the MoU with Andhra Pradesh Power Generation Corporation (APGENCO) for implementation of pumped hydro storage projects and renewable energy projects in Andhra Pradesh, according to the statement. The MoU was signed at Tadepalli in Andhra Pradesh's Guntur on Wednesday in the presence of Chief Minister Y S Jagan Mohan Reddy. The agreement was signed by NHPC Director (Finance) R P Goyal and APGENCO Managing Director K V N Chakradhar Babu. The MoU seeks to achieve the national objective of generating clean and green energy of 500 GW from non-fossil fuel sources by 2030 and of realising the net-zero target by 2070. The pumped storage system utilises surplus grid power available from thermal power stations or other sources to pump up water from lower to upper reservoir and produces electricity during peak demand.

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Prospects and Constraints of Longwall Mining at Shallow Depth of Cover – A Case Study

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ABSTRACT

The article highlights the importance of increasing coal production in India to meet the growing demand for energy. While India has a significant reserve of coal, a large portion of it is located at shallow depths, making open-cast mining the dominant method. However, there is a need to strike a balance between open-cast and underground mining methods to optimize coal production. The article proposes adopting the mass production method known as Longwall (LW) technology for underground coal mining in shallow-depth areas. LW technology is globally recognized and has the potential to increase production and productivity while ensuring safety in the mining operations. To address the challenges associated with LW mining in India, the article considers various technical variables. These variables include strata control, subsidence problems, and other factors that may hinder the successful implementation of LW mining under shallow depths. By analysing and understanding these variables, the article aims to develop a generalized formula that can mitigate the adverse effects of these factors and create favourable working conditions.

Overall, the article emphasizes the importance of adopting advanced mining methods like LW technology to increase coal production in India. By considering technical variables and developing strategies to overcome challenges, the article aims to achieve higher production rates, improved productivity, and enhanced safety in underground coal mining operations.

INTRODUCTION

Coal is an important energy source of countries across the globe. In India 50 % of primary commercial energy demand is fulfilled by the coal. Apart from power generation in India, coal is used in the production of steels, gasification, liquefaction and for domestic use. India is the third largest coal producing country in the world. But the quantity of coal produced by the nation is not sufficient to meet the growing demand of the industries. Therefore, coal is imported to meet the increasing demand of the nation. India has the potential to meet the ongoing coal demand, but this could not be achieved due to lack of mechanization. Coal extraction in India is done either by Opencast or Underground method of mining. Coal seam at a higher depth is extracted by Underground method, whereas the coal seam at shallow depth of cover is extracted by Opencast working. Shallow depth here is not a fixed concept, it is a depth at which the fundamental element of the load can be worked safely. The forecasting for probable energy generating resources in India for the present century clearly indicates towards very high requirement of coal for the nation. This requirement will be fulfilled only by adopting highly mechanized methods of mining. Considering the Indian situation in which large reserve of coal seam is within shallow depth of cover where in general opencast mining is not feasible. But in such particular situation where opencast is not feasible either technically or economically there underground mining founds a place for its implementation. In the context of underground mechanization in coal mines we must appreciate the global acceptance of this Longwall mining technology. Longwall is preferred in Underground not only as a tool for higher productivity, but also for greatly reducing the accident rate due to ground movement in the waste and over the area of supported roof. Even though from Indian context the production from Longwall mining is very limited; nearly 2% (Naidu et al, 2020). The feasibility of extraction by Longwall method at shallow depth of cover was not considered earlier as the majority of the topmost seam in almost all the coalfields in India were developed by Bord and Pillar method. Moreover, the earlier experience of extraction by Longwall method in few mines with H/T ratio less than 10 (Where H is hard cover and T is extraction thickness) was not favorable as extraction could not be completed due to irregular caving even after induced blasting for surface was restored to (Mozumdar et al, 2015). Hence, we need to analyze thread bearing reasons for introduction of Longwall mining in Indian Underground coal mines and

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make attempts for gainful utilization of this world acclaimed technology in our country. Therefore, due consideration must be given to the problems to the problems which are coming as obstructions between the ambitious perspective of achieving higher productivity with tremendous safety potential by introducing en-masse Longwall technology in our country.

APPLICABILITY OF LONGWALL MINING

We would consider the following areas where Longwall mining can be adopted in large scale in our country.

i. Extraction of coal seam from 0.7-5.5m with single lift.
ii. But the most favorable thickness is 2.5-3.5 m.
iii. Gradient up to 20 degree (Jangara et al, 2021).
iv. For very deep and gassy seam Longwall is the right option.
v. There should be large blocks of coal without major geological disturbances.
vi. Roof condition should be stable but regularly cavable. If the roof condition is very hard, the support capacity requirement will very high.
vii. Normal geo-mining condition having large reserve of
viii. coal with substantial reserve of atleast 20- 25 Mt having no measure geological disturbance.

PROSPECTS OF LONGWALL UNDER SHALLOW DEPTH OF COVER

In Indian mining context though the use of Longwall mining is very much limited till date, it is extremely important to introduce mechanized mining methods. In general, and Longwall mining in particular for the survival of Indian coal mining industry as a whole. For this purpose, it is extremely essential to analyze and find out the particular situations in which Longwall mining Can be introduced most gainfully.

Considering the Indian situation in which large reserve of coal seam is within shallow depth of cover where in general opencast mining is not feasible. But in such particular situation where opencast is not feasible either technically or economically there underground mining founds a place for its implementation. We shall verify the possibility of Longwall mining at shallow depth of cover. We shall also consider the situation where opencast mining is already being done to extract upper seams but the below lying seams are not viable to be extracted further by opencast method.

In such case we can introduce Longwall to extract below lying seams from the floor of worked out opencast mines. This means that these properties may be under shallow depth of cover when considered the RL of the floor of worked out opencast mines.

PROBLEMS ASSOCIATED WITH LONGWALL TECHNOLOGY UNDER SHALLOW DEPTH OF COVER

i. In consideration of caving mechanics in Longwall mining, we know that the loads on support are mainly from active caving zone and fragmentation zone. In the normal condition, the total load of active caving zone and nearly 30 % load of the fragmentation zone is transmitted vertically on the support system. But when the Longwall mining is done under shallow depth of cover the fragmentation zone may enter into the unconsolidated strata near surface resulting in total dead weight of the strata, imparting the load on the support system. This will enormously increase the load on support system.

ii. So far as subsidence is concerned normally the angle of draw can be estimated through hard strata and the total impact of subsidence on surface can be well predicted but under shallow depth of cover there may be problem in prediction of subsidence span because of the presence of unconsolidated strata.

iii. At shallow depth of cover there is possibility of formation of pot holes directly from the surface requiring intensive surface management.

iv. Due to the presence of unconsolidated strata above the working seam, sometime pot holes, cracks and crevices make a direct connection with the surface, causing the Longwall panel to be prone to self-heating.

So, while adopting Longwall mining technology under shallow depth of cover, we should take all these matters into consideration and frame effective measure.

PAST EXPERIENCE IN JHANJRA PROJECT

One of the basic features of Longwall mining at Jhanjra is related with the working at such shallow depth of cover, as has rarely been conducted in any other country in the world. The first Longwall panel W-1 was worked with a minimum depth of 47m. In the subsequent panels, the depth of cover further decreased in W-4 and E-3 panels, it decreased to the bare minimum limit of 15m hard cover only, the shallowest in the world. Therefore, it can be rightly said that Jhanjra has experienced the ground control problem of Longwall mining at shallow depth of cover.

Before dealing with the past experiences at Jhanjra it is important to understand the caving mechanism and impact of low depth of cover on loading pattern of the support system. With the extraction at Longwall face, the overlying roof strata is disturbed in order of severity, there are three zones.

i. Caved zone
ii. Fractured zone
iii. Subsiding zone

The blocks of rock that caves in and fills up the goaf is the caved zone. The thickness of rock necessary for filling the goaf completely depends on the bulking factor of the roof rocks and this can be estimated by the formula.

$$T= h/K-1$$

Where,

h - Height of extraction.

K - Bulk factor and,

T- Thickness of caved zone

The bulk factor of rock varies greatly with the compressive strength of rock, Rock type, In-situ porosity, Particle size distribution, Rock Strength, Rock mass structure, Height of fall of individual blocks and type of strata lamination (Ofoegbu et al, 2008). For rocks with low to moderate compressive strength, the bulking factor is assumed to be 1.2, which means five times the height of extraction would be necessary to fill up the goaf completely. The laminated rock of high compressive strength and massive rock has been found to break in big pieces. Bulking factor of such rock may be around 1.1 to 1.15, which means that approximately seven to ten times the height of extraction would be needed to fill the goaf completely. The block of roof which caves and fill the goaf completely called as caving block (caving zone). Caving is not a continuous phenomenon, after considerable amount of extraction, the insitu stress acting on the void area keeps on increasing, leading to the sagging of roof rock. Further advancement or retreat of the Longwall panel causes caving of the roof strata behind the support system. A longitudinal section depicting the caving phenomenon of a Longwall Coal Face is shown in **Figure 1**.

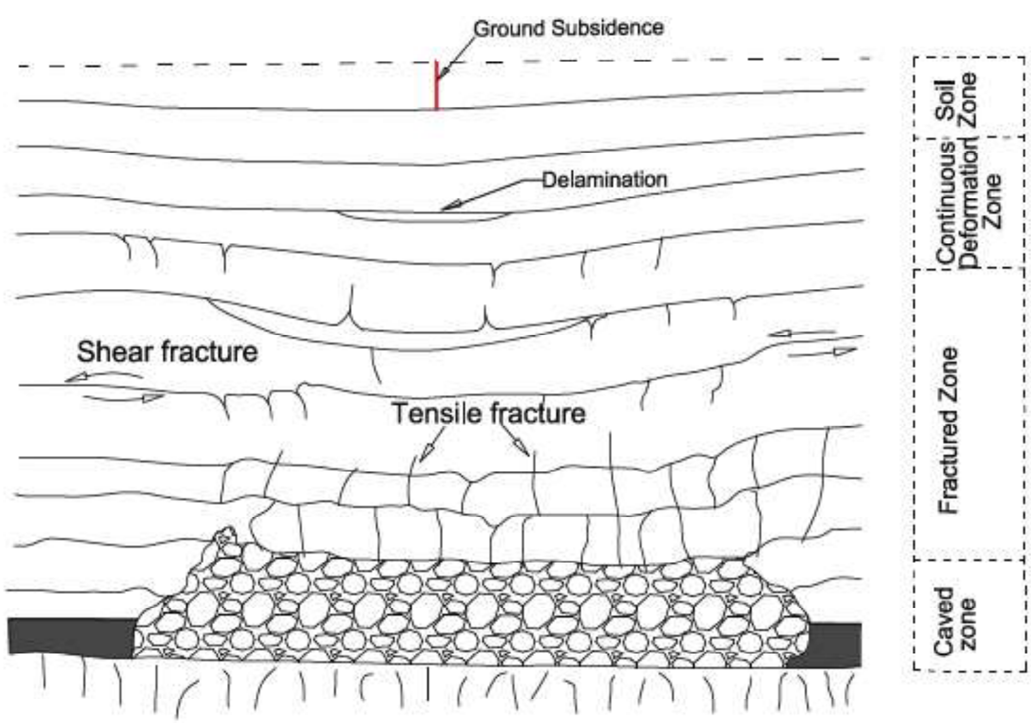

Figure 1 – Longitudinal section of Longwall Coal Face

The rock overlying caving zone settles on the caved goaf and during the process of settlement, it gets fractured as the goaf start getting compacted. Nature and type of fracture development in the fragmentation zone depends on the characteristics of rock mass and rock lithology. But the author's observation in a large number of locales decreasing from the periphery of the caving zone upwards and frustration pattern becomes less significant over 18 to 20 times the height of extraction. Above the fractured zone is subsiding zone, the rocks which are not fractured extensively, but subsides gradually on the fractured block. In laminated strata, the caving block mainly determines the load coming on the support. However, in presence of massive strata over the caving block, it may also impose additional load on the support system. In case, there is some source of additional load like remnant pillar or loose alluvium within the confine of caving zone or fractured zone, there is every likelihood of its additional load on the support. In case, the source of additional load is above the subsiding zone, it may not have significant effect on load condition at the face and support.

ACHIEVEMENT OF JHANJRA PROJECT

The achievement of Jhanjra project in successful introducing and dealing with Longwall method of mining relates to already extracted ten Longwall panels in R-VII seam (top most seam of Jhanjra block).

Geo-Mining Condition Of R-VII Seam

i. Mineable reserve of coal in the seam (sec A) – 800 MT
ii. Thickness of the seam - 1.27 to 4.23
iii. Height of extraction - 3.1 to 3.8 m
iv. Depth range from surface to seam - 42 m to 60 m (Average = 53 m)
v. Gradient of the seam - 1in 10
vi. Length of panel - 900 m
vii. Development of panel along strike
viii. Thickness of alluvial soil - 6 m
ix. Thickness of hard cover above the seam - 15 to 40 m
x. Maximum compressive strength of the bed overlying strata – 386 kg/m^2
xi. Caving index of the strongest bed - 2426

Specification of Russian supports (KM130 COMPLEX) used in majority number of panels in R - VII seam

i. Model M130 (2 Legged sections)
ii. Type - Free base chock with single telescopic leg
iii. Open height - 4.14 m
iv. Closed height - 2.5 m
v. Yield load of legs - 2×314 m
vi. Support resistance - 51- 58 T/m^2

Table 1.: Reduction of compressive strength under influence of water

`	Dry compressive strength	Wet compressive Strength (after 3 days soaking)	Reduction percentage (%)
Medium grained sandstone	334	263	15.0
Fine grained sandstone	543	476	12.3
Fine grained sandstone	764	674	12.0
Medium grained sandstone	251	128	49.0

CASE STUDY

Extraction of Longwall panel in R-VII seam, below VII seam and ongoing experience of strata control problem: -

With the exhaustion of R-VII seam in sector-A, extraction of coal by Longwall method has been introduced in the under lying R-VII A seam. The first Longwall panel in this seam is AW1, which lies below the goaved out area of W1 panel in overlying R-VII seam.

Position wise, W1 and AW1, are slightly staged and length of AW1 panel being 43 m more than the W1 panel, the installation chamber and first few meters (about 38m) of

AW1 panel are below solid coal barrier. We have tried to highlight the present status of working of Longwall panel under shallow depth of cover.

Characteristics of AW-1 Longwall Panel

Name of Panel	AW-1 (bellows the goaf of W-1 panel of R-VII seam)
Length of the Panel	850 m
Width of the Panel	120 m
Name of the Seam	R-VIIA
Depth of the Seam	100.16M asd per BH No AW1/99
Thickness of the Seam	2.80 m to 4.26 m
Parting between R-VII & R-VIIA	44 m
Existence of overlying or underlying worked out area.	Above lying W-1 panel was extracted by Longwall caving method. A A maximum subsiden of 2.10 m was observed over W1 panel. Below lying R-VI seam is virgin.

Extraction of AW – 1 Longwall panel and manifestation of strata control problem in this panel

The extraction of AW-1 panel started on 6th June 1999 gullick support (ex-Dhemomain) was deployed at the face. The support resistance being marginal for 2.4 m height of extraction is (86 T/m^2), DGMS permitted for adhering strictly to 2.4 m extraction height, leaving the rest of coal in the goaf. As apprehended by CIMFR (erstwhile CMRI) study, the first fall fill the goaf occurred after 47.5 m of progress and the first main fall occurred after the face advanced by 80.5 m from the installation chamber. After 135m of face progress, there was an occurrence of fire/ heating in the goaf, resulting in the stoppage of the face for considerable period (about 6 month). The face restarted after dealing with fire on March,2000 and the subsequent rate of progress of the Longwall face is very high. Within three months, the face progressed from 135 m to 435 m (i.e. 300 m progress) without any manifestation of strata problem during the entire period, fall in the goaf was regular. Load on the chock were nominal and as such there was no cavity formation in the front of the canopies at face. But in the 3rd shift on 16-06-2000 (morning on 17-06-2000) at a face position 435 m from the installation chamber there was an occurrence of severe weighting on the chocks, resulting in: -

i. Convergence of chocks in different degrees ranging between powered support (PS) No17-66, with chocks between PS no 33-50 (mid zone) being affected the maximum.
ii. Formation of cavity in front of canopy measuring 1.3 m to1.8 m in height and about 1 m in web
iii. 7 numbers of canopies were badly damaged and 36 legs were bend, requiring their replacement.

It took tireless effort for some days to recover the face started operating from 29-06-2000.

Some observation made and searching for genesis of the strata control problem at AW-1 face

In trying to figure out the genesis of the problem certain observations have been made and some concepts have been developed.

i. As per CIMFR (erstwhile CMRI) report for demomain equipment for AW-1 rated support resistance is marginal at 10,000 cycles for this support. Further the support capacity was enhanced by 10% with the higher capacity bleed valves & rapid yield valves by this manufacturer.

ii. The immediate roof (about 7 m above the coal seam) is moderately cavable (as per data from different boreholes) followed by the strong bed of higher caving index and high RQD, which has dominating influence on the load over the supports.

iii. Though the immediate roof up to 4 m was blasted down in the goaf, but over laying strata is breaking in blocks causing serve loading on chocks gradually.

iv. The phenomenon of first severe weighting occurred at 473 m of face progress, this region is characterized by a geological inconsistency in the form of swelley.

v. Every subsequent periodic weighting at an interval of 19 m to 23 m was associated with the phenomenon of chocks closure in certain zones of the face, though the degree of severity was varying as stated above.

vi. All the damages of the chocks as could be seen was due to lateral load created cantilever formation in the roof.

Future prospect of Jhanjra

Jhanjra block has immense mineable reserve and the grade of coal is high in the seams lying at greater depth. Therefore, potentiality of this project is very high, this requires successful liquidation of the present working seam (R-7th A seam) before entering the seams with higher reserve and higher-grade coal. The initial data collected from the bore holes up to R – 6th seam refers to massiveness of the overlying strata and higher support resistance requirement. So, it is important to develop the proper method of support resistance calculation and also find out the method for hard roof management. Potentiality of Jhanjra project is shown in **Table 2**.

Table 2.: Potentiality of Jhanjra block can be seen from the following table

Seam	Thickness (m)	Parting	Extr. Res. (MT)	Coal grade
R-7th	1.27-4.23	20-47	8	D-E
R-7thA	0.45-4.11	57-93	6	D-E
R-6th	2.15-5.60	15-60	38	B-C
R-5th	2.90-6.64	40-45	46	B-C
R-4th	3.75-11.27	26-44	65	B-C
R-3rd	2.25-4.60	0-35	23	B-C
R-2nd	1.00-2.41	40-60	9	C
R-2nd /3rd	3.45-5.53	-	17	B-C

FACTORS AFFECTING THE EFFICIENT & ECONOMIC USE OF HIGH MECHANIZATION IN INDIA UNDERGROUND MINES

i. Improper planning leading to selection of wrong technology/wrong place of application.

ii. Lack of proper infrastructural facility.

iii. Lack of proper training facilities, failure to generate required skill

iv. Policy failure to develop proper cadre scheme suitable for high mechanization.

v. Apathy in developing proper indigenous spares and equipment for gradual substitution of imported items.

vi. Improper utilization of fund, leading to ultimate dearth of fund.

vii. Lack of Research & Development facilities, resulting in insufficient study of rock characteristics and load distribution design, leading to strata control problem in several faces.

ACTION TAKEN BY JHANJRA FOR SUCCESSFULLY INTRODUCING MECHANIZATION

i. Upgradation of information system

ii. Development of planning process and skills.

iii. Improvement in management technique and skill

iv. Improvement of resources utilization.

v. Improvement in rate of development and production

vi. Improvement in procedure for procurement of stores asst. management

vii. Development of operational plan and introduction of 5 years rolling plan

viii. Development of business plan for Jhanjra.

ix. 18 months operational plan of Jhanjra project by Microsoft project

x. Production program schedule

xi. Longwall production schedule

xii. Development program

CALCULATION OF LOADING PATTERN ON SUPPORT SYSTEM

For calculation of loading pattern on support system we require to calculate the load due to active caving zone.

Nature of intermediate roof

i. Sandstone and intercalation of shale and sandstone with average density (d) of 2.5 T/m3.
ii. Average swelling factor (K) of the roof with fragmentation is 1.3
iii. The roof is easily cavable with regular periodic fall at interval of 13 - 15 m.

Load acting on support system due to active caving zone

i. Calculation of load due to active caving zone when H/T e” 10.
ii. Height of extraction (h) = 2.8 m

Height of active caving zone, H(ac) = (h/K-1)
= 2.8/ (1.3-1)
= 9.3 m

Load due to active caving zone = {H(ac) ×d} T/m^2
= 9.3 × 2.5
= 23.25 T/m^2

Above the active caving zone there is the fragmentation zone which is divided in 2 layers.

i. Lower main roof.
ii. Upper main roof.

Thickness of hard cover above coal seam = 47 m

Considering 10 times of extraction height as lower part of main roof, we may consider as 30% of load of this zone would affect the loading pattern on support system. Remaining load due to movement of the main roof strata would be transmitted horizontally.

Load due to fragmentation zone (fg) = 10 × (h)
= 10 × 2.8
= 28 m

Therefore, total load of fragmentation zone on the support system
= 10 (h) × Density × 0.3 (30% load of this zone)
= 28 × 2.5 × 0.3
= 21 T/m2.

Total load acting on support system = (Load due to active caving zone **+** Load due to fragmentation zone)
= 23 T/m^2 +21 T/m^2
= 44 T/m ^{2}

Support system used in this panel is of 51 - 53 T/m^2 capacity. So, it is clear that the load coming on the supports is less than the support capacity and hence no strata control problem occur in this panel while extraction.

EXPERIENCE IN SUBSEQUENT PANEL

After successful completion of the first Longwall panel at shallow depth of cover the subsequent panels were worked at lower depth of hard cover. From the experience of working at one of the panels were H/T ratio was 8 severe strata control problem was observed and hence failure. In the subsequent panels the av. depth of hard cover above the coal seam ranged between 40 m to below and at about 170 m from the barrier the depth of hard cover was as low as 26 m. The panel started with satisfactorily result but with the progress of work due to lower depth of hard cover the problem started cropping up and at the location with 26 m hard cover the normal working in the panel was totally jeopardized.

Calculation of Load at 26 m Hard Cover by maintaining extraction Height of 2.8 m

Height of extraction (h)= 2.8 m
Geo-mining condition is same
Hard cover above the coal seam = 26 m
Alluvium height = 6 m
Height of active caving zone, H(ac) = h/K-1
= 2.8/1.3-1
= 9.3 m

Av. density of roof rock (d) = 2.5 T/m3
Load due to active caving zone = {H(ac) ×d} T/m^2
= 9.3 × 2.5
= 23.25 T/m^2

Height of lower main roof is 10 times the height of extraction
= 10 × 2.8
= 28 m

In this case, the thickness of hard cover above extraction height is only 26 m due to which load acting on the support system will be increased tremendously. Here, the horizontally transmitted load is between 45 - 50 % due to absence of hard cover above the seam.

Load due to 26 m hard cover = 26 × 2.5 (S.G) × 0.5 (50 % load of this zone) = 32.5 T/m^2

After larger extraction of coal seam, void is formed behind the powered support system. The in-situ stresses acting on that area due to coal extraction are disturbed, allowing the lower main roof to cave in to distribute the unexpected loads. The cavable height needed to fill the goaf formed behind support depends on the rock swell factor. In this geo mining condition, the cavable height of lower main roof needed to fill the goaf is 28 m. But, the actual thickness of hard cover is only 26 m which means the lower 2 m alluvium is within the lower main roof. Alluvium is loosely bonded they are easily cavable. Once the hard cover will subside soon the alluvium just lying above it will start subsiding. Alluvium don't break in fragments, once the hard cover will subside, the entire alluvial layer of 6 m thickness will disintegrate and sink. Due to the subsidence of the alluvial layer, the load on the support system will increase significantly, due to which the support system will fail.

Av. density of dry gravel = 1.6 T/m3
Av. density of wet gravel = 1.8 T/m3
Thickness of alluvium = 6 m
Load due to alluvium part on support = 6 ×1.6
= 9.6 T/m^2

Total load acting on support system is sum total of = Load of active caving zone + Load of lower main roof + Load of alluvium
= (23.25 + 32.5 + 9.6) T/m^2
= 65.35 T/m^2

Total load acting on the support system was 65.35 T/m^2 but the Support system used in this panel was of 54 T/m^2 capacity. So, it is clear that the load coming on the supports system is greater than the capacity of support, causing failure of the support system.

Nature of problems encountered

i. Severe weighting was experienced on the chocks.
ii. The chocks on the mid zone almost collapsed and the convergence of the chocks were severe by applying and the situation was controlled by functioning of rapid yield valves
iii. Severe lateral pressure was encountered on large no. of chocks which practically lean forward on the face
iv. Cavity was formed in front of face and the cavity height was (4-6) m
v. All this phenomenon indicated severe load on the support system which was beyond the capacity of powered support density.

DEALING WITH THIS EMERGENCY SITUATION

i. To cope with this situation small girders were placed in the gap between the support which helped to build up additional resistance of the support system and also prevented the falling of stone blocks at the face.
ii. Height of extraction was reduced and thereby decreasing the required support resistance in this zone for this purpose the height of extraction was maintained.
iii. In place of bidyke method of cutting monodyke system was adopted to minimize the roof exposure area.
iv. With quick movement salvaging chamber was formed and the Longwall equipment set was withdrawn keeping the balance panel length unextracted.

DERIVED FORMULA

Before taking decision for deploying specific capacity of powered support, there is need to calculate the load acting on the powered support on the basis of available data.

Calculation of Load on Support System

When H/T ratio >10

i. Height of active caving zone (H_{acz})
h = Height of Extraction
k = Bulking Factor
d = Average Density of Roof Rock
H_{acz}= h/k-1 .... **I**

ii. Load due to active caving zone = (H_{acz}×d) **T/m²** = h×d/(k-1) **T/m²** ....**II**

iii. Load due to fragmentation zone = 10×h×d×0.3
Total load on the powered support to be decided

Load on support = h×d/(k-1) + 3×h×d

K = say (1.1,1.2,1.3)
= h×d [1/(k-1) +3]
= h×d {1+(3k-3)/(k-1)}
= [(3k-2) × h × d/k-1] **T/m²** ....**III**

FOS considered = 1.2
Support capacity = 1.2×Load on support
= [1.2×(3k-2) ×h×d/k-1] **T/m²** ....**IV**

DERIVATION AND FORMULA WHEN H/T < 10

i. Load due to active caving zone =
H_{acz} = h×d/(k-1) **T/m²** ....**V**

ii. Load due to fragmentation zone = 10×h×d×0.4

When H/T ratio is less than 10 the fragmentation zone reaches up to the surface due to which load acting on support is increased tremendously i.e. why we are considering here 40% of load with low hard cover the figure even may changes to 60-70 %.

iii. Load due to unconsolidated strata when H/T is <10 100% of load of alluvium will act on the support

= Height of unconsolidated strata(H) × av. Density $(D)_{\text{unconsolidated strata}}$
Load acting on support = h×d/k-1 + 10×h×d×0.4 + $(H\times D)_{\text{unconsolidated strata}}$ **T/m²** ....**VI**

SUGGESTION

For the successful implementation of Longwall at shallow depth of cover following measures are to be considered during planning and working of the seam.The data obtained from different bore holes describing the lithology of overlying strata showing variation in lithology within the active caving zone and fragmentation zone. These data should be carefully studied in the lab for determination of physio-mechanical properties of the hard cover above the coal seam. So, we should consider the following variables:

i. Height of extraction

ii. Bulking factor – It is the ratio of volume of rock on breaking with its volume in situ. The empirical relationship used to estimate bulk factor

K-1 = $(n/R_c)^{1/3}$ ....**VII**

Where, K= Bulk factor
N = Number of homogeneous strata layers
R_c = Uniaxial compressive strength.

iii. Mean rock density of hard rock strata
iv. Average thickness of unconsolidated strata
v. Thickness of unconsolidated strata if existed with in fragmentation zone.
vi. Minimum thickness of hard cover above the coal seam.
vii. Rock quality designation

viii. Compressive Strength - The average compressive strength of rock in the active caving zone varies widely between 250 to 1000 kg/cm² in Raniganj measures of Raniganj coalfield.

ix. Cavability - No theoretical approach has been found to be reliable to assess the cavability and estimate the support requirement at a Longwall face. Almost every major coal producing country has got their own empirical norms which presumably may be valid and may give good results under the particular geomining conditions in which it has been developed.

Cavability of overlying strata depends on weak parting layers, laminated nature and massiveness of the roof rock formation (Banerjee et al, 2016). The cavability characteristic of a bed is designated by "Cavability Index" which has been expressed as a function of the parameters mentioned above. The index has been correlated with the span at which first major weight is expected to take place and subsequent interval between periodic weightings (caving interval).

Projection of Caving Behaviour

Singh in his research paper (Conventional approachs for assessment of caving behaviour and support requirement with regardto strata control experiences in Longwall tecnologies) quantified the caving behavior of a rock by: (Singh, 2015)
The cavability index may be expressed as: -

$I = [\sigma L^n t^{0.5}]/5$ ....**VIII**

Where, σ - Compressive Strength

L - Average Core Length
t - Thickness of bed

n - Varies between 1.1 to 1.3 depending on massiveness of the bed as quantified by RQD

On the basis of the cavability index, the pattern of caving may be projected. the span (Sa) at which the bed breaks for the first time is given by relationship:

$Sa = 0.72\ I^{.51}$

Where Sa, is the span of first breakage.

ACTIONS TAKEN FOR SUCCESSFULLY INTRODUCING LONGWALL MINING AT SHALLOW DEPTH

i. Borehole extensometer have been installed to correlate the loading pattern with movement of overlying strata.

ii. Systematic subsidence measurement is being organized as to correlate the phenomenon of weighting with subsidence

iii. The use of double drum cutting can indeed help enhance immediate support and increase support resistance. In underground mining, immediate roof support is crucial to ensure the safety and stability of the working area.

iv. Geological mapping is being done to identify the zones with varied lithological characteristic.

v. Development of planning process and skills

vi. Improvement of resource utilization.

CONCLUSION

Deriving the minimum depth of cover for effective and safe Longwall operation is an important aspect of this research. The research paper focuses on identifying the critical variables that impact the feasibility and safety of Longwall mining in shallow depths. By considering these variables, the paper aims to provide suggestions and pave the way for further development in this area based on factual data. The Longwall mining method is indeed widely recognized as an efficient technique for extracting coal from deeper deposits. However, when considering its application in shallower depths in India, it is important to critically evaluate the strata behavior and adapt the method to suit the specific geo-mining conditions of the region. This evaluation can be carried out through various approaches and experimental trials. Additionally, incorporating innovative strata control monitoring techniques and strata behavior monitoring methods can further enhance the safety and effectiveness of Longwall mining in shallow depths.

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An Overview of Current Statutory Regime for Blasting in Opencast Coal Mines in India

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ABSTRACT

Blasting technology with extensive use of explosives is one of the most popular and preferred means of rock disintegration in commercial mining. During the last few decades, coal mining in India has undergone a paradigm shift from underground to opencast mining technology, resulting in manifold enhancement in coal and overburden production and hence also a sharp jump in the consumption of explosives. Use of bulk explosives has become a common affair and there has been adoption of several state-of-the-art practices in blasting. The mining safety legislation has also been updated from time to time to keep pace with the changing scenario in technology and the quantum of excavation. This paper aims to give an overview of the current blasting related statutory obligations on safety in Indian opencast mines, especially the coal mines.

INTRODUCTION

Minerals generally occur in nature in the form of ores, which are predominantly and typically entrapped within the surface of mother earth as intact rock. Extraction of any ore, therefore, involves extensive excavation work not only to make access up to the seat of the ore body but also to dislodge the ore from its original position. Adoption of a systematic methodology of rock disintegration thus acquires significant importance for accomplishing the mining operation in a commercially viable manner. On the one hand, blasting technology using high explosives remains one of the most popular and preferred means of rock disintegration. However, there are certain blastless technologies used in commercial mining. In the Indian coal mining industry, the last few decades have witnessed a paradigm shift from underground to opencast mining technology. The volume of coal production and overburden have also enhanced manifold. The mining safety legislation has also been updated from time to time to keep pace with the changing scenario in technology and quantum. Therefore, a better understanding of all the applicable statutory requirement is essential for achieving an improved level of compliance.

REGULATION OF BLASTING IN OC COAL MINES UNDER THE OLD REGIME OF SAFETY LEGISLATIONS

Prior to 27th November, 2017 the coal mines were regulated by the Coal Mines Regulations, 1957. The old regime of safety legislations, which had come into force when the mining was predominantly by underground technology, was devoid of adequate provisions for regulating opencast mines and most of the aspects of blasting, such as deep hole blasting, use of site mixed slurry/emulsion explosives, use of ANFO, blasting within danger zone, sleeping of charged holes, were governed through the mechanism of permissions granted by DGMS.

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CURRENT SAFETY LEGISLATIONS GOVERNING BLASTING IN OC COAL MINES

The Coal Mines Regulations, 2017 (CMR, 2017), inter alia, addressed most of the issues related to opencast mining, including blasting in opencast coal mines. Many practices and methodologies adopted in blasting which over the time had become common in opencast mining have been properly covered in the regulation. Apart from the general provisions pertaining to drilling, explosives and blasting, appropriate provisions for the subjects specific to blasting in opencast mines, such as deep hole blasting, transport of explosives in bulk, use of Ammonium Nitrate Fuel Oil (ANFO), use of Site Mixed Slurry (SMS) / Site Mixed Emulsion (SME), sleeping holes, blasting in hot strata, etc. have been made in the Coal Mines Regulations, 2017 and the General Orders issued thereunder by the Chief Inspector by notification in the Official Gazette.

Deep Hole Blasting

Unlike the previous statutory framework, presently no permission is required to be obtained for conducting deep hole blasting in the mine. Conditions for conduct of deep hole blasting in opencast coal mines are prescribed under Regulation 194 of the CMR, 2017 and the notification no. GSR 985(E) published in the Gazette of India on 4th

October, 2018, wherein detail guidelines for drilling, charging, stemming and firing have been stipulated in addition to the general guidelines. The general guidelines for conducting deep hole blasting are mentioned below:

(1) A prior scientific study in designing deep hole drilling and blasting system shall be conducted for the mine taking into consideration all mining parameters, machinery parameters and long-term stability of pit and dump slope.

(2) The Owner and Agent of the mine where deep hole drilling and blasting is proposed shall indemnify the occupants/owners of houses/dwellings/buildings or other structures and the public authority concerned, if any, against danger to their property or injury to them or other persons, arising out of the operations conducted in respect of deep hole drilling and blasting.

(3) The Owner, Agent and Manager of the mine where deep hole drilling and blasting is proposed shall ensure that-

a) Only properly trained persons who are authorised in writing by the manager for the purpose are deployed.

b) Adequate personal protective equipments as required to be used by persons deployed in this connection are provided and also used.

c) Adequate scientific equipments for measurement of the health of machinery deployed, ground movements and various blasting parameters like ground vibrations, noise, etc., are provided.

(4) Entire operations connected with deep hole drilling and blasting in the mine shall be placed under the overall charge of a “blasting officer” holding first class manager’s certificate of competency, assisted by adequate number of persons holding at least overman’s certificate.

(5) The mines manager, in consultation with OEM and explosive manufacturers, shall formulate suitable “Code of Practice” in respect of drilling, charging, stemming, warning of persons, taking shelter and firing of shots.

Transport of Explosives in Bulk

The conditions and other details for transport of explosives in bulk for blasting in opencast coal mines are prescribed under Regulation 188 of the CMR, 2017 and the notification no. GSR 982(E) published in the Gazette of India on 4th October, 2018. Following precautions are required to be taken where transport of explosives in bulk is proposed:

(1) Only trained and authorised persons shall be deployed.

(2) The transportation shall be placed under the overall charge of a competent person holding at least an Overman’s certificate of competency.

(3) Transport of explosives in bulk to the priming station or the site of blasting shall be done only during day light hours.

(4) The quantity of explosives to be transported in bulk at one time to the site of blasting shall not exceed the actual quantity required for use in one round of shots, and also not before 30 minutes of the commencement of charging of holes.

(5) Only a vehicle duly approved by the Competent Authority shall be used for transport of explosives in bulk.

(6) All conditions stipulated by the licensing authority in respect of the vehicle deployed for transportation and handling of explosives in bulk shall be strictly followed.

(7) Such vehicle shall be in safe operating condition and should be driven by competent licensed drivers duly authorised by the Manager.

(8) At least two fire extinguishers of suitable size and capable of fighting electrical and petroleum fires shall be provided in each vehicle in an easily accessible position and maintained in a state of readiness.

(9) Before transporting explosives in bulk, the competent person authorised in this regard shall personally search every person engaged in the transport and use of explosives and shall satisfy himself that no such person has in his possession any cigar, cigarette, biri, or other smoking material or any match or any other apparatus like mobile phone etc., of any kind capable of producing a light, flame or spark.

(10) During transport of explosives in bulk, the vehicle transporting explosives shall be -

- properly earthed with chain links while loading.
- well locked except during times of placement and removal of stocks of explosives.
- kept in isolated places while loaded.

(11) The vehicle loaded with explosives shall not be –

- left un-attended.
- taken into garage or repair shop and shall not be parked in a congested place.

(12) The vehicle transporting explosives shall not be –

- overloaded.
- driven at a speed exceeding 25 kilometers per hour.
- refueled except in emergencies; even then it’s engine shall be stopped and other precautions taken to prevent accidents.

(13) Every vehicle used for the transport of explosives in bulk shall be carefully inspected once in every 24 hours by a competent person, to ensure that –
- fire extinguishers are filled and are in place
- the electric wiring is well insulated and firmly secured
- the chassis, engine and body are clean and free from surplus oil and grease
- the fuel tank and feed lines are not leaking and
- lights, brakes and steering mechanism are in good working order

(14) A report of every inspection shall be recorded in a bound paged book kept for the purpose and shall be signed and dated by the competent person making the inspection.

(15) The mine manager shall frame a suitable code of practice for handling and transportation of explosives in bulk.

Use of Ammonium Nitrate Fuel Oil (ANFO)

Regulation 183(2) of the CMR, 2017 enables use of ANFO in opencast coal mines without obtaining any permission under the CMR, 2017. The conditions for use of ANFO in coal mines have been prescribed under regulation 193 and the notification no. GSR 984(E) published in the Gazette of India on 4th October, 2018. It covers elaborate guidelines regarding storage and handling of ANFO, mixing or impregnating of Ammonium Nitrate with diesel oil, charging and firing, misfires, fighting fires involving explosives, hazard of static electricity and supervision during the use of ANFO in mines.

Use of Site Mixed Slurry (SMS)/Emulsion (SME) Explosives

Like ANFO, regulation 183(2) of the CMR, 2017 enables use of SMS/SME explosives in opencast coal mines without obtaining any permission. Provisions stipulated for deep hole blasting under regulation 194 coupled with the guidelines prescribed under the notification no. GSR 985(E) published in the Gazette of India on 4th October, 2018 should be followed during use of SMS/SME. The above notification and also the regulation 110 requires the mines manager to formulate suitable "Code of Practice" in consultation with OEM and explosive manufacturers, in respect of drilling, charging, stemming, warning of persons, taking shelter and firing of shots.

Sleeping Holes

In mining industry, the terminology "sleeping holes" is used for the holes charged on any one day but not fired on the same day for whatsoever reason. Earlier when the charging process was mostly manual, the practice of using sleeping holes was needed more while conducting large blasts. However, presently on introduction of bulk use of explosives with mechanised means, the charging process have become very fast. Hence, to eliminate the associated hazards this practice should be avoided as far as practicable.

The guidelines and precautions for sleeping holes have been stipulated in notification no. GSR 985(E) published in the Gazette of India on 4th October, 2018, under Regulation 194 of the Coal Mines Regulations (CMR), 2017. As per the above guidelines all holes charged on any one day shall be fired on the same day. However, in case of specific problems which may lead to the charged holes to sleep over night, the following conditions shall be strictly complied with:

(1) The shot holes charged with explosives in coal faces and the overburden bench immediately above coal seam SHALL NOT be kept sleeping and shall be blasted off on the day of charging.

(2) The total duration of sleeping of holes with explosives shall not exceed the hours as designated by the Manager in writing after consultations with the manufacturer of the explosive(s).

(3) Elaborate standing orders shall be formulated by the Manager on the various precautions to be taken during the sleeping of shot holes. The standing orders shall clearly spell out the responsibilities of various officials and supervisors including the Blasting Officer of the mine in maintaining the sleeping shot holes in safe condition.

(4) Before deciding to allow shot holes on sleeping at any place in the mine other than the coal benches and the immediate overburden bench over the coal seams, following precautions shall be taken:
- Care shall be taken to ensure that there is no heating of strata anywhere along the depth of the shot holes.
- Details of bottom hole temperatures of all shot holes proposed to be charged with explosives and later kept sleeping shall be recorded.
- For regularly monitoring the bottom hole temperature for any increase, a few pilot holes shall be left uncharged.
- If the bottom hole temperature shows any sign of abnormal increase, then immediate steps shall

be taken to blast all the shot holes at the earliest.
- The details of all such abnormal increase of temperature, shall be recorded in the register separately kept for the purpose and also signed and dated by the blasting officer and the Manager.

(5) The area where the shot holes are kept sleeping with charged explosives shall be conspicuously marked by such an arrangement which could be clearly visible during any time of the day or the night.

(6) No person shall be allowed to be present anywhere within 100 m of the shot holes sleeping with explosives. The entrance to such areas shall be effectively cordoned off to prevent any inadvertent and unauthorized entry.

(7) The fuse wire emanating out of the shot hole charged with explosives shall be covered with any incombustible material so as to prevent any accidental ignition due to the influence of static electricity form lighting etc.

(8) Adequate arrangement of water under high pressure shall be made available near the area of sleeping of shot holes for dealing with any exigencies, including flushing out of charged explosives from the shot holes and for de-sensitizing the explosives in the charged shot holes.

Blasting in Fire Area

Regulation 202 of the CMR, 2017 and general order issued thereunder by notification no. GSR 986(E) published in the Gazette of India on 4th October, 2018, lay out detail guidelines on precautions to be taken for blasting in fire areas in opencast coal mines. Following are the general guidelines prescribed for working by opencast method in the fire area:

(1) Blasting operations shall be carried out under direct supervision of an Assistant manager in charge of blasting operations.

(2) Persons trained in the job of blasting in Fire area only shall be deputed for the said operation.

(3) No explosive other than slurry or emulsion explosives shall be used.

(4) Blasting shall be done with detonating fuse down the hole.

(5) Temperature inside the blast holes shall be measured (before filling with water) and if the temperature exceeds 80°C, in any hole, such hole shall not be charged. Records of measurement of temperature in each hole shall be maintained in a bound paged book and shall be signed by the Assistant manager incharge of blasting operations and countersigned by manager.

(6) All blast holes shall be kept filled with water. When any hole is traversed by cracks or fissures, such hole shall not be charged unless it is lined with an asbestos pipe and the hole filled with water. In addition, bentonite or any other effective material shall be used for sealing any cracks at the bottom of the holes.

(7) Detonating fuse shall not be laid on hot ground without taking suitable precautions which will prevent it from coming in contact with hot strata.

(8) Hottest holes shall be loaded last. Uncharged holes shall be filled with water/ sandy material.

(9) Carbonaceous material shall not be used for stemming.

(10) The charging and firing of the holes in any one round shall be completed expeditiously and in any case within 02 hours.

(11) Regular monitoring of Carbon Monoxide (CO) shall be done by a competent person authorized by the manager, during charging of the holes. If CO is more than 50 ppm, all persons from the area shall be withdrawn.

(12) Water spraying/quenching arrangements shall be kept available at the blasting site to deal in case of emergency.

(13) Precautions while drilling in Overburden or Coal over the underground workings in Opencast Mines.

The fire in opencast workings of coal mines may be due to the existence of previously developed or mined-out belowground workings, for which special precautions have been stipulated in addition to the above.

Blasting Within Danger Zone of Permanent Surface Structures

Existence of permanent surface structures in the vicinity of opencast mines is a very common feature. As per regulation 196 of CMR, 2017, the area of mines lying within a radial distance of 500 m from such structures has been defined as blasting danger zone.

Where any part of a public road or railway lies within the danger zone, the shot-firer shall not charge or fire a shot unless two persons are posted, one each in either direction at the two extreme points of such road or railway which fall within the danger zone who have by an efficient system of communication or hooter intimated clearance of traffic to the shotfirer and have also warned the passersby and vehicles, if any, which have passed by such road or railway.

Further, in case of an opencast working, where any permanent building or structure not belonging to the owner of the mine lies within the danger zone, the aggregate maximum charge per delay and per round shall not exceed the amount fixed by the DGMS, by a permission granted on the basis of a scientific study. The purpose of restricting the aggregate maximum charge is to basically control the blast-induced ground vibration and fly rocks to within the prescribed acceptable/permissible limits which may be incapable of damaging the structures. The blast-induced ground vibration is measured terms of peak particle velocity (ppv) in mm/s. Depending on the type of structures and dominant excitation frequency, the maximum permissible peak particle velocity (ppv) at the foundation level of structures have been prescribed under DGMS (Tech.) Circular No. 7 of 1997, which is given in the table below:

Table: Permissible Peak Particle Velocity (ppv) at the foundation level of structures in mining area in mm/s

Type of Structure		Dominant excitation frequency, Hz		
		<8Hz	8-25Hz	>25Hz
(A)	**Buildings/structures not belonging to the owner**			
(i)	Domestic houses/ structures (Kutcha, Brick & Cement)	5	10	15
(ii)	Industrial Buildings (R.C.C. & Framed Structures)	10	20	25
(iii)	Objects of historical importance and sensitive structures	2	5	10
(B)	(B) **Buildings belonging to owner with limited span of life**			
(i)	Domestic houses/ structures (Kutcha, Brick & Cement)	10	15	25
(ii)	Industrial Buildings (R.C.C. & Framed Structures)	15	25	50

General Precautions Regarding Drilling, Explosives and Blasting

As per regulation 216(3) of CMR, 2017, truck mounted drill machines designed for tube well drilling for sources of water shall not be used and only proper type of blast hole drill machine, especially designed for mining purpose, shall be used in the mine.

General precautions regarding explosives have been laid down under regulation 207 of CMR, 2017, which are as follows:

(1) No person, whilst handling explosives or engaged or assisting in the preparation of charges or in the charging of holes, shall smoke or carry or use a mobile phone or light other than an enclosed light, electric torch or lamp.

(2) No person shall take any mobile phone or light other than an electric torch or an enclosed electric lamp into any explosive magazine or store or premises.

(3) The owner, agent or manager shall take adequate steps to prevent pilferage of explosives during its storage, transport and use in the mine.

(4) No person shall have explosives in his possession except as provided for in these regulations or hide or keep explosives in a dwelling house.

(5) Any person finding any explosives in or about a mine shall deposit the same in the magazine or store or premises and every such occurrence shall be reported to the manager in writing.

(6) Shot-firers and their helpers shall-
  (a) not use battery operated watches, mobile phone, synthetic clothes and socks;
  (b) use only conductive type of foot-wears; and
  (c) in case of leather shoes or boots, the sole shall also be of leather and without hobnails.

CONCLUSIONS

The present paper has highlighted the overview of salient features of the current statutory provisions for drilling and blasting applicable to opencast coal mines, especially deep hole blasting, transport of explosives in bulk, use of ANFO, use of SMS/SME, sleeping holes, blasting in hot strata and general precautions regarding explosives, drilling and blasting. This may be useful to the practicing mining engineers for developing a better understanding of the statutory obligations which, in turn, will be help in improving the compliance standards.

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Best Practices for Conducting Blasting in Fire Areas (Hot Holes) in an Opencast Coal Mines

Satish Jha* Teja Ram**

INTRODUCTION

Opencast coal mining is employed in areas where coal seam is close to the surface and allow for recovery of larger coal deposits . In opencast mines, Overburden is removed to expose the coal seam that is then drill, blasted and hauled for economic purpose.

Surface coal mining operations are often executed on virgin ground and above old underground workings . The shot holes drilled are either cool or hot . Cool holes have in-hole temperature below 400C . Hot holes have in- hole temperature of more than 400C.

i.e. 40°C. minimum temperature of hot holes

80°C threshold temperature :- persons are required to evacuate the place

110°C grey-white fumes :- down lines melts .

130°C -160°C white fumes observed
140°C -220°C product ejection around the collar of holes

180 0C - Orange brown noxious fumes 220°C -260°C Exothermic reactions : eruption

A number of fires are known to be active in different coalfields . Any attempt to win coal from these areas involving blasting is fraught with danger.

During the experience at different open cast mine like :

™ Majri OCM – Virgin Seam (Thick seam)

™ Majri OCM : Underground to OCM – Developed pillars – Thick seem

™ Telwasa OCM : Virgin Seam (Thick seam)

™ Ghorawari OCM : Underground to OCM developed pillar

™ Dudhichua OCM : Multi seam working

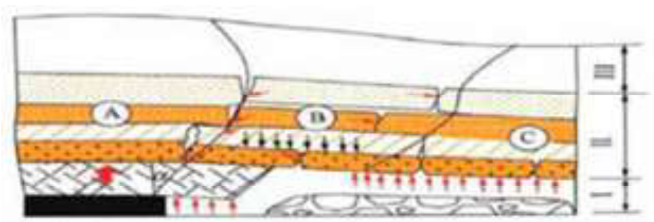
Based on the experience gained so far , it is recommended that while blasting in hot strata (either in OB or coal) many precautionary measured required to be taken.

It is natural process that coal seam got spontaneous heating after its incubation period. In big OCM, large exposure of coal is the need of company to produce the coal on demand so spontaneous heating will always exist in many mines.

**Area GM, Dudhichua Area*
***ASO, Dudhichua Area*

DIFFERENT REASONS OF SPONTANEOUS HEATING / HOT HOLES

i) Large exposure of seam to atmosphere
ii) Cracks developed in seam due to back break of previous blast .
iii) Cracks developed due to subsidence over underground working
iv) Poor dressing of coal seam during coal extraction.
v) Under cut /coal bench failure and accumulation of loose coal at toe of seam.



DANGERS DUE TO HOT HOLES

The temperature at which explosive is stored and used may have detrimental effect upon its ultimate performance during the use . The explosive used in below freezing temperatures are specially formulated so that they do not lose their characteristics . For example ,dynamite will freeze and become hazardous to tampering, slurries become stiff and insensitive and fail to detonate. All types of NG –based explosives are prohibited to be used in hot holes . Only slurries and emulsions are permitted to be used in hot hole having maximum temperature up to 800C.

Hot holes conditions are those that have a temperature of 400 C or more and a reactivity less than 1 % . The dangers are summarized as -

i) Premature detonation in hat holes
ii) Uncontrolled detonation
iii) Melting of booster ,detonator , detonating fuse .
iv) Burn injuries
v) Heat and humid climate

Health Hazards due to noxious fumes

RECENT FEW INCIDENCES

Sl. No.	Year	Location	Incident description
1	2020	South Africa	Premature detonation in a hot hole
2	2020	SCCL India	Premature detonation
3	2019	South Africa	Premature detonation in a hot hole
4	2018	South Africa	Un control detonation
5	2016	Indonesia	A melted booster in a hot hole
6	2014	Australia	A melted downlines

CONDITIONS FOR CONDUCTING BLASTING IN FIRE AREAS IN AN OPENCAST MINE

1. Blasting operations shall be carried out under direct supervision of an Assistant manager in charge of blasting operations.
2. Persons trained in the job of blasting in Fire area only shall be deputed for the said operation.
3. No explosive other than slurry or emulsion explosives shall be used.
4. Blasting shall be done with detonating fuse down the hole.
5. Temperature inside the blast holes shall be measured (before filling with water) and if the temperature exceeds 80°C, in any hole, such hole shall not be charged. Records of measurement of temperature in each hole shall be maintained in a bound paged book and shall be signed by the Assistant manager incharge of Blasting operations and countersigned by manager.
6. All blast holes shall be kept filled with water. When any hole is traversed by cracks or fissures, such hole shall not be charged unless it is lined with an asbestos pipe and the hole filled with water. In addition, betonies

or any other effective material shall be used for sealing any cracks at the bottom of the holes.

7. Detonating fuse shall not be laid on hot ground without taking suitable precautions, which will prevent it from coming in contact with hot strata.
8. Hottest holes shall be loaded last. Uncharged holes shall be filled with water/ sandy material.
9. Carbonaceous material shall not be used for stemming.
10. The charging and firing of the holes in any one round shall be completed expeditiously and in any case within 02 hours.
11. Regular monitoring of Carbon Monoxide (CO) shall be done by a competent person authorized by the manager, during charging of the holes. If CO is more than 50 ppm, all persons from the area shall be withdrawn.
12. Water spraying/quenching arrangements shall be kept available at the blasting site to deal in case of emergency.
13. Precautions while drilling in Overburden or Coal over the underground workings in Opencast Mines.

A. Where the underground workings are accessible

Before commencement of blasting operations in the quarry.

(1) Such workings shall be surveyed and cleaned of coal dust and thickly stone dusted.

All persons shall be withdrawn from the underground in the same working seam or any other seam or section connected therewith and no work person shall be re-admitted into the said undergroundworkings unless the same have been inspected by a competent person duly authorized for the purpose by the manager and found free from any noxious gases and or signs of fire, etc.

(2) The underground workings to be quarried shall have sufficient thickness of horizontal barrier as stipulated in the Regulation 121(2) of the Coal Mines Regulations, 2017, otherwise shall be iso-lated by explosion proof stoppings such as to withstand the force of vibration of blasting, from any active working area either in the same or different seam or section or /mine as the case may be, so as to prevent transfer of danger of blasting to the said active underground workings.

B. Where the underground workings are not accessible

Before commencement of blasting operations in the quarry:

Such workings shall be treated with incombustible dust ahead of the quarry face fed through surface boreholes and dispersed by compressed air. The following procedure is recommended for treating the inaccessible workings underground with stone dust:

(a) Ahead of the bottom bench in overburden, holes shall be drilled 18 metre apart in grid pattern from top bench in overburden or surface to the underground galleries. The distance between the 1st row of holesand quarry face should be 06 me-tre or less.
(b) After holing through of the galleries in coal, the drill rod shall be withdrawn and at least 02 tonne ofstone dust fed through the borehole.
(c) The drill rod shall then be lowered through the borehole again so that it is well in the heap of stone dustdropped on the floor of the under-ground galleries.
(d) Compressed air shall then be blown at the rate of not less than 20 cu. m. per minute under pressure of at least 3.5 kg/ cm2 for a minimum of 45 min-utes. This time can be proportionately reduced if compressedair at higher pressure is available.
(e) The steps (b), (c) and (d) shall be repeated with 02 tonne or more of stone dust dropped in each hole.
(f) If perimeter of galleries exceeds 14 m, the quan-tity of stone dust dispersed shall be proportion-ately increased by repeating the whole process a second time.
(g) For greater effectiveness, the holes shall be drilled in the junctions of the galleries.
(h) For better dispersability, it is desirable to use pure limestone dust or dolomite dust with least pos-sibly silica content. The stone dust should prefer-ably be water-proofed in humid and wet conditions.
(i) It shall be possible to improve the efficiency of the operation by fabricating special equipment or devicewhich would enable the stone dust to be airborne near about the mouth of the borehole instead of dumping the stone dust at the bottom of the hole and then attempting to disperse it with compressed air as outlined in the procedure given above.

Note: None of the holes put down for stone dusting the underground workings are to be utilised for any other purpose, except for determining the thickness of overburden, etc. and other monitoring purposes.

C. General Precautions

(1) Surveying: Before commencement of the drilling of shot hole over the underground workings in the opencast mine, surveying shall be done to legibly mark the galleries, pillars & staple pits in the blasting area.

(2) Location of holes: The holes drilled in the overburden bench lying immediately above the coal seam (referred to hereinafter as last overburden bench) shall not lie immediately above the galleries in order to ensure that the blast-holes do not directly fire into the underground workings.

(3) Safe parting: The depth of holes in the last overburden bench shall be such as to leave atleast 06 m thick overburden above the coal seam, and to ensure compliance with this requirement, a pilot hole shallbe put for each round of blasting to determine the total thickness of overburden over the coal seam.

(4) Compacting of the galleries: After blasting the last overburden bench over developed galleries, loading operations shall not be started till the blasted area is fully compacted to prevent any chance of pot holing and declared free from any fire and safe by the blast-ing officer. Special care is to be taken to fill the shafts or staple pits whether vertical or inclined.

(5) Workings developed in more than one section: Where more than one section of the seam had been developed on pillars, the shot holes shall not be drilled to within 03 m of a lower section, and care shall be taken that the blast holes do not directly fire into any underground gallery.

(6) Delay detonators not to be used: Unless otherwise permitted by DGMS in writing and subject to such conditions as may be imposed, no delay action det-onators shall be used in coal, and the manner of extraction of pillars shall be by drilling and blasting holes in coal pillars only from top downwards.

(7) Use of water ampoules/moist sand: All holes in the last overburden bench and/or in coal shall be charged with water ampoules or with moist sand of at least 0.6 m in length at the bottom of the hole.

(8) Where there is any doubt and particularly where there are cracks and crevices, the bottom 02 m length ofthe hole shall be filled with sand.

(9) No person including shot-firer shall take shelter within 100 m of the quarry opening and such shelters shall be of stable and strong construction to provide safe shelter to the shotfirer and his helpers.

(10) Sleeping of holes shall not be permitted.

(11) No PETN/TNT based cast booster shall be used for initiating non-cap sensitive slurry/emulsion explo-sive in coal benches and overburden benches of a fi-ery coal seam.

(12) Overburden benches immediately above the coal seams and other fiery areas in the mine, the explo-sive charge shall be fired by detonator attached to the detonating cord at the surface and not within the shot hole.

(13) All explosives, cast boosters, detonators and deto-nating cord shall be subjected to proper testing in an approved laboratory in respect of temperature sensi-tivity, impact sensitivity for safe handling in mines. A certification to that effect shall be supplied for each batch

FLOODING OF UNDERGROUND HOT WORKINGS BEFORE DRILLING AND BLASTING

BLASTING PROCEDURES IN HOT HOLES: ADDITIONAL PRECAUTIONS DURING DRILLING ,CHARGING AND BLASTING OPERATION

The following steps wise approach may be carried out for successful blasting operation in hot holes :-

Step-1 : Select the number of holes properly so that the total blasting operations should not exceed 2 h from the charging of first hole.

Step -2 : Measure the temperature of the holes.

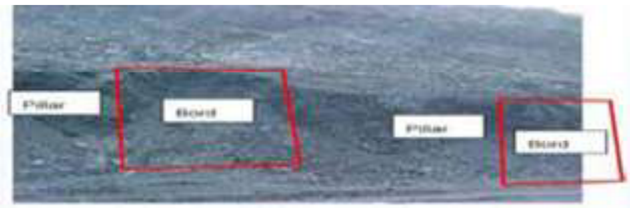
Step-3 : Use water at least 12 h before blasting to flush hot holes till the temperature comes down below 800C

Step-4: Record the temperature of holes at a regular interval of time.

Step-5: Use a mixture of betonies, sodium silicate and water in holes which do not retain water to seal micro –fractures and cracks. Guar –gum up to 5% may also be used for the same purpose.

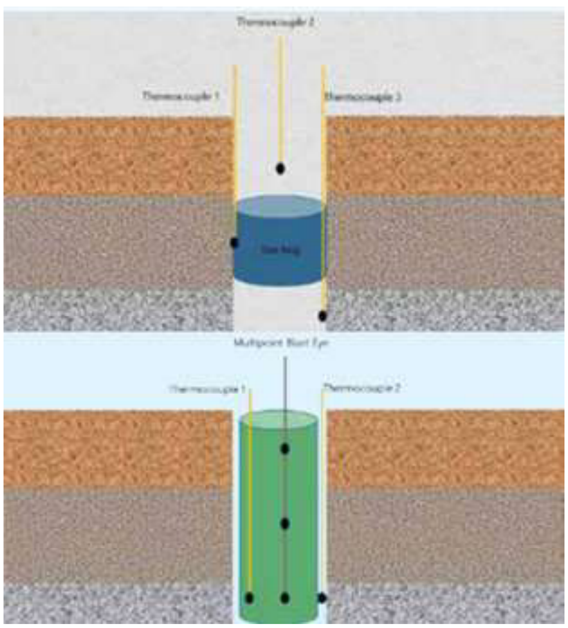
Step -6 : Check before charging whether the detonating cord (or other suitable device) is detached from the main reel or not ? if not then detach it immediately before charging operation starts.

Step-7 : Stemming operation should be done after charging all holes or use gas bags or foam plugs in place of steaming

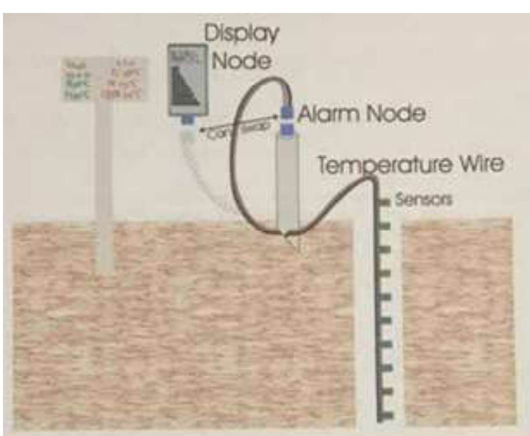


Coal operation without buffer blasting) and with buffer blasting

Types of temperature measurement infrared devices



The set up for testing a sleeve in a hot hole. The set up for testing a gas bag in a hot hole



The Hot Hole Management System accessories

Foam expander plug used to close off a hot hole

Conditions at the two test sites

Three types of gas bags

Pumping of water into the PVC sleeve in the hole

Improvement required /Common mistakes :

A. Drilling: Front hole burden

™ If more – back breaks ,boulder formations

™ If less – fly rocks produced and damaging shovels, cables etc.

™ High wall side row should be in a straight line as far as practicable adjustment of holes should be done towards free face .

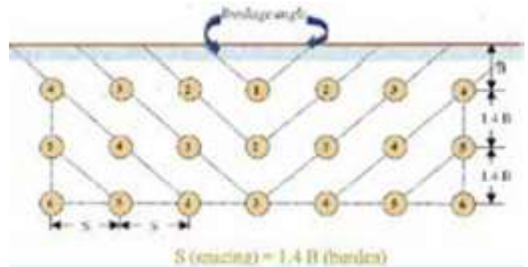
B. Charging- Drill cutting to be removed to exposes the solid collar

Advantages are - Measurements mistakes are minimized wateris flushed out

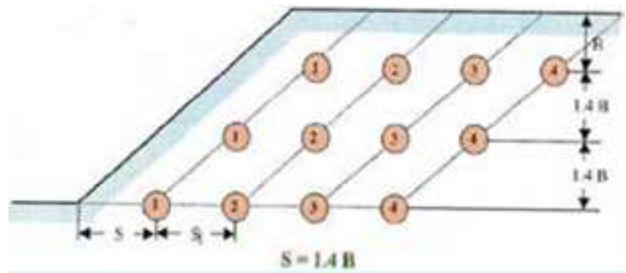
C. Blasting - Effective burden to be consider white considering the different type of initiation system – like diagonals or V-Cut etc.

Once back break occurs – thick over how to arrest back break in future .

V-Cut Initiation



Ordinary Line Initiation



Best practices to reduce the hazards

i) To prevent air from coming into contact with coal containing sulphides .
ii) Cooling agents :- Use of high presser water etc. to douse surface fires and reduce the temperature in hot holes .
iii) Sealing agents : Such as calcium chloride, Bentonite to reduce spontaneous combustion .
iv) Inert gas : Use of Nitrogen and carbon dioxide to flush hot holes
v) Dozing over : Dumping of sand /OB on the highwall using dozer .
vi) Use of moist sand / Black cotton soil for sealing of crakes .
vii) Buffer blasting : Use of explosive to heave the Overburden , collapsing old underground pillars to close –off ventilation routes .
viii) Cladding : Placing OB /Top soil over high walls to limit air flow in to the old under- ground workings through voids /cracks .
ix) Buttress Blasting: cast blasting in sealed off voids .
x) Just in time drilling : To ensure that newly exposed coal is excavated immediately .
xi) Training and procedure of SOP preparation :.
xii) Risk assessment of the place.
xiii) Hole temperature measurement and monitoring – Use of different devices

xiv) Hole loading sequence – It is advisable to load the holes closest to the initiation point first to firing in the event of change in the conditions . Also it is advisable that hottest holes should be loaded last to reduce the amount the time the explosive expose to high temperature .

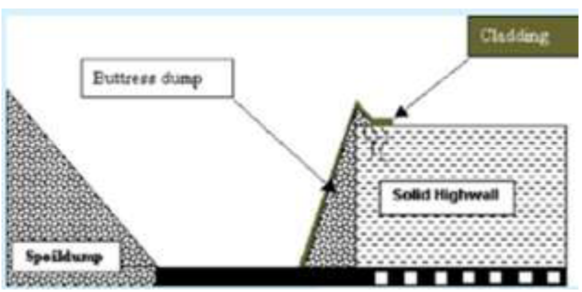
xv) Steaming: Hot holes should not be steamed to avoid a pressure build-up in the hole

xvi) Minimum explosive sleep times : To prevent the heating –up of explosive

xvii) Minimizing spillage : spillage of explosive should be avoid to reduce the risk of explosive decomposition .

xix) Use of hot holes management tools: like to monitor the change in temperature of emulsion, PVC sleeves ,scanner & monitors , gas bags etc.

xx) The selection of explosive and initiating system : it should be based on risk assessment of hot ground conditions , reactive ground conditions . Hot hole specific emulsion should be used for blast in hot holes.



Buttress blasting and cladding applied to a highwall

Dozing of a highwall with soil

CONCLUSION

Opencast mining operation often encounter hot mining areas . The hot hole temperature need to be measured & monitor and take precautionary measures from the time they are drill until charging / blasting to prevent premature detonation of the charge explosive.

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Integrated Hydrogeological Characterization of Rugged Terrains: A Case Study of the Bhatti Mine Area

Ravindra Pratap Singh* Nirmala R** Jitendra Kumar Dixit***

ABSTRACT

This proposed research aims to comprehensively investigate the hydrogeological dynamics of rugged terrains, focusing on the Bhatti Mine area as a case study. The study will employ an integrated approach that combines Double-Ring Infiltrometer Tests and Vertical Electrical Sounding surveys to analyze infiltration behaviour and subsurface properties. The research will address the intricate interplay between surface conditions, vegetation, and geological features in shaping hydrological patterns. The study seeks to enhance our understanding of water movement, absorption, and percolation in such challenging terrains, contributing to improved environmental assessments and land management strategies.

This research will contribute to advancing our understanding of hydrogeological dynamics in rugged terrains, shedding light on the complex interactions between surface conditions, vegetation, and geological features. The integrated approach will provide valuable insights for groundwater management, environmental assessments, and land management strategies in similar challenging landscapes. By addressing the limitations and challenges associated with data collection in rugged terrains, the study will also offer innovative solutions for conducting field research in difficult-to-access areas.

Keywords: infiltration behaviour, Double-Ring Infiltrometer Tests, Vertical Electrical Sounding, hydrological assessment, land management.

INTRODUCTION

Delhi, historically referred to as Inderprastha holds a pivotal role in the rapidly advancing economy. Its evolution from ancient times to a contemporary urban centre has been fuelled by swift urbanization and burgeoning industrialization. This transformation has triggered a surge in population growth, rendering Delhi the most densely populated city in India, with a density of 11,297 persons per square kilometre as recorded in the 2011 Census. This population upsurge has exerted immense pressure on natural resources, particularly the vital resource of drinking water. However, the available surface water sources are proving inadequate to satiate this growing thirst, leading to an intensified reliance on groundwater.

Recent research underscores a disconcerting trend the city's escalating dependence on groundwater is outpacing its capacity for replenishment. This valuable resource is facing a dual threat of depletion and contamination, a phenomenon acutely exacerbated in the National Capital Region due to escalating demand and the influx of pollutants from various industries (Dutta et al. 1996; Dutta 2005). Amid these challenges, a glimmer of hope persists: sustainable groundwater usage is contingent upon effective replenishment mechanisms using untainted water sources. However, the urban landscape in Delhi introduces further complications, including urban flooding and unregulated pollutant mixing. These factors conspire to detrimentally impact groundwater quality and availability. For instance, the proliferation of concrete infrastructure impedes natural groundwater replenishment by inhibiting the percolation of rainwater. Simultaneously, the limited existing groundwater reservoirs face contamination risks from diverse pollutants. This reckless exploitation of groundwater has precipitated an enduring decline in the water table, compounding the region's water scarcity (Kumar et al. 2006).

Addressing this intricate hydrogeological conundrum necessitates an integrated approach. This involves comprehensive mapping of hydrogeomorphic variances, delineation of hydrogeological boundaries, and nuanced consideration of tectonic influences on aquifer dynamics. These factors, in conjunction with surface and subsurface water interactions, shape the hydrogeological landscape (Singh 2003). To handle the circumstances this, endeavour

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inherently hinges upon acquiring a dependable dataset, particularly encompassing surface and sub-surface conditions. A proficient entity in conducting feasibility studies pertinent to artificial groundwater recharge modelling.

To delve into the specifics, the application of Vertical Electrical Sounding (VES) methodology has been chosen, targeting 15 distinct mine pits within the Bhatti area. This selection aligns with the intent to unravel the complexities of the geological strata. Executed through the Schlumberger Array, this technique involves at least 2 measurements, adaptable to the layout of abandoned Quartzite rock pits. The framework for the survey encompasses the utilization of a DC Resistivity meter, the *Terrameter*, with ensuing data subjected to intricate analysis via computer software. This analysis intends to discern potential subsurface rock fracture zones, delving up to a depth of 100 meters below the pit surfaces. Moreover, the assessment endeavours to establish the depth at which the groundwater table manifests below the pit bottoms, as well as to unveil vertical variations within resistivity layers and their corresponding thickness. A crucial facet of this evaluation pertains to estimating water saturation levels within the surveyed pits, while concurrently quantifying fracture porosity. Ultimately, the findings are encapsulated within a comprehensive report, elucidating the nature of fractured quartzite rock, the distribution of water saturation, and the nuanced characteristics of geological strata.

The study integrates soil infiltration tests conducted at 15 designated abandoned mine pits. This assessment is meticulously executed using a double-ring infiltrometer, effectively gauging the infiltration rates of the methodology employed. Specifically targeting the Quartzite rock formation within the Bhatti area, this endeavour underscores the pragmatic approach of the research. After the fieldwork, the test results undergo thorough evaluation and interpretation, culminating in the preparation of an encompassing report. This report not only encapsulates the numerical outcomes but also offers insights derived from the field interpretation method. The holistic documentation encompasses the entire spectrum of results, juxtaposed against a comprehensive table that contextualizes the findings for each of the examined pits.

STUDY AREA

The geographical positioning of the Delhi state is situated along the western bank of the Yamuna River, encompassing the coordinates of 28°12' to 28°53' north latitude and 76°50' to 77°23' east longitude. It shares boundaries with the Thar Desert to the southeast, the Indo-Gangetic plains to the northeast, and the Aravalli range to the south. Notably, the Delhi Ridge, extending 32 km in length, is an integral component of the city's landscape (Figure 1 and 2). Throughout history, the Delhi Ridge has held significant historical and ongoing importance for the city. The preservation of Delhi's forests primarily focuses on environmental and ecosystem services rather than production forestry. Delhi and NCR occupy a landlocked position within the northern plains of the Indian subcontinent.

The climate of Delhi is predominantly influenced by its inland location and the prevalence of continental air masses, which bring about extreme weather conditions. The continental air is unfavourable for lush vegetation growth, resulting in a semi-arid climate characterized by marked temperature fluctuations, high saturation deficit, and moderate rainfall. The nearby desert regions of Rajasthan to the west and southwest, along with the Gangetic plains of Uttar Pradesh to the east, contribute to the climatic dynamics of the area as monsoon air traverses these regions before reaching Delhi. The climatic patterns in the region exhibit distinct periods, including a dry and increasingly hot season from March to June, dry and cold winter from October to February, and a pronounced warm and rainy monsoon period from July to September (Figure 2).

The Southern Ridge of Delhi, known for its quartzite and sand deposits, has faced significant degradation due to extensive mining activities. In response, a ban on mining quartzite and Badarpur sand was implemented, leading to the establishment of the Asola Bhatti Wildlife Sanctuary in 1991. Covering 4845.57 acres, this sanctuary serves as a crucial conservation area within the national capital region, preserving its natural flora and fauna. The mining activities had severely depleted natural vegetation and soil quality in the Bhatti mines area.

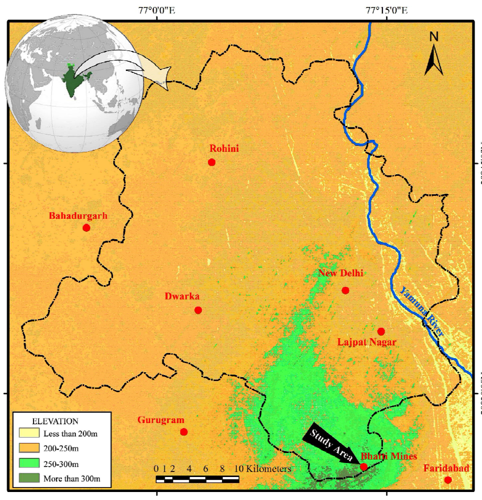

Figure-1

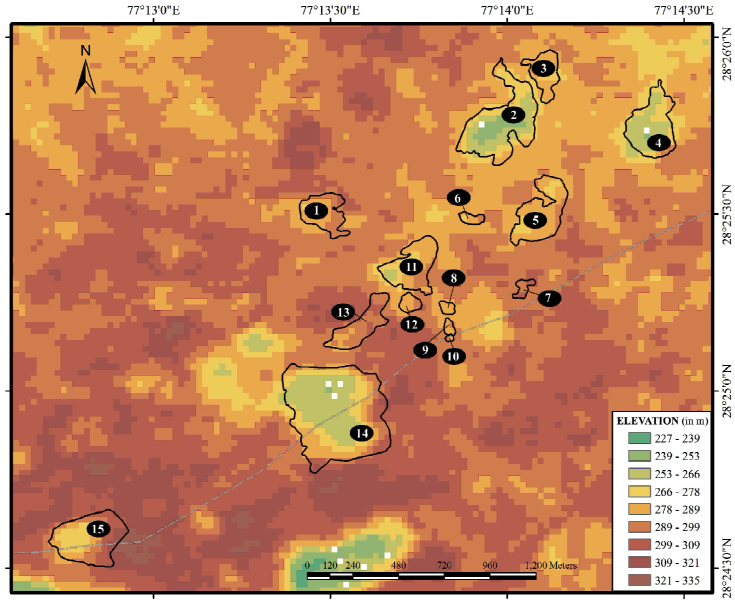

Figure-2

Figure 1: Location map of the study area. Figure 2: The plan map of sites in the Bhatti Mines region.

Groundwater in the Delhi Ridge remains relatively fresh across various depths, contributing to the continuous recharge of adjacent areas. The region's geological features, such as faults and the coarse nature of sediment (weathered quartzite), enhance lateral permeability. However, extensive mining has disrupted aquifers, impacting water levels in the Bhatti mines area.

Relative humidity follows a seasonal pattern, with minimum levels during dry months (April and May) and maximum levels during the monsoon months (July, August, and September). While high relative humidity typically supports plant growth, Delhi's combination of high temperatures and low humidity limits this favourable environment. Consequently, the prevalence of higher temperatures coupled with reduced humidity encourages the proliferation of xerophytic plant species adapted to arid conditions.

Delhi's physiography can be classified into four distinct divisions: Khadar (riverine zone), Bangar (well and canal irrigated area), Dabar (low-lying, rain-fed areas), and Kohi or Pahari (hillsides). The first three divisions are located on the plains and have become hubs for human activities such as agriculture and habitation. In contrast, the Kohi or Pahari division comprises rocky and undulating terrain, encompassing areas like Delhi Ridge, Tughlaqabad, Mehrauli, Fatehpur, Beri, and Dera Mandi. This low plateau is characterized by exposed and unconsolidated micaceous rocks, resulting in dry, sandy soil with limited humus content and sparse vegetation.

METHODOLOGY

The combined utilization of the Double Ring Infiltration and VES survey has been instrumental in fulfilling the objectives of this study, allowing for a comprehensive understanding of the hydrogeological properties of both rock exposures and soil covers at the Bhatti Mine site. These methodologies provide valuable insights into the behaviour of water infiltration and subsurface resistivity, contributing to a holistic analysis of the site's hydrological dynamics.

The Double Ring Infiltration Survey is a robust method for assessing the rate at which water penetrates the ground surface. The study accurately measures infiltration rates by employing the constant head method within a double-ring infiltrometer, this technique has been widely adopted across various applications, including land drainage, sports surface design, and waste isolation layers. The double-ring infiltrometer's design, featuring concentric metal rings and a perforated metal plate, ensures controlled and accurate measurements. The choice of this method aligns with the ASTM D3385-03 standard test, providing credibility and consistency to the collected data.

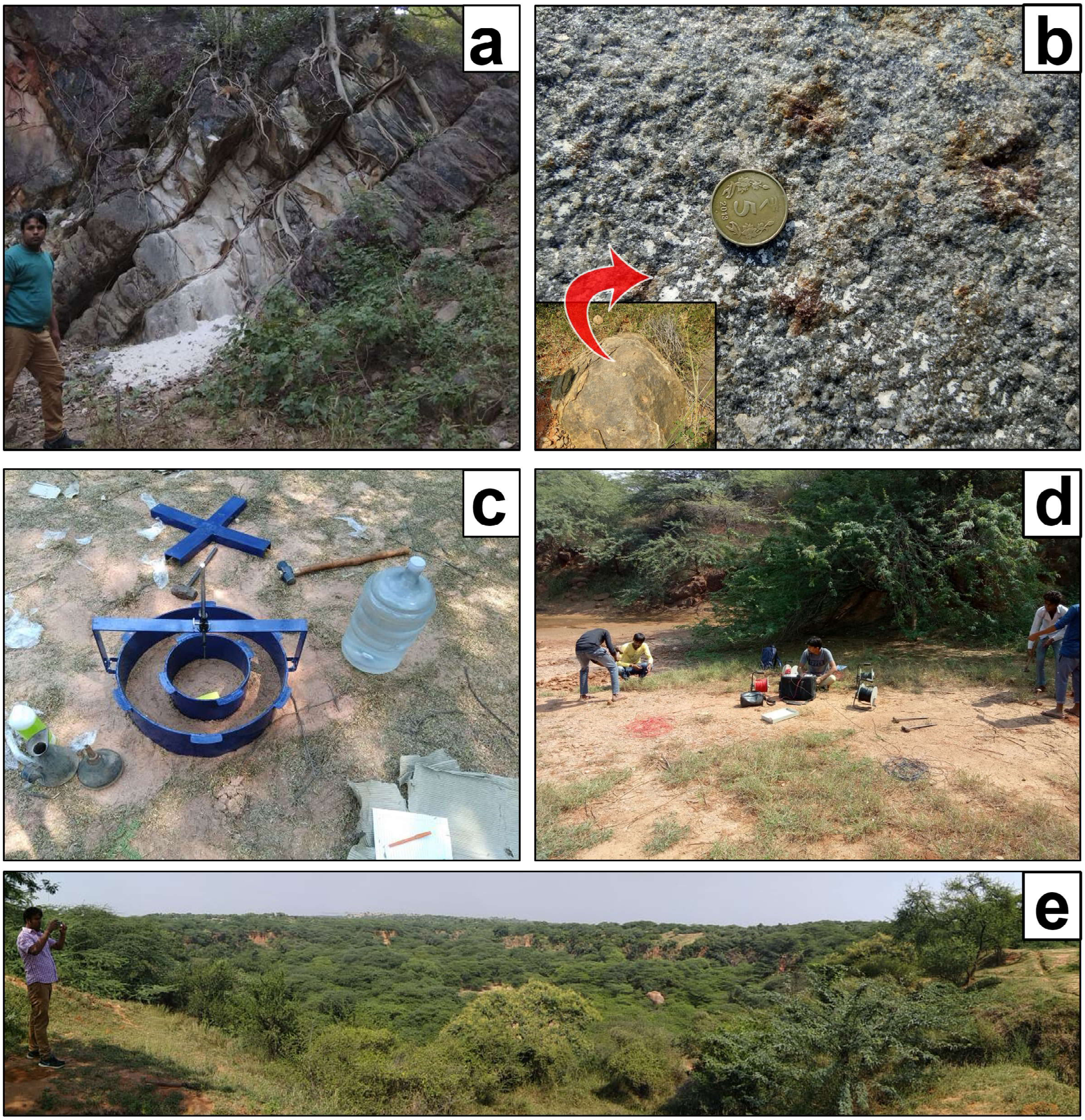

Figure 3(a) Weathered white quartzite with silica sand; (b) granitic quartzite exposure at the location; (c) double ring infiltrometer instrument and its components ready to conduct the survey; (d) double ring infiltrometer instrument and its components ready to conduct the survey; (e) abandoned mining Pit.

Through this methodology, the study has identified significant variations in infiltration rates across different surface types and soil compositions within the Bhatti Mine area. The observed rates highlight the influence of soil aggregation, vegetation cover, and even rock exposure on water infiltration. Notably, the results indicate that well-aggregated sandy soils exhibit higher infiltration rates, while the presence of vegetation tends to moderate the rate. Conversely, hard rock exposure significantly hampers water penetration, emphasizing the role of surface conditions in controlling infiltration dynamics.

The geophysical survey, on the other hand, employs the Schlumberger array to measure subsurface electrical resistivity. This approach provides a valuable tool for delineating water-bearing strata and understanding the geological composition of shallow layers. By recording 1D resistivity distributions, the study gains insights into the site's subsurface properties, particularly in areas underlain by water-bearing strata.

The application of the field surveys has offered a comprehensive and multifaceted exploration of the hydrogeological characteristics of the Bhatti Mine area. The methodologies chosen, with their well-established protocols and adaptable designs, have enabled the collection of accurate and meaningful data. The results underscore the intricate interplay between surface conditions, soil types, and water infiltration rates, providing a foundation for informed decision-making in land management, environmental assessments, and water resource planning. This integrated approach contributes to a more holistic understanding of the site's hydrological behaviour, ultimately guiding future studies and actions in the region.

The soil infiltration can be calculated using the constant head method or the variable head method and for this purpose, a single-ring or double-ring infiltrometer (Figure 3c) may be used. In a double-ring infiltrometer, the infiltration rate is calculated from the drop-in water level per unit of time or the amount of water required to maintain the specified depth or head of water per unit of time and the maximum rate of entry of water into the soil is measured. In the present study, the soil infiltration rate is calculated with the help of a double-ring infiltrometer, which is a simple instrument used for determining water infiltration of the soil measured according to the ASTM D3385-03 standard test method. The double ring infiltrometer is a widely used method of infiltration test used in many applications; i.e. design of land drainage pipes, design of sports surfaces, golf courses, isolation layers of the communal waste, etc. This infiltrometer consists of two concentric metal rings which are driven into the soil, and of a perforated metal plate.

The measurement is taken in the inner cylinder; the outer cylinder is used only as a tool to ensure that water from the inner cylinder will flow downwards and not laterally. The soil surface in the inner cylinder can be covered by a perforated metal plate which is used in order to dissipate the force of the applied water, distribute water uniformly inside the ring and prevent disturbance of the soil surface. The rings are partially inserted into the soil and filled with water, after which the speed of infiltration is measured.

The infiltration is the process of water penetrating the ground surface. The intensity of this process is called the infiltration rate. The infiltration rate is expressed in terms of the volume of water per ground surface and per unit of time (L/T, for instance, mm/hr). The infiltration capacity of the soil indicates the maximum infiltration rate at a certain moment. Under certain circumstances, it may be necessary to determine the infiltration capacity of the soil, for instance in infiltration areas or infiltration basins. The double ring infiltrometer is suitable for almost any type of soil and is applied in irrigation and drainage projects, groundwater and infiltration basins, in optimizing water availability for plants and to determine the effects of cultivation.

Electrical prospecting is the art of measuring the electrical properties of rocks in the study of the structure and composition of those layers of the earth which are sufficiently shallow to be exploited by man. For the present study, 1D resistivity distribution is recorded by the vertical electrical sounding method. The method is applied using a Schlumberger array in the field to delineate water-bearing stratum in the areas underlain.

DATA COLLECTION

A double ring infiltration survey and vertical electrical sounding survey were performed to fulfil the objectives of the study. Both surveys describe the hydrogeological properties of the rock exposure and/or soil cover at the and collected this essential dataset through two significant field expeditions conducted within the designated region. The preliminary field excursion done on September 22,

2017. During this preliminary survey, all 15 excavation sites were inspected to ascertain the appropriateness of employing VES and Infiltration Tests.

The comprehensive collection of field data was accomplished using a suite of specialized instruments from 3^{rd} to 9^{th} October 2017, encompassing all 15 designated mining pits. A total of 12 sampling points was strategically chosen to facilitate 13 infiltration tests at the respective sites. Each mine pit was represented by at least one selected sampling point, with the exception of pits numbered 5, 6, and 9, where accessibility was hindered. The locations chosen for conducting infiltration tests were deliberately selected to encompass both exposed soil and rock surfaces. This strategic selection aimed to unveil disparities in the infiltration rates within the region.

The terrain within the study area has been subjected to extensive exploitation, leading to a highly uneven topography that poses challenges for geophysical surveys. Despite these constraints, a determined effort was made to gain insights into the subsurface characteristics. As such, a total of 19 VES were conducted, with some instances achieving electrode spreads of up to 240 meters. Out of these, 11 VES measurements were undertaken at the pit level depth, while the remaining measurements were distributed across higher elevations of the unexplored regions of the pits.

The analysis provided offers a comprehensive perspective on the field research conducted in the Bhatti Mine area, focusing on infiltration characteristics and subsurface electrical resistivity. The study examines various aspects such as surface conditions, vegetation influence, and the challenges encountered during data collection, presenting valuable insights from the field surveys.

The Bhatti Mines region, characterized by a forest cover with limited rock exposures, exhibits a geological setup marked by three sets of joints and strata dipping at an angle of 50-55° towards the southeast. The white quartzite, frequently found in moderately to highly weathered states, is often accompanied by white silica sand deposits. The presence of seepage and damp rock conditions along pit walls signifies the hydrological dynamics within the region.

For data collection, two major instruments, the Double-Ring Infiltrometer and *Terrameter* for VES using the Schlumberger array, were employed, supported by a range of field equipment. The infiltration test proved challenging due to the rugged terrain and limited water availability. Despite these hurdles, meticulous data recording was executed, yielding valuable insights into the hydrological behaviour of the region.

Double-ring infiltrometer is used to calculate the soil infiltration rate at the site for the valuation of soil hydrological properties. The experiment shows loose sand and reddish sand (also known as *Badarpur*) have the highest infiltration rate in the study area. The initial readings at the site show an infiltration rate anomalously high (i.e. more than 2000mm/hr), though these anomalously high values become steady with time. The detailed analysis of the infiltration rate shows sandy areas have high infiltration rates (30mm/hr) in sandy areas and low infiltration rates (as low as 2mm/hr) in the clayey sand region.

The infiltration tests were conducted at multiple locations within the Bhatti Mine area, elucidating the effects of varying surface conditions on infiltration. These tests revealed that soil cover, vegetation, and rock exposure significantly influence the rate of water movement into the subsurface. The analysis offered insights into each pit location's infiltration behaviour, ranging from moderate to limited based on the prevailing surface conditions.

Similarly, the geophysical surveys, conducted using the Schlumberger array and showcased the electrical resistivity properties of the subsurface. The study strategically selected suitable locations for geophysical surveys, revealing the intricate relationship between surface characteristics and electrical resistivity. Ground-level and pit-level surveys were employed to capture variations in the subsurface, and the challenges posed by dense vegetation and undulating terrain were evident in the limited electrode spread in some areas.

In conclusion, the research in the Bhatti Mine area, through field surveys, provides valuable insights into both hydrological behaviour and subsurface electrical resistivity characteristics. The interplay between surface conditions, including soil cover, vegetation, and rock exposure, emerged as a critical factor influencing infiltration and resistivity. The findings contribute to a more comprehensive understanding of the region's hydrological and geological dynamics, with implications for future environmental assessments and land management strategies. The challenges encountered during data collection underscore the complexities of field research in rugged terrains, highlighting the need for innovative methodologies and cautious data interpretation.

DATA ANALYSIS AND RESULTS

The detailed analysis of individual pits within the Bhatti Mine area reveals diverse infiltration characteristics influenced by varying surface conditions, vegetation, and rock exposure. The findings underscore the intricate interplay between these factors in shaping the hydrological behaviour of the region. Moreover, the challenges encountered during data collection highlight the importance of innovative methodologies and cautious interpretation for successful field research in rugged terrains. The combined insights from field surveys contribute to a holistic understanding of hydrological and geological dynamics in the Bhatti Mine area, with implications for future environmental assessments and land management strategies.

The application of resistivity techniques in this study, encompassing various methods such as VES curve interpretation, transverse resistance estimation, formation factor calculation, and apparent resistivity polygons, has provided a comprehensive assessment of the subsurface hydrogeologic conditions. These methodologies have proven effective in delineating aquifers, identifying lithological variations, and characterizing subsurface trends. However, the interpretation process requires careful consideration and validation against available subsurface data to ensure accurate results.

The study demonstrates the versatility of VES curves in providing valuable insights into subsurface conditions, particularly in aquifer delineation. Nevertheless, it's essential to acknowledge potential challenges in interpretation, especially when differentiating between distinct subsurface materials with similar resistivity values. For instance, dry coarse sand, gravel, and hard rock can exhibit high resistivity values, while dry clayey *kankar* may be misinterpreted as fine sand with saline water due to its low resistivity. Such intricacies emphasize the need for thorough analysis and validation of interpretation results.

During curve matching interpretation, the occurrence of discontinuities in the layers can lead to deviations from the fitted curve, highlighting the importance of recognizing abrupt changes in subsurface properties.

The geophysical study has successfully identified thin layers of high resistivity indicative of granular materials, as well as low values of transverse resistance associated with buried paleochannels. This ability to detect subtle subsurface features contributes to a more comprehensive understanding of the site's geology and hydrogeology.

The utilization of apparent resistivity polygon plots, as exemplified in Pit No. 5, has facilitated the interpretation of anisotropy in subsurface geologic materials. This approach provides insights into fracture trends in bedrock zones and the pattern of granular zones. The integration of such interpretations with nearby hard rock terrain and VES curve results enhances the accuracy of subsurface information, allowing for a more refined depiction of actual conditions.

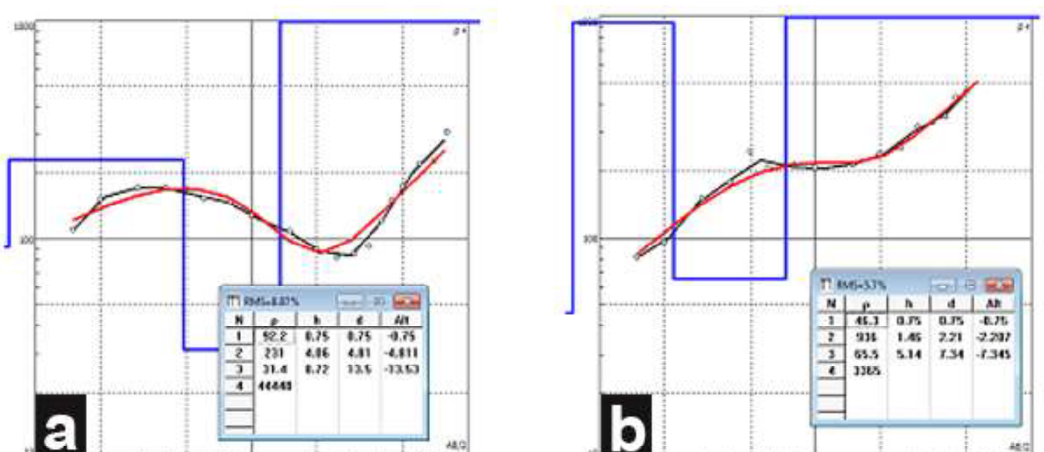

Figure 4: Schlumberger sounding curves at (a) Pit-2 and (b) Pit-14

Pit-wise apparent resistivity results reveal distinct patterns across various pits. In Mine Pits 1 and 2, for instance, the resistivity data point towards the presence of weathered hard rock at certain depths. In Mine Pit 3, the availability of larger electrode spacing aids in obtaining more comprehensive subsurface information. Conversely, challenges in accessing certain areas, such as Mine Pit 5, led to the application of ground-level VES surveys, highlighting the adaptability of the methodology.

The study concludes that resistivity techniques, particularly VES curve interpretation and apparent resistivity polygons, offer a powerful tool for subsurface characterization and hydrogeological assessment. These methods contribute valuable insights into lithological variations, aquifer delineation, and fracture patterns, which are crucial for informed decision-making in groundwater management and environmental planning. By acknowledging the limitations, verifying results, and integrating interpretations with existing data, these techniques serve as an indispensable asset in understanding subsurface dynamics and optimizing resource utilization.

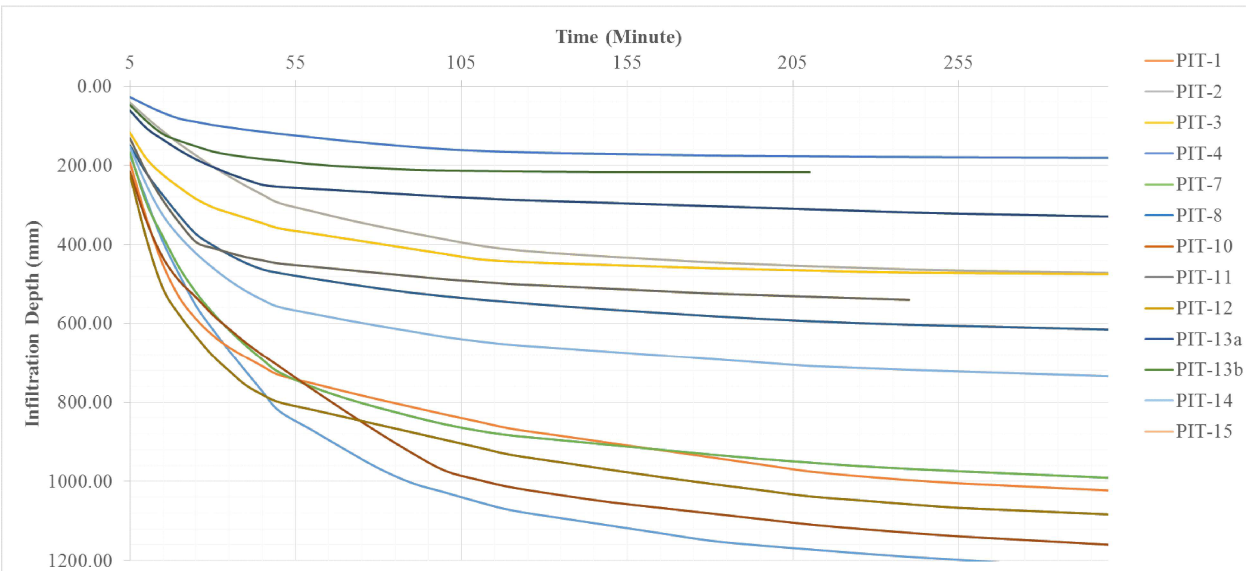

Figure 5: Infiltration rate curves for the study area

The data collection during regional studies showing the study site is belonging to the South Delhi Fold Belt, exposing quaternary alluvium and weathered grey quartzite. Stratigraphically, the grey to brownish grey, massive (to thinly bedded at places), hard, compact, highly jointed and weathered quartzites belong to the Alwar Group of Delhi Supergroup. The quartzites are ferruginous and gritty types which on weathering and subsequent disintegration give rise to coarse sand. Chemical weathering of deeper horizons is also common in the area. The quartzite is overlain by unconsolidated quaternary to recent sediments comprising sand, silt, clay, silty clay and *kankar* beds with various proportions. Hydro-geologically, the latter plays an important role, which represents the major water-bearing horizon in the area.

The field data provides valuable insights into the infiltration rates observed across various pits within the Bhatti Mine area. The study focused on understanding the interaction between different surface types and soil characteristics to determine the rate at which water penetrates the ground. The analysis of this dataset reveals significant variability in infiltration rates, shedding light on the hydrological behaviour of the study area.

The comprehensive investigation of multiple mine pits within the study area has shed light on their hydrogeological properties, offering a nuanced understanding of their infiltration rates, sediment compositions, and geological structures. This analysis has unveiled intriguing patterns and trends that provide significant insights into the hydrological behavior of these pits and their surrounding environments.

Pit 1 has unveiled a soil-covered surface characterized by moderate infiltration rates. This observation hints at the potential for efficient water absorption and percolation into the subsurface, indicating favorable hydrological conditions that may support various ecological processes.

Similarly, Pit 2 presents a comparable scenario with its soil-covered setting and moderate infiltration rates. This consistency underscores the recurring patterns of water movement within these specific areas, suggesting a degree of uniformity in their hydrological characteristics.

Pit 3 introduces an interesting dynamic with its vegetated soil cover, contributing to heightened infiltration. This finding underscores the positive influence of vegetation in enhancing water absorption, aligning with established knowledge regarding the pivotal role of plants in shaping hydrological processes.

Building upon this observation, Pit 4 reaffirms the significance of vegetation by displaying infiltration behavior akin to that of Pit 3. The presence of dense vegetation over soil-covered surfaces appears to facilitate water movement into the subsurface, demonstrating the interconnectedness of biological and hydrological elements.

However, the challenging conditions presented by Pits 5 and 6, marked by dense vegetation and steep slopes, have hindered the effective execution of infiltration tests. These limitations have underscored the practical difficulties encountered when studying hydrogeological dynamics under extreme surface conditions.

In Pit 7, the exposure of hard rock has led to minimal infiltration, emphasizing the role of geological features in shaping hydrological interactions. The presence of impermeable surfaces like hard rock can significantly impede water movement, creating localized hydrological constraints.

Pit 8 showcases the beneficial impact of even slight vegetative cover, contributing to moderate infiltration rates. This finding reinforces the notion that vegetation, regardless of its density, can exert a notable influence on water absorption and percolation.

Pits 9 and 10 underscore the challenges posed by difficult surface conditions, rendering infiltration tests impractical. These instances highlight the complexities involved in conducting hydrogeological studies in areas with adverse terrain and vegetation.

Pit 11 introduces another instance of moderate infiltration behavior, akin to other sites with similar surface conditions. This regularity in hydrological patterns suggests a degree of consistency in water movement within these specific contexts.

Pit 12 further bolsters the role of vegetation in influencing infiltration, as its vegetated soil cover aligns with moderate infiltration rates. This correlation echoes the patterns observed in Pits 3, 4, and 10, reaffirming the importance of plants in mediating hydrological processes.

Pit 13 presents a unique scenario with variable infiltration behaviors resulting from the interplay between dry and vegetated soil covers. This observation highlights the intricate interactions between different surface conditions and their implications for water movement dynamics.

The influence of geological features takes center stage in Pit 14, where highly weathered quartzite restricts infiltration. This finding underscores the pivotal role of rock exposure in shaping hydrological behavior and emphasizes the importance of considering subsurface characteristics in hydrogeological analyses.

Lastly, Pit 15 adds to the overall picture by displaying moderate infiltration rates akin to other soil-covered areas. This consistency further supports the notion of recurrent hydrological patterns within such settings.

The detailed examination of these diverse mine pits has provided valuable insights into the complex interplay between surface conditions, geological features, and vegetation in shaping hydrogeological dynamics. These findings enhance our understanding of the region's unique hydrological context and contribute to the broader body of knowledge concerning water movement, absorption, and percolation in varied geological and environmental settings.

The geological characteristics of the area are primarily shaped by weathered grey quartzite, which is covered by decomposed rock fragments known as *Badarpur*. The infiltration rates of the quartzite and the overlying sediments have been studied extensively, leading to the classification of the sediments into four distinct classes based on their infiltration rates: sand, sandy and silty loam, and clayey loam.

The sand layer exhibits the highest infiltration rate, exceeding 50mm/hr, while the sandy and silty loam layers display moderate infiltration rates ranging beyond 20mm/hr. On the other hand, the clayey loam or clayey soil layer has the lowest infiltration rates, measuring less than 20mm/hr.

CONCLUSION

The results and analysis of the vertical electrical sounding surveys show the pits in the region have a top loose soil cover of 0.75 to 3.13m thick. This loose top soil cover overlain to a sandy/silty/clayey or weathered rock (mostly dry) up to the depth of 13.5m. The fractured and/or hard quartzite is present at varying depths from 7.0m to 13.5m below the pit surface underlying the overburdened cover of soil and weathered rock. The field infiltration tests reveal that the top loose soil has very high infiltration rates (more than 22mm/hr) while the quartzite has the least infiltration rates (less than 4mm/hr).

The analysis of the field results can be concluded that the hard quartzite in the area has 2mm/hr and the weathered quartzite have a 4mm/hr infiltration rate. The studies done by Atkinson (2002) show the infiltration rate of water in a pit (or pit lake) is a function of the underlying rock (which is fractured or hard quartzite in the Bhatti Mines area).

So, in the study area, the infiltration rates for the pits should be considered as 2mm/hr for hard quartzite and 4mm/hr for weathered to highly weathered quartzite, which is derived from the field infiltration tests. The detailed field investigations done by Wolff (1982) and Fazelalavi (2013) show the horizontal infiltration rates are almost negligible as compared to the vertical infiltration rates. So, in the present study, only vertical infiltration rates should be considered for any further computation.

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Unravelling the Dynamics of Landslide Occurrences and Joint Plane Influences in the Lesser-Himalayan Terrain: A Case Study of Ramban-Gul Road

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ABSTRACT

In the context of research focused on the structurally anisotropic Sub-Himalayan regions, the frequency of landslides or slope failures exhibits significant variation from one location to another. This variability can be attributed to underlying structural features and neotectonic activities. A notable case study of Ramban-Gul Road lies within the Jammu Himalaya, where a series of impactful landslides have occurred in recent times. Noteworthy among these are ongoing events like the Khuni Nala Rock-fall, Dhamkund Landslide, and Kora Pani Landslide. The manifestation of landslide activities in the Jammu Himalaya, including processes like falling, toppling, slumping, debris flow, debris slide, and land creeping, predominantly occurs in regions characterized by Low-hills exhibiting moderate to steep slopes. Within this geographical setting, the occurrence of landslide events is influenced by a range of intrinsic and extrinsic factors. These factors encompass geomorphological attributes, morphometric characteristics, lithological composition, structural arrangements, tectonic dynamics, and joint conditions. The interaction of these factors operates in conjunction with slope gradients, a relationship highlighted by previous studies. In the examined transect of the study area, numerous instances of landslides have been observed. However, a majority of these occurrences have undergone treatment interventions. Curiously, the treatment approach employed across the area remains relatively uniform, regardless of the diverse lithological compositions and varying overburden loads present in different areas. As part of this research endeavour, a subset of nine particularly unstable slopes from the identified landslide sites was selected for in-depth analysis. This analysis seeks to elucidate the specific role played by joint planes in the initiation and progression of these landslides. The ensuing article presents a comprehensive exploration of the details pertaining to these selected landslides, shedding light on the intricate relationships between joint plane characteristics and landslide occurrences.

Key words: Landslide, Jammu Himalayas, Slope Stability, NSMR, RMR

INTRODUCTION

In the realm of engineering projects, the maintenance of any endeavour is intricately tied to the geotechnical understanding of the project site and the surrounding geological and morphological conditions. However, in mountainous terrains, both geological and morphological conditions are subject to substantial variations, which can exert direct or indirect influences on the success of a project. This situation becomes particularly complex in the context of hydroelectric projects (HEPs), as evidenced by prior research (Krishnaswami and Singh 2005), owing to the presence of significant water bodies. Among the critical components demanding meticulous examination within a HEP framework, the Catchment Area Treatment (CAT) plan takes a prominent position. The formulation of CAT plans hinges on two fundamental data inputs, updated periodically: water (both surface and groundwater) and the sediment load carried into the reservoir. This sediment load bears a direct impact on the storage capacity of the reservoir, consequently amplifying the construction, production, and maintenance costs associated with HEPs.

This sediment load, often referred to as silt load, is intrinsically linked to erosion processes upstream, which in turn are influenced by the properties of the upper soil cover. Specifically, areas with highly jointed rock strata or regions characterized by low values of geotechnical classifications tend to be particularly susceptible. Thus, conducting exhaustive geotechnical investigations assumes paramount importance. To address this, comprehensive geotechnical surveys were systematically

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conducted within the Chenab River basin, focusing on selected regions. The field data gleaned from these geotechnical studies play a pivotal role in not only orienting the understanding of unstable slopes but also in delineating their geometric characteristics. Furthermore, these surveys offer a means to identify areas of heightened erosion risk through the identification of low geotechnical classification values. Through this holistic approach, this research endeavours to contribute to the enhancement of sustainable maintenance strategies for engineering projects, especially within the context of mountainous terrains and hydroelectric projects.

METHODOLOGY

The classification of rocks based on geotechnical properties has undergone continuous refinement by pioneering researchers since its inception (Terzaghi 1946; Løset 1992). These classifications have evolved over time to account for various geological and geomorphological conditions. Examples include Rock Structure Rating (RSR; Wickham et al. 1972), later renamed Rock Mass Rating (RMR; Bieniaswki 1973, 1976), Q-system (Barton et al., 1974), and Laubscher's Classification (1977). These systems have seen further amendments with the incorporation of numerous parameters, aiming for more precise characterization of ground conditions and more accurate load assessments. Examples of these modified systems include Mining Basic RMR (MBR; Kandorski et al., 1983), RMR_b (Basic RMR; Bieniawski 1989), Modified RMR (MRMR; Unal 1990), Rock Mass Index (RMi; Palmstrom 1982, 1995), and Rock Engineering Systems (RES; Mazzoccola and Hudson, 1996). Researchers have also identified limitations inherent in these geotechnical classifications (Palmstrom and Broch 2006).

One of these prominent classifications, RMR (Bieniaswki, 1989), served as the basis for Romana's (1993) proposal of a slope classification system, known as Slope Mass Rating (SMR). This classification system assesses slope stability and has found widespread application globally, including in India (Prakash et al. 2003). The SMR technique has been site-specific and has undergone refinements based on rock conditions (Moon et al. 2001) and topography.

The adoption of these classification systems necessitates the consideration of specific parameters, for which data collection techniques have evolved alongside advancing instrumentation. These parameters encompass rock strength (Brown 1981; Hoek 2012), rock quality designation (RQD; Deere and Deere 1988), joint orientation and spacing conditions (Palmström, 1995; Laubscher, 1984), water saturation (Hoek, 2012), weathering (Williamson and Kuhn, 1988), and roughness (Stimpson, 1982).

In the context of this study, base maps were primarily generated using satellite images and digital elevation models (DEMs). Where accessible, outcrops were surveyed using total station equipment and processed using AutoCAD. The collected dataset also incorporated information about landslide occurrences, their locations, and frequency. This study encompasses an examination of the impact of these landslides on local lifestyles and property. Data collection occurred through predefined data sheets (Appendix I), including details about the types of sliding, the primary factors (both direct and indirect) contributing to sliding at the sites, and geological information collection.

Sample Preparation and Data Collection

The geotechnical analyses, the bedrock rests upon the meticulous gathering and assessment of field data. This process of geotechnical data collection and subsequent evaluation holds paramount importance in the domain of rock mechanical analysis. Within the context of this current research, the acquisition of data was meticulously executed through the utilization of diverse field datasheets. These data sheets were thoughtfully designed to capture the intricate rock joint parameters as prescribed by the Rock Mass Rating (RMR) system introduced by Bieniaswki in 1989, as well as the slope parameters stipulated by the Slope Mass Rating (SMR) system proposed by Romana in 1993. In addition to the systematic data collection from the field, acute attention was directed toward recording the nuances of outcrop sketches and overarching geomorphic conditions encountered during the field survey. These on-site sketches and observations contribute critical contextual insights to the dataset, enhancing the richness and precision of the gathered information. Subsequent to the successful completion of the field data collection phase, the research journey advanced into the laboratory domain. Herein, a dual trajectory commenced, encompassing the processing and examination of both collected field data and rock samples. This multi-faceted endeavour aimed to address a diverse spectrum of parameters essential for comprehensive analysis and insightful interpretation.

Discontinuity Data

Within the scope of this research paper, comprehensive data pertaining to joint characteristics and discontinuities was gathered. The terminological breadth of "discontinuity" encompasses an array of features such as minor and major joints, bedding planes or foliation, shear zones, faults, and other surfaces of vulnerability (Matula and Holzer 1978; Hoek et al., 1992; Milne et al., 1992). The data collected encompasses parameters of continuity, aperture, and frequency, alongside their respective orientations. Traditional methodologies designate the orientation of bedding planes as joints, denoting them as 'S0'. Notably, well-defined structures penetrating rock strata are termed master joints (Bell, 1992), while minor joints are denoted as "J1, J2, …, Jn" in sequential order based on a clockwise joint strike direction (Nawani 2011).

The investigation area exhibited four sets of joints, primarily consisting of cross-joint sets, with discernible longitudinal joint sets as well. To quantitatively analyze these features, a total of 39 slopes were examined for joint orientation and spacing. Each slope location entailed around 60-70 joint orientation measurements within a 5×5m² area, contingent upon the available free surface. The joint data was categorized according to four joint properties: joint orientation (kinematic analysis), joint frequency (volumetric analysis), joint continuity (spacing measurements), and joint aperture (observations on separation and infilling). These properties are presented and discussed sequentially.

The collected slope and discontinuity data were subjected to kinematic analysis by projecting dip and strike data of discontinuities and slope surfaces onto stereographic diagrams. While the primary focus of this study is the wedge failure analysis of slopes, the procured discontinuity data can also be qualitatively employed to assess various aspects of discontinuities pertinent to slope stability, in accordance with Barton and Choubey (1977). The kinematic analysis furnishes insights into daylighting, dip-slope interactions, and wedge-friction behaviour (Hoek and Bray, 1981), thereby offering crucial indicators of slope stability. Given the impracticality of obtaining Rock Quality Designation (RQD) data via borehole cores for the extensive Chenab river basin, an alternate approach of joint volumetric analysis was adopted. This method necessitates joint frequency data within a specific volume; consequently, the study involved manual counting of joints along a 2-meter stretch on different exposed rock surfaces.

Moreover, the present research included the measurement of spacings between parallel surfaces, following the approach proposed by Bieniawski (1989). The overall condition of a discontinuity is predicted by joint condition factors, determined by the frictional characteristics of block faces and the relative scale effect (Palmstrøm 1995). This condition is influenced by factors such as length, roughness, separation, weathering of the wall rock or planes of weakness, and infilling material. These factors collectively reflect the extent and impact of a discontinuity, recorded numerically through tabular representations and field-prepared block diagrams and sketches. Interestingly, a prevailing trend of joint separation was observed in the study area, where smaller separations tended to enhance rock stability. This phenomenon arises from increased interlocking of asperities on the rock walls, bolstered by contributions from both filling material and the rock itself (Barton et al., 1974). Given the tectonic activity within the Himalayan terrain, the roughness of discontinuity surfaces provides insights into terrain stability. Furthermore, observations spanned a spectrum from fresh rock exposures to decomposed rock material, with different weathering states (as noted by Lama and Vatukuri 1978) further influencing slope instability conditions.

Rock Strength Data

It is widely recognized by global researchers that the surface strength of intact rock plays a crucial role in the realm of deformation, as indicated in the works of various authors (Coates, 1964; Piteau, 1973; Bieniawski, 1989; Palmstrøm, 1995). Given the diverse range of rock formations present in the study area, a comprehensive approach was undertaken. Approximately 10 to 15 samples from each litho-type were meticulously collected during field expeditions. Subsequent laboratory compression tests were conducted in accordance with the recommendations of the International Society for Rock Mechanics (ISRM 1978). The scarcity of extremely robust rocks within the study area led to the predominance of moderate rock strength classifications. Due to the expansive nature of the study region, it was neither feasible nor practical to assess compressive strength at every location. Consequently, the implementation of the Schmidt Hammer (also known as the Schmidt Impact Hammer, Sclerometer, or Swiss Hammer) facilitated a quantitative extrapolation of strength values, a methodology advocated by Brown (1990). The Original Schmidt Hammer - Type L (S/N 7281), calibrated in February 2012, was utilized during fieldwork. This cost-effective, expeditious, and reliable technique (Haramy and De Marco 1985) generates R-values (dimensionless values) with each impact. Approximately 20-30 R-values were gauged on an approximately 2x2m^2 surface at exposure sites, with the subsequent average serving as the basis for further calculations. Following this, an empirical relationship was established between the measured R-values and the Load Bearing Capacity for distinct rock types. This relationship function was subsequently applied to each location to derive compressive strength, a pivotal parameter in the computation of the Rock Mass Rating (RMR; Bieniawski 1989).

The rock specimens procured from the field underwent trimming at the geotechnical laboratory housed within the Department of Geology at the University of Delhi. These specimens were trimmed to dimensions of 150x20x20mm^3. Post-elimination of flawed and cracked samples, a total of 116 samples were subjected to load bearing capacity analysis. The "Micro Controller Based Compression Testing Machine," designed by Aimil Industries Limited, Delhi, and calibrated in May 2012, played a vital role in these assessments. This testing machine boasts a load capacity spanning 50kN to 5000kN, with capabilities for peak stress calculations. The machine provided the maximum load at the point of failure (P), and subsequently, the Uniaxial Compressive Strength (UCS) of each sample was computed using a designated equation. The average value of UCS for each rock type was then determined and applied in subsequent analyses.

UCS= Maximum Load at Failure / Cross-sectional Area of the Sample

Further, the data was analyzed statistically to determine the variability in the results. The rock samples were collected and tested from different lithological formations to evaluate the correlations between the UCS or Compression test results and the corresponding Smidth Hammer test results.

Water Condition Data

The infiltration of water through fractures and discontinuities within rock formations yields a consequential reduction in the effective normal stress along these joints, thereby diminishing the overall stability of rock slopes. Given the absence of comprehensive groundwater data within the study area, our approach involved visual assessments of seepage conditions, classified into categories such as completely dry, damp, wet, dripping, and flowing. These conditions were evaluated based on a rating system advocated by Bieniawski (1989). To ensure the accurate collection of seepage data from discontinuities, we adhered to the guidelines outlined by Barton et al. (1974).

Nonetheless, it is noteworthy that the development of seepage forces along joints, coupled with instances of rockfall from mountain cliffs as documented in various regions, often exhibits correlations with elevated water pressure and peak rainfall events. Consequently, within the study area, subsurface water may exacerbate instability through various mechanisms. Specifically, these mechanisms encompass but are not limited to, the following four pathways. Furthermore, it's important to acknowledge that water's presence can induce softening and potential outwash, particularly in the context of softer fillings such as clay stones, phyllites, and rocks characterized by a high mica content.

DATA PROCESSING

The Himalayan region is subjected to significant tectonic stress, which in turn generates varying degrees of deformation in the rocks within the designated geotechnical mapping area. This deformation data is crucial and is captured in terms of discontinuities, which serve as fundamental units for geotechnical assessment. Discontinuities mark shifts in the engineering properties of the surrounding rock mass. Therefore, meticulous records of discontinuity orientations and frequencies have been both documented and graphically plotted onto the base map.

For each landslide area, typically characterized by uniform lithological conditions and identified by a specific location number, a comprehensive base map has been meticulously produced using total station survey methods, covering the entire span from crown to toe. These base maps have been enriched with pertinent geological and geotechnical field data as aforementioned.

The accumulated dataset is subjected to analysis through diverse statistical and Geographic Information System (GIS) software tools, enabling the assessment of the New Slope Mass Rating (NSMR; Singh et al. 2013) across the study area. The subsequent sections expound upon the processes involved in software utilization and the resultant analyses.

Kinematic Analysis

The kinematic analysis was facilitated through the utilization of the DIPANALYST software, a readily available resource as highlighted by Admassu and Shakoor (2013). This software mandates specific data inputs, including strata orientation, slope face orientation, and the friction angle of the slide. By processing this information, the software generates results in the form of a failure index. This quantitative approach to kinematic analysis offers insights into the nature of failure (be it plane, toppling, or wedge) by generating corresponding failure indices. In the analysis of each landslide location, more than 100 joint surface orientations (dip angle and direction) were combined with the overarching slope orientation gleaned from slope and aspect maps. Additionally, friction angle values were integrated into the analysis. The interrelation between the friction angle, slope angle, slope azimuth, and failure indices were also scrutinized, with distinct plots aiding in the visualization of these relationships.

Rock Mass Condition Analysis

The Basic Rock Mass Rating (RMR_b) system, introduced by Bieniawski (1989) and also embraced by the South African Council for Scientific and Industrial Research, is employed to numerically characterize the rock condition within the study area. In the current investigation, five distinct parameters were systematically gathered and their correlation with the RMR_b system, as outlined by Bieniawski (1989), is explored.

$$\mathbf{RMR_b = P1 + P2 + P3 + P4 + P5 - P6}$$

Where, P1 is rock strength rating parameter; P2 is rock quality designation rating parameter; P3 is discontinuity spacing rating parameter; P4 is discontinuity condition (sum of five joint parameters, i.e. joint persistence, joint separation, infilling, joint roughness and joint weathering condition rating) rating parameter; P5 is groundwater condition rating parameter; and P6 (introduced by Wickham et al. 1972) is the effect of discontinuity orientation which is a relation between the orientation of dip strata and the orientation of construction. But for the natural slope, this factor is zero so P6 = 0 was used in the above formula.

Slope Stability Analysis

The slope stability is calculated using Slope Mass Rating (SMR; Romana 1993). This slope geomechanical classification provides a systematic approach for quantifying the adjustment factors, and for analyzing the failure susceptibility of rock slopes. The function used for the calculation of SMR is (Romana 1993):

$$\mathbf{SMR = RMR_b + (F1.F2.F3) + F4}$$

Where F1 depends on parallelism between joints and slope face strikes; F2 refers to the joint dip angle in the planar mode of failure; F3 reflects the relationship between slope face and joint dip; and F4 is the excavation factor for the method of excavation.

RESULTS:

As elucidated at the outset of this paper, a comprehensive compilation of geotechnical data encompassing 39 slopes was undertaken. However, due to a significant proportion of these slopes undergoing treatment, their analysis might not yield precise outcomes. Consequently, a focused approach was adopted, concentrating on nine accessible sites exhibiting comparatively minimal or negligible treatment. These selected sites were then subjected to in-depth analysis. The ensuing sections present the outcomes of these analyses for each of the nine landslides, encompassing parameters such as the Basic Rock Mass Rating (RMR_b), the Slope Mass Rating (SMR), and failure indices.

Gul Rock Topple (33°16'0.67"N, 75°2'5.07"E

Situated in close proximity to the Gul village, this landslide is prominently visible along the southeastern-facing slopes. It poses a significant threat to the inhabitants of nearby villages, including Gul and Thatarka. Spanning across a ridge line oriented in a north-south direction near Thatarka village, this landslide area comprises alternating bands of thickly bedded grey sandstone and red claystone belonging to the Murree Formation. The crown of the landslide lies to the north of a small lake adjacent to Budhan village.

Notably, the sandstone beds in this region exhibit considerable variability in thickness, with the thickest beds measuring up to 5 meters, a substantial contrast to the relatively thinner siltstone and claystone beds, measuring around 2 meters and less than 1 meter respectively. The orientation of the bedding plane (S0) here is predominantly east-west, dipping at an angle of 30-50 degrees to the north. Additionally, two joint planes are evident: joint plane J1 exhibits a northeast-southwest strike and dips at 25-30 degrees northwest, while joint plane J2 has a northwest-southeast strike and a dipping angle of approximately 45 degrees northeast.

In close proximity, an old landslide has been observed, previously documented by Singh (2009), near the Gul village along the road leading to Sangaldhan. Here, the road has experienced a shift of approximately 20 meters downwards. The sandstone in this region is extensively jointed, particularly pronounced near the toe of the slope (Jv = 63). Meanwhile, the crown of the landslide is predominantly veiled by inclined pine trees, an indication of ongoing creeping activity. Sizeable sandstone boulders have traversed the slope alongside a wet clay matrix, implying a wet to dripping moisture condition in the upslope areas.

Upon conducting a kinematic analysis of the area, the type of failure is determined to be a rock fall, with respective failure indices of 0.03 and 0.33 for wedge and topple failures. This interpretation aligns with the presence of south-dipping beds associated with wedge failure, while slopes characterized by steep angles are linked to toppling failure mechanisms (Figure 1).

Geotechnical Data Summary for Gul

No. of Data	**: 135 Readings**	Friction Angle	**: 25°**
Slope Azimuth	**: 45°**	Plane Failure Index	**: 0**
Slope Angle	**: 63°**	Wedge Failure Index	**: 0.03**
		Toppling Failure Index	**: 0.33**

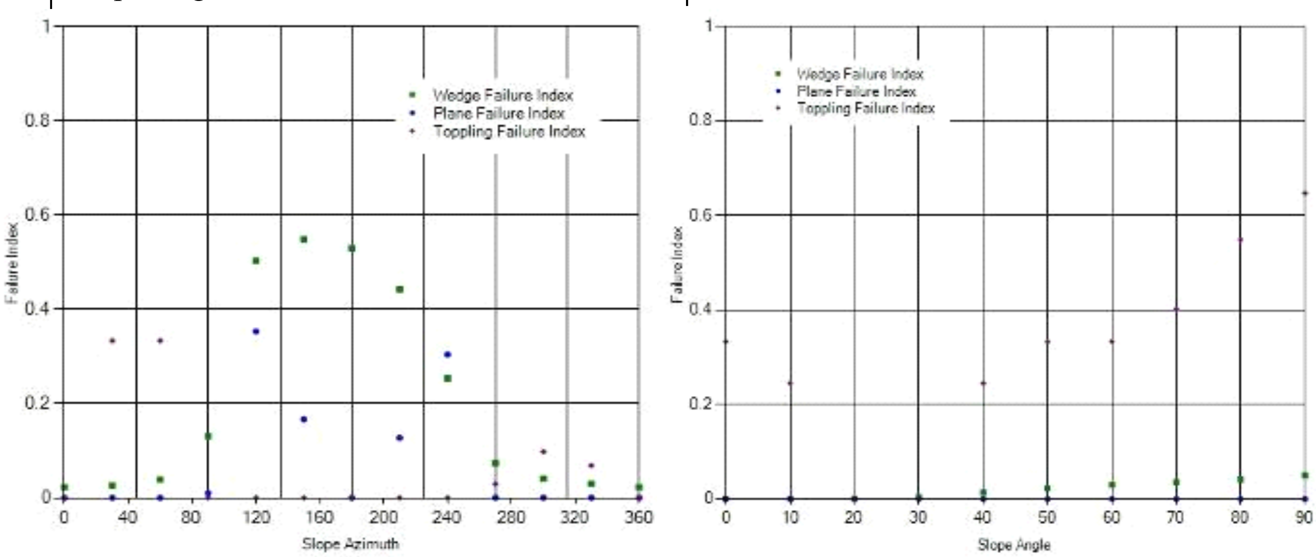

Figure 1: Representation of kinematic data along the Gul road.

ANALYTICAL CONCLUSION

In conclusion, the findings of this study shed light on the prevalence of landslides within the Lesser Himalayan region, particularly in the Murree and Salkhala Formations. While the majority of landslides occur within these rock units, exceptions are reported in areas adjacent to significant geological structures, often involving rocks from the Siwalik Formation. The occurrence of landslides is notably heightened during the monsoon season, although toe cutting emerges as a substantial contributor to landslide reactivation. Notably, regions with moderate elevations within the central part of the study area exhibit a higher frequency of landslides compared to areas of lower or higher elevations.

An intriguing observation is that certain slopes, despite being assigned fair rock conditions based on the Rock Mass Rating (RMR) and Slope Mass Rating (SMR) systems, do not accurately represent the actual site conditions. This underscores the need for improved evaluation techniques, particularly along the Lesser Himalaya.

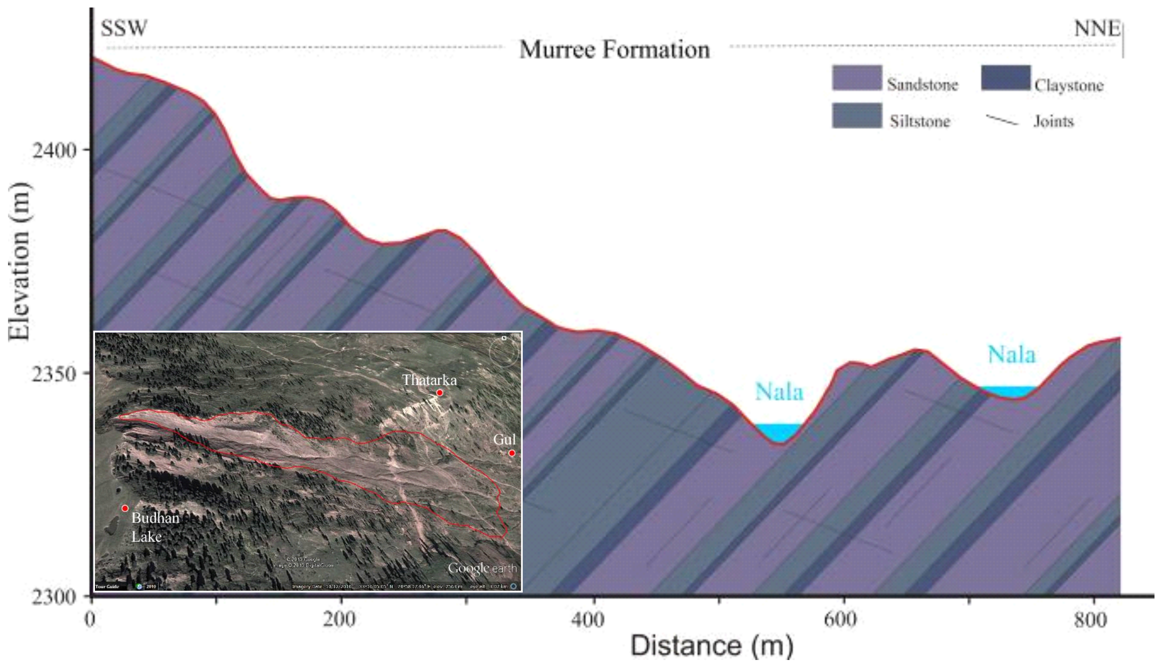

Figure 2: A cross-sectional profile along the Study area (Index map shows the location of study area).

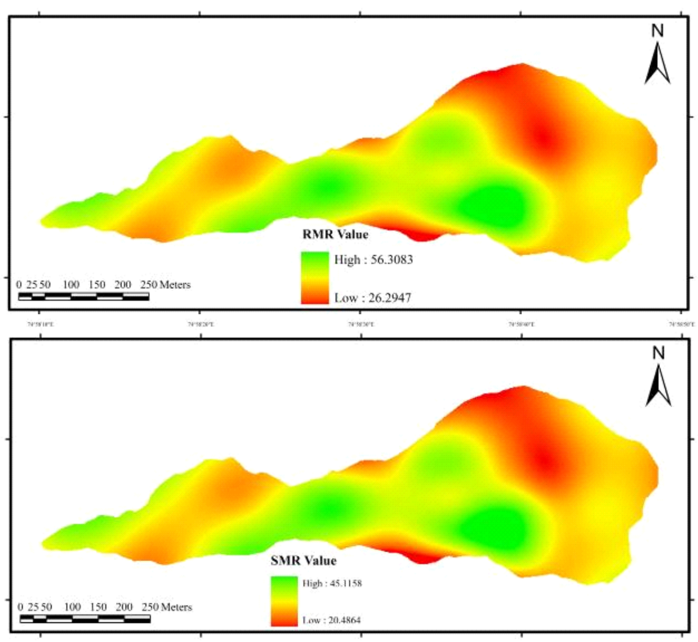

Figure 3: RMR and SMR contours along the study area

Furthermore, the specific case of the landslide near Gul village along the southeast-facing slopes was investigated in detail. This landslide poses a threat to the inhabitants of surrounding villages such as Gul and Thatarka. It is situated amidst alternating bands of thick grey sandstone and red claystone from the Murree Formation, aligned along a ridge line extending north-south near Thatarka village. The crown of this landslide lies northward of a small lake in proximity to Budhan village.

The sandstone beds at this site display significant variations in thickness, with the thickest reaching up to 5 meters, in stark contrast to the siltstone and claystone beds measuring around 2 meters and less than 1 meter respectively. Various planes are observable, including the bedding plane (S0) oriented almost east-west, dipping at 30-50 degrees to the north. Joint planes J1 and J2 exhibit different orientations and dipping angles.

An additional historical landslide, previously documented by Singh et al. (2013), is noted near the Gul village along the road leading to Sangaldhan. This location has experienced a shift of approximately 20 meters downward.

The sandstone in the area is heavily jointed near the toe, while the crown is covered with pine trees exhibiting creeping behaviour. Boulders of massive sandstone mixed with wet clay matrix have slid along the slope, indicating a wet to dripping moisture condition. Kinematic analysis classifies this area as a rock fall type of failure, featuring failure indices for wedge and topple failures.

In summation, this study provides a comprehensive understanding of landslide occurrences, characteristics, and underlying factors within the study area, offering insights that can contribute to effective mitigation and management strategies in similar geological contexts.

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