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Smt. Soma Mondal has assumed the position of Chairman, SAIL w.e.f. 1st January 2021. Smt. Soma Mondal has the distinction of not only being the first woman Functional Director of SAIL, but she is also the first woman Chairman of the Company. A graduate in Electrical Engineering from National Institute of Technology, Rourkela, in 1984. She has over 35 years of experience in the metal industry. She commenced her career as a Graduate Engineer Trainee at NALCO and rose through the ranks to take over the mantle of Director (Commercial) at NALCO in the year 2014.

She joined SAIL in March, 2017 as Director (Commercial), at SAIL, she spearheaded the implementation of the marketing strategies emerging from the Comprehensive Turnaround Roadmap for the Comapny since 2017, which witnessed SAIL increasing its sales and expanding the market reach progressively year on year. SAIL achieved best ever sales volume consecutively for the three Financial Years from 2017-18 to 2019-20, and the momentum is still continuing in the current financial year of 2020-21, despite the challenges posed by COVID-19. Promoting the branding efforts of the various products of the Company. She was instrumental in the launch of new brands viz. "NEX" and "SAIL SeQR" to promote PF STructural Sections and TMT Bars respectively. Sensing the need for improving sales through the retail channels, she has made relentless effort to set up two-tire distribution network. To educate and tap the vast potential of rural India, the "Gaon Ki Ore" workshops were organised in almost all the States and Union Territories across the Country.

For meeting the evolving expectations of the challenging domestic market, Smt. Soma Mondal has introduced timely reforms in the Marketing Organisation Sructure of SAIL. For better management and marketing of the enhanced volumes after the modernization & expansion of SAIL, she created three verticals viz. Sales, Marketing and Services. This was envisaged to bring in focus and effective micro management in the marketing operations. She is a member of the CII-National Committee on Steel and Chairperson of the CII sub-committee on 'Safeguard for Tariff and Non-Tariff Barrier".

COAL NEWS

COAL INDIA RAISES CAPEX BUDGET BY 30% TO RS 13,000 CRORE FOR FY21

Coal India Limited has raised its capital expenditure budget by 30% to Rs 13,000 crore for the ongoing fiscal amid government directions to all Central PSUs to step up their spend to stimulate economic activity. The miner's capex target for 2020-21 was Rs.10,000 crore. Of the additional Rs 3,000 crore injected into CIL's capex budget, South Eastern Coalfields Ltd- the largest coal producing subsidiary of the company, accounts for Rs 800 crore followed by CIL headquarters with Rs 585 crore and Mahanadi Coalfields Ltd with Rs 550 crore, a company executive said. Central Coalfields Ltd will spend Rs 460 crore. The major heads CIL has identified for capex are land acquisition, procurement of heavy earth moving machinery, upgrade of rail evacuation infrastructure and mine development. CIL has posted 166% growth at Rs 7,801 crore during the first nine months of the fiscal ending December'20. "CIL has utilized 78% of its total original capex budget during April-December'20. CIL was directed by the coal ministry to achieve Rs 7,500 crore capex utilisation by the closure of December'20 against which the actual capex utilization was Rs 7,801 crore," the executive said.

CIL's capex for the current year's Q3, ending December'20, at Rs 2,778 crore posted a 90% growth against Rs 1,463 crore of the same quarter last year. For Q2 of the current year CIL logged 312% capex growth and a growth of 86.5% in Q1. CIL's capex during the current financial year makes it one of the top spenders among the Indian PSUs. Coal India's land possession and civil construction jobs, among other activities, were affected during the Covid -led slowdown. Subsequently, headway could be made with the situation improving post unlock, nudging its increased capital expenditure. "CIL will be closely monitoring the progress of the capital expenditure to achieve the revised target of Rs 13,000 crore in the current FY," said the executive.

INDIA'S COAL IMPORT DROPS 17% IN APRIL-NOVEMBER: MJUNCTION

India's coal import declined by 17 per cent to 137.16 million tonne (MT) in the April-November period of the current fiscal. The country had imported 165.35 MT of coal in the year-ago period, according to provisional compilation by mjunction, based on monitoring of vessels' positions and data received from shipping companies. mjunction, a joint venture between Tata Steel and SAIL, is a B2B e-commerce company that also publishes research reports on coal and steel verticals. The country's coal import in November also dropped to 20.35 MT from 21.72 MT in the corresponding month of previous fiscal, it said. "India's coal and coke imports during November 2020 through the major and non-major ports are estimated to have decreased by 6.3 per cent over November 2019," it said. Of the total imports during November, non-coking coal was at 13.77 MT, against 15.32 MT imported in the same month last year. Coking coal imports were at 4.28 MT, up from 4.09 MT in November last fiscal.

During April-November period, non-coking coal import was at 91.44 MT as compared to 114.05 MT during the same period of the previous fiscal. Coking coal imports were recorded at 28.18 MT, lower than 32.72 MT imported during the same period a year ago. "Coal demand from utilities witnessed a modest easing following the festive season, leading to an increase in coal stockpile in the system in December. This coupled with the recent surge in seaborne thermal coal prices in the international markets is expected to restrict the volumes, going forward," Vinaya Varma, MD and CEO, mjunction services said while commenting on the coal import trend. Coal India, which accounts for over 80 per cent of domestic coal output, is aiming at substituting imported dry fuel of 80-85 million tonne with more domestic supplies in the current fiscal. The miner has asked power plants in the coastal areas to submit proposals for a gradual increase of its supplies to these units to reduce foreign exchange outgo. The country had imported 248 million tonne of coal in 2019-20, resulting in an outflow of around Rs 1 lakh crore of foreign exchange.

MINING NEWS

DOMESTIC PRIMARY NON-FERROUS METAL INDUSTRY'S CREDIT PROFILE SHOWS SIGNS OF IMPROVEMENT: ICRA

Rating agency Icra said cash flow visibility due to a sharp recovery in non-ferrous metal prices has led to a revision in the credit outlook of the industry to stable. "The credit profile of the domestic primary non-ferrous metal industry has shown signs of improvement in the last few months as prices of the metals have registered sharp recoveries from the lows witnessed in the months of March and April," Icra said in a statement. Currently, international prices of aluminium, copper and zinc are up by 12 per cent, 28 per cent and 17 per cent, respectively on a Year-on-Year (Y-o-

Y) basis. The increase is 38 per cent, 70 per cent and 47 per cent from their respective lows registered in March-April last year.

The increase in prices in turn is a result of a steady turnaround in demand conditions, Icra said, adding that prices may have also been influenced by the ample liquidity in the global financial markets, which play a role in determining non-ferrous prices. "The current buoyancy in prices is a result of the improvement in global macroeconomic sentiments with the beginning of vaccination drive against COVID-19. The favourable momentum is likely to continue in the next 12-15 months, thereby, resulting in a steady turnaround in the risk profile of the industry," Icra Senior Vice-President and Group Head Corporate Sector Ratings Jayanta Roy said. In the first half of FY21, the pandemic had severely impacted the global automobile, construction and electrical machinery industries, which together contribute 75-85 per cent to the global non-ferrous metal demand. Consequently, during that period, consumption of these metals had contracted significantly, ranging from 3-4 per cent for copper and zinc, and up to 8 per cent for aluminium on a Y-o-Y basis.

Currently, global demand is on the path to recovery led by a turnaround in demand conditions in China. Although global growth in non-ferrous metal demand on a Y-o-Y basis is estimated to have remained in the negative territory in the third quarter of this financial year, growth outlook is favourable for the coming quarters. The recent improvement in non-ferrous metal prices coupled with a correction in input costs would support consolidated operating margin of the domestic industry, which is likely to improve to 21 per cent in FY21 and subsequently to 23 per cent in FY22 from 15 per cent in FY20. "Downside risks, however, remain as the macroeconomic uncertainties due to the COVID-19 pandemic are yet to dissipate. Nevertheless, prices are unlikely to reach the lows of Q1 FY2021, thereby supporting margins," Roy said.

TOP FOUR STEEL PLAYERS' PRODUCTION RISES 6 PER CENT TO ABOUT 15 MT IN OCT-DEC

The country's top four steel makers jointly produced 14.95 million tonne (MT) steel in the October-December quarter of the current fiscal, registering a 6 per cent year-on-year rise. The total steel output of JSPL, JSW Steel, SAIL and Tata Steel India was 14.09 MT during the same quarter of 2019-20. During the quarter under review, the total sales of the steel producers - excluding JSW Steel - surged 2.25 per cent to 10.88 MT, as against 10.64 MT in the year-ago quarter, according to the data provided by the companies.

Among all four steel players, Tata Steel India was the top producer in the October-December period of FY21. Its total output from India operations was 4.60 MT during in the quarter. At 4.60 MT, the company's output was 3 per cent higher compared to 4.47 MT steel it had produced in the year-ago period. Its sales from India operations slipped 4 per cent to 4.66 MT from 4.85 MT. Steel Authority of India Ltd (SAIL) production grew 9 per cent to 4.37 MT steel during October-December compared to 4 MT a year ago. Its total sales were at 4.32 MT, up about 6 per cent from 4.09 MT in the same quarter preceding fiscal. JSW Steel's output during the period under review rose 2 per cent to 4.08 MT, as against 4.02 MT in the year-ago period. Jindal Steel and Power Ltd NSE 1.82 % (JSPL) output surged 18 per cent to 1.9 MT in the October-December quarter from 1.6 MT a year ago. Its sales increased by 12 per cent to 1.9 MT from 1.7 MT. JSPL, JSW Steel, SAIL and Tata Steel India jointly contribute about 45 per cent to India's total steel production annually.

CEMENT DEMAND EXPECTED TO GROW BY UP TO 20 PC IN FY22: ICRA

Cement demand in India is likely to increase by 18%-20% in FY2022 over FY2021 with the volumes reaching back to around FY2019-FY2020 levels, amid a strong rural demand including affordable housing and a recovery in the infrastructure segment. “After a sharp contraction in volumes in FY 2021, ICRA NSE 0.16 % expects it to grow by 18%-20% in FY2022. The rural offtake is likely to be supported by the positive farm sentiment with the timely rabi sowing and favourable groundwater and reservoir levels, which are likely to boost rabi yields,” said Anupama Reddy, Assistant Vice President, ICRA.

NTPC ARM MAKES 50 MW SOLAR PROJECT IN KASARGOD COMMERCIALLY OPERATIONAL

NTPC said that its arm THDC India has made its maiden solar project in Kerala commercially operational. "The maiden solar power project of 50 MW capacity situated at Kasargod Solar Park at Kasargod, Kerala of THDC India Limited (a subsidiary company of NTPC Limited), is declared on commercial operation from December 31, 2020," a BSE filing said. With this, the commissioned as well as commercial capacity of the THDC India Limited and NTPC group has become 1,587 MW and 62,975 MW, respectively. NTPC has planned to have a 130 GW power generation capacity by 2032. Its non-fossil fuel based capacity would be 30 per cent.

L&T HYDROCARBON ENGINEERING BAGS UP TO RS 5,000-CRORE ORDER FROM ONGC

L&T Hydrocarbon Engineering (LTHE) said it has bagged an order worth up to Rs 5,000 crore from Oil & Natural Gas Corporation (ONGC). The engineering and construction company, however, did not provide the exact value of the contract, but as per its project classification, the value of a large order ranges between Rs 2,500 crore and Rs 5,000 crore. "L&T Hydrocarbon Engineering Limited (LTHE), a wholly-owned subsidiary of Larsen and Toubro, has secured a contract from Oil & Natural Gas Corporation (ONGC) for their new living quarter (LQ) and revamp at 'NQ Complex' project," L&T said in a regulatory filing. L&T said the engineering, procurement, construction, installation and commissioning (EPICC) contract is for a new living quarter platform, 'NQL Platform' of 120 men capacity, bridge (with intermediate support) to existing 'NQO Complex' and major revamping/replacement of existing process systems / facilities at 'NQ Complex' in ONGC's Mumbai High Asset on the West Coast of India. The contract has been awarded through international competitive bidding on a lump sum turnkey basis, L&T said in a statement.

Subramanian Sarma, Whole-time Director and senior EVP (Energy), L&T and CEO and MD of LTHE said, "We have been delivering several large and mega projects for ONGC over the past decade. Our world-class fabrication facilities at Hazira (West Coast) and Kattupalli (East Coast) enable us to maximize the local content, entirely supporting the Government's AatmaNirbhar Bharat Policy."

SAIL REPORTS 9%GROWTH IN CRUDE STEEL PRODUCTION

Steel Authority of India reported a 9% growth in crude steel production for the December quarter of FY21 at 4.37 million tonnes as against a production of 4 mt during the same period last year. SAIL's sales volume registered a growth of 5.6% during the Q3 of FY 21, at 4.32 million tonnes. "The first quarter was impacted due to the onset of the pandemic but gradually we have scaled up our performance by enhancing the volumes. It is heartening that the pre-covid levels have already been reached and the production has grown in the last quarter," said the newly appointed chairman of SAIL, Soma Mondal in a statement.The company's domestic sales increased 9% to 4.05 mt during the December quarter and exports came down by 25% to 0.27 mt, the company's exports during the previous quarter was recorded at 0.67 million tonnes. "The domestic steel consumption has a positive outlook as the economy is reviving and all sectors have started to pick-up. We are confident of seizing the unfolding opportunities in the steel market," Mondal added.

Speaking to ET, SAIL's ex-chairman, Anil Kumar Chaudhary earlier said that the company is on a path to deleverage its balance sheet and is planning to bring down the debt levels to Rs 45,000 crore by the end of December 2020 and to Rs 40,000 crore by March 2021 from its current level of Rs 50,638 crore. "SAIL has significantly reduced the net debt to Rs 44308 crores on 31st Dec 2020 ...The company continues its efforts to deleverage further," said the company. Amid an iron ore shortage in the market, the ministry of mines has allowed SAIL to sell 25% of its total iron ore production calculated on the basis of cumulative production of all captive mines in a state, as well as sub-grade minerals lying at the mine pit heads.

"In compliance of this notification, SAIL has already sold approximately 2.16 Million Tonnes (MT) of fresh fines through auction during the current financial year from its various mines. Around 0.3 MT of dump fines and tailings have also been successfully auctioned during this period," the company said in a statement, adding that this move has helped to alleviate the shortage of iron ore in the market. Yet another top steelmaker, JSW Steel achieved crude steel production of 4.08 mt in the third quarter of FY 21, registering a growth of 6% quarter on quarter and 2% year-on-year. While the primary steelmakers are getting back to pre-covid levels, with an increase in production of crude steel on a quarterly basis, the production at the end of 9 months of FY21 is still falling short due to the peak lockdown period during the initial few quarters and due to a shortage of iron ore. JSW Steel's crude steel production at the end of 9 months ending December fell by 10% to 10.89 million tonnes, as against a production of 12.09 mt during the same period last year. SAIL's production for the period of April-December was recorded at 10.60 million tonnes, a fall of 10.33% year-on year.

DEVELOPMENT OF SIKAR LIGHT FOR UG COAL MINES BY IIT(KHARAGPUR)

A historical moment for Jhanjra underground coal mine of ECL, the first-underground coal mine in India illuminated without battery or electricity, by carrying solar light. Our research team on Optical Fiber-Based Solar Illumination of Underground Mine with Simultaneous Communication Potential have developed a novel solar energy driven illumination system for underground mines as a part of a research and development project sponsored by Coal India Limited. With the help of this system bright illumination, much higher than what is required as per safety norms in the pit-bottom, is achieved without the use of any electrical components in the underground mine. The team comprising of junior research fellows Mr. Sarbojit Mukherjee, Mr. Dushasan Kundu, project officer Mr. Uttam Kumar Ghara, faculty members Dr. Shivakiran Bhaktha B.N. (Dept. of Physics) and Prof. Khanindra Pathak (Dept. of Mining Engg.) have installed the illumination system successfully in Jhanjra underground coal mine. At the mine surface, sunlight is coupled into a multimode optical fiber with the help of a lab-built solar tracker and a Fresnel lens, designed and developed at IIT Kharagpur. We are going to use the same light carrier for communicating also.

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Feasibility Study of Chromite Mine Overburden Material as a Paste Backfill Material

S.P. Kumar* A. K. Verma* P. C. Jat*

ABSTRACT

In this paper, feasibility study of waste overburden material is carried out to establish its use as a paste backfill material. The importance of this engineering solution is pertinent when there is scarcity of land for dump disposal, environmental concern and hazards related with the failure of dump. For this study, dump material are characterized by investigating its physico-mechanical properties and elemental composition. Later on, the dump material are mixed with water to test its flowability characteristics like slump test, rheological behaviour and pipe loop tests. The dump material mainly consists of Limonite having percentage of Fe varies between 45% to 60%. From the laboratory test, it is found that the waste dump material is poorly graded, having low permeability, bleeding potential and settling characteristics during pipe loop test. The preliminary investigation shows that the dump material is not pastable.

Keywords—waste dump, limonite, paste backfill, laboratory testing, pipe loop test

INTRODUCTION

Storage of overburden material from an open pit mine as a waste dump is a challenge for mine planners and management. This is not only requires the land but also requires its maintenance to control its failure. In this study, the open pit mine has reached upto its ultimate depth and hence the mine management is planning for further excavation by underground methods. Here, during underground excavation, waste dump can be utilized as a backfill material for ground control. The waste dump material manly consists of weathered limonite. There are varies methods by which backfilling can be done and the most popular methods are rock filling, slurry fill and paste fill. Deb and Panchal (2018) mentioned that the paste backfill is the most popular method because the solid concentration is about 70~85% and hence dewatering operations are minimized.

Waste dump samples are tested in the laboratory to ascertain its feasibility as a paste backfill material. Paste backfill is defined as a non-segregating, low-plasticity and high- density material produced with two basic ingredients, namely solid materials (between 70% to 85% by weight) and water.

A total of 950 kg of samples was collected from the dump in 16 bags in the month of January 2019. The collected dump material is the host rock of chromite called limonite which is mostly friable due to weathering and look like compacted soil. The waste dump samples are re-sampled in the laboratory by coning and quartering sampling method (see Figure 1) for its laboratory characterization. The purpose of material characterization is to study the flowablility characteristics of waste material during pipe flow and strength evaluation behaviour after placement into the underground stope. In this paper, the flowability characteristics are evaluated by as received moisture content, particle size distribution, slump test, rheological behaviour and Scanning Electron Microscopy - Energy Dispersive X- Ray Spectroscopy (SEM-EDS) test.

Fig 1: Coning and quartering of waste dump sample

The laboratory tests results shows that, waste dump is iron rich material where the particle size distribution is poorly graded with the deficiency of finer particles, settling and bleeding characteristics during flow. Hence, this material is not suited for paste backfill.

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In future, laboratory experiment will be extended with different binders and super plasticizer, which may improve the particle size distribution and helps in increasing the finer particles and also help to attain the better flowablity and rheology during pipe transportation and acquired the required strength after placement into an underground stopes.

CASE STUDY OF DUMP

The case study, Chromite mine is located in Sukinda valley, Jajpur District of Odisha is studied. This open pit mine which already reached upto its ultimate depth. The host rock or the overburden of this open pit mine is which mainly consist of limonite as shown in Figure 2. The overburden rock is too friable and weathered. Because of constraint is expansion of the open pit operation on the surface, company is planning to continue the excavation by underground mining method. In case of underground mining, the waste dump can be used as a backfill material. This paper discusses the general overview of laboratory results of waste dump material which includes its physical, chemical, and rheological.

Fig. 2: Mine waste dump

Paste backfill technology is progressively used because it provides better ground control and adding least water to the underground. Paste backfill improves the ground support for mine structures with high recovery of ore bodies, increased stability of underground excavations, reduction in mine operating cost, eliminates dewatering process during curing time, increased safety, efficiency and productivity for mine operators by preventing from caving and roof falls and also reduces the storage requirements for overburden waste dumps (Belem and Benzaazoua, 2002).

LABORATORY TESTS

The properties of paste backfill material is essential for the design of cost-effective, safe and durable paste backfill structures. The primary properties and characteristics of paste backfill are categorized by their physical properties (such as particle size distribution, permeability, porosity, void ratio, degree of saturation, density), rheological properties (such as slump, viscosity, yield shear strength during pipe flow) and mineralogical properties aspects (such as particle size and shape, mineral content) (Ghirian and Fall, 2013). There are three main phases of paste backfilling system, i.e paste preparation, then transportation through pipe line and finally placement into the stopes.

A. Particle Size Distribution

Literature shows that the weight percentage of fines plays an important role to decide the flow behaviour the solid particle through pipes. Finer particles help to float the coarse grains into a paste form and allow the particle to be suspended (Kuganathan, 2005). Paste backfill material can be categorized into three classes, namely, coarse, medium and fine based on fine (<20 ìm) percentage present in fill material (Landriault, 1995). Well graded material produces a densely packed paste backfill.

Generally, backfill material with at least 15% by wt. of finer particles (<20 ìm) holds sufficient amount of water which helps to create a non-segregating mixture and facilitate plug flow, otherwise the particles will bleed and settle (Landriault, 2001). Coefficient of uniformity (Cu) of backfill material varies between 4 and 6 shows improved packing density and reduced porosity (Sargeant, 2008).

B. Particle Shape

Particle size of the backfill material can be of different shape such as flat, round or angular and smooth or rough in texture. Particles that are flat in shape will generally settle down slowly than particles which are round in shape with equal specific gravity, thus affecting the thickening and consolidation, and the drainage time of paste backfill (Henderson and Revell, 2005). Particle shape can also affect the void ratio and path connections to hold and transport fluids.

C. Hydraulic Conductivity

Literature shows that the hydraulic conductivity of paste backfill lies in the range of 10^{-5} and 10^{-6} cm/s (Newman et al., 2001). It is also found that hydraulic conductivity of paste backfill decreases with increase in curing time and increase in binder content or decrease in the water cement (w/c) ratio (Fall et al., 2009) Low permeability inhibiting leaching of toxic and heavy elements from the paste backfill.

D. Porosity and Void Ratio

The loss of water through drainage may cause solids to settle down (causing increase in the packing density) and may consequently reduce the total porosity and void ratio of the backfill material. Past studies (Kesimal et al., 2003; Fall et al., 2004; Ercikdi et al., 2013; Cihangir et al., 2014) shows that the overall porosity of backfill tends to decrease with increase in fines content.

E. Specific Gravity

The solid mass concentration of paste backfill for a specific slump has been found to be greatly influenced by the specific gravity of the backfill material (Yilmaz et al., 2007). There is a linear relationship between paste backfill strength and the specific gravity of backfill material.

F. SEM-EDS test

SEM coupled with EDS, can provide elemental composition and map out the lateral distribution of elements from selected area in the sample. A very small portion of representative sample is to be chosen for SEM-EDS (Scanning electron microscopy and energy dispersive spectrometry) test. The SEM uses a focused beam of high energy electrons that produces different signal on the surface of the sample, which derive from electron-sample interaction carries information about the samples such as external morphological texture, particle angularity and chemical composition of sample (Panchal 2018). This test is carried out using ZEISS EVO 60 and ZEISS MERLIN as shown in Figure 3.

Fig. 3: SEM EDS System

G. Slump height

Standard slump test is an indirect method of characterizing the rheology in terms of flowability. Slump can be defined as a measure of the drop in height when released from a truncated metal cone of standard dimension according to ASTM standard (ASTM C143/C143M-12) as shown in Figure 4. The ideal slump for good workability of paste backfill falls between 150 and 250 mm (Landriault, 1995). In case of underground paste backfilling systems, optimum slump generally varies between 178 and 203 mm (Belem and Benzaazoua, 2008).

Fig. 4: Slump test

H. Rheological Properties

Rheological properties of the paste mixture is determined by Rheometer (see Figure 5) to find the yield shear stress and viscosity. Generally paste backfill are assumed to be Bingham plastic fluid in which, the yield shear stress is exponentially proportional to the solid concentration of the fluid. It usually exhibits yield stress ranging from 200 to 700 Pa (Potvin, 2005; Sofra and Boger, 2002). The yield stress must be large enough to generate laminar flow during pipeline transportation and prevent settling of the solid materials, but not so high that requires high flow velocity, turbulent flow, high friction factors and friction loss and subsequently creates start-up problems (Potvin, 2005). Plastic viscosity of paste mixture at lower shear rate lies between 1.0 and 3.0 Pa s but at higher shear rate it varies from 0.3 to 0.6 Pa s (Kuganathan, 2007).

Fig. 5: Rheometer test

RESULTS AND DISCUSSIONS

The insitu moisture content of as received sample was carried out and found to be 12.41% and varies between 9.24% and 16.43%. After coning and quartering laboratory samples were prepared for other tests like permeability, specific gravity, particle size distribution analysis, slump test, morphology and elemental content by SEM-EDS (Deb and Verma, 2016). From the particle size distribution curve of dump material is found to be poorly graded with an average coefficient of uniformity (C_u) of 9.64 and the average coefficient of curvature (C_c) of 0.75 as shown in Figure 6.

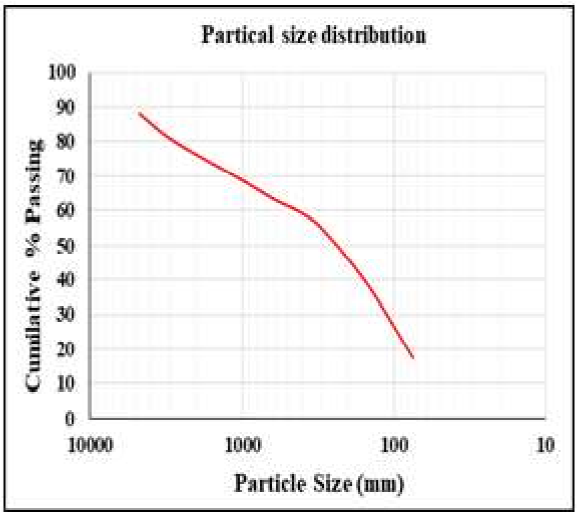

Fig. 6: Particle size distribution

The specific gravity of dump material is found to be 2.85. The specific gravity influences the solid mass concentration of paste backfill for a specific slump. Hydraulic conductivity is coming out to be an average value of 0.0002 cm/s. These values of hydraulic conductivity is coming under low degree of permeability and hence it is not suitable for paste backfill. During slump test, slump height varies between 95 mm and 125 mm with the corresponding mixing water percentage of solid mass concentration being between 46% and 48% respectively as shown in Figure 7. Here it is observed that it is taking much water during mixing to get the required slump for fluidity and flowability of paste mixture but after 10-15 minutes it starts to bleed and almost 7% to 8% of total water were discharged. Scanning Electron Microscopy - Energy Dispersive X-Ray Spectroscopy (SEM- EDX) shows the element content and the microscopic picture of the limonite dump material. The results of SEM-EDX made clear that it is an iron rich material of around 45% to 55% by weight and from the microscopic picture, it verifies the particle size distribution that there is very less finer particle as shown in Figure 8. Finally pipe loop tests are conducted to evaluate the time dependent flow behaviour of paste mixture through pipe and also to measure the pressure gradient across the pipe line during paste flow. It has been observed that the paste flow is good for 10 minutes after that three phenomena are observed (a) disintegration of the material, (b) settling of paste material and (c) bleeding of water as shown in figure 9.

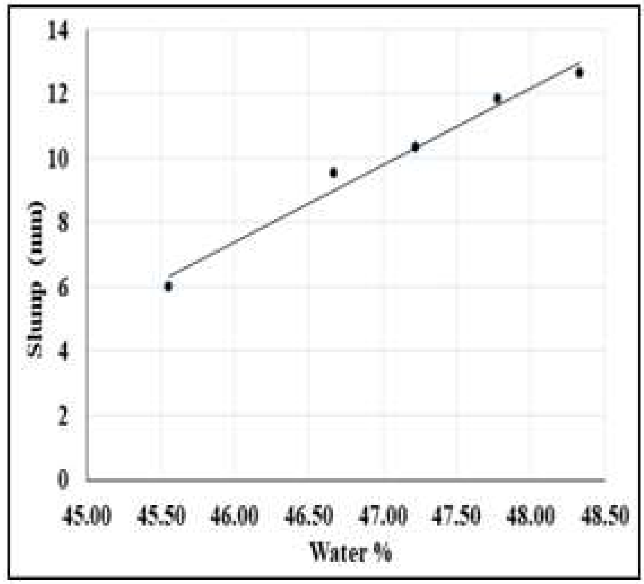

Fig. 7: Variation of slump height with water percentage

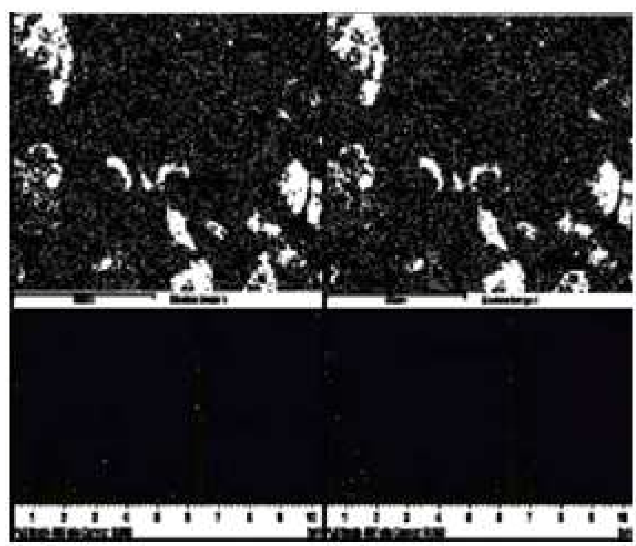

Fig. 8: SEM-EDX results for elemental composition and shape of microstructure

Fig. 9: Consistency of paste mixture

CONCLUSIONS

From the laboratory test for material characterization and pipe loop tests, the following conclusions are drawn

- As received average moisture content is 12.41% that means there is some water content is already present in the dump material
- From particle size distribution, it is found the that this material is poorly graded and fine percentage is very less
- SEM-EDS shows that the iron (Fe) content in the sample is about 45% to 60% and the shape of the material is spherical. It is also verifying the results obtained from PSD test that
- Slump cone test shows that the average slump height of 110mm with almost 47.5% of water.
- The visual observation of the pipe loop test shows that the material is not flowable and the pipe get chocked during the experiment.
- Finally, it is concluded that this is an iron reached material and it gets settle due to its high density during pipe flow and it is not paste-able. Hence, it needs some binders to ease its flowability characteristics.

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Support Design of a Panel During Depillaring with Sand Stowing using Numerical Modelling and Empirical Approaches

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ABSTRACT

In a depillaring operation, coal pillar's size reduces by driving split/slice galleries which encounter strata control issues like roof deformations and redistribution of in-situ stresses around the working face. The strata control issues depend on nature of overlying/underlying rock masses and goaf management approach. The goaf is managed either with sand stowing or regular caving. In both cases, load is transferred towards working area that deteriorates the competency of immediate roof of roadways and natural supports too that warrants additional support for smooth mining. Here, applied support for proposed depillaring with stowing is designed on the basis of available empirical approaches and verified by numerical modeling using Fast Lagrangian Analysis of Continua in three dimensions (FLAC3D) package. Rock load at slice/split/original galleries and junctions is estimated and proposed support is designed considering safety factor more than 1.5. For the considered site, support safety factor found to be 1.82 to 4.80 and 2.1 to 5.58 from empirical approaches and numerical models respectively. These studies revealed that rock load to be supported in depillaring with stowing is comparatively lesser than caving due to less span of exposed void around the goaf line. Discussing the geo-mining conditions, a support design for depillaring with stowing is proposed in JK Nagar(R) Colliery of Eastern Coalfields Limited.

Keywords— Depillaring; Stowing; Rock Load; Numerical Modeling; Support; Safety factor

INTRODUCTION

Semi-mechanised technology is being used to extract developed coal pillars of Bogra seam (R-VI) in conjunction with hydraulic sand stowing at JK Nagar (R) Colliery. The colliery is under Satgram Area of Eastern Coalfield Limited (ECL), Raniganj, West Bengal. Semi- mechanized technology consists of drilling and blasting and Side Discharge Loader (SDL)/Load Haul Dumper (LHD) which are being used for coal evacuation from face. Average thickness of the working seam is 2.74 m, which is just below the immediate roof of around 2 m sandy shale. It is planned to depillar panel BSP-4 of the Bogra seam, which are situated at depth of around 190 m. Average pillar size and gallery width of the proposed panel are 30.5 m x 30.5 m and 4.2 m respectively. Two overlying virgin seams, namely Century and Narainkuri (thin) are lying at around 54 m and 80 m above the working seam respectively. However, the Nega seam (R-VII) has been exhausted and water logged, which is lying at a distance of around 125 m above the working seam. The underlying Satgram (R-V) seam below the proposed panel BSP-4 is virgin.

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Based on the collected geo-mining details of the panel, support system (CMR 2017) is designed using available empirical and numerical modelling approaches. Numerical modelling is a popular tool ((Murali et al., 2001; Coggan et al., 2012; Basarir et al., 2015; Singh et al. 2016, Bai et al., 2017; Kumar et al., 2019) to study of underground structures behaviour during mining activities. Attempts are made to evaluate rock load height (RLH) on numerical models using Fast Lagrangian Analysis of Continua in three dimensions (FLAC3D) package (Itasca, 2012). RLH is measured at the split, slice, junction and at the goaf edge from the numerical simulation. Elastic constitutive model is assigned to all the considered layers in simulation and RLH is measured using the safety factor contours up to 1.5 in the immediate roof. Empirical approaches established in Indian coalfields are considered for the support design.

PANELS DETAILS

The proposed panel is located near leasehold boundary towards South-West side of the mine (Fig. 1). The panel is to be depillared in conjunction with hydraulic stowing following diagonal line of extraction. Borehole No. 0350/CM/011 (Fig. 1)is the nearest to these panels which shows that 1.88 m thick sandy shale as immediate roof of 2.15m thick Bogra seam (Fig. 2a). However, in the proposed the panel thickness of the seam is up to 3.46 m (Fig. 2b). The seam was developed along floor leaving around 0.6 m to 1 m thick coal band in roof against the sandy shale. Geomining details of the panel are summarized in Table 1.

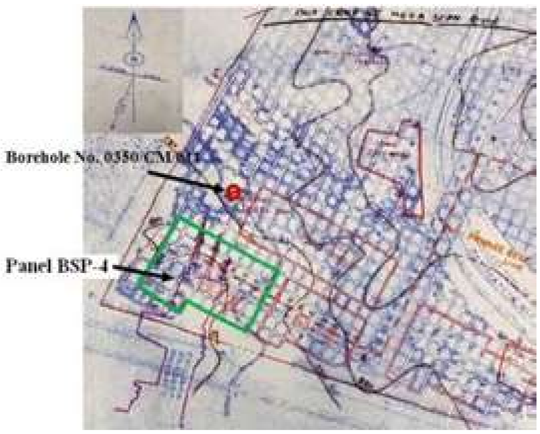

Fig. 1: Part plan of developed Bogra seam (R-VI) showing the location of proposed depillaring panel BSP-4.

Table 1: Geo-mining details of the panel BSP-4

Parameters	BSP-4
Name of the seam	Bogra seam (R-VI)
Seam thickness, m	2.15 - 3
Gradient of Seam	1 in 16
Depth of cover, m	187- 194
Average size of pillars (centre to centre)	30.5 x 30.5
Number of pillars	43
Working height, m	2.4 to 2.7
Gallery width, m	4.2
Immediate roof	Sandy shale
Immediate floor (Thickness, m)	Coal and Sandy shale
Rock Mass Rating	52.5

PROPOSED MANNER OF PILLAR EXTRACTION

A pillar is to be divided into two equal parts by driving a level spilt gallery of not more than 4.2 m width. The half part of the pillar i.e. fender/stook is to be extracted systematically by driving rise-dip slices of 4.8 m width against in-bye rib left towards the goaf side. Height of extraction in the depillaring panels shall be in order to maintain at least 0.6 m thick coal band in roof against sandy shale lying just immediate to the coal seam. Quick

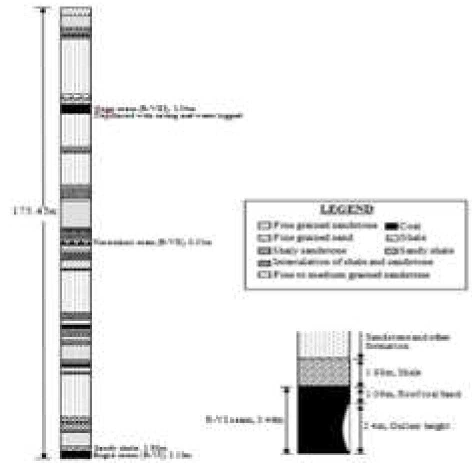

Fig. 2: (a) Overlying strata of working Bogra seam (R-VI) and (b) Section of the Bogra seam in the proposed panel.

setting cement grouted roof bolts are to be installed in every freshly exposed roof during the slicing operation. For the considered size of a pillar (Fig. 3), the width of first and last ribs in a fender has to be kept 3 m; however, middle ribs between two consecutive slices should be of 2.7 m in width. In case of variation in pillar size, width of left- out middle rib shall not be less than 2 m, whereas, first and last rib in a fender shall not be less than 3 m. Void created after completion of a slice, is to be filled with sand before extraction of consecutive next slice maintaining a diagonal line of extraction. Volume of void without sand stowing should not exceed 1500 m^3 at any stage of the depillaring operation. The proposed manner of pillar extraction in conjunction with sand stowing is to be practiced in the panel BSP-4 as shown in Fig. 3.

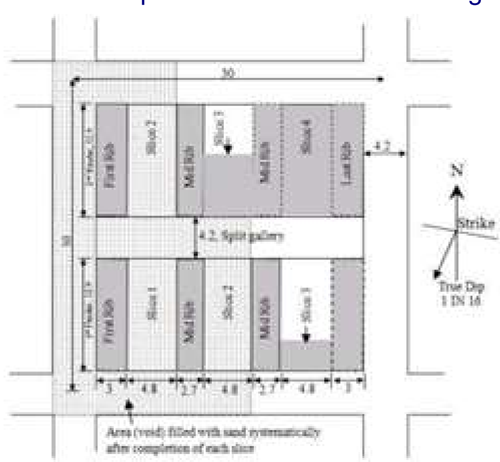

Fig. 3: Proposed manner of pillar (30 m x 30 m, centre to centre) extraction in conjunction with sand stowing.

PROPOSED SUPPORT DESIGN IN DEPILLARING PANELS

In the semi-mechanised depillaring panels, an appropriate design of applied support is required at the vulnerable places in addition to the design adopted during development working. The vulnerable places like split/ slice gallery, junction, goaf edge and other working places from the line of extraction to the extent of active mining zone (AMZ), require suitable support design for efficient pillar extraction. Further, presence of geological discontinuities (slips/weak planes) in immediate roof causes instability, are to be additionally supported with W-straps with roof bolts prior to commencement of pillar extraction.

An attempt is made to design support system using available CIMFR empirical approaches and also studied RLH on numerical models in FLAC3D software package considering the site conditions. RLH and rock load at different vulnerable places are estimated using the empirical approaches developed for depillaring with caving. The required support resistance against the estimated rock load at different working places is calculated using the available bearing/anchorage capacity of the applied supports. The support design is done considering support safety factor up to 1.5. Basically, support design in a depillaring panel depends on various parameters, namely, gallery width, rock mass rating (RMR), unit weight of immediate roof strata, *in-situ* stress and depth of cover.

Support safety factor

To establish the effectiveness of the support system, the support safety factor is calculated using the equation 1.

$$\text{Support safety factor} = \frac{\text{Support Resistance}}{\text{Rock Load}} \quad (1)$$

Support resistance

Mine management provided 10 t as bearing capacity of the quick setting cement grouted roof bolt to be used as applied support to the immediate roof. Bearing capacity of other applied supports (reactive supports) especially steel chock is considered to be 30 t for the design purposes. Support density is estimated using number of support (roof bolt/chock) multiplied by anchorage/bearing capacity of that support. Considering these parameters, support resistance is calculated using equation 2.

$$\text{Support Resistance } (S_r) = \frac{\text{SN} \times \text{C}}{\text{A}} \quad (2)$$

where, SN=Number of support, C=Capacity of each support, A= area supported by one row of supports

Rock load and rock load height estimation

Efficacy of the proposed support system for split/slice galleries, slice junctions and goaf edge during depillaring of Panel BSP-4 is estimated using the empirical relationships (equations 3-8) developed by CSIR-CIMFR (Venkateswarlu et al., 1989, Kushwaha et al. 2010, Ram et al., 2017)In this empirical formulations, RMR, depth of cover, *in-situ* stress, unit weight of immediate roof strata and width of gallery are the influencing parameters for rock load estimation. Recently, CSIR- CIMFR also developed an empirical formulation (equation 8) to design roof bolt- based breaker line support (RBBLS) for mechanised depillaring, which works efficiently under shadow of stable adjacent rib/natural supports. RLH of the considered working places is simply estimated without using unit weight of the immediate roof rock. Rock load and RLH are estimated using the equations 3-8.

(i) For split gallery:

$$SLD_{sp} = \frac{\gamma H^{0.52} K^{0.59} W^{1.12}}{R^{1.02}} \quad (3)$$

(ii) For slice gallery:

$$SLD_{sl} = \frac{\gamma H^{0.67} K^{0.84} W^{1.74}}{R^{1.42}} \quad (4)$$

(iii) For slice/split junction:

$$SLD_{jn} = \frac{\gamma H^{0.50} K^{0.64} W^{1.17}}{R^{0.90}} \quad (5)$$

(iv) For original junction:

$$SLD_{ojn} = 1.5W\gamma(1.7 - 0.037R + 0.0002R^2) \quad (6)$$

(v) For goaf edge (conventional):

$$SLD_{ge} = \frac{\gamma H^{0.54} K^{0.49} W^{0.89}}{R^{0.79}} \quad (7)$$

(vi) For goaf edge (RBBLS):

$$RLH = 11.05W\, H^{0.31} R^{-1.26} \quad (8)$$

where, SLD is required applied support load density (t/m2), is unit weight of the immediate roof rock, H is depth of cover, K is ratio of horizontal to vertical in-situ stresses, horizontal in-situ stress in MPa is equal to 2.4+0.01H and vertical stress in MPa is equal to 0.025H, W is width of gallery, R is adjusted RMR.

Applied support resistance

Roof bolts and chock supports are to be used as an applied support during depillaring operation. Support resistance required at different vulnerable working places are given below:

Split and slice galleries

The split gallery of 4.2 m width is to be supported by full column quick setting cement grouted roof bolt having 10 t bearing capacity. Four number of roof bolts of 1.5 m length are to be installed at 1.2 m x 1.2 m grid pattern. The distance between side bolt and edge the pillar has to be kept 0.3 m. The distance between nearest row of roof bolts and working face should not exceed 0.6 m at any stage of the working. Therefore, support resistance in split gallery (S_r) = (4x10)/(4.2x1.2)= 7.94 t/m^2 (equation 2). This support resistance will provide 2.82 as support safety factor in split gallery. The support plan for such dimension of split galleries is shown in Fig. 4.

The slice gallery of 4.8 m width is to be supported by full column quick setting cement grouted roof bolt having 10 t bearing capacity. Four number of roof bolts of 1.5 m length are to be installed at 1.2 m x 1.2 m grid pattern. The distance between the side bolt and edge of the pillar has to be kept 0.6 m. The distance between nearest row of roof bolts and working face should not exceed 0.6 m at any stage of the working. Therefore, support resistance in split and slice galleries (S_r) = (4x10)/(4.8x1.2)= 6.94 t/m^2 (equation 2). This support resistance will provide 1.82 as support safety factor in slice gallery. The support plan for such dimension of slice galleries is shown in Fig. 4.

Original, split and slice junctions

The original junctions are to be supported by full column quick setting cement grouted roof bolt of 1.5 m in length and two sets of chock/cog supports. Five number of roof bolts are to be installed in a row at 1 m distance and a spacing of 1 m is to be kept between two consecutive rows. Thus, 25 numbers of roof bolts are to be installed at 1 m x 1 m grid pattern as shown in Fig. 4. In addition to these roof bolts, two sets of steel chock should be installed at every junction diagonally as shown in Fig. 4. Therefore, support resistance at junctions (Sr) = {(25x10)+(2x30)}/(4.2x4.2)= 17.57 t/m2 (equation 2). This support resistance will provide 4.55 as support safety factor at the junction.

At split junction (3-way), five number of roof bolts are to be installed in a row at 1 m distance with a spacing of 1.2 m between two consecutive rows. Thus, 20 numbers of roof bolts are to be installed at 1 m x 1.2 m grid pattern (Fig. 4). In addition to these roof bolts, a steel chock should be installed at every 3- way junction. Therefore, support resistance at 3-way junction (S_r)= {(20x10)+ (1x30)}/(4.2x4.2)= 13.04 t/m^2 (equation 2). This support resistance will provide 3.00 as support safety factor at the split junction.

At the slice junction, in addition to existing roof bolts (1.5 m length), one chock support is to be erected at the corner of the junction as shown in Fig. 4. Therefore, support resistance at the slice junction (S_r) = {(16x10)+ (1x30)}/(4.5x4.5)= 9.38 t/m^2 (equation 2). This support resistance will provide 1.99 as support safety factor at the slice junction.

Goaf edge support

In conventional semi-mechanised depillaring panel, generally, skin to skin chock/cog supports are erected at the goaf edge. Sometimes, a row of props is also set closely in addition to the chock supports.In mechanised depillaring (MD) panel using continuous miner and shuttle/ram car, only roof bolt-based breaker line supports (RBBLS) are installed at the goaf edge. Fast coal recovery during MD and most importantly adjacent in-bye rib of slice under extraction remains stable/intact, which enhances the efficacy of the installed bolt. In case of depillaring with caving in semi-mechanised working, it is difficult to use only RBBLS due to loss of intactness in adjacent in-bye rib of the slice under extraction.

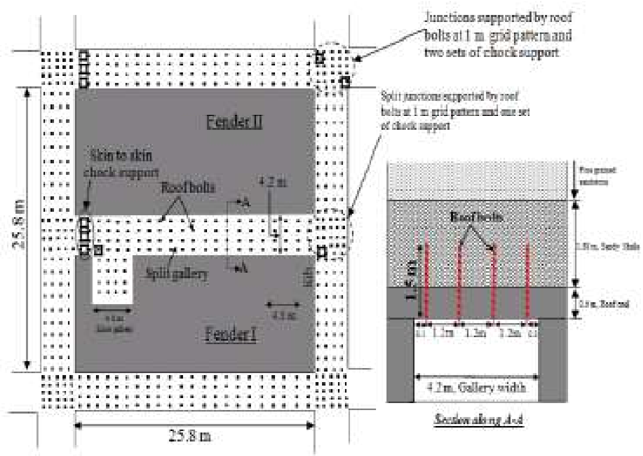

Fig. 4: Support system to be installed during pillar extraction in Panel BSP-4.

However, in case of back-filling (sand stowing) in goaf, less volume (around 1500 m^3) inside the goaf remains void during different stages of the depillaring operation. Further, span between edge of stowed goaf and line of extraction is also less. Thus, ribs remain intact at different stages of the working in depillaring with stowing, which can provide efficacy to the applied RBBLS. Therefore, two options for goaf edge support are given and any one to be applied for the purpose.

Option 1: Conventional Goaf edge support

Steel chock supports are to be installed skin to skin as a conventional goaf edge support as shown in Fig. 4. Three number of steel chock supports (capacity 30 t of each set) proposed to be installed at the goaf edge. Roof bolts installed to support a developed gallery at the goaf edge is also considered for the support resistance calculation. Therefore, support resistance at the goaf edge (S_r) = {(4x10) + (3x30)}/(4.2x2.4)= 12.90 t/m^2 (equation 2). This support resistance will provide 2.15 as support safety factor at the goaf edge.

Option 2: Roof bolt-based breaker line support

It is proposed here to install two rows of roof bolts of 1.5 m length at 0.9 m grid pattern along middle position of the rib as shown in Fig. 5. Five roof bolts are to be installed in a row and one chock support is to be erected in the middle of gallery at the goaf edge. Therefore, support resistance at the goaf edge (Sr) = {(10x10) + (1x30)}/(4.2x1.8)= 17.19 t/m2 (equation 2). This support resistance will provide 4.48 as support safety factor at the goaf edge. It is worth to mention that practice of RBBLS will enhance the efficiency and reduce the cost and cycle time.

Support design on the basis of estimated RLH, rock load, and support resistance for the abovementioned vulnerable places is summarised in Table 2.

Table 2: Support design for Panel BSP-4

Place to be supported	Hs	Vs	K	H	W	R	RLH	Y	Rl	ASr	FOS
Split gallery	4.34	4.85	0.89	194	4.2	52.2	1.28	2.2	2.82	7.94	2.82
Slice gallery	4.34	4.85	0.89	194	4.8	52.2	1.73	2.2	3.81	6.94	1.82
Split junction	4.34	4.85	0.89	194	4.2	52.2	1.98	2.2	4.35	13.04	3.00
Original junction	4.34	4.85	0.89	194	4.2	52.2	1.98	2.2	4.35	17.57	4.54
Slice junction	4.34	4.85	0.89	194	4.5	52.2	2.14	2.2	4.72	9.38	1.99
Goaf edge (Conventional)	4.34	4.85	0.89	194	4.2	52.2	2.73	2.2	6.01	12.9	2.28
Goaf edge (RBBLS)	4.34	4.85	0.89	194	4.2	52.2	1.74	2.2	3.84	17.19	4.80

H_s= Horizontal in-situ stress (2.4+0.01H), V_s= Vertical in-situ stress (0.025H), K = ratio of horizontal to vertical in-situ stresses, H = Depth of cover, W= width of gallery, R= Rock mass rating (adjusted), RLH= Rock load height, = unit weight of the immediate roof rock strata, RL= Rock load, ASR= Applied support resistance.

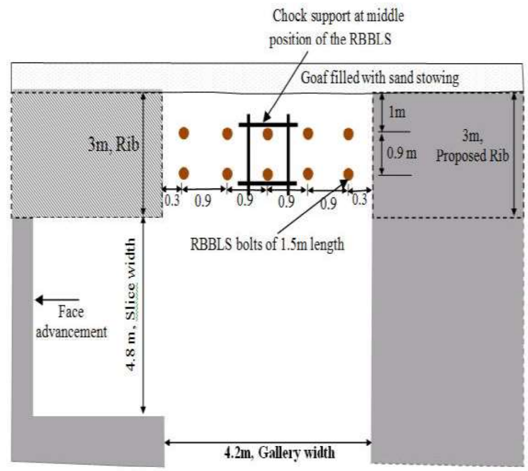

Fig. 5: Position of roof bolt-based breaker line (RBBLS) and chock supports at the goaf edge

NUMERICAL MODELLING

Geo-mining conditions of the proposed panel BSP-4 are considered for numerical modelling study. A model of 306 m length, 244 m width and 63 m height are generated and simulated for existing rock mass properties of the panels. Considering the hardware constraints of a computer like memory, runtime and the effect of overlying strata around 15 times of working height, only 50 m of roof and 10 m of floor have been modelled. Width of the barrier in the panel is kept as 31 m and the size of pillar is kept as 31 m x 31 m (corner to corner). In this model 3 m thick coal seam is considered and working height is kept as 2.5 m leaving 0.5 m coal in roof against sandy shale. Width of original/split and slice galleries are kept as 4 m and 5 m respectively. A truncated load (0.025 x depth of cover, MPa) for the unmodelled portion of the overlying strata is applied over the roof of the model. The sides and bottom boundaries of the model are fixed and the top one is kept free. The properties of rock mass used in the models are determined in laboratory using procured rock samples from the mine, are mentioned in Table 3.

Table 3: Properties of rock mass

Parameter	Rock mass			
	Sandstone	Sandy Shale	Shale	Coal
Bulk modulus (Pa)	4.67e9	3.33e9	2.24e9	1.67e9
Shear modulus (Pa)	2.80e9	2.00e9	1.348e9	1.00e9
Poisson Ratio, υ	0.25	0.25	0.25	0.25
Density (kg/m2)	2500	2250	2100	1440

Initially, *in-situ* model is simulated for the considered depth of cover of the panel BSP-4 followed by development through bord and pillar method. This model is then depillared with sand stowing (Fig. 6). In this model, unstowed void inside the goaf remained around 1500 m^3 at the time of RLH estimation. The model was executed till equilibrium and RLH is measured in the gallery (split/ slice), junction and goaf edge. Here, RLH in the numerical model is defined as the height of the roof strata up to 1.5 safety factor contour in the model which is to be supported. RLH is measured without using applied supports in the model.RLH in the immediate roof of slice/ split and junction is examined for the location (marked in dotted red rectangle) shown in Fig. 6. It is measured during extraction of 7th line of pillar extraction as shown in Fig. 7. RLH in split, original gallery, goaf edge, slice junction and original junction are observed to be 1.12 m, 1.15 m, 1.40 m, 1.70 m and 1.85 m respectively (Fig. 7). After installation of roof bolts (as mentioned in Table 4 and Fig. 4) in the model, significant improvement in safety factor contour in immediate roof is observed at different working places (Fig. 8). The RLH in immediate roof over slice gallery and split junction is observed to be 1.50 m (Fig. 9). After installation of roof bolts, improved safety factor contour in immediate roof over slice gallery and split junction including other working places is shown in Figure 10. This observation revealed that 1.5 m length of roof bolt is found to be sufficient except at junctions and goaf edges. At the original/split/slice junction and goaf edge combined supports of 1.5 m bolt length and chock supports should be installed as mentioned in Table 4. In supported slice, safety factor contour in lower roof horizon (within bolting height) is considerably improved ($\geq$1). In the numerical model, considerable improvement in safety factor contour of immediate roof is observed after installation of roof bolts, reflects the efficacy of proposed applied support. Further, results of the modelling revealed that AMZ is found up to around one pillar ahead (Fig. 6) from the pillar under extraction, needs to be supported well in advance.

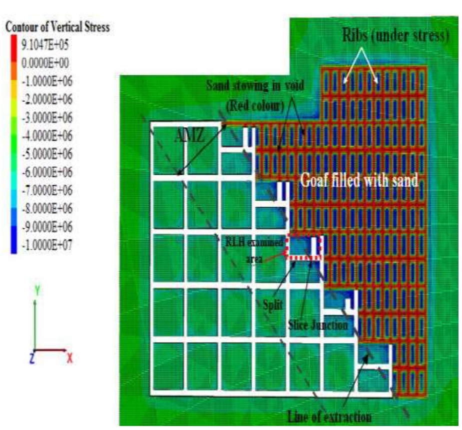

Fig. 6: Pillar extraction with sand stowing and locations marked for measurement of rock load height in the numerical model of panel BSP-4.

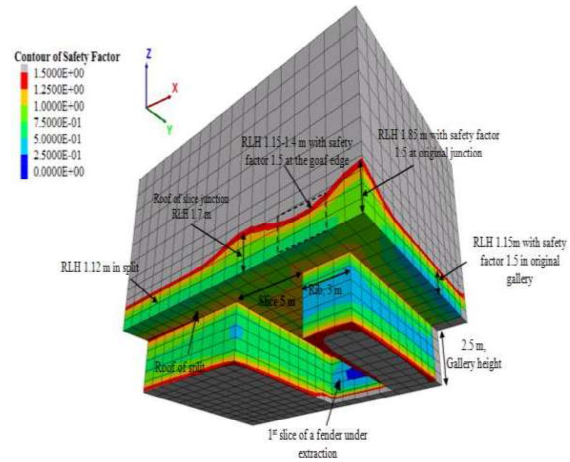

Fig. 7: RLH observed in unsupported immediate roof of original/split galleries, original/slice junctions and at goaf edge in the numerical model.

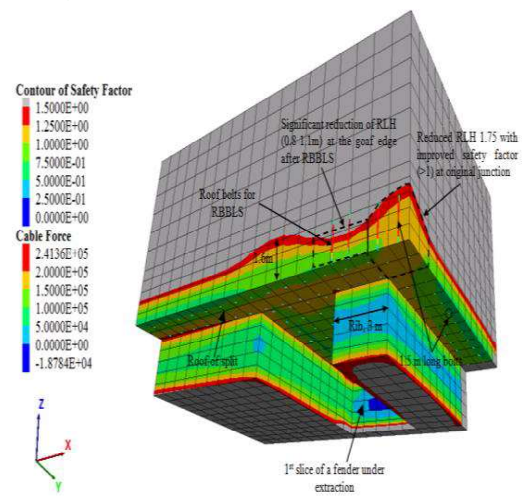

Fig. 8: Improved safety factor contour at original/split galleries, original/slice junctions and goaf edge after application of roof bolts in the numerical model.

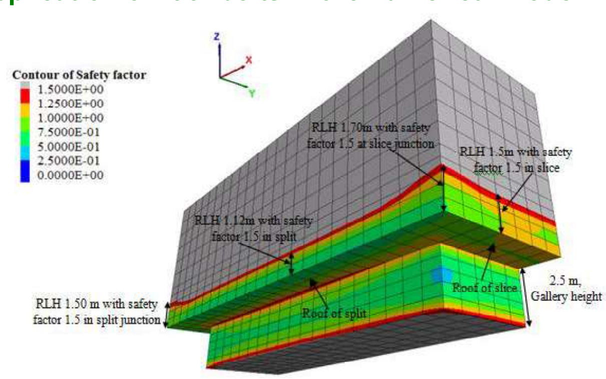

Fig. 9: RLH observed in unsupported immediate roof of slice/split galleries and slice/split junctions in the numerical model.

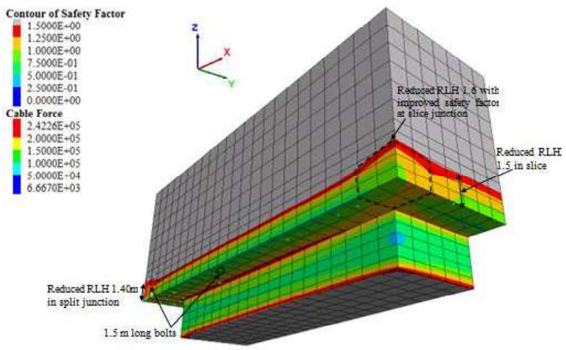

Fig. 10: Improved safety factor contour observed in slice/split galleries and slice/split junctions after application of roof bolts in the numerical model.

Rock load and proposed roof bolt and other supports pattern is given in Table 4. The rock load (RL) is simply estimated by multiplying the measured RLH in numerical models and unit weight of the immediate roof rock mass (2.2 t/m3). A comparison of support safety factor estimated through numerical modelling and empirical approach is shown in Fig. 11. Safety factor obtained from the numerical modelling approach is better than empirical one. It reveals that the proposed support design is efficient to support different working places during depillaring of panel BSP-4.

Table 4: Support design for Panel BSP-4 on the basis of numerical modelling study

Place to be supported	RLH (m)	RL (t/m^2)	ASR (t/m^2)	Safety Factor	Required length of roof bolt (m) and other supports pattern
Split gallery	1.12	2.46	7.93	3.22	1.2 m x 1.2 m grid pattern
Slice gallery	1.50	3.30	6.94	2.10	1.2 m x 1.2 m grid pattern
Split junction	1.50	3.30	13.04	3.95	1.2 m x 1.0 m grid pattern
Original junction	1.85	4.07	17.57	4.32	1.0 m x 1.0 m grid pattern and two sets of chock supports
Slice junction	1.70	3.74	9.38	2.51	1.2 m x 1.2 m grid pattern and one set of chock support
Goaf edge (conventional)	1.40	3.08	12.9	4.19	One row of 1.5 m bolt length at 1.2 m spacing and three sets of chock supports to be erected skin to skin
Goaf edge (RBBLS)	1.40	3.08	17.19	5.58	1.5 m bolt length at 0.9 m x 0.9 m grid pattern one set of chock support at middle of the gallery

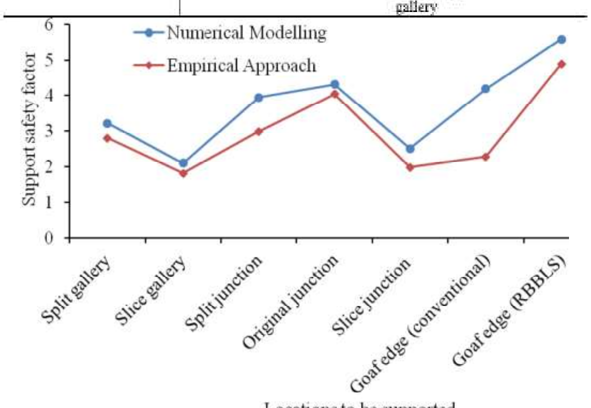

Fig. 11: Comparison of support safety factor estimated through numerical modelling and empirical approach

INSTRUMENTATION AND MONITORING

Behaviour of the strata during depillaring of panels BSP-4 is to be studied through field instrumentation and monitoring (Fig.12). The purpose of the field instrumentation is to enhance safety of the working to protect men and machinery from any risk and also to increase production and productivity of the mine. It may provide some quantitative values of strata movement in and around the workings for better understanding of the rock mass behaviour and further, in optimization of the design.

It is always good to opt for a simple approach of instrumentation and monitoring. However, attention is to be paid for selection of durable and robust instruments, suitable for the underground coal mining environment. An attempt is to be made to monitor the readings of the instrument installed in the AMZ frequently.Details of the instruments proposed to be used for strata controls monitoring in these panels are given below.

Rotary tell tales

Rotary Tell Tales (RTTs) are to be installed at every split junction except the pillar adjacent to surrounding barrier of the panel (Fig. 12). Anchor of the RTTs is to be fixed at a horizon of 8 m from the ceiling of the roof. This instrument magnifies the roof displacement through circular observation scale and remains very close to the working face. Thus, roof displacement observation by this instrument becomes vital for safe working. It should be kept under continuous visual observation by the person associated with strata control study and supervisor during extraction of nearby slices.

Single height tell tales

This instrument provides visual indication of roof strata movement in the opening of a coal seam. The cut-off values are also designated on the instruments, which gives warning of possible roof failure. Remedial actions may be taken if the observed value exceeds the cut off value. It is proposed that these instruments should be installed at every original junction except junctions under barrier pillar in the proposed panel (Fig. 12) to confirm the stability of the roof strata. It is suggested that the anchor of single height telltale is to be fixed at 5 m.

CONCLUSIONS

Panel BSP-4 of Bogra seam at JK Nagar (R) Colliery is proposed to be depillared by conventional splitting and slicing method with sand stowing under the existing geo-mining conditions. As per available empirical approaches and results obtained from the numerical modeling of the site conditions, manner of pillar extraction (Fig. 3) and a suitable support design is suggested. Safety factor-based support design at different working places is considered using proposed applied support resistance and rock load. Rock load estimated using the empirical approaches and numerical modeling.

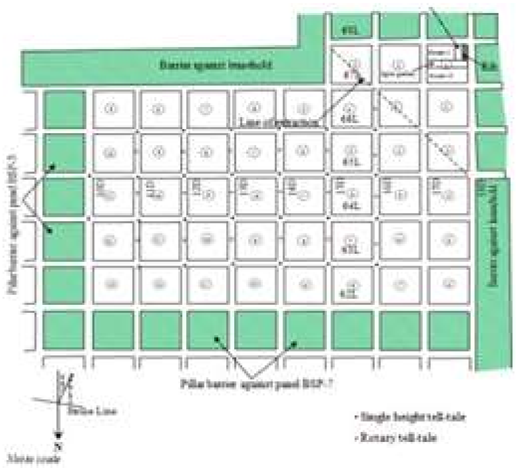

Fig. 12: Plan showing the locations of proposed geo-technical instruments for strata movement study in panel BSP-4.

For the condition of the panel, the range of rock load estimated through the empirical approaches, which were developed on the basis of caving system of goaf management, at different locations varied from 2.82t/m2 to 6.01t/m2. However, the numerical modeling of the considered depillaring panel is carried out with filling of goaf/void area by low properties material and rock load evaluated through themodelsat different locations found to be 2.46 t/m2 to 4.07 t/m2. It may be due to regular and systematic sand filling inside the goaf in order to maintain the permissible void not exceeding 1500 m3 at different stages of the working in the model.

For support design in the panel, roof bolt of 1.5 m length is proposed to install at 1.2 m x 1.2 m grid pattern in immediate roof of original/split. At 4-way junction, combined support comprises roof bolt of 1.5 m length at 1 m grid pattern and two sets of chock support is proposed. At split junction, roof bolt of 1.5 m length is proposed to install at 1 m x 1.2 m grid pattern and one set of chock support is proposed. At slice junction, it is proposed to install roof bolt of 1.5 m length in 1.2 m grid pattern with one additional chock support. Two options of goaf edge support are suggested i.e. conventional and combined (RBBLS and a set of chocks). The colliery management can adopt any one option as per their convenience as the safety factor in both the conditions are above 2.RLH obtained from the numerical modeling is found to be 1.12 m to 1.85 m height in immediate roof horizon considering 1.5 safety factor contours. It is found that 1.5 m bolt length is sufficient to support at the slice/split gallery, goaf edge and junctions. However, original junction and slice junctions are to be additionally supported by two sets of chock supports.

The proposed support system is found to be suitable as per the given site conditions during depillaring of panel BSP-4 for efficient coal recovery with safety. Geo-technical instruments should be installed at proposed locations in the panel for safe working at different stages of depillaring operation. The total design is based on the point information (borehole data) and therefore, the design should iteratively be optimized as per the actual behavior of the rock mass at different stages of the working.

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Mechanical Impact of a Roadheader

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ABSTRACT

Roadheader is a hybrid machine that has excellent maneuverability in comparison with tunnel boring machine, can cut in harder and more abrasive rocks, in different tunnel profiles and adapts easily to changing operational conditions. Roadheader is widely used in underground tunneling and underground mining and can avoid shock loads to the surrounding rock mass and minimize the over break due to its operational ease. The pick spacing on the cutterhead is an important design variable of a roadheader. Lot of literature is available that propose the method of defining the pick spacing and their influence on cutting rate. This paper reviews significant literature from the published domain in order to discern the most important parameters defining the pick spacing and their influence on the cutting rate. The paper briefly touches the procedures and the approaches used for cutterhead design while reviewing cutting actions of the cutters.

Keywords— roadheader, pick spacing, advance rate, literature review

INTRODUCTION

Roadheader (RH) is a hybrid mechanical machine used as a rock excavator in underground workings consisting of a boom- mounted cutting head with loading part usually involving a conveyor and a crawler travelling track to move the entire machine forward into the rock face. RH is the most widely used partial face mechanical excavator for soft to medium strength rocks. It is mainly used for development and production particularly in coal, industrial minerals, and evaporitic rocks to form main haulage drifts, roadways, cross- cuts etc, in both civil and mining industries since. In civil construction, RH is extensive used for excavation of railway, roadway, sewer, diversion tunnels, etc, as well as for enlargement and rehabilitation of various underground structures. The high advance rates, safety measures, mobility, reliability, minimum strata disturbance and less labour deployment are some of the advantages of RH.

The versatility of roadheader is governed by the higher cutting power density by having smaller cutting drum diameter. In spite of the above advantages, RH has some disadvantages such as high pick consumption in abrasive rocks. Under such conditions the RH excavation usually becomes uneconomical due to frequent bit changes with increased machine vibrations and maintenance costs (Couper et al., 1998). The cost of the machine is also quite high. Thus, if RH is not selected as per the geological conditions, the consequences can be economically detrimental to a project. The machine selection is dependent on the tunnel dimensions and its ground conditions that in turn determine the production rate and pick (bit) consumption. Hence, the machine performance must be predicted before starting a tunnel project to move towards more profitable, productive and competitive arena. Roadheaders were first developed for mechanical excavation of coal in the early 50s. Today's advancements make roadheaders more efficient in the mechanical application of hard rock tunnelling also. Mechanical improvisations in roadheader such as increase in machine weight, size, automation and remote-control features, cutterhead, improved design and boom, advances in a hydraulic and electrical system, muck pick and loading system, efficient cutter head design and metallurgical cutterhead design are recent advancement in the RH (Couper et al., 2016).

According to Tucker (1985) roadheaders are classified by their weight and compressive strength of the rock to be cut. For example, weight up to 30 tonnes and the cutting capability up to 70 MPa fall under light duty RH category. Similarly, weight up to 34 to 45 and over 45 tonnes with compressive strength 100 MPa and up to 150 MPa can be classified under the medium duty to heavy duty RH, respectively.

Atlas Copco-Eickhoff established the classification according to weight (Schneider, 1988) and their respective classes such as 20 tonnes comes under 0 class, 20-30 tonnes are in I class. Classification of boom type roadheaders such as small size up to 30 tonnes, mid-size between 30 to 70 tonnes and large size between 70 to 120 tonnes, respectively is provided by Neil et al. (1994) with cutting action of longitudinal type- milling/borer/axial

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head and transverse type- ripping one. The classification of two main types of cutting heads is mentioned in Table 1.

Table 1: Comparison between longitudinal and transverse cutting head

Longitudinal				Transverse			
Cutting mode	Boom forces	Angles recommended	Machine and tunnel parameter	Cutting mode	Boom forces	Angles recommended	Machinea nd tunnel parameter
Sumping, overcutting, lifting, undercutting, lowering	Horizont al, axial, vertical	Max boom position angle	Moment of turning around the vertical axis and stability	Sumping, arcing, lowering, lifting	Horizontal, axial, vertical	Max Boom position angle	Moment of turning to the side direction and stability

Between the two types of roadheaders, the ripping (transverse) and the milling (inline or axial), the ripping type is more suitable for hard rock cutting. This is due to more efficient cutting during sumping as the resultant force act along the boom. This results in optimal use of machine mass and more efficient cleaning of the face. The design parameters include the spacing between bits, location of bits, practical usage and shapes of cutting heads and are further classified as spherical, conical, cylindrical or as a combination of these. Cutting head geometry is determined by the tilt angles of the picks on the head. Tilt angle is the angle between pick and the assumed plane that is perpendicular to rotation axis of the head. Accordingly, the literature available on the roadheader can be classified into different fields and sub-classes based on machine design, drum or pick design and its applicability (Table 2).

The classification of roadheader based on the review of published literature shows the inventive possibilities and future possibilities of improvisation in tune with its respective classes and subfields. Also, improved prediction of cutting performance and bit consumption is essential as the rock conditions might result in risk.

Table 2: Classification of literature available on roadheader

Sl. No.	Principal Class	Sub-field	Sub-classes
1.	Machine Design	1. Model 2. Stability of roadheader/development 3. Excavation 4. Performance/prediction	1. Cutting force measurement 2. counter cutting force 3. Excavation rate
2.	Drum/Pick design	1. Cutter type/comparison/developm ent 2. Cutting efficiency/cutter performance	1. Drum specification 2. Circumferential pick spacing 3. Selection of economic cutting speed
3.	Productivity/application	1. Production 2. Comparative analysis 3. Operational stability	1. Field application 2. Relation to other excavators

PERFORMANCE PREDICTION METHODS FOR ROADHEADERS

The predicted cutting performance of a mechanical excavator for any type of excavating conditions or rock formation in which the machine is deployed is one of the main factors determining the economics of a mechanized mining operation. Table 3 logs important findings of the respective authors.

Table 3: Roadheader performance as defined by various authors

Sl. No.	Author/year	Important Findings
1	Matti, 1999	Suggested that RH is more preferable for low to medium hardness and not suitable for hard cutting condition
2	Schnider, 1998	Compressive strength if the rock intended to be cut by a mechanical excavator may be predicted by directly relating rock cuttability to rock properties from massive rock formations
3	Frenyo and Lange, 1994	As the weight of Roadheader increases RH be used in higher strength rocks, since the weight of the machine has a direct relation to cutting head power and boom forces
4	McDermott, 1988	If weight is less than the machine stability is affected negatively and instability issues may occur
5	Aisawa *et al.* 1987	Suggested that roadheader is a dimensional constraint machine and it is more favourable for low to medium hard rocks
6	Gibson *et al.* 1985	To increase the machine stability, side and rear stabilizer pistons are generally used
7	Kogelman, 1982	Suggested that machine weight is increased for more powerful machines due to increased boom reaction force
8	Kogelman, 1982	Increase of weight causes a rise in initial cost of the machine and also problems of sinking of the machine in wet ground

Table 3 gives an insight into the performance of RH in various formation types and the economics associated with these. It can also be observed from the table that operating range referring to geology has a direct bearing on excavation costs. Accordingly, costs associated with machine are related to weight of the machine.

PHYSICO-MECHANICAL TEST FOR THE PERFORMANCE PREDICTION OF ROADHEADER

In order to assess the performance of the machine in field several tests are to be performed. This ensures the prediction of production in the field and allows to pre-empt the machine and cutter requirements. Nowadays researchers focus mainly on the performance prediction of Roadheader. The performance prediction has mostly been done with the help of physico-mechanical tests such as UCS, BTS, Point load, Schmidt hammer, ultrasonic, density, porosity, small/ full scale linear cutting and Cerchar abrasivity tests.

Comakli et al. (2014) suggested that on the specified core sample five iteration of UCS test should be done and the average vale must be recorded as a result. Tiryaki (2008) used UCS as an individual predictor in non-linear regression analysis. Kahraman et al. (2003) suggested the stress rate application limit of 0.5-1.0 MPa/s should be used during testing as also concluded by Bilgin et al. (2004). Similarly, in case of BTS Comakli et al. (2014) suggested testing six samples of the same lithology and the average of the iterations needs to be quoted as result. Balci et al. (2004) showed that abrasivity of soft to hard and non- abrasive to abrasive rock range from 0.2 to 11.6. Tiryaki (2008) used BTS as an individual predictor in non-linear regression analysis. In case of point load test, Comakli et al. (2014) recommends repetition of tests upto 7 times with average value recorded as a result.

Kahraman (2003) predicted the RH performance by taking core dia of 33 mm and 66mm in length. Similarly, for Schmidt hammer test Balci et al. (2004) shows the correlation coefficient for the Schmidt hammer rebound value for 5 mm and 9mm depth of cut values the correlations are 0.62 and 0.79, respectively. Kahraman et al. (2003) conducted N-type Schmidt hammer tests for prediction of RH performancewhile iterating 3 times for a rock specimen. Balci et al. (2004) shows the summarization of properties in the range varies from soft to hard and non-abrasive to abrasive rock from 1.49 to 4.13 gm/cm3. Comakli et al. (2014) used dynamometer and specific energy by dividing mean cutting force to the yield force. Cooper (2003) test was carried out with a conical cutter had a gage of 80mm, flange dia 64mm, shank dia 35mm, tip dia 22mm and primary tip angle of 80R".

PERFORMANCE PREDICTION OF ROADHEADER

The performance prediction of roadheaders is the main area to focus for determining the economics of underground excavation projects. The researchers have focused on developing the performance prediction models ascertain the cutting rate of Roadheader. Many parameters have been found to be essential to simulate the cutting results that include instantaneous cutting rate, UCS, RQD, specific energy etc. Table 4 summarizes the contributions of various authors in performance prediction of roadheader.

Table 4 identifies compressive strength and specific energy as the main parameters for performance prediction of a roadheader. However, specific energy (SE) determination from the small-scale or full-scale cutting tests is very difficult and expensive. This is the reason that some researchers investigated the relation between SE and some rock properties and suggested empirical equation for the estimation of SE. It was determined that

Table 4: Performance Prediction of Roadheaders as per various authors

Sr No.	Author/Year	Formula	Comments	Machine/ cutterhead type used
1.	Balci *et al.* (2004)	$k = 0.8$	Concluded that k is the energy transfer ratio usually assumed as 0.8 for roadheaders	Cutting head not defined
2.	Thuro and Plinninger, 1999	$ICR = 75.7 - 14.3 \ln \sigma c$	derived a prediction model based on the performance of a 132kW	Transverse type roadheader
3.	Rostami *et al.* 1994	$ICR = k\frac{P}{SE}$	Determine the specific energy (SE) method is a simple procedure for the quick performance prediction of roadheaders	Cutting head not defined
4.	Ozdemir and Rostami, 1994	$ICR = K(\frac{P}{SE_{opt}})$	concluded that cutting rate of any excavation machine to use cutting power, optimum specific energy and energy transfer ratio is expressed	Universal
5.	Bilgin *et al.* 1990	$ICR = 0.28P\,(0.974)^{RMCI}$ $RMCI = \sigma_c(\frac{RQD}{100})^{2/3}$	based on the in-situ observation of many tunnelling and mining projects, suggested a performance prediction model	Axial type roadheader
6.	Gehring, 1989	$ICR = \frac{719}{\sigma_c^{0.78}}$	presented a performance prediction model based on the performance of a roadheader with a 250 kW	Transverse type cutterhead
7.	Gehring, 1989	$ICR = \frac{1739}{\sigma_c^{1.13}}$	also presented a performance prediction model based on the performance of a roadheader with a 230 kW	Axial type cutterhead

Where,

k - is energy transfer ratio, p - cutting rate, SE_{opt} – optimum specific energy in kWh/M³, ICR = Instantaneous cutting rate in m³/h, P = Installed cutterhead power inKw or HP, RMCI = Rock mass cuttability index, σ_c = Uniaxial compressive strength in MPa, RQD = Rock quality designation in %.

the relation between SE and the product of UCS and BTS has a better correlation than that of the relation between SE and both UCS and BTS when treated separately. Balci et al. (2004) showed that abrasivity of soft to hard and non-abrasive to abrasive rock range from 0.2 to 11.6. Tiryaki (2008) used BTS as an individual predictor in non-linear regression analysis. In case of point load test, Comakli et al. (2014) recommends repetition of tests upto 7 times with average value recorded as a result.

CUTTER HEAD AND DESIGN

Table 5 shows the comparison of transverse and longitudinal cutter heads which includes cutting power of the cutting heads and their stability, sensitiveness of the stability of the cutting head and weight of the RH. This gives an idea to choose the type of roadheader for deployment in a particular geological condition.

STATE OF THE ART

As mentioned above, determination of stability of the roadheader is an important parameter for the efficiency of excavation as maximum stability maximizes the production. The most important parameter for a RH is

Table 5: Transverse and longitudinal cutting head

Sl. No.	Transverse Cutting head	Findings	Longitudinal cutting head	Findings
1.	Gehring, 1989; Kleinert, 1982; Kogelman, 1982; Menzel and Frenyo, 1981	Under similar cutting power of this type roadheader can cut higher strength rock than the longitudinal type RH considering the stability	Frenyo and Lange, 1994; McDermott, 1988	Since this type of roadheader can cut in vertically direction also, the vertical stability should also be considered while evaluating the stability of RH
2.	Kogelman, 1982; McDermott, 1988; Menzel and Frenyo, 1981	More sensitive to stability in vertical direction	Gehring, 1989; Kleinert, 1982; Menzel and Frenyo, 1981	More sensitive to stability in the horizontal direction than transverse type
3.	Gehring, 1989; Kleinert, 1982; Menzel and Frenyo, 1981	More stable in vertical direction than longitudinal head roadheaders	Gehring, 1989; Kleinert, 1982; Kogelman, 1982; Menzel and Frenyo, 1981	Suggested to utilize the full weight of the machine and hence require 20–25% more weight than the other type

weight which is used to determine the type of RH to be deployed. Other parameters such as machine width, the width of track, cutterhead power, size and allowance of bit on the cutter and tracking speed sumping/lifting etc. are empanelled under engineering parameters. Different modes of cutting such as such as lifting and lowering give an idea of the slewing reaction force (SR) and arcing reaction force (AR).

Table 7 presents the state of art in application of RH and its predictive regime as employed by different authors. The works of different authors have been classified in pick configuration and engineering parameters and the number of such variables used have been tabulated thereof. This gives a broad idea of the use of variables for developing a predictive regime and to explore further inventive possibilities. Some works have also been summarized further.

Table 6: Mechanical parameters for pick spacing

Sl. No.	Authors	Experimental details	Pick spacing 1	2	3	4	5	6	7	8	9	10	T	Engg. parameters 1	2	3	4	5	T
1	Evans, 1972	Experimental analysis/ ND					√	√					2	√	√				2
2	Roxborough, 1973	Boom type roadheaders					√	√					2	√	√				2
3	Hurt, 1981	Boom tunnelling machine	√				√	√					3	√	√				2
4	McAndrew, 1981	Boom tunnelling machine	√				√	√					3	√	√				2
5	Hurt, 1982	Boom tunnelling machine	√		√	√	√	√					5	√	√				2
6	McAndrew, 1982	Boom tunnelling machine	√		√	√							3						0
7	Hekimoglu, 1984	Boom tunnelling machine cutting heads								√		√	2				√	√	2
8	Morris, 1985	ND								√			1						0
9	Hurt, 1985	Boom tunnelling machine cutting heads	√	√			√	√		√	√		6	√	√				2
10	McAndrew, 1985	Boom tunnelling machine cutting heads	√	√			√	√			√		5	√	√				2
11	Hekimoglu, 1991	Boom tunnelling machine cutting heads			√	√			√			√	4			√	√	√	3
12	Fowell, 1991	Boom tunnelling machine cutting heads			√				√				2			√			1
13	Eyyuboglu, 2000	Boom type roadheaders having cylindrical cutting heads											0			√			1
Total cites			6	2	4	3	7	7	2	3	2	2	40	7	7	3	2	2	21

Pick configuration: 1. pick spacing, 2. lacing type - graded lacing, 3. circumferential pick spacing, 4. equal spacing, 5. line spacing, 6. pick imposition, 7. lacing 8. pick grouping, 9. pick force, 10. boom force

Engg parameters: *1. area, 2. sweep volume, 3. angle of wrap, 4. torque, 5. specific energy*

It can be observed from Table 6 that line spacing, pick imposition and pick spacing assume greater importance in studies among the pick configuration properties. Among the engineering properties the area and sweep volume have been used by authors the maximum times.

Hekimoglu and Fowell (1991) suggested a lacing pattern based on long-term lab investigations and considering the size of the available tool-holders up to 26 or 27 heavy-duty picks. The picks may be laced at equal circumferential spacing on medium-duty roadheader cutting heads to give improved production rate. They also pointed to the fact that tramming of the tool holders was introduced to obtain equal circumferential pick spacing for 360° and over and concluded that there will be a tool holder overlap at nose section due to insufficient space on cylindrical cutting head.

Li et al. (2016) introduced spiral line for pick arrangement on RH by taking mean values of traverse force, vertical force and resultant force on the cutting edge with unequal pitch angle increase by 12.83%, 5.52% and 7.60%, respectively, in comparison to equal circumferential pick spacing. Eyyuboglu (2000) suggested the most reliable angles of wrap were 360° and 776°.

Hurt and MacAndrew (1985) concluded that force variation on cutting head are maintained at minimum level by providing pick spacing around the entire cutting head periphery. They suggested that in order to balance the pick force, application of graded lacing at the corner cutting must be evaluated. Hurt and Morris (1985) suggested that cutters are arranged as a group in the cutting head which affects the performance of RH. Hekimoglu (1984) showed that if the cutting head geometry changes, the torque, boom forces and specific energy of machine also change. Hurt et al. (1982) after analysing force balance suggested that the circumferential pick spacing must be kept equal around the cutting head periphery. Evans (1972) suggested that by imposing force on picks, line spacing between the picks controls cross-section area swept by picks.

FACTORS AFFECTING THE CUTTING PERFORMANCE

Machine excavation operations and skill of machine operator affect the machine utilization time that plays an important role in determining the advance rate and economics of the project as well. Machine utilization time for roadheaders increases to 60 % with a well- organized machining site and an experienced operator.

The mechanism of the cutting is further explained with the help of Fig 1 that also shows the drum design in cutting condition where the drum rolls down towards the rockmass.

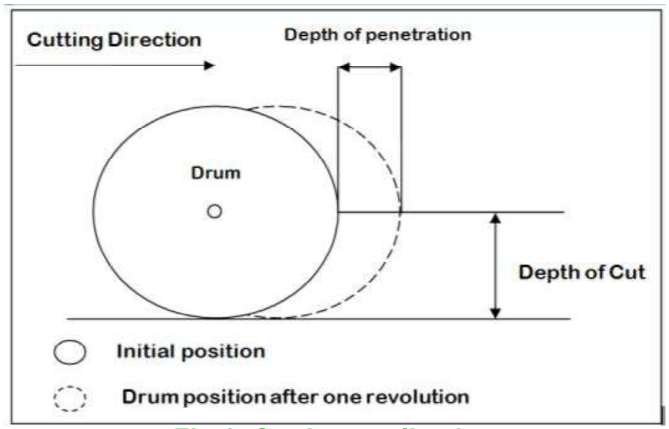
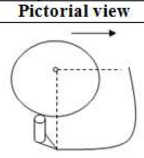
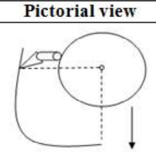
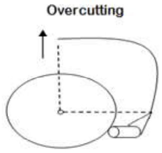
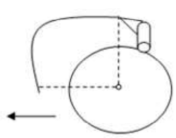


Fig 1: Cutting application

When drum rotates or takes one complete revolution, it penetrates the rock mass to some depth. The initial position of the drum with the final position describes the depth of penetration also called as depth of cut. To achieve better cutting efficiency and to prevent from under size and over size chipping in the process of cutting, sharp tool should be used with suitable spacing. Worn out or damage tools cause higher cutting forces leading to rapid damage to tool holder drum and gearbox, consequently reducing haulage speed apart from producing more dust and fines. Conical picks are mainly used to design drum because of the drum design criteria emphasis on the chip size.

Table 7 shows the pictorial view of cutting conditions with their respective direction such as overcutting, lifting, lowering and undercutting.

Table 7: Cutting conditions of a Roadheader

Sl. No.	Method	Pictorial view	Sl. No.	Method	Pictorial view
1.	Overcutting		3.	Lowering	
2.	Lifting		4.	Undercutting	

ICR defines net cutting rate obtained when the machine is in cutting mode concluded by (Bilgin et al., 1983) which shows cutting conditions of RH and the boom reaction force exerted to the rock mass showing different cutting actions.

PRODUCTION RATE

Prediction of instantaneous cutting rate and machine utilization time, determining daily advance rates plays an important role in the time scheduling of the tunneling projects. Table 8 has been compiled based on works of different authors that can be helpful in determining the economy of tunnel excavation, conditions with action of cutting.

Table 8: Roadheader production rate

Sl. No.	Author/Year	Important findings for advance rate/performance
1.	Sandback, 1985 and Douglas, 1985	Use of rock classification system to predict the changing advance rate in an inclined drift
2.	Bilgin, 1983	Developed a model based on specific energy obtained from drilling rate of a percussive drill
3.	Schnider, 1988	RH cutting rate decreases while increase in the rock compressive strength.
4.	Thuro and Plinninger, 1998 and 1999	For given cutting power of RH there is a decrease in cutting rate with increasing UCS
5.	Coupur *et al.* 1997 and 1998	The power and the weight of the roadheader were considered together in addition to rock compressive strengththat in turn give the net cutting rate of RH
6.	Farmer and Garrity, 1987 and Poole, 1987	For a given power of roadheader, excavation rate in solid bank m³/cutting hour might be predicted using specific energy values

Contribution of the different authors as cited above for the production rate shows that variety of criteria has used such as specific energy, cutting power and cutting rate of the RH, RH power.

CONCLUSION

Roadheaders are mainly deployed in the partial face excavation form soft to medium hard rock conditions. Significant references are addressed in this paper to understand the issues arising regarding RH. Specific parameters like weight of the machine in relation to hardness of the rock, drum design and pick spacing and inclination have been identified as major factors controlling production in varied rock conditions as propounded by different authors. The detail analysis of pick spacing to entire cutting head periphery is reviewed with the parameters used by different authors discussed to show the essentiality and applicability of the cutting actions during excavation. The paper is intended to provide a basis for future developments and innovations in deployment of roadheader in a particular geological condition.

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Utilization of Coal Mine Produced Water Using Different Water Treatment Process: An Approach

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ABSTRACT

India is the third largest coal producing country in the world. Jharkhand with coal production of 113.04 million tones retained as second position in coal producing state in India. Bokaro, Rajmahal, Karanpura, Jharia and Ramgarh coalfields are the main coal producing areas in the Jharkhand. During the coal mining process, a massive amount of mine water drained out, which has caused not only the wastage of ground water assets, but also creates several distress on water pollution. Coal mining produced water discharges into rivers, streams and nallahas, which contain mainly several soluble minerals present in coal or associated rocks. The coal mine water needs to be necessarily treatment and purification before used for drinking, domestic and agriculture purposes.

In this paper different methods and technological process like coagulation, sedimentation, filtration and disinfection etc. are discussed for coal mine produced water treatment and purification. By use of these different mechanical and chemical processes of water treatment, the TDS of treated water reduce below 500 ppm for use of drinking purpose and less than 1000 ppm is use for agriculture purpose. Which may overcome the water scarcity in coal mine areas as well as reduce the impact of mine produced water on environment.

Keywords— coal mine produced water, water treatment, tds, hardness, turbidity

INTRODUCTION

Nature has provided a wonderful gift in the form of coal- a plentiful resource to meet energy demand of the country. Coal is a fossil fuel and has been formed as a result of alteration of vegetation under action of combined effects of pressure and heat over millions of years to form coal seams. It's chiefly composition of C, H, N, S and O, beside other non-combustible inorganic matter.

India is the third largest coal producing country in the world. Coal is vital for sustainable development of our country. Coal production influences economic, industrial as well as social development of our country. The GDP contribution of the mining industry varies from 2.2% to 2.5% only, but going by the GDP of the total industrial sector it contributes around 10% to 11% (Forest Research Institute, 2015). It is the most widely used energy source for electricity generation and an essential input for steel production. However, due to the mining activity balance of nature has been disturbed severely. There are several direct or indirect distress imposed on ecology, agriculture, forest lands, ground water and hence ultimately to the human being.

Coal mining in the country is carried out by both opencast and underground methods. Opencast mining contributed about 93% of the total production. In case of open cast mining, land surface is completely disturbed, whereas underground mining have limited losses on surface except ground subsidence. However, both the mining method affects the quality and quantity of surface and ground water resources in coal mining region of India [VD Choubey (1991), DC Gupta(1999), R Khan et. al.(2005), G Singh(1994), GSingh(1998), AKSingh et. al.(2007), AKSingh et. al.(2010)]. By and large, groundwater gets contaminated mainly due to leaching and percolation [RKTiwary (2001)]. Mining's impacts on the natural water ecosystem may be observed throughout the life cycle of a mine and even after long time of mine closure [PKSrivastava et. al. (2002)]. The large volumes of water can be released from aquifers during opencast and underground coal mining operation. Water pollution in mining areas is mainly due to overburden (OB) dumps, surface impoundments, mine water, industrial effluents acid mine drainage, and tailing ponds (Singh et al, 2007). With increase in human population there is tremendous pressure of good quality water for drinking, domestic and agriculture uses. So it is very important to treat waste mine water for its optimal use and to save our mother nature.

PHYSICAL AND CHEMICAL CHARACTERISTICS OF WASTE MINE WATER

The quality of good water plays an important role in

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agricultural production and on human health. Quality of water is depends on the use of water, hence different uses requires different criteria of water quality. The quality of water is known by some features like total dissolved solid (TDS), total hardness (TH) due to presence of cations (Ca²z , Mg²z) and anions (HCOƒ { , F{ , Cl{ , NOƒ { , SO²{). The chemistry of anions and cations indicates that ionic presence as: HCO3- > SO4->Cl-> NO3->F- and Mg2+>Ca2+> Na+> K+ in the coal mine water. The coal mine water also contains suspended solid & high level of TDS ranging from 430 to 1090 mg per litre. The coal mine water needs to be necessarily treated and purified before used. Since more than 70% of pollutants from the mining industry are discharge into water, the removal of these contaminants prior to discharge is very necessary. It is critical to avoid a discharge of toxic components into the environment and subsequently back to the food-chain.

For treatment of waste mine water in effective manner to utilise it for domestic and agriculture purpose, it should be imperative that to know the physical and chemical properties of water. It is equally important that to assess the impurities mainly present in the mine waste water.

The quality of water contains mainly following characteristics for its utilisation of domestic as well as agricultural purposes:

a) Acidic and basic strength of water, which is measure on pH scale.
b) Colour of water, which represents the different dissolved matter present in water.
c) Total dissolve organic and inorganic solid i.e. Total Dissolved Solid (TDS) present in water.
d) Total hardness (TH) of the water due to presence of cations (Ca^{2+}, Mg^{2+}) and anions ($HCO2^-$, F^-, $C1^-$, $NO3^-$, SO^{2-}).
e) The bio-degradable amount of organic content of water is measure by BOD (biological oxygen in demand).

f) Total organic and inorganic pollutant of waste water, which can oxidized and measure by COD (chemical oxygen in demand).

METHODS USED FOR TREATMENT OF MINE WASTE WATER

Before planning for waste water treatment, it is significant to identify the contaminants, which generally are present in the mine waste water. The most probable pollutants needs to remove from waste coal mine water are as follows

a) Physical impurities, like Suspended solid, colloidal particals.
b) Organic impurities, like Coal, Oil, grease and phenolic compounds etc.
c) Inorganic impurities like Heavy metals (Cr, Hg, Cu, Cd, Pb, Zn, Ni etc.)
d) Dissolved salt.
e) Biological impurities like Bacteria, Viruses, Protozoa, Algae, etc.

For removal of above impurities from waste mine water different stages of treatment should be performed.

1. Preliminary Treatment: In the first step of water treatment, the removal of physical and organic impurities has been carried out from waste mine water.

(a) Screening-The mine waste water is passed through screens, having large no of tiny size holes, then floating matters are retained by them.

(b) Skimming- Impurities which are lighter than water like oil, grease are removed by mechanical skimming. For this waste mine water is passed through skimming tank, which is so design to remove oil and grease from waste water flow.

2. Primary Treatment: In the primary treatment of waste mine water, the following are main water process has been used.

(a) Sedimentation-This is a process of allowing water to stand for 2-6 hours undisturbed in big tank about 5m deep, when most of the suspended particle settle down at the bottom due to the force of gravity. The clear supernatant water is then drawn from tank with the help of pumps. Sedimentation is generally carried out in continuous flow type tank.

(b) Flocculation-The mine waste water is passed through a tank were it retain for 30 min.
This tank is fitted with paddles rotating at a speed of 0.5m/s due to this steering the finely divided suspended solids comes together and form large particles which settle down with this process.

(c) Coagulation- This is the process of removing fine suspended and colloidal impurities by the addition of requisite amount of chemical (coagulants).Coagulants when added to water form an insoluble gelatinous, flocculants precipitate. These precipitate descents through water absorbed and entangle very fine suspended impurities forming bigger flocs, which settle down easily. Coagulants mostly used for waste mine water treatment are Al2(SO4) (Alum), Ferrous sulphate, Sodium Aluminates. Reaction of these coagulants in water is given as

$Al_2(SO_4) + Ca(HCO_3) \rightarrow$
Alum
$2Al(OH)_3 + 3CaSO_4 + 6CO_2$
Flocculants ppt.

$NaAl_2O + 2H_2O \rightarrow$
Sodium Aluminate
$Al(OH)_3 + NaOH$
Gelatinous Floc

Copper or ferrous sulphate ($FeSO_4 . 7H_2O$) act when pH < 8.5
$FeSO_4 + Mg(HCO)_3 \rightarrow$
$Fe(OH)_3 + MgCO_3 + CO_2 + H_2O$

In water

(d) Neutralization- Acidic or alkaline mine waste water is neutralize before its discharge. Acidic water is neutralizing by adding lime stone or caustic soda and alkaline water by adding sulphuric acid.

3. Secondary Treatment: In this method, the process includes for waste mine water treatment are:

(a) Oxidation Pond- Oxidation pond is also called lagoons or stabilisation pond. Ponds are generally constructed of brickwork with relatively small depth .Waste water treatment occurs naturally by interaction of sunlight, bacteria and algae .It efficiently remove bacteria, biodegradable organic and inorganic impurities from waste water.98-99% of BOD reduction is often possible.

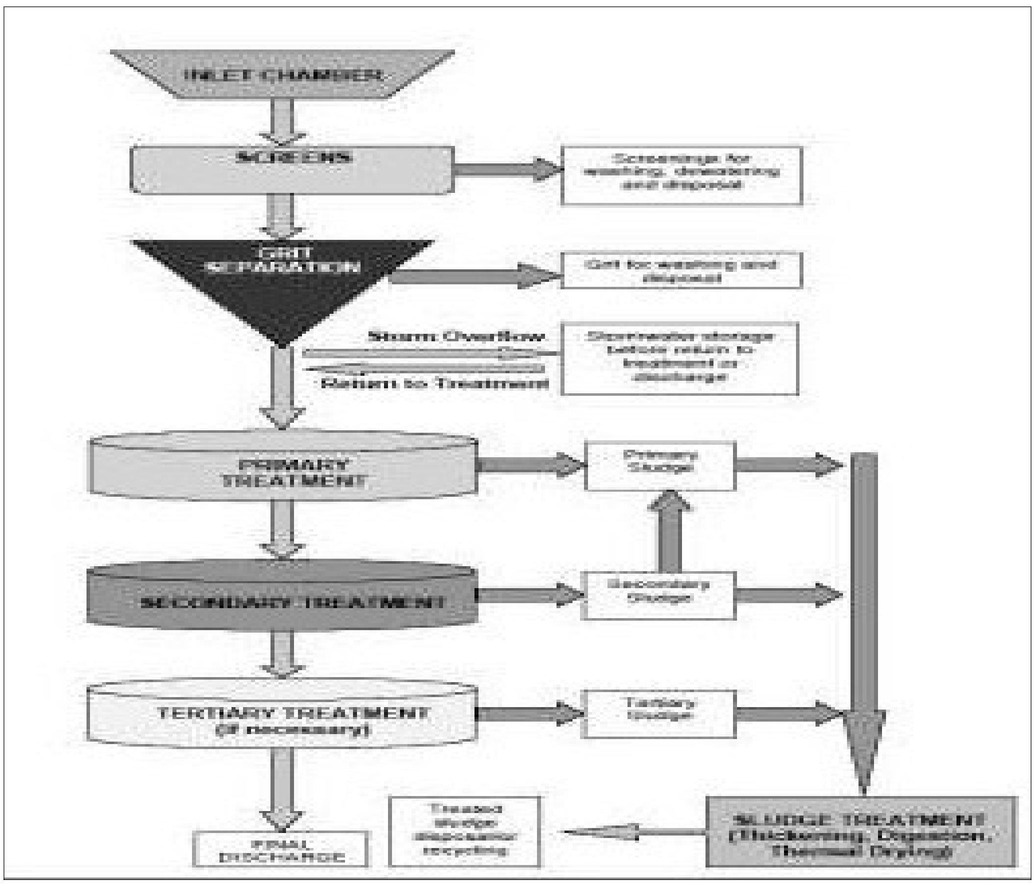

Fig.1: Flow chart of coal mine water treatment methodology

(b) Trickling Filter-It is also known as sprinkling filter. It has a well like structure made up of brick of a circular rectangular bed, 3-5cm at the top and 10-15cm at the base with a depth of 1-3m. This aerobic process is used to remove organic matter from waste mine water by use of microorganism attached to surface of well.

4. Tertiary Treatment: This method of waste mine water treatment includes

(a) Chlorination- Chlorine (as gas or concentrated solution), bleaching powder or chloramine is added to the waste mine water. These chemical produce hypochlorous acid, which is germicide and bactericidal.

$Cl_2 + H_2O \rightarrow HOCl + HCl$
$HOCl + \text{Bacteria} \rightarrow$ Bacteria are killed (b)Coagulation Sedimentation-After
primary and secondary treatment of water, waste mine water is again coagulated using chemicals, so that remaining colloidal particle settle down by means of gravity.
(c) Adsorption-This is a physical process used to remove low concentration of impurities from water that is difficult to remove by other means. In this process porous activated carbon is used, which have very large surface area available for adsorption of impurities (7.3 million sq.ft/lb).
(d) Reverse Osmosis-This is a water purification method, using semi permeable membrane to remove ion, unwanted particles and molecule against pressure gradient. In this process impure water is placed across semi permeable membrane and pressure is increases by use of high pressure pump on impure side. The pure water flows through the semi permeable membrane and salt molecules in water is retained by the membrane.
(e) Electro Dialysis- In this process electro dialysis cell is used to purify water containing concentration of ionic impurities. When current is passed cations get deposited in cathode & anions on anode and central compartment carries water which is free from ionic impurities.

RESULT AND DISCUSSION

An enormous amount of mine water drained out from the coal mining, which has the wastage of ground water assets. Coal mining produced water discharges into rivers, streams and ponds creates several adverse impacts on surface water pollution because it contains several soluble minerals present in coal or associated rocks with other associated impurities. To utilize the coal mine water for drinking, domestic and agriculture purposes, it necessarily needs treatment and purification.

In primary treatment water, oils, greases, colloidal particles, suspended particles are removed from mine waste water. The pH level of water is balanced by use of acid and base. The water quality after primary treatment has TDS approximately1000ppm, which can be used for agriculture purpose. Whereas, after secondary treatment this treated water, it became almost free from biological and biodegradable organic and inorganic chemicals. This water can be used for domestic purpose. Moreover, the tertiary treatment removes all inorganic impurities at molecular level from impure water. Hence, the TDS level of water has become less than 500ppm, which can be used for drinking purpose.

CONCLUSION

The crisis of drinking water is the facing by all most all part of the world, due to increase in population. India also has tremendous pressure of good quality water for drinking, domestic and agriculture purposes. Henceforth, each and every drop of water is precious. An enormous amount of mine water drained out from the coal mining, which has the wastage of ground water as sets. So it is very important to treat waste mine water for its optimal use and to save our mother nature. These waste mine produce water can be fulfill our daily water requirements for vary purposes using above water treatment methodologies. The treatment of coal mine produce waste water will resolve the water crisis problem in respective coal mine areas of Jharkhand and balance our ecosystem as well.

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Assessment of Status of Unapproachable Underground Old Mine Workings Using Electrical Resistivity Tomography: A Case Study

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ABSTRACT

An assessment of sub-surface ground condition is a serious problem in Indian coalfields due to old underground mine workings. Unfortunately, most of these are uncharted as no proper mine plan is available. The location of hidden openings like galleries, goaves, shafts inclines etc. may pose great threat at surface and in turn affect the local environment. In the present study, 2D electrical resistivity tomography (ERT) technique has been used to detect underground mine workings to identify the location of the underground workings as well as status in terms of water fill conditions. ERT survey has been conducted at Jamadoba 6 & 7 Pit coal mine of Jharia coalfield, India, using Wenner-Schlumberger array. This technique helped in detecting the location of water logged areas in underground working, isolating by contrast resistivity.

Keywords—: electrical resistivity tomography, cavity, wenner-schlumberger array, mining

INTRODUCTION

ERT or Electrical Resistivity Imaging (ERI) method has wide application in the field of near surface and engineering geophysics. This imaging method is capable of detecting subsurface cavity, geological features such as faults, formation boundaries and depth of bed rock. Subsurface cavity, detection is very important for coal mining especially for shallow depth old mine workings safety prospective. Last date of full paper submission is Day-Month-Year. The author should submit the paper in full by no later than the mentioned date, identifying it with last name of the author. Paper can be only be accepted on the understanding that at least one of the authors shall register for the national seminar and will present the paper. ERT is a geophysical technique that helps to get subsurface distribution of electrical resistivity. Subsurface cavities may be air filled or water filled that could easily be detected due to high resistivity contrast with surroundings (Zhou et.al 2002; Van Schoor 2002). Also, this technique is most popular due to low survey cost and provides effective result with less time. ERT is very efficient to image subsurface complex geology compared to conventional geoelectrical approach (Griffith and Barker 1993; Zhou et al. 2000). ERT is also used to detect subsurface sinkhole and cavities (Kruse et al. 2006; Bharti et al 2016a, 2016b). Maillol et al. (1999) used ERT for uncharted mine galleries detection in West Bengal, India.

STUDY AREA

The study was carried out over a part of Jamadoba 6 & 7 Pit coal mine located in Jharia coalfield in Dhanbad district as shown in Figure.1.

The rock formations of Jharia coalfield unconformably overlying the Archean basement, mainly belong to the Lower Gondwana group rocks of Permian age comprising Talchir, Barakar, Barren measures and Raniganj formations, from bottom to top.

The Barakar Formation is main coal bearing formation. The rocks of Barakar Formation are mainly sandstone of variable grain size, argillaceous sandstone, intercalation of sandstone and shale, carbonaceous shales, jhama, mica-peridotite and coal seams Affiliations.

METHODOLOGY

A. Layout

2D ERT data area have been collected along 480m long profile AA' using Syscal Pro Standard & Switch (24-48-72-96-120) version (IRIS Instrument) system multi electrode resistivity meter over the possible old mine working in a part of Jamadoba 6&7 Pit colliery as shown in Figure 2. Wenner-Schlumberger array, with 5m electrode spacing have been considered for prospecting purpose. Total length of electrode profile length is 480m.

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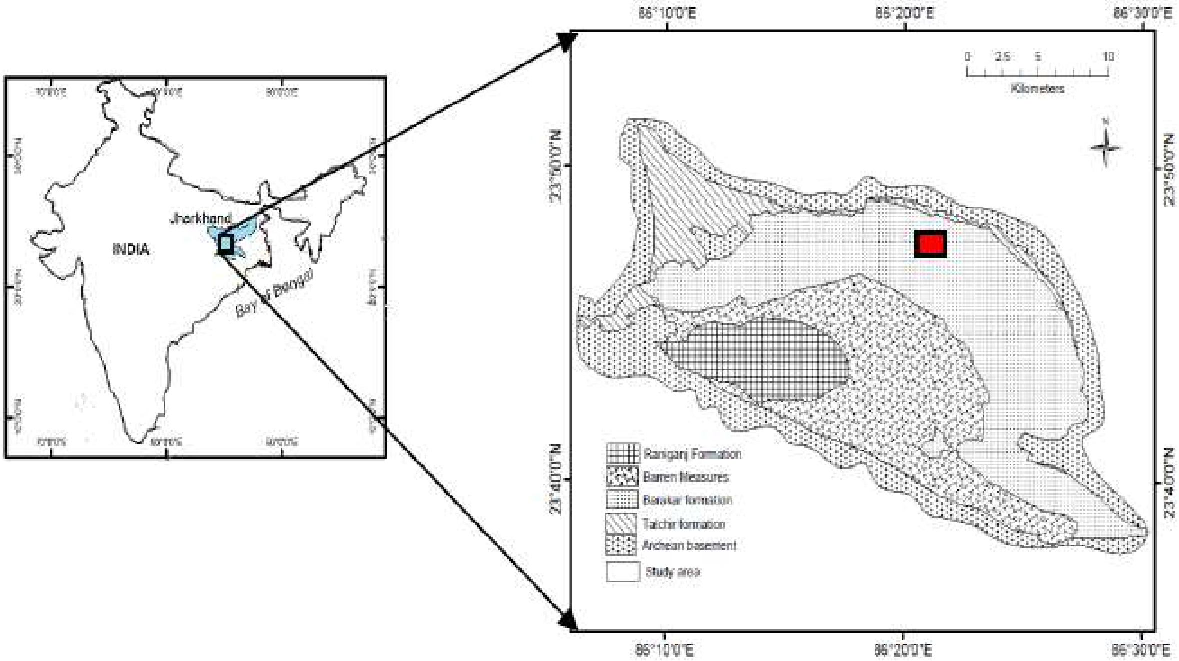

Fig. 1: **Location map of study area along generalized geological map of Jharia coalfield**

Fig. 2: Layout of the instrument arrangement along with the surface location plan

B. Concept of Wenner-Schlumberger Array

The Wenner-Schlumberger array is hybrid between Wenner and Schlumberger arrays (Pazdirek and Blaha 1996). This array is moderately sensitive to both horizontal and vertical structures. In areas where both types of geological structures are expected, this array might be a good compromise between the Wenner and the dipole-dipole array. The median depth of investigation for this array is about 10% larger than that for the Wenner array for the same distance between the outer electrodes. It has a slightly better horizontal coverage compared with the Wenner array. The signal strength for this array is smaller than that for the Wenner array, but it is higher than the dipole-dipole array.

FIELD INVESTIGATION

The acquired ERT data has been procced. The spicks and noise have been eliminated from apparent resistivity data set. Then data sets have been inverted based on regularized least square optimization technique using RES2DINV (Loke and Barker, 1996) software.The inverted geoelectrical sections have been interpreted based on the variation of anomalous high and low resistivity values. The inverted geoelectrical sections along AA/ computed using Wenner-Schlumberger array is shown in Figure 3.

The geoelectrical section of this array indicated soil/ alluvium/ weather layer up to ~10m from the surface. Moderately low resistivity of about ~70 Ωm was observed up to 160 m from the starting point indicating no underground working which validates with the surface plan as shown in Figure 3. Expected waterlogged caved zone was identified at reduced distance (RD) of about 160m to 230m with relatively low resistivity of about 05 Ωm to 24 Ωm near the depth of about 40m. The underground working is at an average depth of 40 m in this area as per the plan. The developed underground working was observed at reduced distance 230m to 480m with relatively high resistivity of about 186 &!m indicating non-inundated area.

In addition, an abandoned incline was also observed at reduced distance (RD) of about 320m to 425m with relatively high resistivity. It showed the air-filled cavity near the depth of about 18m. Though there is no indication of incline at the surface but validates with the plan shown. Hence, this zone is unsafe from stability point of view as well as is a possible source for inclusion surface water to the underground workings.

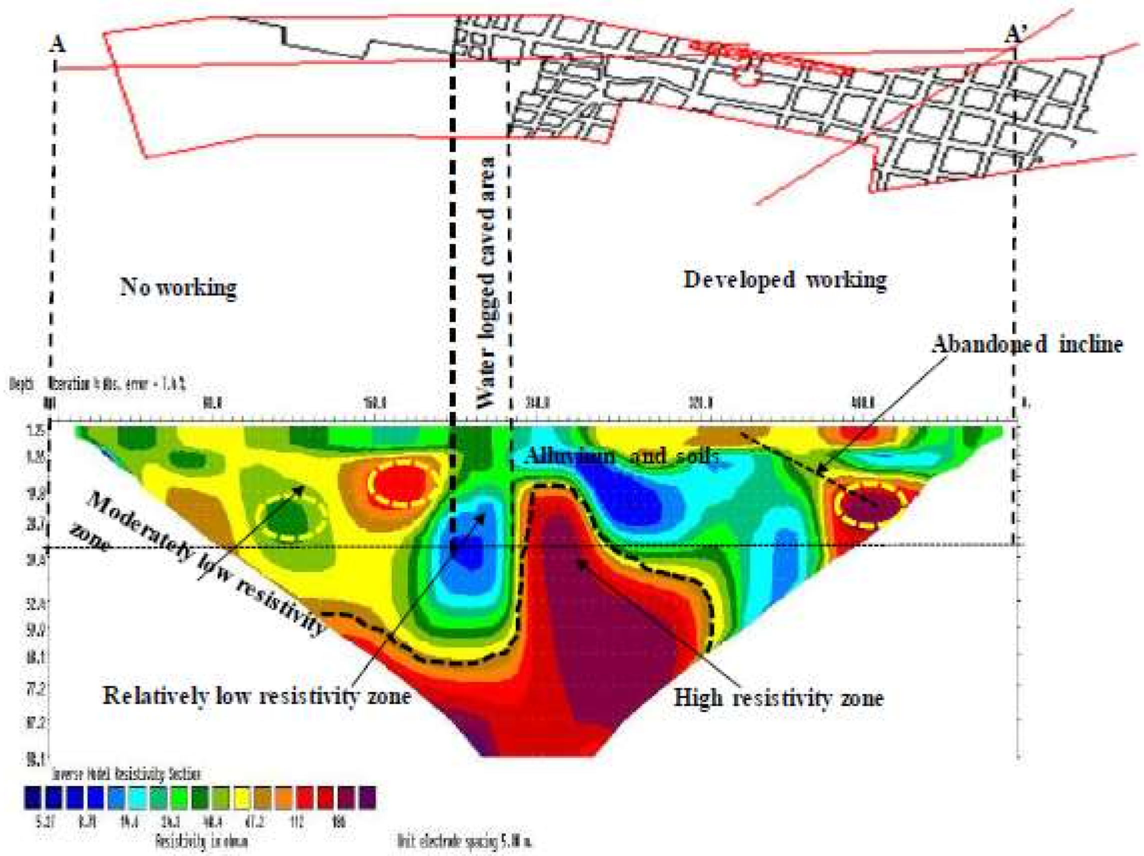

Fig. 3: 2D Electrical Resistivity Profile showing the general conditions of subsurface encountered around a section of 6 & 7 pit mine

CONCLUSIONS

2D ERT data have been acquired along a profile over the possible old mine workings in a part of Jamadoba colliery using Wenner- Schlumberger array. The geoelectrical models reveals that though high resistivity value has been considered of air-filled cavities, but it indicates to be relatively conducting as old mine workings are filled with moist debris and air.

Most of the cavities are might be fully or partially water filled as it shows low resistivity value. At last, it may be concluded that the proficiency of the 2D resistivity imaging survey is a useful technique and more effective for determining and mapping subsurface cavities associated with the old mine workings using Wenner–Schlumberger array.

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Coal Mining for Sustainable and Techno-Economic Mining Solution in India

M. P. Dikshit*

ABSTRACT

In India, commercial Coal Mining dates back to 1774 and first started in Raniganj Coalfields (W.B). It advanced with many changes in technology adoption depending on need and geo- mining conditions at the locale. Coal mining subsequently spread to all most all the coal bearing states in India. Technology was Bord and Pillar (B&P) in Underground (u/g) and small Open Cast (O/C) mines by overburden removal manually earlier and now mechanized in both U/G and o/c coal mining. Technology has to be safe, productive, Techno-economic and Eco-friendly to the best. Time has come for continuous coal wining method avoiding cyclic u/g operations. In o/c coal mining, high capacity HEMM (240Te dumper, 42Cu.m shovel, 400m Dia Drill) for o/b removal where adoptable and surface miner for continuous coal wining avoiding explosives are the need of the hour. In India coal inventory is around 321Bte. Our u/g mining need immediate revisit for its economic operation by suitable technology adoption. Where mine geometry permits Surface Miner is the best option for bulk output economically in o/c mines. Mine safety status in India can be improved by technology up-gradation in both u/g and o/c coal mining. The future coal mining solution has to be an appropriate Technology adoption in Indian coal Industry.

Keywords— techno-economic mining, bord and pillar, coal mining, coal production.

INTRODUCTION

In India, commercial Coal Mining dates back to 1774 when John Sumner and Suetonious Grant Heatly first started coal mining in Raniganj Coalfields. It advanced with many changes in technology depending on geo-mining conditions prevailing at a locale. Mining thereafter spread to all the coal bearing states in the country. Coal India Ltd (CIL) formed in 1975 with HQ at Kolkata after nationalization of coking coal mines (Oct'71) and non-coking coal mines (Jan'73). In India now CIL and SCCL (The Singareni Collieries Co. Ltd) are the two major public coal miners and CIL contributes around 84% in India's coal production. SCCL operates with the coal deposits in southern states to cater thermal coal there.

Coal Mining Technology adopted earlier were Bord & Pillar in Underground (u/g) mining and small Open Cast (o/c) mines with usual Overburden (O/b) removal manually. Subsequently there was mechanization in u/g mining with bucket loading of coal by SDL/LHD to conveyors and use of Low Capacity Shovels in o/c mining for o/b and coal removal. Later the u/g mine mechanization advanced towards continuous coal wining by continuous miner and use of High Capacity HEMM (Heavy Earth Moving Machinery) to formulate high capacity o/c mines in India.

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Bulk output and high capacity u/g mine is not possible by cyclic mining (drilling/blasting) in coal wining with bucket loading at face. Continuous coal wining equipment avoiding cycle mining practices are coming up for relatively safe & economic u/g mining and mega o/c mines using high capacity HEMM as available in the globe.

India needs appropriate technology adoption for mine operation economically with safety in coal mining. Technology moves with time and time has come to operate mines safely with techno-economically and ecofriendly to the best.

COAL MINING IN INDIA

Majority of the u/g coal mining operation continues with bucket loading by SDL/LHD. There is attempt for continuous coal wining avoiding cycling mining practices. Even Powered Support Longwall equipment 1st tried in 1978 at Monidih mine and subsequently at Dhemomain, Seetalpur, Jhanjra, Churcha, Kottadih, Rajendra, New Kumda, Balarampur in CIL and GDK, VK, PVK at Singareni and lately at Adriyala. Results now are highly encouraging. Continuous miners operated in u/g mining at Chirimiri, Kurja, Haldibari, Vijay West, Sarpi, Jhanjra, Churcha, Raniatari mines successfully.

In o/c mines, high capacity shovels, drills, dumpers, surface miner (in coal) have shown rapid growth in India. Now 3.4 Mte. u/g mine with PSLW has become first time a reality in India with techno-economic operation. Even o/c with 42 Mte. capacity mega mine is operated using 42 Cu.m shovels, 240Te. dumpers & 400mm drills economically.

There are advances in coal evacuation from u/g till wagon loading. The use of high capacity steel cord belt, u/g strata bunker, SILO, mega CHP, Merry-go-round for continuous and faster wagon loading with eco- friendly approach is a solution in mega coal mines today. Eco-friendly option with bulk output economically for both u/g and o/c mining has to be the solution for its sustainability in the country.

TECHNOLOGY IN COAL MINING

Coal Mining Technology-wise are broadly
1) Underground 2) Open Cast Mining. In u/g mining the following coal wining practices are adopted in our coal Industry.

a) Mining with Blasting at coal faces in Bord & Pillar.

b) Blasting free Mining – non-cyclic mining.

- Longwall Mining
- Short wall Mining
- Continuous Miner operation in B&P Layout.
- High wall Mining.

In India the major share of u/g coal production around 90% is from conventional B&P Mining with drilling and blasting. The share of continuous coal wining by blasting free mining is in increasing trend and likely to come up for large scale u/g mining in future for its numerous advantages. In continuous miner, there may be "Standard Height Miner" or "Low Height /Low Capacity" serving the same purpose in coal wining but of lesser capacity.

In O/C, the following options are adoptable for O/C coal mining in India.

i. Both O/b and coal wining by drilling and blasting.
ii. O/b with drilling & blasting, coal wining by Surface Miner.
iii. O/b removal with suitable equipment and coal by surface Miner. Such combination is just tried at some locales and likely to get wider application.

Table 1: National coal production and CIL's coal production/productivity (last 10 yrs.)

Year	National coal Production (Mte)			CIL's Coal Production & Productivity					
	U/G	O/C	Total	Production (Mte)			Productivity		
				U/G	O/C	Total	U/G	O/C	Overall
2009-10	58.52 (10.99%)	473.522 (89.01%)	532.042	43.25 (10.02%)	388.01 (89.98%)	431.26	0.78	9.51	4.47
2010-11	54.86 (10.29%)	477.832 (89.71%)	532.692	40.02 (09.27%)	391.3 (90.73%)	431.32	0.77	10.06	4.73
2011-12	51.96 (09.62%)	487.99 (90.38%)	539.95	38.39 (08.80%)	397.45 (91.20%)	435.84	0.75	10.40	4.89
2012-13	52.206 (09.36%)	505.501 (90.64%)	557.707	37.78 (08.35%)	414.41 (91.65%)	452.19	0.77	11.48	5.32
2013-14	49.65 (08.77%)	516.12 (91.23%)	565.77	36.11 (08.46%)	426.3 (91.54%)	426.41	0.76	12.18	5.62
2014-15	50.20 (08.20%)	516.8 (91.80%)	612.00	35.04 (07.08%)	459.19 (92.92%)	494.23	0.79	13.13	6.20
2015-16	48.5 (07.59%)	583.68 (92.41%)	638.18	33.786 (06.27%)	504.754 (93.73%)	538.75	0.79	14.83	6.88
2016-17	44.35 (6.69%)	618.44 (93.31%)	662.79	31.47 (05.67%)	522.659 (94.33%)	554.135	0.84	14.58	7.65
2017-18	43.30 (6.25%)	63.16 (93.75%)	680.46	30.54 (5.38%)	536.82 (94.62%)	567.36	0.86	14.66	7.84
*2018-19	42.06 (6.00%)	667.40 (94%Approx)	710	30.48 (5.0%)	576.40 (95%)	606.88	.94	14.50	8.22

Open Cast coal wining by Surface Miner is increasing fast and shares major coal production from O/C now. In some of the major coal Producing Co's like MCL, the share of Surface Miner is over 90% in the last fiscal. The option is best adopted in other subsidiaries of CIL also.

PRODUCTION AND PRODUCTIVITY

In India, with coal production of 30 Mte. in 1947, reached to 100 Mte. in 1977-78 and crossed 700 Mte. in last fiscal (2018-19). CIL being the largest coal miner in the globe exceeded 606 Mte. sharing around 84% of national production. U/g coal production gradually reduced from 94% at time of independence to about 6% in the last fiscal (2018-19) in the country.

TECHNOLOGY SOLUTION FOR FUTURE COAL MINING IN INDIA

The intermediate technology in u/g mining with SDL/LHD in B&P layout though economic earlier not sustainable now for increase in production cost.In O/C also the Low capacity excavators and tippers are now avoided where high capacity HEMM are introduced on economic grounds. U/g coal mining now needs adoption of suitable technology for economic operation.

The future solution for technology for economic, sustainable and safe u/g mining in India include the following –

- Powered Support Long Wall (PSLW) Mining
- Short Wall Mining
- High Wall Mining
- Mining with Continuous Miner(C/M) and Shuttle Car's Combination
- Low Capacity/Low Cost Continuous Miner (LCCM/ LHCM)

A. Powered Support Long Wall (PSLW)

PSLW commissioned in 1978 but first light of its provenness was while USSR's L/W complex at Jhanjra Mine (ECL) produced 90,000Te in a March'92 and thereafter at SCCL and also in SECL's – New Kumda, Balrampur and Rajendra Mine. Now the Longwall operating at Jhanjra mine (Fig.1) is able to extract 5.5m in single lift which achieves average 6000TPD economically. The technology is able to produce over 1 Mte./Yr.

from single set of equipment for working thickness of 3.5m coal seam.

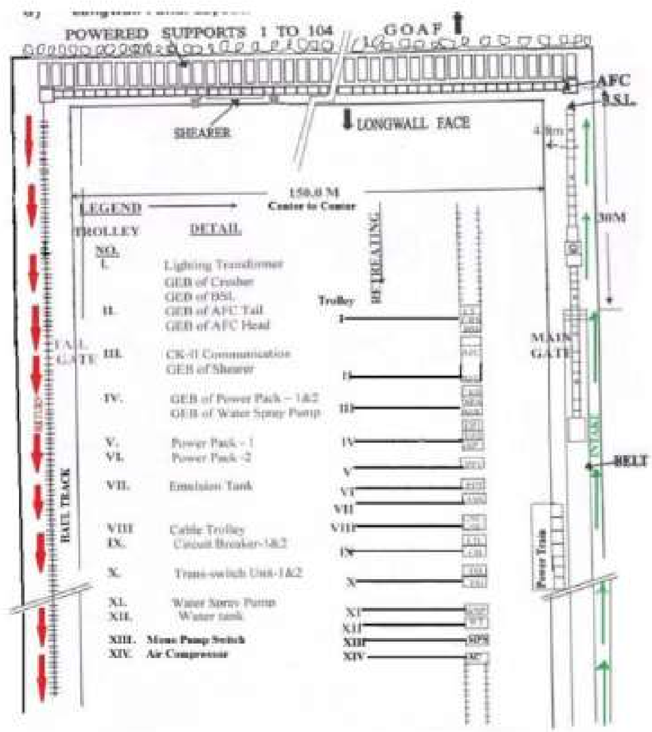

Fig. 1: Layout of a PSLW Panel

The major equipment are Hydraulic Powered Support, shearer, Armoured Face Conveyor (AFC), stage loader, Power Pack, Lump breaker (Crusher), Lighting and signaling system, TSU's – 2 Nos. (Usually 2 Nos. of IMVA each for coal face equipment), Power/Energy train for electricals and hydraulics, HP equipment, Booster Pump, gate belt conveyor and other associated equipment used for monitoring of environment & operations etc. Gate roads within the abutment zone upto 30m from face are usually supported with 40Te open Circuit hyd. props. The production and productivity increases with the increase of extraction thickness and face length. With a face length of 150m, the average annual production of 1 Mte. is possible in a seam (3-3.5m), where the M/power may be around 350 with an overall mine productivity (O.M.S) of around 8.5 This may rise to 5500- 6000TPD i.e. 2 Mte. annually in 4.5 to 5m coal seam with productivity (O.M.S) of 18-22. The operation stands economic, safe and sustainable with proper selection of equipment.

B. Short Wall (S/W) Mining

It is practically a combination of Bord &Pillar and PSLW mining in our country. The developed B&P workings is extracted using PSLW equipment and successfully practiced at Balrampur Mine, SECL, where number of panels were extracted economically and the mine turned to green from red. Manpower requirement will be around

230 heads, achieving annual output of 0.60 to 0.70 Mte. where Production comes to around 1800 to 1850 TPD with Productivity (O.M.S) around 8 for a face length of 100m. The production and productivity may increase to 2100-2200 TPD and 15 respectively by increase in face length upto 120 to 150m. The coal face is operated with an orientation of 10 to 20p from usual gallery orientation as shown in fig.2 and gate roads are supported with open circuit 40Te hyd. props upto 30m & 45m respectively for out bye and in bye gates. Developed galleries are supported in advance by rope stitching or cable bolting in the panel.

a. Thickness of seam = 1.2m
b. Web Pillar width between drivages = 1.5m
c. Extraction = 70%
d. Average Production per day = 2700 Te
e. Manpower = 4 Persons / Shift, others 15
f. Productivity = 84

In India 4 such mines were developed and the first operation started at Sharda Mine, SECL in 2011 and more mines are to come up in the country.

Fig. 2: A Typical Short-Wall Panel Layout

C. High Wall Mining

This is a best option for thin seams where neither o/c nor u/g is economically viable & when the o/c mining reaches to its ultimate pit position and no more economically viable and where coal is below built up area, sterilized in mine boundary. The coal properties in hilly terrain can also be extracted by High Wall Mining as done large scale in USA's West Virginia state.

No person works in u/g workings where the coal comes from u/g by remotely operated continuous miner and driving long headings leaving suitable web pillars between the drivages. Coal is evacuated though conveyors to surface (Fig.3).

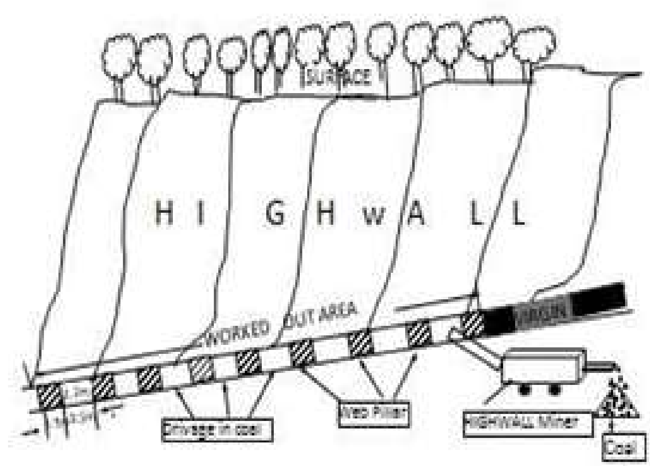

Fig. 3: High Wall Mining Layout in Extraction of Thin Seam

High Wall Mining system to work thin (1.2m) coal seam

a. Drivage Dimension in coal = 3.5m×1.2m

D. Mining by Continuous Miner and Shuttle Cars Combination

Such equipment combination in B&P layout for faster development followed by subsequent depillaring of pillars was first successfully operated in 2002 at Anjan hill

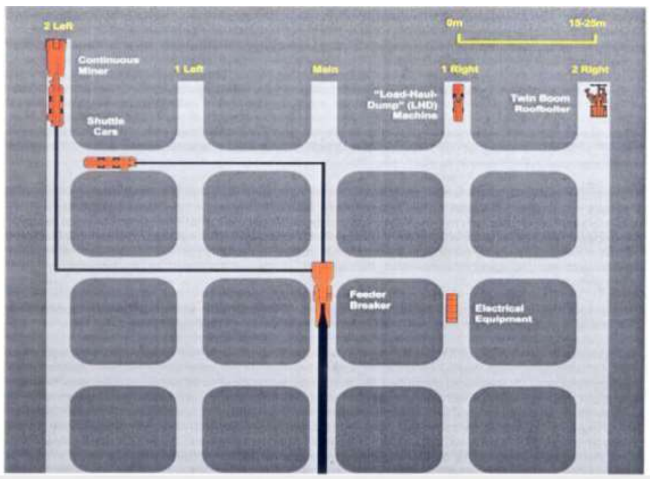

Fig. 4(a): 5- Heading Development Layout by C.M

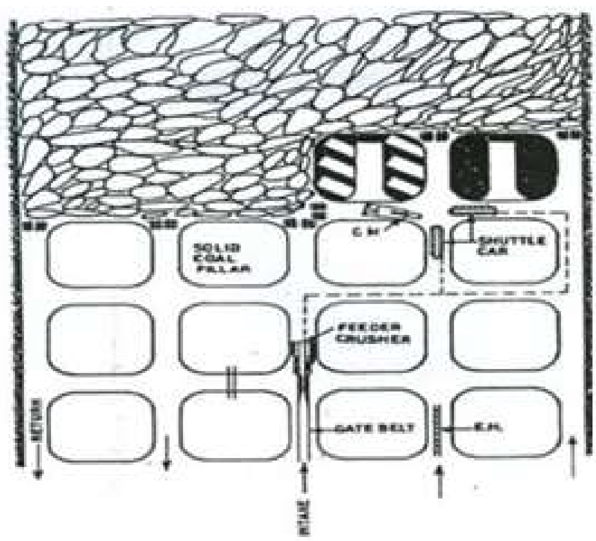

Fig. 4(b): A Typical 5-Heading Depillaring by CM

E. Low Capacity/Low Height Continuous Miner (LCCM/LHCM)

Low capacity/Low Height Continuous Miner (Fig.5) is a continuous coal wining m/c, relatively lesser output capacity operated in B&P layout. It is able to achieve 0.4 to 0.45 Mte./annum coal output (1200-1250Te/day) with productivity (O.M.S) of around 7/8. In a mine, at least 3 such sets will lead to economic viability and to support the overall M/Power requirement. Coal haulers/ Shuttle Car are used to carry coal from face and the exposed

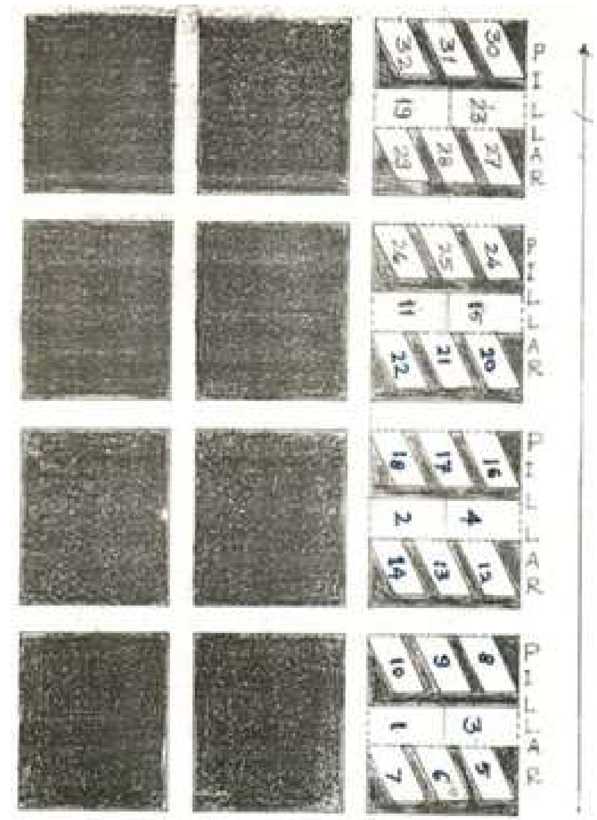

Fig. 4(c): Sequence of Depillaring by CM

roof is supported by roof bolts. In the presently operating u/g mines with B&P layout, such equipment combination is best suited in Indian conditions where coal evacuation by belt conveyor through Incline outlet. It was tried initially at Kumbhar Khani (WCL) and Raniatari (SECL) where the experience gained paved the way for further improvement in the system. Now similar miners are successfully operated at Jhanjra Mine with highly satisfactory m/c performance.

Manpower requirement is nearly 90 per m/c and 40 for overall services. At least 2 (Two) such M/C's if operated in a mine the mine economics will be in an acceptable level. The operation is similar to continuous miner district layout in B&P mining.

AVOIDANCE OF CYCLE U/G COAL MINING

Presently major u/g coal output in India is by cyclic mining with series of activities Drilling, Blasting, dressing, supporting, coal evacuation, face environmental assessment, water spraying for dust suppression etc. & 1.2 to 1.5m face progress per shift. None of the u/g coal mines is possible to be operated economically with such cyclic mining now in the present economics where EMS (Earning/Man-Shift –Rs. 4100/- approx) is very high. Only economic, relatively safe and better environmental control is possible by continuous coal wining technology in our u/g coal mining with improved mines safety performance.

Fig. 5: Layout for 5-Heading Development District with LCCM

ECONOMIC COAL MINING

Mining of coal is always an operation dealing many factors like strata/ground control, environment protection, proper operational planning, safety and health of persons so deployed etc. U/g coal mining, technology with drilling and blasting practices is not felt to be economic with designation-wise manpower deployment and cyclic, low productivity (O.M.S) mining. In opencast mining, mechanized operation with high capacity HEMM gives favourable performance. Surface Miner in o/c coal wining is the best option where Geo-mining environment permits. Economic u/g coal mining never possible with conventional cyclic mining in India while in o/c, surface miner where adoptable can be operated for non-use of explosives, dispensation of drilling & blasting in coal wining and extra revenue from consumer of Rs. 87 per Te for -100mm size product. However, mechanized o/c coal mining with both o/b and coal by drilling & blasting operation can yield economic output now with high capacity HEMM development.

EXTRACTION OF PROPERTY UNDER SURFACE STRUCTURES, BUILT UP AREAS, ROADS ETC.

Coal property below surface structures and built up areas is possible to be extracted with u/g mining by proper compaction/filling of the void from segregated sand of o/b, bottom ash, fly ash, from power plants. Also possible by using qua based compaction material or filling void by crushed o/b with paste material.

MULTI JOB CONCEPT – ALL MEN ALL JOB

In conventional u/g mining with designation-wise job, nearly 50% time every workmen remains idle in cyclic mining. In present situation for economic mining & manpower optimization management must adopt "All Men All Job" concept forthwith. All operating u/g mines need such concept of "All Men All Job" and settlement with respective union functionaries for improvement in u/g mining.

USE OF DIESEL EQUIPMENT IN UNDERGROUND MINE WORKINGS

In many u/g mines in India gas emission is quite less and Diesel equipment can be used for loading, transportation of coal in the district like LHD, Shuttle Cars, Ram Cars, Coal haulers. Output capacity of such m/cs enhances for free movement and easy maneuvering without limitation of cable feeding power to the m/c. Output capacity and percentage extraction gets increased where a better/clean district is maintained.

Diesel equipment is in use in many advanced countries in u/g coal mining where gas emission rate is much higher than our seams and same is possible for adoption in India too.

ECO-FRIENDLY MINING SOLUTION

Coal mining is a pollution-prone Industry and there are several environmental challenges in it. However to avoid the pollution to the best and for eco-friendly mining, steps are necessary against adverse impact on environment. Continuous coal wining options in u/g mining eliminating use of explosives causes less ground control problem and eliminates blasting fumes, dry dust formation from it. Sufficient dust control measures in such m/c's in u/g with high pressure water flow leads mine environment convenient than cyclic mining. O/C with surface miner is an eco- friendly coal wining option and mass production with least environmental effect from it.

O/B removal also possible economically by Ripper mining without avoiding explosives. It helps for mining near built up area and offers a solution towards mining near inhabitation. The operation of Ripper Mining for o/b removal found to be economically favorable over conventional and offers eco-friendly environment than use of explosives in O/B removal. Surface miner in o/c coal mining helps in selective mining, eliminates oxidation hazards in coal benches from shattering effect and coal bench fire. Continuous coal wining in u/g coal mining and in surface mines are environment friendly, safer and economic options compared to conventional drilling, blasting etc. in cyclic mining.

TECHNOLOGY UP-GRADATION & IMPROVEMENT IN MINE SAFETY PERFORMANCE

In coal mining there are several changes and improvement right from pick mining, bucket loading and present trend of mechanized continuous coal wining in both u/g and o/c coal mining scenario in India. Technology up-gradation also up-grades the safety performance in mines (Table-2a. 2b.)

Table 2(a): Last 5 years safety performance in CIL Mines

Year	No of Fatal Accidents	No of Fatalities		Total Fatalities
		UG	OC	
2014	43	19	26	45
2015	38	18	20	38
2016	38	19	42	61
2017	34	16	21	37
2018	33	15	28	43

Table 2(b): No. of Serious Accidents/injuries in CIL Mines (2014-2018)

Year	No of Serious Accidents	No of Serious Injuries			Total Serious Injuries
		UG	OC	Surface	
2014	198	104	38	58	200
2015	137	72	45	24	141
2016	120	74	27	22	123
2017	108	59	29	20	108
2018	87	52	21	23	96

The fatal & serious injury rate gradually declined and serious injuries in u/g are higher than that in o/c mines. Coal India Ltd. incorporated in 1975 sharing around 90% of coal production in the country. There were 177 accidents causing 233 fatalities for a coal production below 100 Mte. In 2018 there were 33 fatal accidents with 43 fatalities where production of coal crossed 606 Mte. It is also from technology up-gradation and improvement on following fronts.

- Elimination of manual loading
- Use of SDL/LHD at face.
- Roof bolting as primary support etc.
- Adoption of surface Miners in O/C mines.
- Around 10% u/g production from advanced blasting free mining.

Case Study

A. Technology Up-gradation & Safety Performance.

Table 3 Performance at Jhanjra Mine (ECL)

Year	No of accident	Injury rate/frequency	
		Per 1000 persons employed	Per Million Tonne Coal output
2012	19	6.0374	18.8889
2013	18	5.6567	17.3133
2014	12	3.77233	9.4674
2015	10	3.0750	6.7803
2016	4	1.2778	1.9402
2018	0	0	0

2012–1CM, 2014–2 CM's, Aug'2016–2 CM + PSLW, 2018–2 CM's + PSLW+2LHCM's

B. Performance at Sarpi Mine (ECL)

Table 4: Technology up-gradation Vs. Safety Performance at Sarpi u/g mine

Year	No of accident	Injury rate/frequency	
		Per 1000 persons employed	Per Million Tonne Coal output
2012	14	13.36	19.91
2013	3	3.51	4.12
2014	4	4.54	7.12
2015	1	1.15	1.37
2016	3	3.48	4.26
2018	1	0.0027	1.55

From 2011 - CM Operation

The trend is Similar in other u/g and o/c mines where technology up-gradation are there - Kurja mine, Churcha Mine, Dipka o/c, Gevra o/c mine, Kusmunda O/C mine, Haldibari, Vijay west mine, Sonepur Bazari O/C etc. and subsequent improvement in mine safety performance.

CONCLUSIONS

The technological options are similar in all the coal mining countries as mining remained International.

The technology has to be sustainable, eco-friendly to the best, potentially safe and economic. Cycle mining needs revisit as continuous coal wining is the only future option.

Besides high capacity HEMM development, the surface miner in coal wining added further Phillip for selective, safer and improved economic option in O/C mining which need wider diffusion. In o/b removal, adoption of suitable equipment like ripper etc. needed to avoid drilling, blasting to the best.

The road ahead in our coal mining as mentioned in Chapter 5 are to find wider diffusion at suitable locales for economic and safer future mining solution.

High Wall Mining are to be tried at more sites economically.

Use of 'Man Riding System' in u/g mines to avoid long, arduous travelling of employees & 'All Men All Job' concept in u/g coal mining is need of the hour for manpower optimization/economic operation.

Coal property below built up surface structures, High ways, water bodies is possible for extraction economically with the help of proven technology by u/g mining & filling the void suitably.

The technology has to be suitable for Geo-mining environment at particular locale and u/g coal mining has to be potentially safer, economically sustainable.

The average loss of around Rs. 3000 per Te. in u/g coal mining in nationalized coal sectors in India need to be reversed with safer, sustainable and techno-economic option.

Future mining both in u/g and o/c in Indian coal mining are to be eco-friendly to the best, safe and sustainable to support bulk output economically by technology up-gradation as practiced in major coal producing countries in the world.

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Reservoir-Induced Alterations and Climate Change Effects: A Case Study of Alaknanda River Basin

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ABSTRACT

The Bhagiratha brought the nectar from heaven and the fortunate Himalaya let it flow over its valleys and recharge throughout the year with the help of south-western monsoon and westerlies. The sacred Ganga, descending from the heavens through the Himalayan valleys, nourishes the land with its perennial flow, fed by the monsoon and the westerlies. But the river basin faces many challenges in the modern world, from pollution to climate change. The lower stretches of the river are tainted by urban waste, while the upper reaches are prone to floods and flash floods that cleanse the river but also cause havoc and destruction. The Kedarnath Disaster (2013) is a grim reminder of the river's wrath. The river basin receives abundant rainfall and snowmelt from both sides of the orographic barrier (i.e. main central thrust zone), which affects the water volume and discharge. This study explores the relationship between the discharge and the disasters (or the extreme events) that occurred in the Upper Ganga River Basin. The data from stations Karnaprayag, Rudraprayag (Alaknanda), Rudraprayag (Mandakini), Chandrapuri, Tehri and Devprayag show that the orographic front side had high discharge values in the years 1998, 2005 and 2010, which coincided with several reported extreme events. The results suggest that the anomalously high discharge value is a good indicator of an extreme event in the region. This indicator can be used as a warning signal for future disasters and to plan for mitigation and management strategies.

Key words: Ganga, discharge, extreme event, precipitation pattern.

INTRODUCTION

The Himalayan River systems have nearly double sediment erosion rate as of any compare to any other river system (Singh and Hasnian 1999). This erosion is attributed to high relief, steep gradient, favourable rock type, intense precipitation, glacial activities and so on. The erosion of sediment is an extensive and slow process where the precipitation and the glacial inputs are generally cyclic. There are a number of recorded events (e.g. Kedarnath Floods in 2013) when a thick pile of sediments is deposited in just couple of days (e.g. as thick as 10 m at Sonprayag; Devrani et al. 2015) due to swift weather changes in the precipitation resulted as cloud bursting. In the Himalayas, this anomalous sedimentation is generally recoded during the south-western Monsoon (generally June to September). This phenomenon is taking place because the basin gets a major part of total precipitation in just one third period of the year, except the water added by westerlies and snow melt. The natural disasters like flash floods and landslide in the Ganga River basin are mostly consequences of extreme events (Paul et al. 2000; Naithani et al. 2002, 2011; Kumar et al. 2007; Gupta and Bist 2004; Rana et al. 2012; Dubey et al. 2013; Gupta et al. 2013) added with snow melting at high reaches (Rautela and Thakur 1999; Naithani 2001; Singh et al. 2005; Rana et al. 2012) and natural damming of the river (Paul et al. 2000; Sati et al. 2011).

The wet season of the year also affect the area or especially the slopes of the area in two ways, firstly the excess pore water increases the pressure along with lubrication of the incompetent layer and secondly the sudden rise in stream water level initiate the toe cutting of slopes. This slope failure mechanism turns-out in natural damming (landslide damming) and the sudden bursting of the same increases the sediment load and erosion in the basin.

The present study is an attempt to predict the natural disasters with the help of precipitation outcomes, i.e. water discharge and sediment yield. The observations are also strengthened by the analysis of precipitation trend in the study area.

STUDY AREA

The South Asian Sub-continent is fed by a number of rivers and the Ganges covers third most important part with 8.61×10^3 km^2 area. The lower reaches of the basin is experiencing flood almost every year, probably due to

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climatic changes and the interference of anthropological elements in the flow region. In the recent past, the encroachment in the upper river valley, to harness the energy and to enhance tourism, has made the situation even worse. So, the major emphasis in the present study is given to the major two river valleys, Alaknanada and Bhagirathi (30°-31°N latitudes and 78°45'-80°E longitudes), where the extreme events and resultant disasters are frequently witnessed.

This area is regionally known as *Garhwal Himalaya* and is sandwiched between the Tibetan plateau and Indo-Gangetic Plains on the north and south respectively. The area covers eight major districts of Uttarakhand state of India and the catchment area is about $11.08\times10^3 km^2$ and $7.65\times10^3 km^2$ for Alaknanda and Bhagirathi Rivers respectively. The geology of area has been studied intensively since 1885 by Middlemiss and the stratigraphic successions are established after Auden in 1934 which was further refined by a number of subsequent workers (i.e. Heim and Gansser 1939; Jain 1971; Rupke 1974; Valdiya 1980, 1995; Srivastava and Mitra 1994; Richards et al.2005, Yin 2006). The study area consists of three geological domains, firstly, the Trans Himalayan Sedimentary sequence is bounded by South Tibetan Detachment Sequence (STDS; Le Fort 1996) and lies on the southern edge of the Tibetan plateau and in the northern part of the Garhwal Himalaya. This area receives scanty rainfall due to the presence of the Higher Himalayan series which prevents the moisture-laden winds from migrating further northward. The second downhill is the Higher Himalayan Crystalline Series, bounded by MCT (main central thrust) in the south (Coleman 1998) and consists of the medium to high-grade metamorphic rocks belonging to the Vaikrita Group and Munsiari formations (Patel and Carter 2009) with pro-grade metamorphism (Vannay et al. 2004) at places.

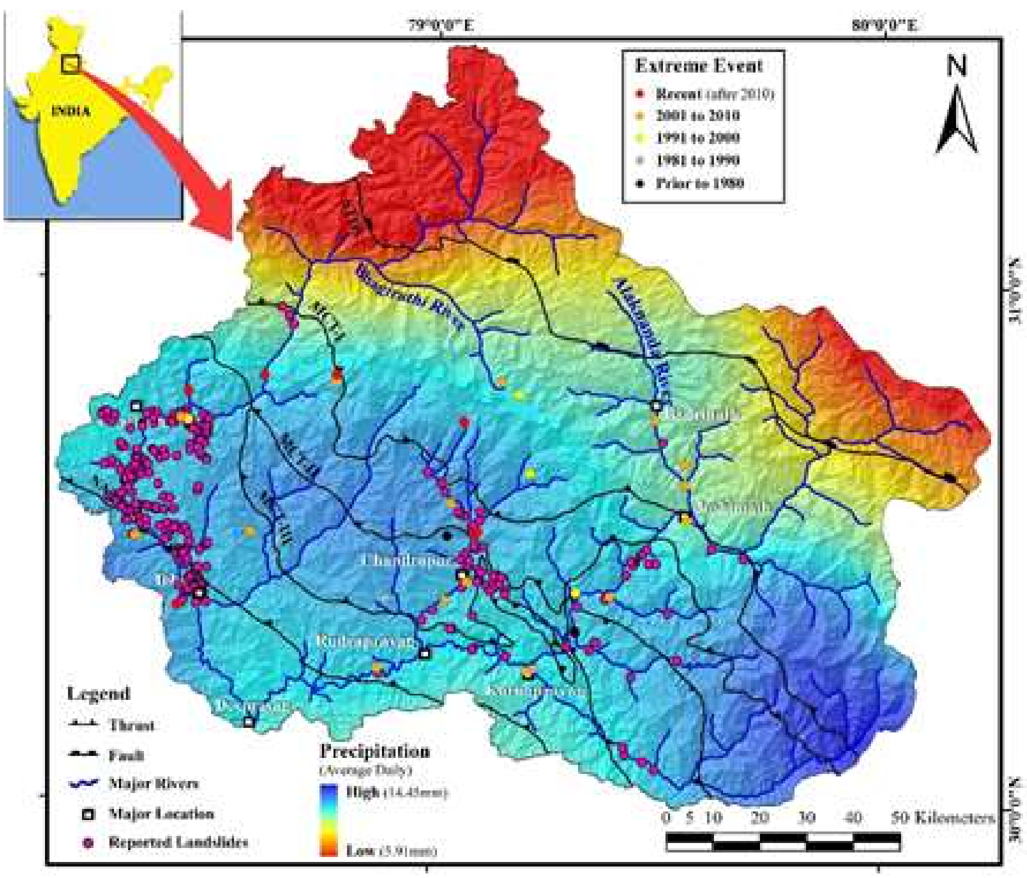

Figure 1: Map showing the rainfall distribution and location of reported landslides and extreme events in the study area.

Finally, towards the foothill side, the Lesser Himalayan Crystalline and Sedimentary are bounded between the MCT II (Munsiari Thrust) in the north and MBT (main boundary thrust) in the south and consist of medium to low-grade metamorphics, calc-silicates, impure limestone and dolomites (Valdiya 1995).

There are several local and regional faults and other structural deformations existing in the study area (Thakur 1992; Rajendran et al. 2000; Celerier et al. 2009, 2010) which also affects the discharge and sediment production in the basin.

METHODOLOGY

The specific sediment yield is calculated according to Haritashya et al. (2006) and Holeman (1968) method with the help of about 34 years (1976-2010) annual river discharge and sediment load data collected by CWC (Central Water Commission) observation stations. Data of the nine observation stations have been selected for the analysis, i.e. Badrinath, Joshimath, Rudraprayag in Alaknanda River; Uttarkashi, Tehri and Devprayag in Bhagirathi River; Chandrapuri and Rudraprayag stations in the tributary of Mandakini River and Karnaprayag in Pinder river (a tributary of Alaknanda River).

The precipitation data is collected from TRMM (Tropical Rainfall Measurement Mission), which is about 40% of the observed data so it is further standardised with the help of observed precipitation data of 1998, 2005 and 2010 collected from IMD (Indian Meteorological Department).

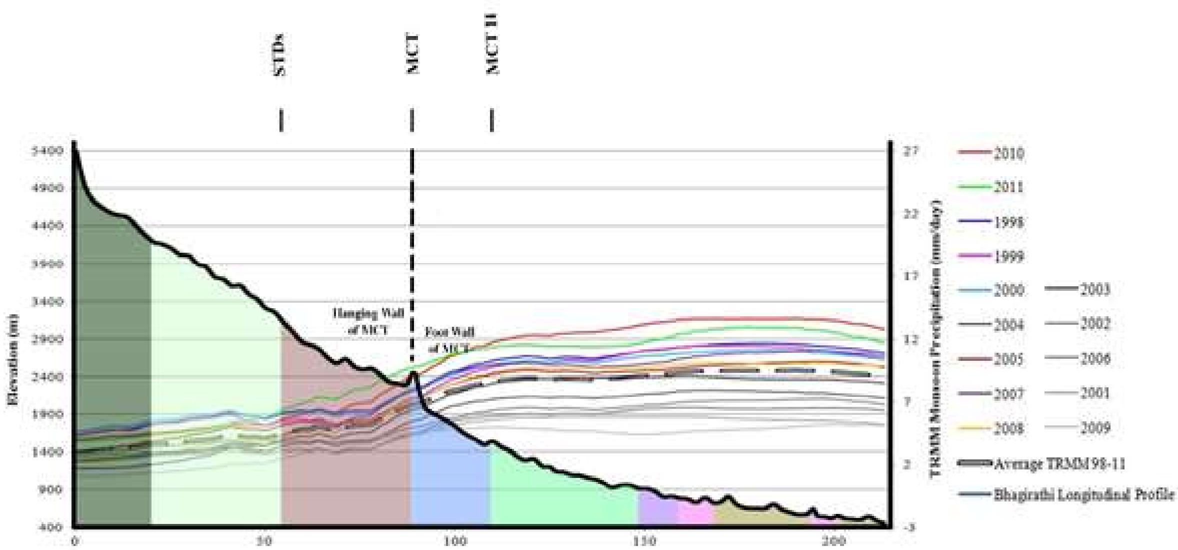

Figure 2:TRMM precipitation data (1998-1999) plotted over the longitudinal profile of the Bhagirathi River, depicting MCT as an orographic barrier (note the peaks in front of MCT).

RESULTS AND ANALYSIS

The present study analyses the contribution of monsoonal rainfall in the disasters that took place in the basin. However, at the same time high values of river discharge and sediment load contributed from the glacial melts cannot be ruled out, especially in Chorbari Bamak glacier, Satopanth and Bhagirathi Kharak Glacier, Khatling Glacier, Tiprabamak Glacier, Gangotri and Dokraini glacial region. It is also noticed that in the years 1998, 2005 and 2010 the area received high river discharge and sediment loads which could be cumulative effect of high glacial melt and high precipitations.

The results shows that on an average per year 5.42×10^6 ton of sediment is eroded from the Bhagirathi River Basin but it shows an anomalously high value of about 2.5 times in 2010. Such anomalous increase have also been observed in 1998 and 2005. The annual sediment erosion is comparatively high in Alaknanda River Basin (6.17×10^6 ton) and the anomalous increase is observed in the same years as witnessed in Bhagirathi River Basin, which is highest (~1.5 times) in year 2010. The data suggests a positive correlation between the discharge and sediment load, moreover the trend is almost similar. Though, some specific storm events have also been documented with anomalously high discharge (as high as 1208m^3/s in July, 1998) as compared to the average monthly discharge of 119 m^3/s.

In the study area, a total of 31 extreme events and 308 landslides have been recorded and/or observed. Among them, half (13) were witnessed during 1998, 2005 and 2010. The steep slopes in the study area also show some failure incidents in the aforesaid period.

If, extreme events and landslide incidents show a positive correlation with discharge and sediment load, it can be directly correlated with the precipitation as well. So, the discharge of the river can be used as a tool for predicting a disaster which is going to happen in the area.

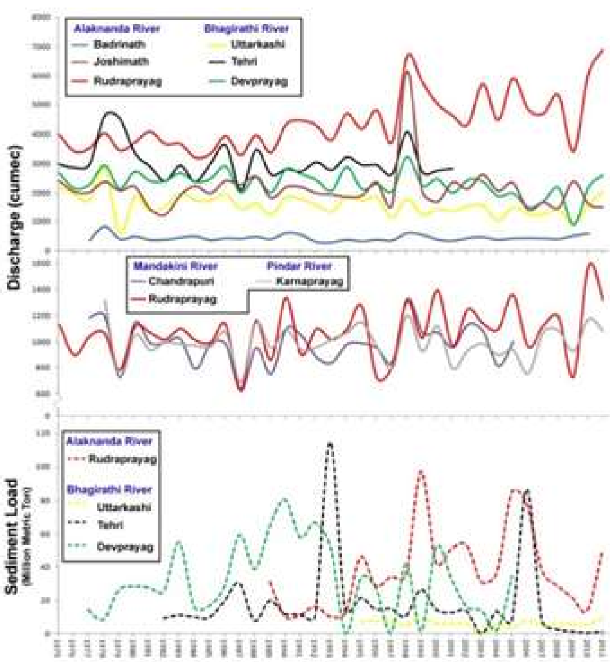

Figure 3: Graphs showing the positive correlation between the discharge and sediment load data (note the peaks correlate well the extreme events of 1998, 2005 and 2010.)

CONCLUSION

As discussed in the previous sections, the Garhwal Himalayas have witnessed numerous intense mass instability and metrological disasters such as flash floods, cloud bursts and glacier avalanches or glacial lake outburst flood (GLOF) during the monsoon, among them, the Kedarnath Disaster can be cited as the recent one. The analysis of the precipitation, discharge and sediment load data from different stations in the concerned basins relates well with the occurrences of extreme event instances. The assimilation of the various relationships drawn among the operating factors and the resultants provided significant insight into the existing knowledge.

The present communication reveals a fair agreement between the high discharge and the subsequent high sediment load and the occurrence of extreme events. So, the anomalously high values of both the discharge and sediment load for a particular location can be treated as a signature or signal for an extreme event that is a preparatory phase and may take place.

In the present scenario, where the humans have penetrated deeply into the natural balance, the effects of extreme events are exaggerated to a great extent. Such in-depth studies on are required to predict the phenomenon and to suggest preventive measures to deal with or minimize the damage to life and property in the Indian Himalayan region for future. So, the utilization of the outcomes of the present research can be utilized as a forewarning to the upcoming devastating extreme event.

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