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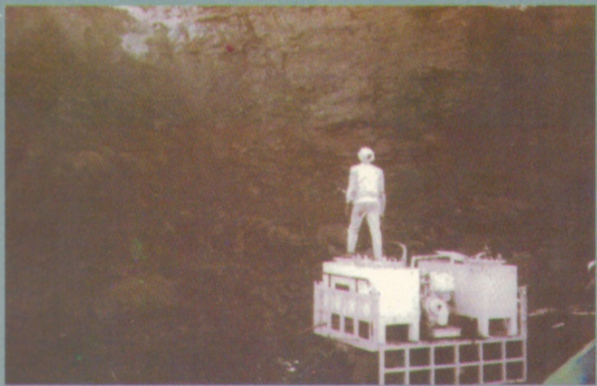

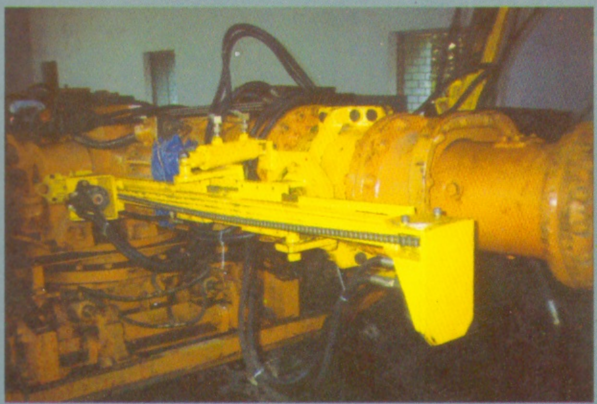

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CABLE BOLT BASED MECHANISED DEPILLARING OF A THICK COAL SEAM

Rajendra Singh*, P.K Mandal* and A.K. Singh*

चिरीमिरी क्षेत्र की एन. सी. पी. एच. खान की 6.8 मीटर मोटी 3 नं. सीम का निष्कर्षण कमजोर कोयला चाल सम्बद्धता, लम्बे अवलम्ब की अनुपलब्धता तथा अन्य संस्तर नियंत्रण की समस्याओं के कारण आंशिक ही था। सीम 3 मीटर की ऊँचाई तक ही विकसित थी तथा स्प्लिटिंग एवं स्लाइसिंग द्वारा 4.8 मीटर की ऊँचाई तक अंतिम निष्कर्षण हेतु स्वीकृत थी और इस प्रकार कुल 40 प्रतिशत प्राप्ति ही संभव थी। संशोधित खनन पद्धतियों को अपनाकर पिलर पर खड़ी मोटी सीम से सुरक्षित तरीके से अधिक से अधिक कोयला निकालने तथा उत्पादन एवं उत्पादकता में सुधार लाने के लिए कई अध्ययन किए गए। खान की भू-खनन स्थिति को ध्यान में रखकर कई विकल्पों पर गहराई से विचार किया गया तथा केबुल बोल्ट से उच्च चाल थम्हाल कर डिपलरिंग को अन्तिम विकल्प के रूप में चुना गया। केबुल बोल्ट की मदद से न केवल बेहतर उत्पादन और उत्पादकता के साथ सकल निष्कर्षण मे वृद्धि हुई बल्कि सपोर्ट की कम लागत के साथ बेहतर सुरक्षा भी प्राप्त हुई। आलेख में केबुल बोल्ट से थम्हाल के लाभ पर चर्चा की गई है।

INTRODUCTION

The NCPH mine of chirimiri area under S.E.C.Ltd. developed 6-8 m thick No. 3 seam extensively on pillars. The depillaring was limited to only 4.8 m along the floor, leaving rest of coal within the goaf with the level of recovery invariably below 40 per cent due to non-availability of effective long supports. Moreover, the conventional supports in form of a jungle of props and chocks obstructed the SDL operation. Due to weak parting along the roof, coal band caved upto full height even in early stage of depillaring when efforts were made to withdraw the easy coal under unsupported roof. Fallen coal in the goaf frequently caused potential danger of spontaneous heating.

In view of the problems, the Central Mining Research Institute (CMRI) undertook the exercise developing suitable method of extraction with better recovery, conservation and safety. The method deploying a piece of old haulage rope as cable bolt was developed to keep the roof supported during mining irrespective of the working height. The critical input to the experimental trial of the method were (a) drilling of long holes in coal/sandstone roof (b) full grouting of the cables ensuring sufficient anchorage strength and (c) stability of the cable bolts even after blasting of the roof coal. The system named Chirimiri Method of Mining was conceived and tried to mitigate support problems during thick seam mining at NCPH mine. The studies cover the reliability of the cable bolts as support, its impact over caving and settlement of the goaf with due regard to safety, conservation, production, productivity, problem of spontaneous heating and overall economics.

The basin has five coal seams developed in Barakar stage including 6-8 m thick No. 3 seam having 9.30 million tons of coal logged in pillars at NCPH mine. The seam No. 3 is overlain by a thin band of shale and 10 cm thick layer of graphite. This is followed by 3-4 m thick coarse sandstone poorly laminated and massive in character. The formation on the whole appeared to be competent and stable. The immediate roof could be 3.0 m thick likely to cave after 30-35 m advance of the face irrespective of the cable bolting. The seam was developed on pillars of size varying from 25 to 30 m centre to centre along the floor up to 3 m height leaving coal/shale/sandstone band in the roof. The developed pillars were divided into artificial panels by the fire proof stoppings and the number of pillars within the panels varied from 22 to 50 pillars.

CONVENTIONAL MINING METHOD

Three panels within the old lease mine were depillared by conventional method of splitting and slicing. During the final extraction, the pillars were split by level drivages followed by slicing upto 4.8 m height. The slicing method was associated with the problem of heightening of roof due to progressive failure and/or separation of the coal band along the graphite roof. As a result in the beginning, high recovery upto 65 per cent was reported within a first few pillars followed by weighting and roof fall in the subsequent pillars when the recovery dropped upto 30 to 40 per cent. The support of even 4.8 m high roof was a real problem in view of non-availability of large size timber props. The ribs left on either side of the slices, were subjected to high stress due to slow winning. Resultant side spalling added new dimensions to the method of extraction of the developed pillars and invariably the

*Scientist, Central Mining Research Institute, Barwa Road, Dhanbad- 826 001.

workings had to be abandoned midway with very poor recovery, low production and productivity.

The depillaring operation was adopted by conventional cycle of drilling and blasting and manual loading of coal on tubs. The loading of the coal was under poorly supported roof due to ineffectiveness of large size props and non-availability of such props in sufficient numbers. Scrapper was subsequently introduced at a stage to improve the safety of the workers when only few of them were exposed to such hazards during anchoring of the scrapper and drilling of roof coal. The method under ideal condition yielded 40 to 45 per cent recovery with production level below 200 t/day and the productivity around 1.4.

MINING CONCEPT WITH CABLE BOLTS

The miners were familiar with slicing method of pillar extraction mechanised with SDL for loading and pit props as the support. Main problem in the exploitation of a thick seam was the support of high roof so essential for the safety of the workers engaged for lifting of the blasted coal. The use of wire ropes under tension while reinforcing the roof was more efficient as a support and cost effective with no scope of flexure or buckling. The suspended support hanging in the roof kept the floor free for the movement of the workers or the loading machines. It was decided to use full column grouted steel ropes as cable bolts for advance support of the immediate coal band in the roof and also the immediate sandstone roof. Its application, however, was fraught with the following operational problems:

1. Drilling of a large number of long vertical holes through coal and sandstone interface.
2. Installation of 6-8 m long rope in the roof.
3. Reliability in the installation and grouting of long cables for adequate anchorage.
4. Impact of the blasting of the coal roof over the anchorage of cable portion left over in the immediate roof.
5. Chances of delayed caving of the reinforcement roof.

DRILLING OF HOLES FOR CABLE BOLTING

In the trial stage of the cable bolting, long holes in the roof were drilled manually using normal coal drills where a crew of 4 persons could drill 2-3 holes in a shift. The experiments of the cable grouting and blasting of the immediate coal roof revealed 50 per cent drop out as their grouting was confined within the coal roof. The anchorage of some of the cables was over 10 t after the installation and anchorage remained over 10 t even after blasting of the roof coal. The preliminary trial revealed the possibility in respect of the following:

a) Possibility of getting adequate anchorage in holes of desired length.

b) Anchorage remained unaffected by blasting of roof coal

The problem was solved by the cooperation of the local engineers when a hydraulic drill was assembled; using the SDL power pack at Korea workshop, SECL. The system is shown in Fig. 1(a), comprising hydraulic system and hoses connected to the drilling machines for lifting with the help of chain drive and a drill mounted on pinion. The system was able to drill 12 vertical hole of 6-8 m length including 1.5-2 m within sandstone roof in a shift. Later, tyre mounted assembly of drilling was fabricated for the purpose of mobility of the machine which is shown in Fig. 1(b).

GROUTING OF CABLE BOLTS

The arrangement for full column grouting of the cables is shown in Fig. 2. The holed of 43 mm diameter were drilled across the coal band and 1.5 m within the immediate sandstone roof. The mount of the hole was rimmed to 80 mm diameter over 20 cm length by improvised coal drilling bit. The holes in general were desired to be vertical. The system improved and modified in stages served fairly well except the mobility of the system which has to be shifted from place to place manually.

The rope of the desired length was inserted in the hole along with (a) vent tube of 5 mm diameter and (b) grouting tube of 12 mm diameter. The assembly was held up in position by a tapered circular wooden block of 5 to 10 cm and length nearly 15 cm. The block had three holes to accommodate the rope (22-25 mm dia), vent hole (5 mm dia) and a grouting hole of 10 mm diameter. The wedge shaped block was tightly fitted at the mouth of the hole. The breathing and the cement injection tubes were loosely tied to the rope keeping the inner end of the breathing tube up to the end of the hole.

The cable assembly was inserted in the hole with one end of the vent tube dipped in water bottle. The gout mix comprising of cement, water and hardcrete in the proportion of 50 kg, 30 litres and 2 litres respectively was prepared in the grouting tank attached to a compressor for injecting the slurry (Fig. 3). The grouting assembly was mounted on another trolley for site injection through hose.

The injection tube was set nearly 15 cm below the end of the cable and as soon as the vent tube set at the extreme end gave indication of slurry flow; the grouting tube was withdrawn.. In order to avoid ungrouted socket; injection was continued even during the withdrawl of the grouting

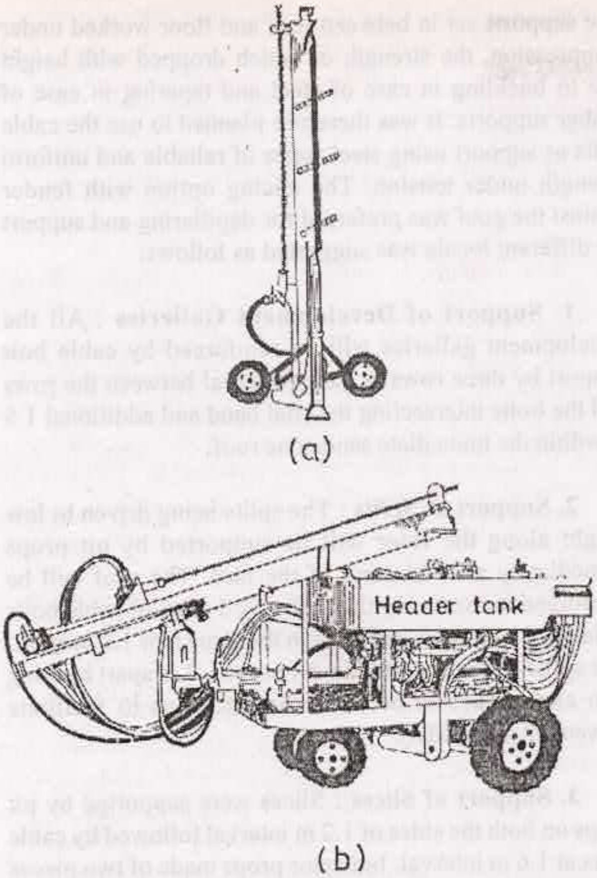

Fig. 1 : Hydraulic drilling machine developed at Korea Workshop.

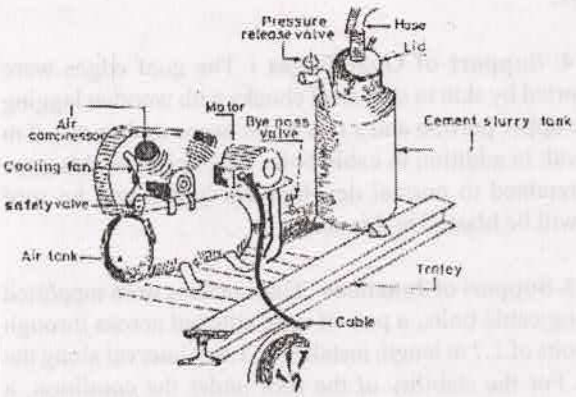

Fig. 2 : Grouting assembly for the cable bolt.

tube. The vent tube was pulled out and the total operation was completed within 2 minutes.

As a modification; the grouting tube was inserted close to the hole mouth inbye wooden block and the slurry was pushed upward upto the end of the hole. With the indication of full column filling; it was taken out and the open end was plugged by jute/cotton waste, It was possible to grout 12-18 holes in a shift in case the setting of the grouting assembly was not to be changed. A crew of 4 persons could pull the trolley; arrange the grouting set-up and grout 12 holes in a shift. The grout started setting within 30 minutes and gained full strength within 4 hours.

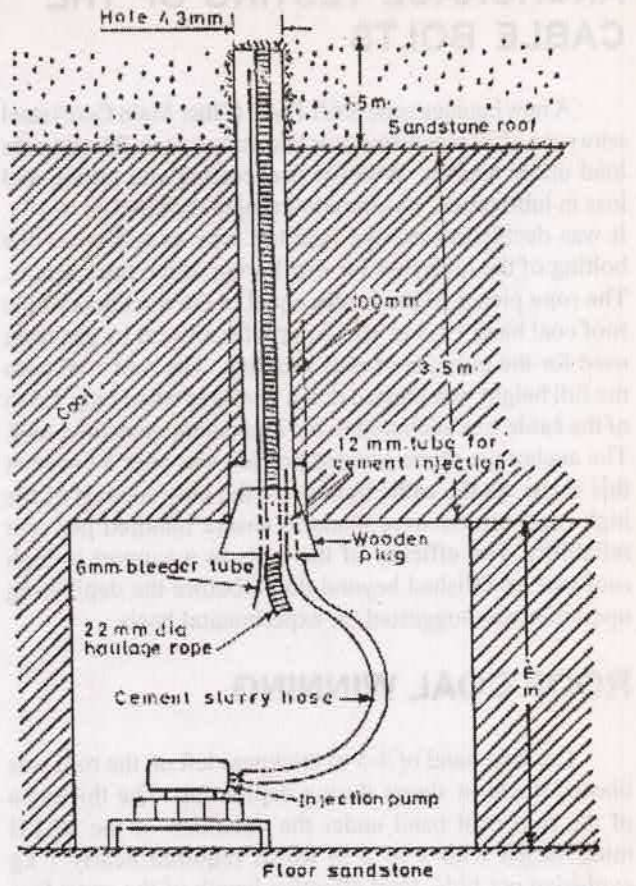

Fig. 3 : Arrangement for grouting of long cable.

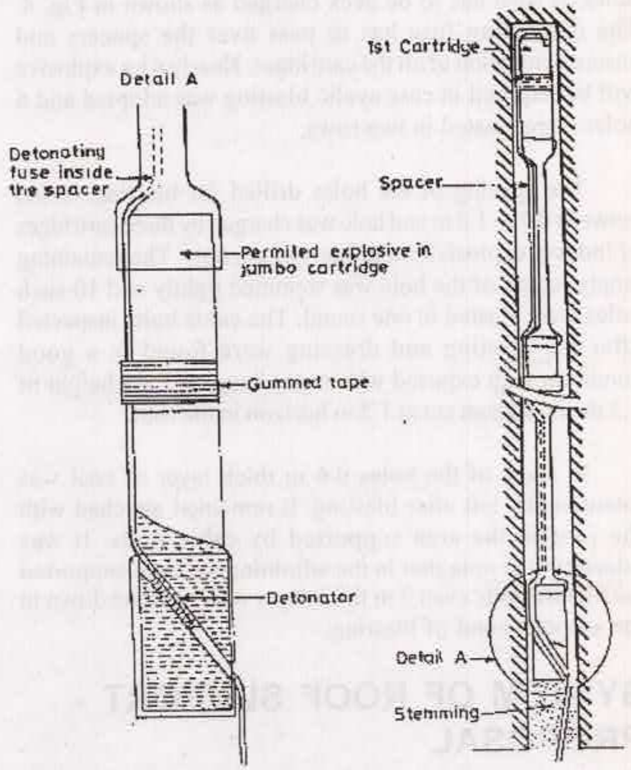

Fig. 4 : Charging and stemming arrangement for a long hole blasting.

ANCHORAGE TESTING OF THE CABLE BOLTS

A new haulage rope 6/6/1 FMC (Fiber Main Core) steel wire rope of 20 mm diameter is known to sustain 20 t ultimate load under tension. With the use, rusting and pitting and loss in lubrication, its ultimate strength dropped up to 15 t. It was decided to use the rejected rope as cables for the bolting of the high roof for depillaring of the thick seams. The rope pieces of the length equal to the thickness of the roof coal band +1.5 m +0.10 m preferably cut by gas were used for the purpose of roof grouting. The roof coal upto the full height was blasted down in stages when only 1.5 m of the cable was anchored to the immediate sandstone roof. The anchorage of the grouted portion was over 6 t even at this stage. As the cable bolts were the only support of the high roof; efforts were made to ensure hundred per cent reliability. The efficacy of the bolts as a support to high roof was established beyond doubt before the depillaring operation was suggested on experimental basis.

ROOF COAL WINNING

The coal band of 3-5 m thickness left on the roof was blasted down in stages during depillaring. The thickness of the roof coal band under the condition of the NCPH mine varied with 3 to 5 m which required nearly 1 kg explosive per hole, total effective length of the cartridges nearly 1 m only. For the purpose, the blasting practice with Uniring Cord and Soligex explosive was recommended. The holes as such has to be deck charged as shown in Fig. 4. The detonating fuse has to pass over the spacers and ensure detonation of all the cartridges. Nearly 6 kg explosive will be required in case cyclic blasting was adopted and 6 holes were blasted in two rows.

The spacing of the holes drilled for blasting varied between 0.7 to 1.0 m and hole was charged by three cartridges of Indocol explosive weighing 140 gm only. The remaining empty length of the hole was stemmed tightly and 10 such holes were blasted in one round. The cable bolts inspected after the blasting and dressing were found in a good condition with exposed wire ropes hanging for a height of 1.5 m and a clean cut at 1.5 m horizon in the roof.

In some of the holes 0.6 m thick layer of coal was intentionally left after blasting. It remained attached with the roof in the area supported by cable bolts. It was interesting to note that in the adjoining area not supported with cable bolts even 2 m thick layer of coal came down in the second round of blasting.

SYSTEM OF ROOF SUPPORT - PROPOSAL

The crux of depillaring exercise in thick seams has been efficient support system effective even in high workings. The supports set in between roof and floor worked under compression, the strength of which dropped with height due to buckling in case of steel and tapering in case of timber supports. It was therefore planned to use the cable bolts as support using steel ropes of reliable and uniform strength under tension. The slicing option with fender against the goaf was preferred for depillaring and support for different locale was suggested as follows:

1. Support of Development Galleries : All the development galleries will be reinforced by cable bolt support by three rows at 1.6 m interval between the rows and the bolts intersecting the coal band and additional 1.5 m within the immediate sandstone roof.

2. Support of Splits : The splits being driven to low height along the floor will be supported by pit props immediately after blasting of the face. The roof will be reinforced immediately thereafter by 3 rows of cable bolts installed 1.6 m apart and bolts in the same row 1.2 m apart. The splits were supported by pit props 1.2 m apart keeping 3 m clear space in between the pit props to facilitate movement of the SDL.

3. Support of Slices : Slices were supported by pit props on both the sides of 1.2 m interval followed by cable bolts at 1.6 m interval. Indicator props made of two pieces of steel pipes joined by wooden pieces were installed in the rib side of a slice to indicate the status of rib loading. Ledges of roof coal were supported by bolts and W straps.

4. Support of Goaf Edges : The goaf edges were supported by skin to skin steel chocks with wooden lagging in the upper portion and a row of pit props outbye at 1.0 m interval, in addition to cable bolts. The height in this zone was retained to normal development height and no roof coal will be blasted in this zone..

5. Support of Junctions : The junctions were supported by long cable bolts, a pair of rope stitched across through eye bolts of 1.7 m length installed at 1.6 m interval along the rope. For the stability of the roof under the condition, a coal stump was left on the goaf side (Fig. 5) in the corner with rope end anchorage.

METHOD OF DEPILLARING

The depillaring system was conventional splitting and stooking, maintaining their height equal to that of the normal development height. The pillars were divided in two stooks and the stooks so formed were extracted by slicing, keeping the height of the slices equal to that of the splits and the development headings. The steps involved in the extraction of the pillars are summarized as follows:

1. Cable bolting of all the development galleries of the panel before the start of the depillaring operation.

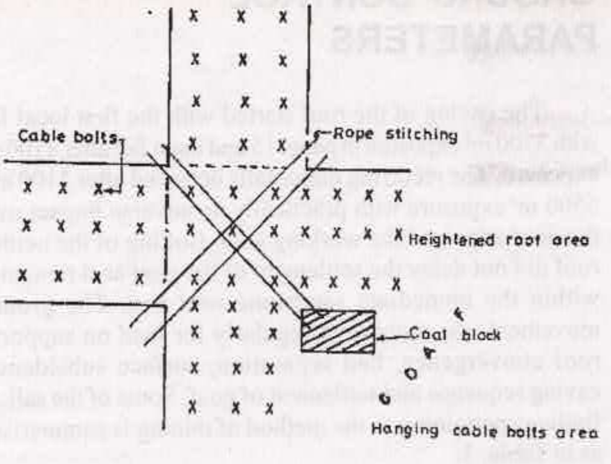

Fig. 5 : Junction support with cable bolts / rope stitching.

2. Systematic support of the development galleries 2 pillars within the goaf line using tabular steel props and timber/steel chocks.

3. Drivage of 5 m wide split gallery for dividing the pillar in two halves and its systematic support with tubular steel props and chocks in addition to the cable bolting.

4. Drivage of 4.8 m wide slick from the goaf edge leaving 2.5 m thick fender and support of the low height roof by steel pit props in addition to cable bolts.

5. Support of the slice edge by steel chock and cross bars and the goaf edge by steel chocks.

The sequence of depillaring suggested with the cable bolt as support for the high roof and conventional pit props and chocks for low height roof is shown in Fig. 6. Extraction of roof coal was resumed on retreat under only cable bolt support from the inbye end of the slice.

EXPERIMENTAL TRIAL

The method of depillaring with cable bolt support for high roof was adopted as a test case for No. 3. seam of NCPH mine. The experimental trial even with all preparations had a number of doubts in the mind of mine operators and safety officials. Some of the doubts and apprehensions were as follows:

1. Chances of overriding of the goaf with the weakening of stooks due to increase in height.

2. Chances of spontaneous heating of loose coal left in the goaf.

3. Chance of delayed caving of the roof due to its reinforcement by cable bolting resulting in air blast.

4. Efficacy of the cable bolts as a support to the immediate roof after blasting of the roof coal to full thickness.

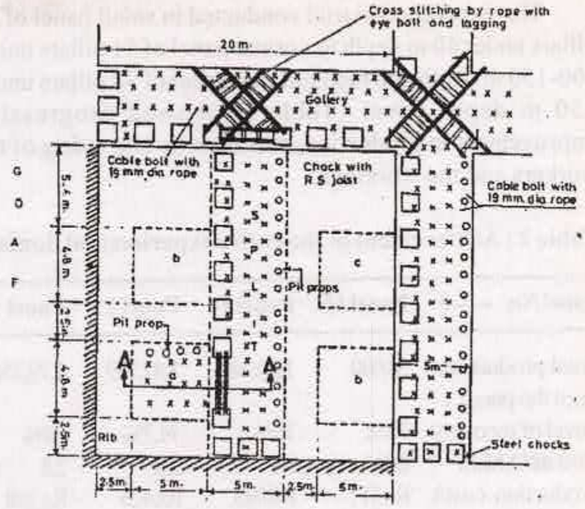

Fig. 6 : Depillaring method suggested with cable bolts.

The experimental panel was located in the farthest corner of the property with natural barrier all around as a precaution against any unforeseen problem. The coal seams of the area had a number of active fires and records of palaeo fires and the estimated incubation period within 9 months. It was, therefore, desired to have small size panel which could be extracted within the incubation period at the lowest expected production level. A panel No. 15 located along the common boundary of Kurasia and NCPH mine appeared to be the ideal site for the experimental trial as it had natural barrier on two sides while stoppings could be made on the rest of the sides. The experiment was continued for another three panels of different geomining conditions of NCPH mine is summarized in the Table. 1.

Table 1 : Geomining parameters for the experimental domain of NCPH mine

Parameters	Panel 15	Panel 16	Panel 17	Panel 18
Panel size, m x m	115x135	200x220	180x190	190x200
No. of Pillars	22	43	52	38
Pillar size, m x m	22.5x22.5	25x25	22.5x22.5	30x30
Seam Th., m	6.5	7.5-8.0	6.5	6.5
Depth cover, m	33-58	37-70	46-103	119-256
Coal reserve within the panel, t	70,000	2,49,000	1,98,885	2,93,000
No. of cable bolts	-	5,005	5,824	-
Cable bolt size, m	5.0	6-6.5	5.0	5.0
Bolt matrix, m x m	1.2x1.2	1.2x1.2	1.2x1.2	1.2x1.2
Bolt anchorage, t	18	>10	>10	>10
General ground condition : Disturbance/ Joints/Fissure/ Pillar spalling	Not Percep-tible	Not Percep-tible	Not Percep-tible	Percep-tible

The experimental trial conducted in small panel of 22 pillars under 40 m depth to normal panel of 51 pillars under 100-150 m depth and medium size panel of 30 pillars under 250 m depth cover (Table 2) showed progressive improvement in production, productivity and safety of the workers and the workings.

Table 2 : Achievement in the entire experimental domain

Panel No. ⟶	Panel 15	Panel 16	Panel 17	Panel 18
Total production from the panel, t	53,000	1,89,240	1,47,759	1,79,358
Level of recovery	75%	76%	74.3%	66%
District O.M.S.	2.01	2.6	2.8	2.8
Production cost/t,	Rs.417	Rs.395	Rs.405	Rs.398

The economic impact with the implementation of this technology as compared to the conventional method of depillaring was found to be very encouraging. The following information in respect of economic impact from NCPH mine support it.

Estimated production from 1.04.97 to 31.03.98	:	1,54,793 t
Sale price of annual production (sale price Rs. 859.82/t)	:	Rs. 13,30,94,117
Estimated cumulative production upto March, 1998	:	8,25,000t
Sale value of cumulative production	:	Rs. 70,93,53,500
Reduction of production cost/tonne	:	Rs. 76
Additional profit with introduction of the technology during trial period	:	Rs. 6,27,00,000

GROUND CONTROL PARAMETERS

The caving of the roof started with the first local fall with 3300 m^2 exposure in panel 15 and main fall after 4100 m^2 exposure. The recurring major falls occurred after 5100 and 5500 m^2 exposure with practically no adverse impact over the goaf edge and the working area. Bolting of the neither roof did not delay the settlement of the goaf as it remained within the immediate sandstone roof bed. The ground movement was monitored regularly for load on supports, roof convergence, bed separation, surface subsidence, caving sequence and settlement of goaf, Some of the salient findings pertaining to the method of mining is summarised as in Table. 3.

CONCLUSION AND RECOMMENDATION

The Chirimiri method of mining was conceived, planned and executed with an aim to transfer the technology in letter and spirit and monitor its techno-economic feasibility. The experiment started in a small panel with 22 pillars in view of past record of heating in the area and conservative estimate for the production from the experimental panel. The advance preparation of the panel by way of cable bolting of all the development galleries, mechanization of drilling and grouting and evacuation of coal by SDL proved to be very advantageous when the targeted production was achieved within three months and the panel which was expected to lost 10 months was extracted within 7 months.

Table 3 : Summarised strata contrl parametres of the experimental domain

Parameters	Panel 15	Panel 16	Panel 17	Panel 18
Maximum convergence, mm				
- Within the working area	8	18	28	37
- Along the goaf line	26	20	36	97
- Inbye goaf	23	55	58	
- Convergence rate before fall, mm/day	2-4	2-3	2-3.5	3-4.5
Load on support, t				
- Maximum up to goaf edge	11	12.5	7.6	18
Maximum stress variation, kg/cm^2				
- Up to goaf edge	11	21	18.2	24
-Within the goaf	21	52	32	31.9
Maximum exposure of the goaf, m^2				
- Local fall	3,300	2,425	1,100	1,350
-First major fall	4,100	5,900	4,900	5,400
- Second major fall	5,100	7,950	6,775	6,180
- Third major fall	5,500	9,200	8,825	10,080
Subsidence record				
- Maximum subsidence, m	2.64	2.40	Not measured	Not measured

The speed of depillaring process has been found to be very essential to keep the stress field to minimum, restrict spalling in stooks, ribs and pillars and to control the chances of spontaneous heating. No slice extraction should be deferred for the third shift and no stook should be left for the next week to keep the damaging parameters within control.'

The main apprehension of anchorage loss to the cable bolts due to blasting of the roof coal band proved wrong and the cable bolts retained adequate anchorage to serve as the advance support to the freshly exposed roof. The roof on the other hand caved very regularly without any sign of weighting.

The performance of the cable bolts remained satisfactory under all the conditions and the production and productivity improved with the experience of the miners. Coincidentally, the formation in one of the panels (Panel No. 18) was geologically disturbed and the roof had two to three sets of open joints. The cable bolt support even in such a condition remained very effective, though the immediate roof coal band was additionally supported by bolts and W-straps. The system proved to be safe and productive under good to disturbed roof condition because of the advance preparation and efficient support planning.

ACKNOWLEDGEMENT

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WIDE STALL MINING: A METHOD TO OPTIMISE COAL RECOVERY FROM A SEAM UNDER SURFACE FEATURES

Rajendra Singh*, P.K. Mandal* and A K. Singh*

इमारतदार सतही संरचनाओं के अधीन कोयले की इष्टतम प्राप्ति हेतु परम्परागत प्रणाली के पिलर के विखंडन एवं स्टूकिंग का कार्य क्षेत्र व्यापक तात्कालिक चाल संस्तर एवं सामान्यत: गहराई के अंतर्गत स्थित कोयले के लिए सीमित है। चाल संस्तर की क्षमता तथा पिलरों के आकार के साथ इनकी क्षमता में तेजी से की जा रही वृद्धि का लाभ उठाकर इमारतदार सतही संरचनाओं के नीचे स्थित कोयले की इष्टतम प्राप्ति की परम्परागत प्रणाली के स्थान पर अब वर्तमान गैलरियों को एक विशेष आकार में चौड़ा करने में उपयोग किया जा रहा है। इसे वाइड स्टॉल कहा जाता है। इसका पहला क्षेत्र-परीक्षण ईस्ट भगतडीह कोलियरी में किया गया। इस प्रणाली से अनेक खण्डों में विकसित 17 मी. मोटाई वाली कोयला सीम के सतह एवं उप सतह को नुकसान पहुँचाए बिना (जोखिम में डाले बिना) लगभग 420 मि0. टन कोयला प्राप्त हुआ। विज्ञान एवं प्रौद्योगिक योजना के अन्तर्गत ईस्ट भगतडीह कोलियरी में इस प्रणाली के सफल क्षेत्र परीक्षण के बाद सतह/उपसतही बाधाओं के अन्तर्गत स्थित कोयले की इष्टतम प्राप्ति के लिए इसका उपयोग विभिन्न कोयला क्षेत्रों में व्यापक रूप से किया गया। इस आलेख में वाइड स्टॉल खनन के साथ-साथ इसके प्रयोगशाला एवं क्षेत्र अन्वेषण के परिणामों पर भी प्रकाश डाला गया है।

INTRODUCTION

The development of a coal seam under varying thickness of overlying strata cover may face two extreme situations where problems related with instability of underground developments become pronounce. First situation arises due to presence of inadequate hard cover over a driven gallery at shallow depth cover. Under this condition there is a chance of collapse of entire rock-mass column over the galleries up to surface (formation of pot holes). Under the second situation, high induced stress dilutes the adequate size of pillars due to side spalling. The chance of occurrence of this type instability problem is more at deeper side of a mine. Once a pillar of inadequate size get crushed at higher depth cover range, redistribution of mining induced stresses may affect the stability of neighbour pillars which may ultimately increase the size of failure enough to disturb the surface structures.

Under surface/subsurface structures, extraction of coal seams of moderate depth cover faces problem of low recovery. In this situation large amount of coal is sterilised inside a panel as natural support in form of pillars/stooks to ensure the stability of overlying strata vis-a-vis surface-subsurface structures. Conventional method of mining for optimal coal extraction under such geomining conditions is splitting and stooking. Formation of stooks during the optimisation of recovery causes considerable drop in strength of the natural support which is not desirable for a long term stability of the underground mining and surface structures. A more difficult situation arises during the optimisation of recovery from a multisection developed thick coal seam where additional problems like: decrease of pillar strength due to increased height of extraction [1], stability of partings and superimposition of pillars make the situation more complex from production, productivity and safety points of view. For Indian geomining conditions, conventional splitting and stooking method does not provide more than 30% coal [2-3[during recovery optimisation from a developed thick seam under built-up surface structures.

Around 60% of the total coal reserve, workable by underground mining, of India belongs to thick seams[1]. Most of these coal seams are under massive sandstone strata and shallow depth cover. The reserve of Jharia coalfield, which contributes nearly 90% cocking coal of the country, faces similar geomining conditions. Further coal seams are below important surface structures and have extensively been developed in single and multiple sections to keep a balance between demand of the coal and safety of the built-up structures. The coal production strategy by formation of pillars was found favourable for the industry also because of low capital investment and involvement of low level engineering support. Development of all these seams is almost complete and now the industry is looking towards the huge amount of coal locked up in pillars[2].

Geomining conditions of the Jharia coal field were simulated in physical and numerical models. These simulated models were tested for a number of alternatives including narrow panel mining, splitting and stooking and

[1] *In India coal seams of 4.8 m thickness or higher are called thick.*

[2] *Around 900 million tonnes of coal are blocked in pillars in Jharia coal field only.*

***Central Mining Research Institute, Barwa Road, Dhanbad, Bihar, India, 826 001.**

widening of galleries. On the basis of extensive laboratory investigations on different simulated models, idea of wide stall mining was conceived and, first time, implemented in field at East Bhuggatdih colliery of the Jharia coalfield for experimental trial. Taking advantage of the presence of massive sandstone roof and rapid increase of pillar strength with the increase of its width height (w/h) ratio[4], splitting and stooking of the pillar was replaced by wide stall formation. Wide stalling provided better recovery and safety (Fig. 1) in comparison to the conventional method so the successful initial field trial at East Bhuggatdih colliery of Bharat Cocking Coal Limited (BCCL) under built up surface structure followed large scale application of this mining method in Indian coal fields to protect a variety of surface features including fragile ecology of the surface [5]. This paper describes the basic philosophy behind wide stall formation[3] along with the results of the experimental field trial conducted at East Bhuggatdih colliery.

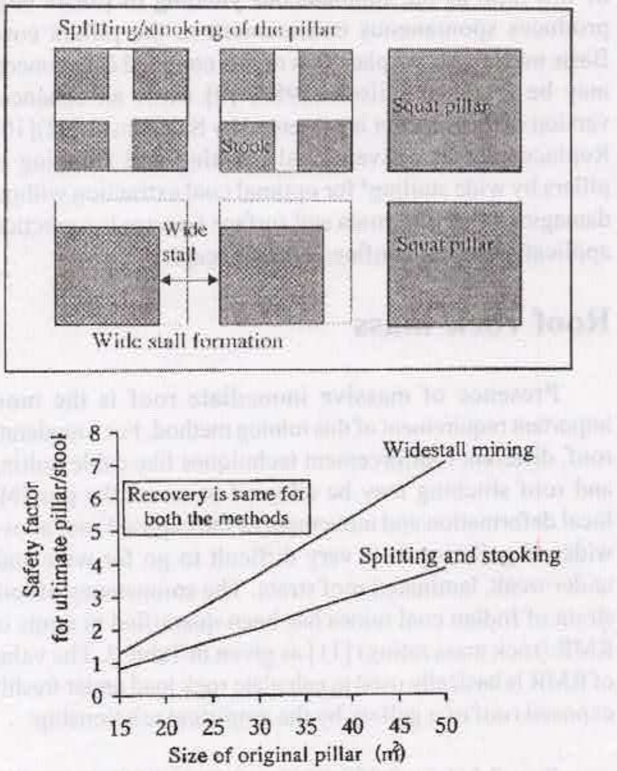

Fig. 1 : Comparison of wide stall mining with conventional splitting & stooking of a pillar.

Details of the site

East Bhuggatdih colliery is situated in Jharia coalfield which falls under BCCL. Jharia township lies in the heart of BCCL property creating problems for liquidation of a number of coal seams underneath. Around 80% leasehold area (246 hectare) of this colliery is under the densely populated part of the Jharia town. The coal seams being liquidated at this colliery are thick and are under shallow depth cover. There are 16 coal seams at different depths (numbered from zero to fifteen in ascending order from the deepest one) in the leasehold of this colliery. Immediate roof of most of the seams of this area is massive sandstone of Barakar formation. The top most four coal seams, XV to XI/XII outcrops within the leasehold area of this mine leaving too little part of XV & XIV seams to be worked at this colliery. Coal seam XIII is jhama and seams XI/XII and X are extensively developed in the area. From VI to zero coal seams of the area are virgin and the status of different coal seams being liquidated at this colliery is given in Table I.

The scope of optimal recovery of coal locked in oversize pillars of VII/VIII seam led the trial of wide stall mining at this colliery. 17 m thick VII/VIII seam was fully developed in two sections; 3.0 m thick bottom section along floor and 2.4 m top section leaving 6.4 m coal parting between the two sections and 5.0 m coal in roof. Total recovery attained by pillar formation (30 m x 30 m, centre to centre) in two sections, having 3.6 m wide gallery, was 7.4%

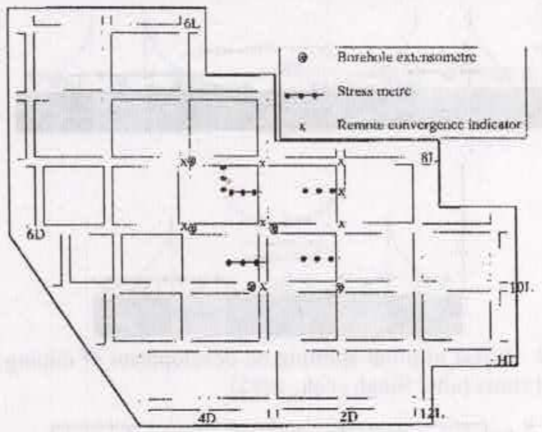

Fig. 2 : Plan view of the experimental panel with instruments for strata control observations.

Table 1 : Salient details of different coal seams being worked at East Bhuggatdih colliery.

Seam	Thickness (m)	Depth (m)	Status of working	Details of working Pillar size (m x m) (centre to centre)	Gallery width (m)
XI/XII	13	27	Caved (30 years ago) over a part of the panel	-	-
X	13.6	68	Standing on pillars	27 x 27	3.2
IX	12	93	Few galleries were driven	-	3.5
VII/VIII	17	110	Developed	30 x 30	3.6

[3] *Major part of this paper has been published in* ***International Journal of Rock Mechanics and Mining Sciences, 36(2), 155-168.***

only. Experimental panel consisted 25 pillars (Fig. 2) at an average depth cover of 110 m and was completely free from geological disturbances. Widening of galleries around the two undersize pillars of the panel was not possible due to long term safety consideration of the mining structure. Thickness of the parting and superimposition of the two developed contiguous sections in the seam were checked at every gallery junction through bore holes drilled at all these junctions. During this examination, the thickness of the parting deviated ± 0.5 m from above mentioned value while superimposition of pillars/galleries of the two contiguous sections deviated up to 3 m.

Wide stall mining

Matching of gallery size with the strength of immediate roof and strength improvement of the natural support by increasing w/h ratio [6] are the two basic constituents of the wide stall philosophy. Strength of ultimate pillars and stability of overlying exposed roof span, both, play important role for long term stability of a wide stall under shallow depth cover. Conventional splitting and stooking for optimisation of recovery during partial extraction under surface features reduces the strength of the natural support and brings core of the resulted stooks under the effect of mining induced stress [7] (Fig. 3). The effective bearing capacity of a pillar is comparatively more than a number of stooks of the equivalent area. Wide stall formation[4] accommodates existing galleries of a developed coal seam

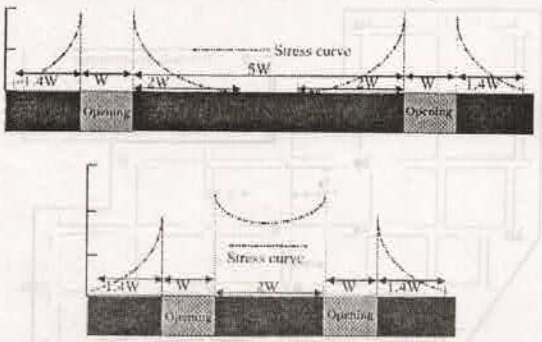

Fig. 3 : Effect of pillar splitting on development of mining induced stress (after Singh et al., 1992).

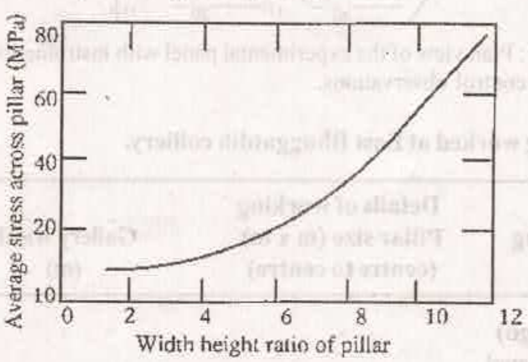

Fig. 4 : Sharp increase of pillar strength for higher values of W/H ratio (after Gale, 1992).

as well as improves recovery and safety of the ultimate mining structure in comparison to the conventional method. Different basic constituents of a wide stall working are discussed below.

Pillar size

It is not very easy to explain the failure process of a squat pillar but the influence of width/height ratio (w/h) on the strength of a pillar is well known. An analysis based on extensive field observation conducted by Gale (1992) [8] showed that the increase of pillar strength with the increase of w/h ratio (Figure 4) becomes rapid for higher values of w/h ratio. This effect of w/h ratio on pillar strength is considered to reflect confinement distribution developed within the pillar. In fact, compressive strength of a pillar of brittle materials, such as coal, is enhanced for higher values of w/h ratio as the simultaneous yielding of pillars edge produces spontaneous confinement to the pillars core. Basic mechanistic explanation of this confined core concept may be found in Wilson (1983) [9] while an enhanced version of this concept is presented by Salamon (1992)[10]. Replacement of conventional splitting and stooking of pillars by wide stalling[5] for optimal coal extraction without damaging overlying strata and surface features is a practical application of the confined core concept.

Roof rock mass

Presence of massive immediate roof is the most important requirement of this mining method. For a moderate roof, different reinforcement techniques like cable bolting and roof stitching may be adopted to arrest the possible local deformation and movement of the exposed roof above widened galleries. It is very difficult to go for wide stall under weak, laminated roof strata. The competency of roof strata of Indian coal mines has been quantified in terms of RMR (rock mass rating) [11] as given in Table 2. The value of RMR is basically used to calculate rock load under freshly exposed roof of a gallery by the empirical relationship:

$$R_t = Bd\,(1.7 - 0.037\,RMR + 0.0002\,RMR^2) \qquad (1)$$

where R_t is rock load in ton/m², B is gallery width (roof span) in m, d is mean rock density in ton/m³ and RMR is rock mass rating.

Table 2 : Geomechanical classification of roof rock mass (after CMRI, 1987)

RMR	Class	Description
0-20	V	Very poor
20-40	IV	Poor
40-60	III	Fair
60-80	II	Good
80-100	I	Very good

[4] *Widths of the existing galleries are increased in a particular configuration to improve coal recovery leaving wider pillars (in comparison to stooks) of intact core.*

[5] *Four stooks are replaced by one pillar of almost equivalent area.*

Unfortunately, this relationship is valid up to 4.2 m gallery width only. This method of roof rock classification does not give any idea about the stability of drivages wider than 4.2 m. A different classification of roof rocks adopted by United States underground mines [12,13] provides relationship between roof span and CMRR (coal mine roof rating) which may be written as:

$$F_r = 0.87B - 0.23\,CMMR - 6.2 \qquad (2)$$

where, F_r is roof fall rate, B is gallery width in m and CMRR is coal mine roof rating. The results of this relationship are shown in Figure 5. This relationship establishes influence of gallery width on the stability of the roof strata of different strength but it was not very straightforward to apply any CMRR type classification of a different country to Indian geomining conditions. In the

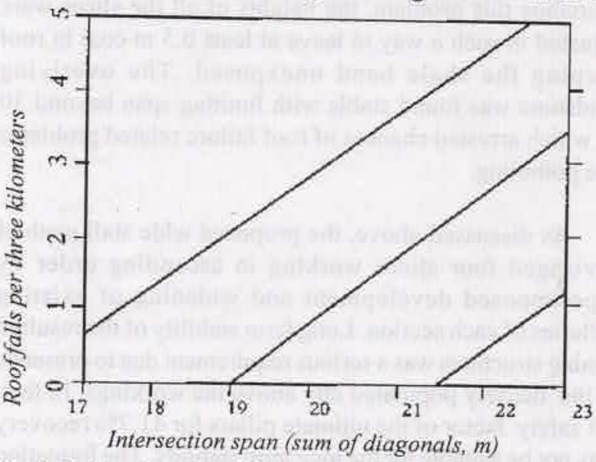

Fig. 5 : Estimation of roof fall rate using CMRR and intersection span (after Molinda & Mark, 1997).

absence of an established guideline to decide the span of a widened gallery, the geomining condition of East Bhuggatdih colliery was simulated on equivalent material mine model [14] and numerical models. During these investigations, it was observed that wide stall mining method can comfortably be practised under a roof strata of RMR class I and II[6]. Investigations on these simulated models [3] showed that a gallery width of 9 m will be most suitable for optimisation of recovery and safety at East Bhuggatdih colliery.

Depth cover

Average depth cover of developed coal seams of Jharia coalfield is around 150 m which is another advantageous condition for the wide stall mining. In fact, the low value of vertical in situ stress for shallow mining condition [15] is favourable for the stability of roof over the widened gallery. However, high value of in situ horizontal stress at low depth cover creates a chance of shear failure for a week roof strata [16]. The possibility of shear failure of roof strata due to high horizontal in situ stress at low depth cover may be overcome by orienting the direction of drivages parallel to the direction of maximum horizontal stress[17]. A detailed study [18] of influence of horizontal stress on the stability of exposed gallery roof of different RMR values shows that the deteriorating effect of high horizontal in situ stress decreases with the increase in RMR value. The influence of high horizontal in situ stress on the stability of exposed roof strata over a wide stall becomes insignificant because the presence of a massive roof strata of high RMR value is the basic requirement for wide stalling.

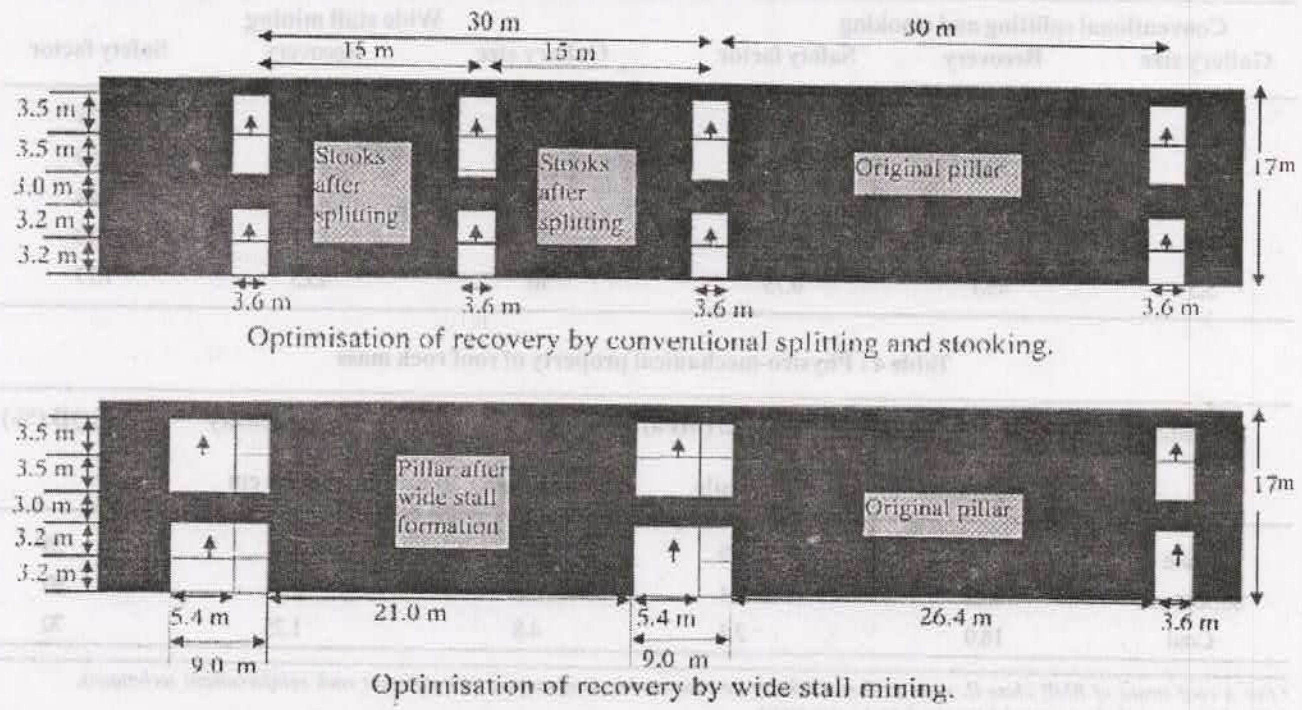

Fig. 6 : Dimensional information for wide stall mining at East Bhuggatdih with a comparison to conventional splitting and stooking.

Design of wide stall for East Bhuggatdih colliery

Testing of simulated models provided initial idea of a safe wide stall formation under the geomining conditions of East Bhuggatdih colliery. But estimation of dimension and strength of the underground mining structures during the wide stall formation is also done with the help of other simple approaches being practised by the industry. The uniaxial compressive strength of coal of the VII/VIII seam, tested as per guide lines of ISRM, was found to be 18 MPa (5 cm^3 coal samples). The seam was already developed in two sections so the optimisation of recovery was planned accordingly in two sections. As per the thickness and positions of the available working horizons in the seam, two slices of equal heights were planned for each section. Working was planned in ascending order from the bottom slice keeping slices heights of bottom and top sections equal to 3.2 m and 3.6 m respectively and leaving an intact coal bed of 3 m thickness between the top and bottom sections as shown in Fig. 6. Due to presence of the three metre thick parting between the two sections, height of only one section was taken into account during the estimation of strength of the ultimate pillars. Computed values of the safety factors for different gallery widths during optimisation of recovery by wide stall method and splitting/stooking of the pillars are given in Table 3. It is clear from this simple exercise that wide stall provides 1.3 safety factor for 41.7% recovery while the conventional splitting and stooking provides 0.90 safety f actor of the ultimate pillars/stooks for this level of recovery.

A nine meter wide gallery was to be driven around the pillars of the panel to achieve 41.7% recovery of coal. This widening of the existing 3.4 m wide gallery brings stability of the exposed roof into picture. Considering the case of a clamped beam, stability of the roof strata over the wide stall is estimated with the help of its tensile strength, thickness, volumetric weight and weakening coefficient For the first three slices in the panel, the immediate roof is coal of 2.4 MPa tensile strength and 70% RQD (rock quality designation). The VII/VIII seam was poorly cleated[7] and was found suitable to remain stable up to 10-15 m span. The immediate roof of VII/VIII seam is a shale band of 1.9 m thickness followed by massive sandstone strata of 30 m thickness. As per physico-mechanical properties of the formation given in table 4, the immediate shale roof has low RQD. Extensive support requirement was felt for 4th slice working (wide stall) under this weak roof strata. To overcome this problem, the heights of all the slices were adjusted in such a way to leave at least 0.5 m coal in roof keeping the shale band unexposed. The overlying sandstone was found stable with limiting span beyond 30 m, which arrested chances of roof failure related problems like potholing.

As discussed above, the proposed wide stall method envisaged four slices working in ascending order by superimposed development and widening of existing galleries of each section. Long term stability of the resulted mining structures was a serious requirement due to presence of the densely populated city above the workings. In fact, 1.3 safety factor of the ultimate pillars for 41.7% recovery may not be suitable for the long term stability. The formation

Table 3 : Safety factors of the ultimate pillar for different methods of mining

Conventional splitting and stooking			Wide stall mining		
Gallery size	Recovery	Safety factor	Gallery size	Recovery	Safety factor
3.6	34.6	1.06	6	29.5	1.75
4.0	37.8	0.98	7	33.7	1.59
4.5	41.7	0.9	8	37.8	1.44
5.0	45.5	0.82	9	41.7	1.30
5.5	49.1	0.75	10	45.5	1.17

Table 4 : Physico-mechanical property of roof rock mass

Formation	Strength (MPa)			Bulk density	RQD (%)
	Compressive	Tensile	Shear	gm/cu. cm	
Shale	55.6	1.9	10.6	2.5	20
Sandstone	46.2	5.4	11.9	2.41	80
Coal	18.0	2.4	4.8	1.25	70

[6] *For a roof strata of RMR class II, wide stall method can be practiced in conjunction with different rock reinforcement techniques.*

[7] *The frequency of cleat varied between 7 to 12 per meter.*

[8] *Gradient of seam dipping was favourable for right stowing.*

of wide stalls of second slice of bottom section was proposed on the sand floor of first slice as all stalls of each slice were proposed to be stowed with sand. Similar approach of stowing and wide stalling was to be followed for the two slices of the top section. Proposed sand stowing is not only to help in upper lift working but to minimise the chances of potholing and subsequent weathering of exposed roof strata and pillars. Side thrust and confinement provided by the stowing[8] is considered to improve the strength of the resulted coal pillars. The wide stall formation in different horizons were proposed to maintain steps along with faces to facilitate transport layout from the existing floor development up to the top section workings.

Working in the panel

Above described norms and parameters to optimise recovery of coal were adopted, without any change, during field trial of wide stall mining in the panel. Conventional system of drilling, blasting and manual loading was adopted to develop and widened the galleries inside the panel. The

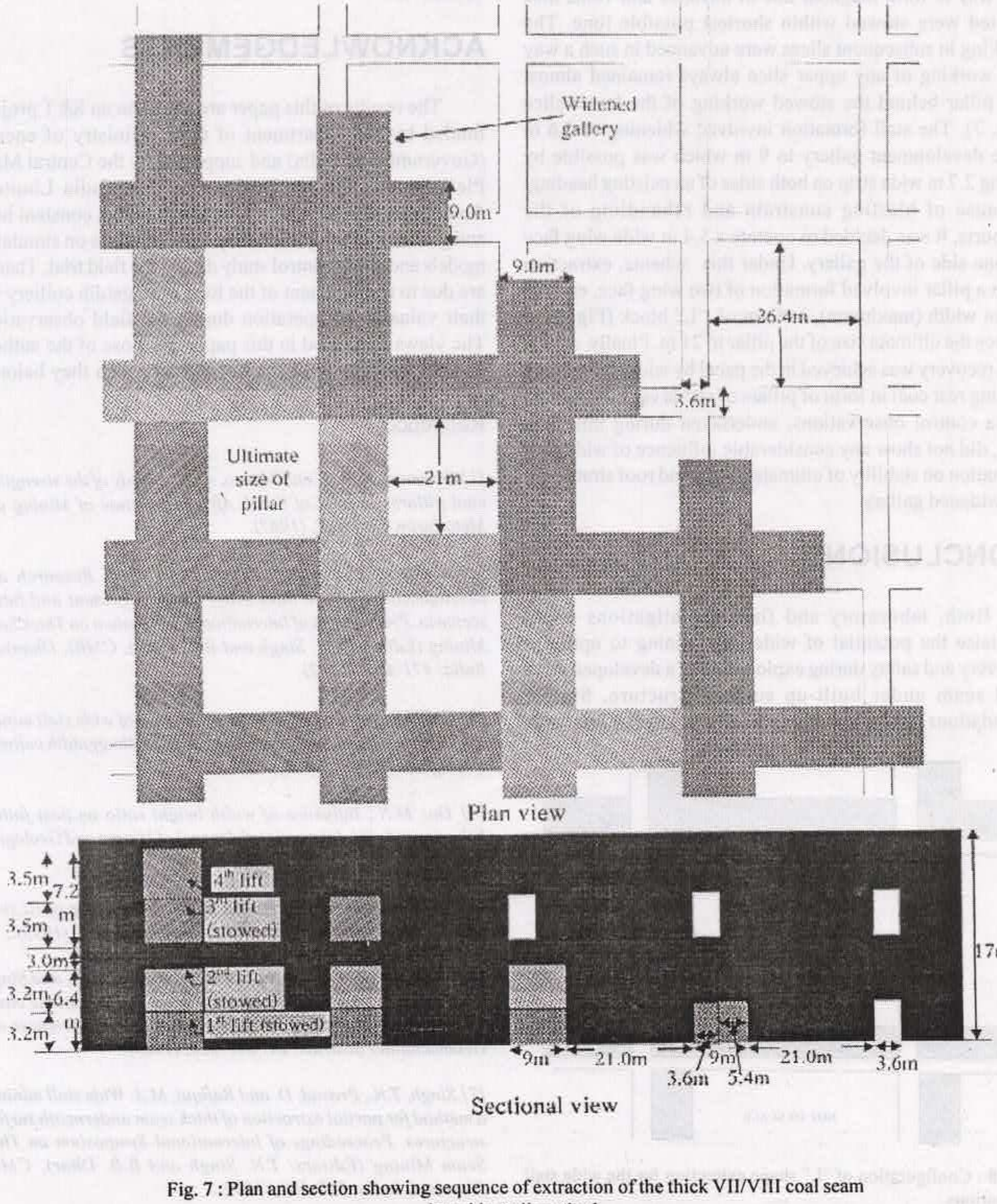

Fig. 7 : Plan and section showing sequence of extraction of the thick VII/VIII coal seam by wide stall method.

stalls of all four lifts were formed in ascending order with stowing of lower slice before the formation of the upper ones. The method of working and sequence of extraction followed in the panel is shown in Fig. 7. Seam thickness in the panel varied from 16.5 m to 16.9 m, so marginal adjustments in the slice thickness were done to ensure 3 m solid parting between top and bottom sections and, at least, 0.5 m coal in roof. Working in the panel followed the conventional depillaring style maintaining diagonal line of face and face advance from dip to rise. Wide stall formation in 1st slice of bottom section started by attacking the pillars in a way to form diagonal line of advance and voids thus created were stowed within shortest possible time. The working in subsequent slices were advanced in such a way that working of any upper slice always remained almost one pillar behind the stowed working of the lower slice (Fig. 7). The stall formation involved widening of 3.6 m wide development gallery to 9 m which was possible by taking 2.7 m wide strip on both sides of an existing heading. Because of blasting constrain and rehandling of the supports, it was decided to operate a 5.4 m wide wing face on one side of the gallery. Under this scheme, extraction from a pillar involved formation of two wing face, each of 5.4 m width (maximum), in form of "L" block (Fig. 8) to reduce the ultimate size of the pillar to 21 m. Finally, 41.7% coal recovery was achieved in the panel by wide stall mining leaving rest coal in form of pillars of stable core. Extensive strata control observations, undertaken during this field trial, did not show any considerable influence of wide stall formation on stability of ultimate pillars and roof strata over the widened gallery.

CONCLUSIONS

Both, laboratory and field investigations could visulaise the potential of wide stall mining to optimise recovery and safety during exploitation of a developed thick coal seam under built-up surface structure. Simple calculations, based on strength of the surrounding rock mass, and results of simulated models may be used to design the dimension of the ultimate pillars and limiting span of roof over the widened galleries. This method has a number of advantages over the conventional splitting and stooking method including improvement in stability of the ultimate mining structures. Successful field trial of this mining method and favourable results of the strata control monitoring could show the strength of this mining method. Drawback of this method is the requirement of a favourable geomining conditions: among which availability of a competent immediate roof strata is must for shallow mining conditions.

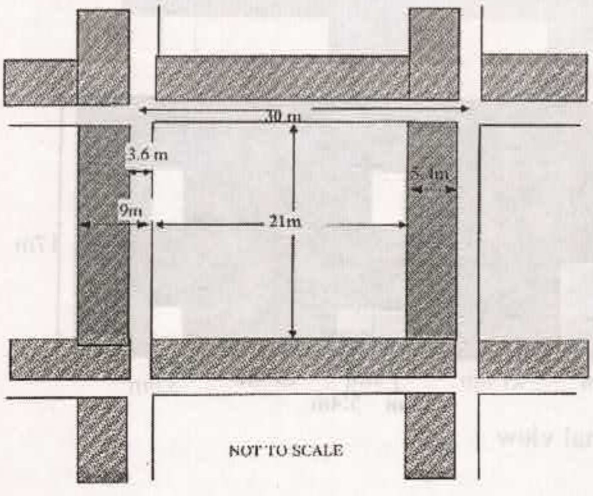

Fig. 8 : Configuration of "L" shape extraction for the wide stall formations

ACKNOWLEDGEMENTS

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STEEL FIBRE REINFORCED SHOTCRETE DESIGN FOR ROADWAYS IN COAL MINE

U.K. Singh*

सामान्यत: रॉक सपोर्ट का तात्पर्य चट्टान में भार प्रतिरोधक क्षमता में वृद्धि को समाविष्ट करना है। थिन शॉटक्रीट लाइनिंग किसी भी चट्टान के विस्थापन को रोकने का कार्य निम्नलिखित तरीके से करता है – (क) चट्टान के समूहों के खुले जोड़ों एवं भ्रंशों को भरकर उसे कठोरता एवं मजबूती प्रदान करना, (ख) आँसजन (चिपकाव) एवं अपरूपण (दबाव) के माध्यम से चट्टान के भार को निकटवर्ती सुदृढ़ चट्टान पर स्थानान्तरित करके, (ग) जब शॉटक्रीट चट्टान बाण्ड निम्न तथा शाटक्रीट संस्तर लगातार हो उस समय चट्टान के झुकाव और तनन हेतु झिल्ली का कार्य करके।

रॉक ब्लाक के आलम्बन हेतु शॉटक्रीट रॉक संस्तर की मोटाई रॉक सतह तथा शॉटक्रीट संस्तर के मध्य जोड़ने की शक्ति पर निर्भर करती है। बहुत सारे समदूरस्थ बिन्दुओं पर आलम्बित एक समान भारित प्लेट के अनुरूप बोल्ट द्वारा धरातल पर सुदृढ़ स्टील फाइबर प्रबलित शॉटक्रीट (SFRC) सतह की वहनीय क्षमता पर विचार-विमर्श किया गया है। कोयला खान में सपोर्ट किए जाने वाले रॉक लोड को सी. एम. आर. आई. भू-यांत्रिकी वर्गीकरण आर. एम. आर. का इस्तेमाल करते हुए परिकलित किया गया है तथा चट्टान के भार को वहन करने हेतु एम. एफ. आई. एम. बोल्ट सपोर्ट प्रणाली को डिजाइन किया गया है। आलेख में टाटा स्टील की कोलियरी में कमजोर दीवार एवं क्षतिग्रस्त चाल के मामलों के अध्ययन का वर्णन किया गया है। यह देखा गया है कि अन्य लाभों के साथ एस. एफ. आर. एस. बोल्ट सपोर्ट परम्परागत स्टील क्रास बार एवं प्राप सपोर्ट की तुलना में रु. 9500/- प्रति मीटर सस्ता है।

INTRODUCTION

The strata control in underground mining plays a very significant role. The efficiency, cost effectiveness and safety of underground mining are largely dependent on support system. The support requirement of underground roadways is dependent on following factors:

a) freshly exposed roof or long standing old roadways where bed separation has already taken place.
b) nearness to already worked out working.
c) depth from surface,
d) geo-technical properties of rock,
e) the period for which the roadway has to remain in use, and
f) method of mining.

The causes of damage to excavations in rock are as follows:

a) The separation of pieces of rock, bounded by geological discontinuities and fractures induced by excavation, from the rest of the rock around the excavation. This is named a loosening ground condition.

b) General shear failure in the rock mass caused by overloading from the existing stress field. Shear failure of the type bearing failure in weak side walls and roof failure due to incomplete arch formation come under this category.

c) Problems of instability due to high stresses. The intensity may very from splitting, spalling, bending and buckling of slabs. Under favourable conditions, explosive failure i.e. rock burst occurs extending far into the rock.

The integrity of rock around an excavation is readily destroyed by any movement in excess of those resulting from its behaviour as a continuum elastic material. If a piece of rock becomes loosened it is no longer capable of contributing to its own support. Opening, separation, and rotation destroy the strength due to interlocking inherent in even the most extensively jointed rock. The stability and safety of an excavation in rock are determined in extensively jointed rock. The stability and safety of an excavation in rock are determined in

i. by the extent to which this disruptive displacement are prevented, and

ii. by the extent to which they can be controlled.

Thus every effort must be made in the design of an excavation to retain the competence of the surrounding rock, and the support must be used to provide additional competence as is required. The term rock support usually implies to increasing the capacity of the rock to resist loads.

The shotcrete is a surface support. The bearing capacity of surface support and its deformation resistance.

* Professor, Dept. of Mining Engineering, Indian School of Mines, Dhanbad-826 004.

toughness have a decisive factor on ability of the surface support to interact with the rock and to stabilize the surface.

Vandewalle (1996) summarizes the principle of tunnel lining using steel fibre reinforced shotcrete as follows:

i. The most important part of the tunnel lining is the ground that surrounds it, and the most important component of the ground is ground water.

ii. The most important element of lining construction is to secure full, continuous contact between the lining and the ground movement, not to carry ground loads.

iii. The most efficient tunnel lining is one that mobilizes the strength of the ground permitting controlled ground deformation.

iv. For multistage linings, the initial construction support is very flexible compared to ground, and can absorb large flexural deformations associated with redistribution of ground stresses.

v. Selection of the type of lining depends on excavation methods that are suited to ground characteristics, of which stand-up time is usually most significant.

vi. The design of tunnel linings must account for large uncertainties and variations in ground conditions, construction procedures and ground behaviour during construction.

vii. These inherent uncertainties are incompatible with the apparent precision of mathematical analysis of stresses. Nonetheless, reliable guidance for economical lining design may be derived from experience with construction of tunnels in similar ground.

BEARING CAPACITY AND DEFORMATION RESISTANCE OF SURFACE SUPPORT

The surface support in the form of strong arches of concrete, shotcrete or steel may be sufficient in itself to stabilise the roadway. In other cases, it is possible to use surface supports of less strength, but it must then interact with the bolt support to achieve the necessary stability of the entire roadway.

The role of a thin layer of shotcrete as surface support in this connection is to give stability to the area between the bolts so that the bolts function in the intended way (Fig.1). If large deformations arise in the surface support, this may lead to a situation where the bolts cannot take up the load from the rock in the intended way. The load will then be transferred to the surface support which is not designed to carry this load and the tunnel will collapse (Fig. 1a). In other load situations and for other failure mechanisms (Fig. 1b), the bolt support will function in spite of the fact that large deformations occur and the rock breaks up. Large deformation in surface support are not disadvantage here (Stille, 1992).

Bearing capacity of the shotcrete against fall of a block

Thin shotcrete lining acts to prevent rock displacement by (Mahar, 1975)

i. stiffening and strengthening the rock mass by filling open joints and fractures,

ii. transferring the rock load to adjacent stable rock through adhesion or shear, and

iii. acting as a membrane in bending and tension when shotcrete rock bond is low and the shotcrete layer is continuous.

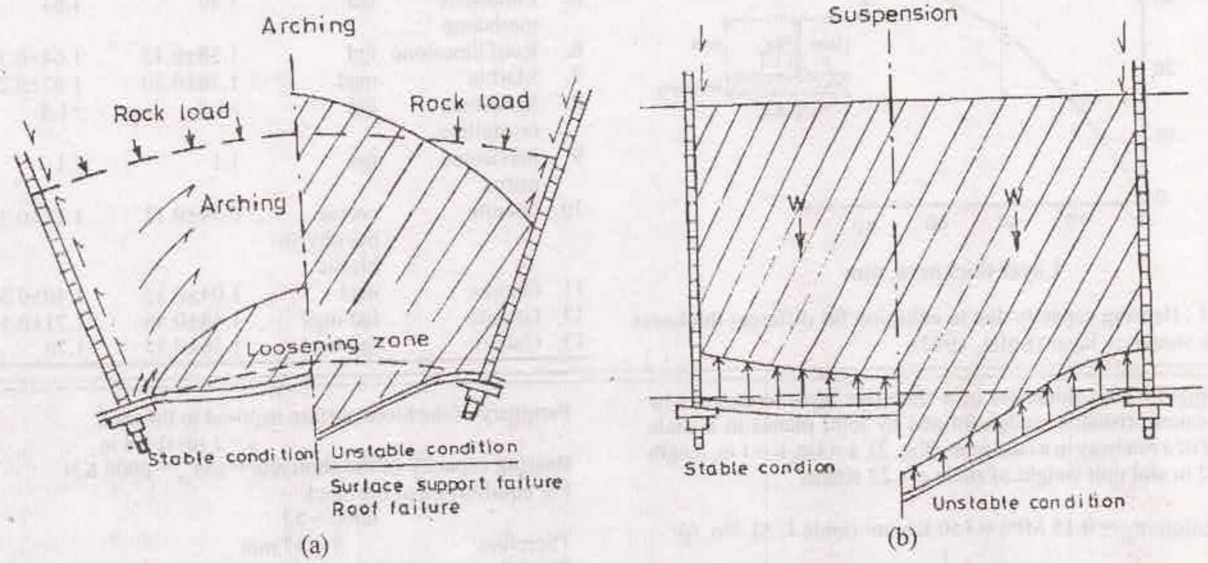

Fig. 1 : Toughness requirements of surface support for (a) arching and (b) suspension (Stille, 1992).

Shotcrete in thin lining having poor shotcrete bond does not have the capacity to carry the full gravity load of large wedges but may provide enough support to reduce displacement and thus help maintain the self supporting capability of rock. In this case shotcrete layer must be reinforced.

Cecil (in Mahar, 1975) proposed an analytical method to compute shotcrete thickness required to support the full gravity weight of a block in flat roof with good shotcrete-rock bond. In his approach, weight of a freely falling block from the roof is restrained by a shotcrete layer applied on the roof. It is assumed that resisting force by the shotcrete is adhesion of shotcrete over a small width & (equal to thickness of the shotcrete) along periphery of the rock block (Fig.2).

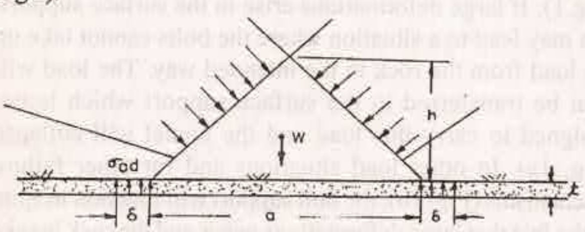

Fig. 2 : Support of a free falling rock block by shotcrete due to adhesion.

On the basis of laboratory scale experiments, Stille (1992) has found that δ is less than the shotcrete layer thickness t, and the relation between t and δ is shown in Fig. 3. Bond (adhesion) strength σ_{ad} between coal measure rocks and steel fibre reinforced concrete layer, tested in ISM laboratory, is given in Table 1, and the bond strength of shotcrete layer on hard rock, tested by Hahn and Holmgren (1979) is given in Table 2.

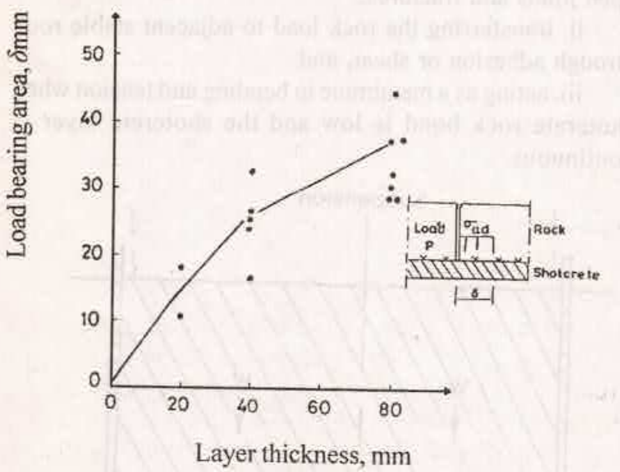

Fig. 3 : Bearing capacity due to adhesion for different thickness of the shotcrete layer (Stille, 1992).

Table 1 : Coal Measure rock-concrete bond strength

Sl. No.	Rock Type	Surface description	Bond failure description	Bond strength (MPa)
1.	Coal	rough	Bond failure; some coal pieces (Vitrain) sticking on the concrete surface	.013
2.	Coal	rough	Bond failure; very few coal pieces sticking on the surface	.013
3.	Coal	old & dusty	Bond failure	0.04
4.	Coal	smooth	Bond failure; some coal pieces sticking on the surface	0.10
5.	Shale	smooth	failure through film of shale, a thin filmy layer of shale was sticking on the surface	0.05
6.	Shale	smooth joint plane normal to bedding	Bond failure	0.15
7.	Sand-stone	rough; coarse grain, poor cementing material	Failure through the sandstone	0.15
8.	Sand-stone	rough; grained strong & hard sand-stone	Bond failure	0.24
9.	Sand-stone	rough; carbo-naceous hard sand-stone	Failure partly through bond & partly through the layer of carbonaceous sandstone	0.22

Table 2 : Hard rock-shotcrete bond strength (Hahn & Holmgren, 1979)

Sl.	Rock Type	Grain size	Bond Strength Smooth surface	Bond Strength Rough surface
1.	Shale	very fgd	0.24±0.18	0.28±0.11
2.	Mica Schist	mgd	0.58±0.19	0.85±0.35
3.	Gneiss ⊥ (001)	mgd	0.19±0.05	0.51±0.11
4.	Gneiss ∥ (001)	mgd	1.53±0.28	1.8
5.	Limestone marlstone	fgd	1.49	1.84
6.	Roof limestone	fgd	1.58±0.12	1.64±0.30
7.	Marble	mgd	1.38±0.30	1.52±0.28
8.	Sandstone crystalline	fgd	>1.8	>1.8
9.	Sandstone porus	fgd	1.1	1.1
10.	Granite	coarse porphyro-blastic	0.34±0.12	1.12±0.20
11.	Granite	mgd	1.04±0.32	1.40±0.26
12.	Granite	fgd-mgd	1.48±0.46	1.71±0.14
13.	Gabbro	fgd-mgd	1.56±0.25	1.70

<u>Problem</u> : Find thickness of a shotcrete layer to support by adhesion prismatic wedge formed by joint planes in a shale roof of a roadway in a coal mine (Fig. 2). a = 1m, h = 1 m, length l = 2 m and unit weight of shale ρ = 22 KN/m³.

<u>Solution</u> : σ_{ad} = 0.15 MPa = 150 KN/m² (table 1, Sl. No. 6)

Weight of the rock block = $\frac{1}{2}$ (ah)l ρ = 22 KN

Periphery of the block surface exposed to the roof,

s = 2 (a+l) = 4 m

Bearing capacity of the shotcrete = $s\delta\sigma_{ad}$ = 600δ KN

For equilibrium of the block

600δ = 22

Therefore δ = 37 mm

Thus, the required shotcrete layer thickness corresponding to δ = 37 mm is 60 mm read from Fig. 3.

The thickness of the shotcrete layer required to support a rock block is very sensitive to the bond strength. For example, the required shotcrete thickness for supporting the same size of block in hard sandstone where σ_{ad} = 0.22 MPa (Table 1, Sl. No. 9), is only 30 mm.

Further, the stability of roof and side walls in a jointed rock depends on stability of a key block. Its failure destroys the natural arch. A thin layer of shotcrete having good bond strength stabilises the key block which results in a stable natural arch. The shotcrete layer applied on the jointed rock surface also mobilises the inherent strength (friction) of the rock by binding and keeping them together (Fig. 4).

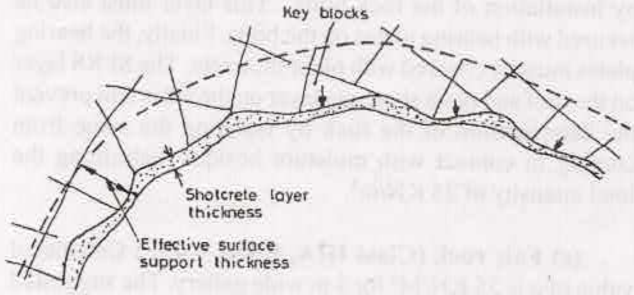

Fig. 4 : Support of "key blocks" by shotcrete and effective shotcrete layer thickness

Bearing capacity of the shotcrete layer supported by bolts

The bearing capacity of a shotcrete layer secured to the ground by bolts can be treated as a uniformly loaded plate supported at many equidistant points. (Fig. 5a). In this case, we obtain a good approximation to the maximum stress and to the stress distribution near a support as follow (Timoshenko, 1956): A part of the plate near the support, bounded by a circle of radius a=0.22c where c is the distance between the supports i.e. bolts, is considered as a circular plate simply supported at the outer edge loaded at the inner edge by a load P=qc² acting upward, and uniformly loaded by a load of intensity q acting downward. This loading is shown in Fig. 5b.

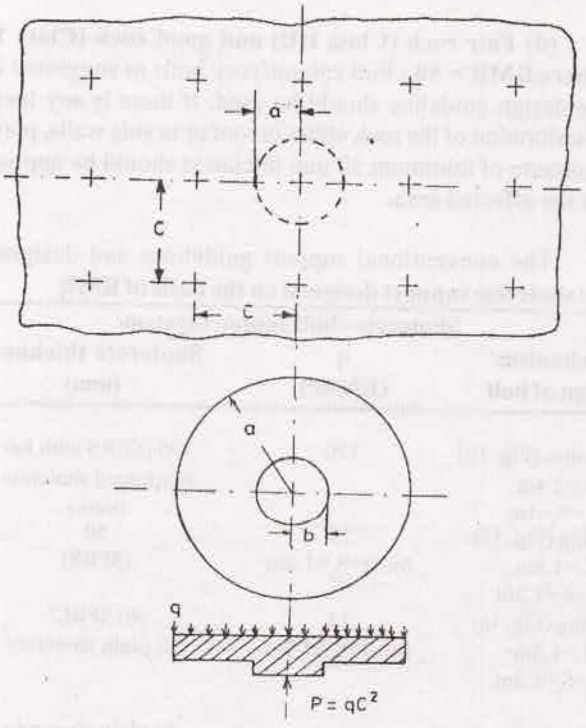

Fig. 5 : A model of shotcrete layer supported at many equidistant points by rock bolts.

In this case the maximum stress in the plate can be represented by formula

$$\sigma_{max} = k\,(qa^2/t^2) \qquad (1)$$

and the maximum deflection is given by

$$W_{max} = k_1\,(qa^4/Et^3) \qquad (2)$$

where t = shotcrete layer thickness, the coefficients k and k_1 are given in Table 3 for several values of the ratio a/b and for Poisson's Ratio ν = 0.3.

Table 3 : Coefficients k and k_1

a/b	k	k_1
1.25	0.135	0.00231
1.5	0.410	0.0183
2	1.04	0.0938
3	2.15	0.293
4	2.99	0.448
5	3.69	0.564
6	4.34	0.646
7	5.09	0.699

DESIGN OF SHOTCRETE BOLT SUPPORT SYSTEM FOR THE ROADWAYS

"CMRI Geomechanics Classification" (CMRI Report, 1987, Venkateswarlu and Raju, 1987) of coal measure rock is widely accepted and used in Indian collieries for design of roof support in galleries. Since it follows the lines of Bieniawaski's Geomechanics Classification, the term Rock Mass Rating (RMR) has been also used in this classification. On the basis of RMR, the rock mass is divided into 5 classes. The rock load intensity q is given as $q = k_1 B \rho$, KN/m².

where k_1 = (1.7 - 0.037 * RMR + 0.0002 * RMR * RMR),
B = roadways width or roof span, m, and
ρ = unit weight of the rock, KN/m³

k_1B is also called rock load height. It means that a support must bear the rock load of height k_1B m. The designed guidelines for permanent galleries based on this classification is given by Venkateswarlu and Raju, 1887. A critical analysis of the guidelines, and design of a suitable Steel Fibre Reinforced Shotcrete (SFRS). Bolt system for the same rock condition has been performed using the data: B = 5 m, p = 25 KN/m^3, the residual flexural strength after initial crack in 40 kg/m^3 Steel Fibre Reinforced Shotcrete (SFRS) = 3000 KN/m^2 and bearing plate diameter = 0.10 m. This is given as follows:

a) Very poor rock (Class V, RMR=10) : It is highly jointed, fractured and loose which can not be hold by rock bolts. Therefore, yielding arches have been suggested to hold back the rock in place and yield under increase of rock pressure if any.

Bolt-Shotcrete support design : Here the bolt works as shown in Fig. 1b where the surface support of the shotcrete has to bear the weight of the very poor loose rock over the gallery width. For RMR = 10 and B = 5m the calculated value of q is 170 KN/m^2 (Eq.3). In order to provide sufficient lateral restrain to the shotcrete layer, the bolts must be longer in the weak rock. The minimum length should be 2 m, preferably 2.4 m (Laubscher, 1984). Thus full column grouted bolts of length L = 2.4 m should be installed at S_b = S_r = 1 m where S_b is bolt spacing along the gallery width and S_r is the same along the length. The SFRS layer thickness can be calculated using Eq.1. It is 95.1 = 100 mm for q = 170 KN/m^2 The total thickness of the shotcrete should be applied in 2 layers. The first layer should be applied in each round followed by installation of the rock bolts. The final SFRS layer should be applied when some movement has occurred in the gallery. The time of installation shoould be decided on the basis of closure monitoring of the gallery. If necessary, 2-3 nos. of reinforcing steel bars of dia. 25 mm may be attached to the bolts across the gallery and they should be covered with an additional 50 mm thick plain shotcrete, forming a continuous bar reinforced arch in the gallery at 1 m interval.

(b) Poor rock (Class IV, RMR = 30) : Calculated value of q is 100 KN/m2 for 5 m wide gallery. The suggested support system is full column grouted bolt with wire-netting, W-strap and props. The bolts of length L=1.8 m are to be installed at sb = Sr = 1 m. Here, wire-netting will not be able to provide immediate restrain to the roof and also will not be able to prevent deterioration of the shale roof due to the moisture in air.

Bolt-Shotcrete support design : Here, the bolt works as shown in Fig. 1a where the surface support has to restrain weight of the rock under the stable arch formed between the bolts. The rock bolts of length L = 1.8 m should be installed at S_b = S_r = 1.2 m. Therefore, B has to be taken equal to S_b = S_r = 1.2 m. The calculated value of q to be restrained by the surface support is 25 KN/m^2 (Eq.3). The calculated SFRS layer thickness is 50 mm. This layer should be applied immediately in each round of advance followed by installation of the rock bolts. This layer must also be secured with bearing plates on the bolts. Finally, the bearing plates must be covered with plain shotcrete. The SFRS layer on the roof and plain shotcrete layer on the sides will prevent the deterioration of the rock by isolating the same from coming in contact with moisture besides restraining the load intensity of 25 KN/m^2.

(c) Fair rock (Class IIIA, RMR = 45) : Calculated value of q is 55 KN/M^2 for 5 m wide gallery. The suggested support design guideline is roof stitching supplemented with grouted bolts (L = 1.5 m, S_b = 1m, and S_r = 1.2 m) and wooden sleepers.

Bolt-Shotcrete support : The bolts works in the way similar to the one for poor rock described above. The rock bolts of L=1.8m are installed at S_b = S_r = 1.2 m. Therefore, calculated value of q is 15 KN/m^2 for the shotcrete layer. The SFRS layer thickness is 40 mm for restraining the q. However, a minimum 50 mm thickness of plain shotcrete should be applied instead of 40 mm SFRS.

(d) Fair rock (Class IIB) and good rock (Class 1) where RMR > 50 : Full column rock bolts as suggested in the design guideline should be used. If there is any local deterioration of the rock either in roof or in side walls, plain shotcrete of minimum 50 mm thickness should be applied on the affected area.

The conventional support guidelines and designed

Table 4 : Comparison of the conventional support and the shotcrete support designed on the basis of RMR

Rock type Class/RMR	Rock load q for B=5m (KN/m^2)	Support guidelines	Shotcrete - bolt support system: Mechanism/ design of bolt	q (KN/m^2)	Shotcrete thickness (mm)
Very Poor V 10	170	Steel arch	Suspension (Fig. 1b) L=2.4m S_b=S_r=1m	170	100 (SFRS with bar reinforced shotcrete beam)
Poor IV 30	100	L=1.8m S_b=S_r=1m wire-net, 'W' strap & prop	Arching (Fig. 1a) L=1.8m S_b=S_r=1.2m	25 for B=S_b=1.2m	50 (SFRS)
Fair IIB 45	55	L=1.5m, S_b=1m, S_r=1.2m roof stitching supplemented with grouted bolt and wooden sleeper	Arching (Fig. 1a) L=1.8m S_b=S_r=1.2m	15 for B=S_b=1.2m	40 SFRC/ 50 plain shotcrete
Fair & Good RMR>50	-	Rock bolt where needed	Arching (Fig. 1a)	-	50 plain shotcrete where needed

S_b = Bolt spacing; S_r = Bolt row spacing; L = Bolt length

SFRS-bolt support system on the basis of RMR have been compared in Table 4. The performance of the SFRS support in rock having RMR<50 should be monitored by measuring deformation and closure of the roof and side walls. If the shotcrete layer gets fractured, then the ground should be immediately stabilised by grouted rock bolts or cables preferably pre-tensioned followed by an additional layer of the SFRS.

The designed SFRS - Rock bolt support system works more than anticipated because the shotcrete binds the jointed and fractured rock pieces together and with the shotcrete layer. Thus the effective shotcrete thickness increases and it becomes able to take higher rock load than anticipated (Fig. 4). This, in turn, increases the ground stability. Further, it is important to note that a continuous SFR or plain shotcrete maintain-integrity of the fractured rock which mobilises the inherent frictional strength of the rock.

SFRS SUPPORT OF A ROADWAY IN DIGWADIH COLLIERY

Digwadih colliery is one of the five collieries of TATA Steel in Jharia Coal Field. It is situated about 14 km from Dhanbad on Dhanbad-Sindri road. The method of working is predominantly based on Board and Pillar system with loading of coal manually as well as by Side Discharge Loaders and Road Headers in tubs and on to chain conveyors. Underground access to the coal seam is through vertical shafts. The support system used in the galleries, wherever required, mostly is conventional ones consisting of timber, steel and rock bolts. In good ground, the developed galleries normally span 4.2 m without any support.

The Endless Haulage roadway of XIV seam in the colliery was selected for the SFRS trial support. This roadway is disturbed from 7th to 13th Dip. These are worked out and stowed panels on both dip and rise sides of the roadway. The panels were worked out in the year 1993-95. The shotcrete trial was conducted between 9th and 11th Dip. The site specifications are :

Depth from the surface	= 225 m,
Excavated roadways width	= 3.6 m,
Roadways height	= 1.8 m,
Roof rock RMR	= 35,
Side wall	= coal with well developed cleats.

Ground condition

Presence of the goaves on both sides of the roadway has induced compressive stress in the side walls. This has resulted in extensive fracture and spalling from the side walls in the area. The induced stresses have caused further separation in the layers of shale in the roof. The slip planes and other geological discontinuities have resulted in severe damage of the roof. The shale is also prone to deterioration due to moisture and air, and this has further aggravated the problem. The endless haulage roadway's width has increased to 4 m due to the side spalling. The roadway's height has also increased to 4.2m at some places due to caving in the roof.

The conventional support system and problem

The roadway is supported through out the length by steel crossbars over goal-posts (Fig. 6 and 7). Three props are used between which two haulage tracks are laid. Even with this support system, a number of times, there were severe roof falls which were cleaned and the area was re-supported. At one of the sites selected for the shotcreting, a large cavity in the roof was formed due to the roof fall. The fall was recovered and area was re-supported by wooden chocks over mild steel sheets which were laid over the steel crossbars (Basu, et.al. 1999). This conventional support was giving too many problems. It renders obstruction in operation of the mine haulage and the mine return air current. Further, frequent rehabilitation of the conventional support system has proved to be very expensive and disruptive. There was also no alternative support to prevent the side spalling.

The shotcrete support design

The ground belongs to the Class Poor (RMR = 35). The SFRS-bolt support system was designed following

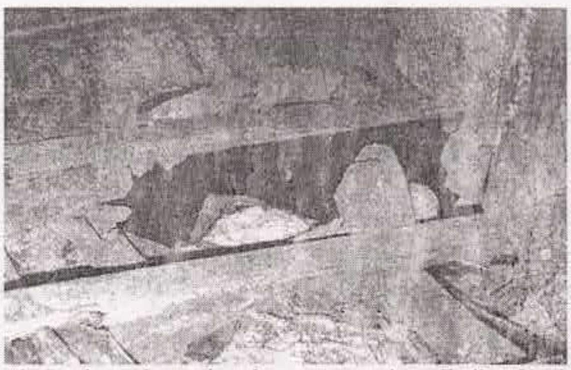

Fig. 6 : Steel crossbar and goal-post support in an Endless Haulage roadway in Digwadih Colliery (courtesy TISCO, Jamadoba, Basu, 1999).

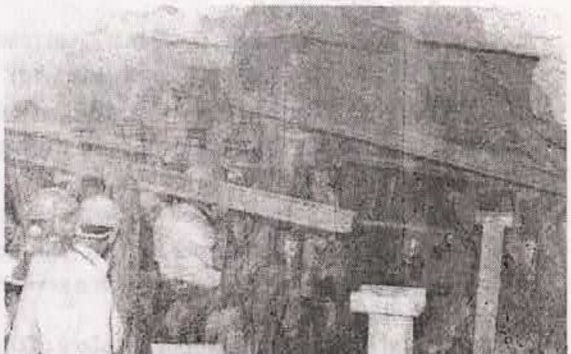

Fig. 7 : The same area as shown in Fig. 6 after SFRS-bolt support installation (courtesy TISCO, Jamadoba, Basu, 1999).

the approach described in Section 3. The design is given as follows:

Roof : SFRS; 75 mm thick; 60 kg/m^3 steel fibre in the mix.
:Bolt; bolt length, L = 1.8 m, bolt spacing, S_b =Sr = 2.0 m; bearing plate diameter = 150 mm (100 mm was considered for the design),
Side wall: Plain shotcrete; 100 mm thick.

The base mix proportion of the SFRS was 2.6:1.9:1 (Coarse chips(-10 mm): Damodar river sand: slag cement) by weight. 150 mm cube compressive strength of the base mix was 25 MPa. The 60 kg/m^3 steel fibre in the mix results in 40 kg/m^3 steel fibre in the shotcrete layer after loss of the fibre in rebound. The flexural strength of the shotcrete layer was 5.5 MPa and the residual flexural strength of the 75mm thick 40 kg/m^3 SFRS layer remained 3 MPa up to 2-3 mm wide crack in the shotcrete layer (Singh and Mishra, 1998 & 1999). Using Eq.3, the calculated rock load q was 32 KN for the RMR=35 & B=S_b=2m, and using Eq. 1, the computed bearing capacity and the residual bearing capacity of the designed shotcrete-bolt system were 32 KN and 17 KN respectively.

Method of shotcreting

The coal on both the side walls was highly fractured, loose and spalling due to the induced stresses and well developed cleats in the coal. No effort was made to remove the immediate fractured coal from the walls, as it would have been resulted in further widening of the roadway. The surface was thoroughly cleaned off the dust by a stream of water and compressed air just before beginning the shotcreting. Thus the shotcrete was sprayed on the already fractured coal in the sides walls. The initial fine sand, cement and Condensed Silica Fume grout penetrated into the fractures. The subsequent shotcrete layers were deposited on this thin sticky layer. The net result was a stable side wall. Further, the side and roof convergence reference points were also installed in the area before the shotcreting.

At first, both the side walls were supported by the plain shotcrete. 100mm thick layer was deposited in a single pass. For shotcreting in the roof, a few lagging between the roof and the cross bars were carefully removed. This resulted into fall of already loose rock pieces from the roof. No effort was made to dress down the roof. While removing the lagging, it was ensured that the roof did not become unsupported. For doing the shotcreting, the roof surface was cleaned by a stream of water and compressed air. 75mm thick SFRS layer was deposited in two passes of 30 and 40mm. The rock bolts in the roof were installed in the shotcreted roof and the bearing plated were covered with plain shotcrete. After supporting a substantial area by 75mm SFRS and the rock bolts, the remaining laggings were removed and their shadows on the roof were cleaned off from the loose shotcrete and were covered with fresh SFRS. The same area of the roadway is shown in Fig.4 after the SFRS-bolt support installation. The steel girder and props seam in this figure were later removed . The roof and side convergence were regularly monitored and the monitoring has shown no movement at all after replacement of the conventional support by the SFRS one.

Table 5 : A Cost Comparison of different types of support in coal mines roadways

Sl No.	Support	Cost (Rs. / m^2)
1.	**Bolt** : 22 mm φ, 1.5 m long , 1.2 x 1.2 c/c grouting with 3 cement capsules	115.00
2.	**'W' Strap** : 2.6 mm x 45 cm x 3 m at 1.2 m interval in a roadway of 4.2 m width	70.00
3.	**3 Piece Girder Support** : Roadway size 3.6 x 2.4 m, Support interval = 1 m (no side support)	1320.00
4.	**Steel arch** : at 0.9 m interval for 16 m^2 X - section roadway (Barve, 1995)	2407.00
5.	**Girder support** : (Barve, 1995) Roadway size = 4.2 x 4.2 m • Girder (200 x 100 mm) fixed in side wall with concrete at 0.7 m. • channels (100 x 50 mm) placed over the girder. • no side support	2390.00
6.	**Cost of shotcrete support** : in 4.2 m x 2.4 m roadway in NMC - 3 (WCL) Mine (Barve, 1995) • 50 mm plain shotcrete in side wall • 50 mm Netlon (CE - 153, 45 mm x 45 mm) reinforced shotcrete support in roof	388.00
7.	**SFRS support cost for a week roof and stressed side wall** (4.2 m x 2.4 m roadway) (Singh & Mishra, 1998)	
	• **SFRS support cost per m^2** (bar reinforced SFRS beam at 1.5 m interval)	896.00
	• **SFRS support cost per m^2** (bar reinforced SFRS beam at 0.9 m interval)	1016.00
8.	**Steel Cross-bars over 3 steel props in a severely damaged endless haulage** (4 x 2 m). (Basu, 1999)	15500.00/m length
	The cost of the shotcrete support in place of the above conventional one	
	• 100 mm plain shotcrete on side wall	6000.00/m length
	• 75 mm 60 kg/m^2 SFRS in roof with 1.8 m rock bolts at 2.0 m centre to centre distance	600.00/m^2 (approx.)

Cost benefit

A total 18 m length of the endless haulage roadway was shotcreted as per the design. The shotcreting cost was Rs. 4500/m^3 inclusive of material and man power. The cost of shotcrete bolt support system was Rs. 6000/m roadway length where as it was Rs. 15500/m for the conventional steel support. A comparison of the cost of different types of support in coal mine roadways (Table 5) reveals that the shotcrete is costlier than the rock bolt and 'W' strap ones. However, it is definitely cost effective over the conventional steel girders and arches for poor and very poor strata.

CONCLUSION

* The shotcrete support has been related to the CMRI Geomechanics Classification System (RMR).

* A SFRS support design methodology has been developed for roadways in coal major rocks.

* The role of a bearing plate with bolt is very important in increasing the restraining capacity of a shotcrete layer.

* The SRFS-bolt support for the poor ground in Digwadih Colliery was cost effective over the conventional steel support. There is a saving of Rs. 9500/m of the roadway length over the conventional one.

* In general, It is cost effective for the poor and very poor ground when we consider long term stability, and time and cost for repair of the conventional supports.

ACKNOWLEDGEMENT

The management of the TATA Steel, Jharia Division is duly acknowledged for giving permission for using data and photographs of the Digwadih Colliery in this paper. Prof. V.P. Singh and Prof. R.C. Mishra are also duly acknowledged for the helping in shotcreting work at the colliery, Manish Nath, final year B.Tech, Mining Engineering student has helped in formulating the bearing capacity of the shotcrete layer.

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FLOTATION COLUMN - A NOVEL TECHNIQUE FOR FINE COAL BENEFICIATION

S. K. Biswal*, P. S. R. Reddy*, S. Prakash*, S. K. Bhaumik* and S. R. S. Sastri*

इस्पात निर्माण हेतु कम राख वाले कोयले की प्राप्ति के लिए प्रति वर्ष लगभग 30 मिलियन टन कोयले की धुलाई की जा रही है। धुलाई की क्रम में 15-20% कोयला चूर्ण हो जाता है जिसमें राख की मात्रा लगभग 35-40% रहती है। यह इस्पात निर्माण में सीधे प्रयोग नहीं होता है। चूर्ण रूप के कारण इस कोयले के परिष्करण के लिए प्लावन (फ्लोटेशन) तकनीक एक स्थापित तकनीक है। आज के परिदृश्य में खनिज संसाधन उद्योग में विशिष्ट लाभ के लिए पारंपरिक सेल विधि के स्थान पर नवीनतम फ्लोटेशन कॉलम तकनीक का प्रयोग किया जा रहा है। सुदामडीह, वेस्ट बोकारो और भेलाटाँड से प्राप्त कोयला चूर्ण पर फ्लोटेशन कॉलम का विस्तृत अध्ययन किया गया है। प्रयोगशाला में प्राप्त परिणामों के आधार पर इस प्रक्रिया के कार्यनिष्पादन का मूल्यांकन करने हेतु सी.एम.पी.डी.आई., राँची तथा टिस्को, जमशेदपुर के सहयोग से सुदामडीह कोल वाशरी तथा वेस्ट बोकारो कोल वाशरी II में एक मीटर आकार के एक प्रायोगिक फ्लोटेशन कालम संयंत्र की स्थापना की गई। इस प्रायोगिक संयत्र के अनुभव के आधार पर उपयुक्त वाश वाटर डिस्ट्रीब्यूटर तथा स्पार्जर प्रणाली विकसित की गई है जिससे कि कॉलम की दक्षता न सिर्फ कोटि (ग्रेड) एवं प्राप्ति बल्कि सुविधाजनक यांत्रिक रखरखाव के लिए भी बढ़ाई जा सके।

INTRODUCTION

Flotation column differs radically from conventional mechanical flotation cell both in design and operating philosophy. The basic principle of flotation column is the counter current flow of air bubbles and solid particles and behaves as a plug flow reactor in contrast to conventional cells which more or less resembles a continuous stirred tank reactor. The air is injected at the bottom of the column to form the bubbles and allow to raise them through a downward flowing slurry. It has been fairly well established, that column has certain distinct advantages over the conventional mechanical cells such as simple construction and easy to fabricate locally without any moving parts, less capital cost as material requirement is low, less power consumption, low operating and maintenance cost, ease to control with better computer control facilities, reduction in number of stages of operation for required grade and recovery, requirement of relatively small floor area, ability to handle fine particles and better performance with respect to grade and recovery. During the last three decades, there have been significant developments relating to design, operation, scale up, simulation and optimization and application of technology to extract maximum advantages of this machine in the mineral industry[1-3].

A major difference of flotation column is the cleaning zone in comparison with mechanical cell. Wash water is added in the cleaning zone to eliminate the gangue minerals from the concentrate so that the grade of the concentrate increases.

A specific to coal fines a lot of R&D work have been carried out [4-31]. Based on these results, number of commercial plants have come up in different countries for beneficiation of coal fines. The high lights of some of the commercial plants are given below.

A coal preparation plant in Pennsylvania is operating a 2.4 m diameter and 5.2 m height flotation column for the recovery of—28 mesh coal. It produces a concentrate with 6.3% ash from a feed containing 26% ash at 85% yield [3].

The 1.7 m diameter, 5.5 high column was installed at Riverside and commissioned in September—October 1986. Through the use of wash water, a single stage of column flotation can produce 10% ash concentrates from 40-45% ash ultra fine feed at 70-80% recovery of combustibles [25].

The Middle Fork Preparation Facility located in Russel County near Carbo, Virginia, USA has installed five 3 m diameter microcel flotation columns to process approximately 100 tph of minus 150 microns raw coal. It is able to produce the ash content 8% from 45-55% feed coal. Comparison with conventional cell, it can reduce 7% ash further by increasing 27% combustible recovery [26].

Four flotation columns of 2-4 m diameter and 6.7 m height each were installed at Mayflower preparation plant of Powell Mountain coal company at Virginia. These flotation columns are processing about 18 tph of fine coal refuses from 500 tph coal cleaning plant and producing about 5.4 to 7.2 tph clean coal at 6% ash [19].

*Regional Research Laboratory, Bhubaneswar -751013

10 and 22 cm diameter and 250 cm height glass flotation columns were used in the laboratory experiments for coking coal fines. Based on laboratory results, 1 metre diameter flotation column was designed and installed at Sudamudih coal washery, BCCL, Dhanbad and West Bokaro coal washery-II, TISCO, West Bokaro at the capacity of 2.5 tonnes of feed per hour [14, 29]. In West Bokaro, the column was used to recover the carbon values from plant tailings. In this paper the results of both laboratory as well as pilot scale experiments have been discussed.

MATERIALS AND METHODS

Raw Materials

The coking coal fines from Sudamudih and Bheltand coal mines and plant tailings from West Bokaro coal washery were used for the experiments [14, 28-29]. The ash and size distribution of each samples are given in Table 1.

Reagents

Commercial grade light diesel oil was used as collector whereas MIBC supplied by National Organic Chemical Ltd., Bombay, India as frother.

Flotation study at laboratory

Batch flotation tests were carried out using Denver D- 12 sub-aeration flotation machine with 10 litre capacity of cell. About 800 gm of sample was taken in each experiment. The sample was made 40% solid concentration with water and conditioned with required amount of diesel oil as collector at 1000 rpm. Then the solids concentration was brought down to 10% by adding additional water in the cell. Then the required amount of frother was added and conditioned for two mm. Then the flotation was carried out using the induced air. The experiments were carried out at different operating parameters. The flotation concentration and tailings were collected separately in each experiment and both were dried, weighed and analysed for ash according to ASTM standard using Fisher coal analyser model 490.

Continuous flotation studies were carried out employing 100 mm and 220 mm diameter of flotation columns. The schematic diagram of the whole set up is shown in Fig.1. The total height of the column is 2.5 m, which consists of 1.5 m flotation zone and 1.0 m of cleaning zone. The diffuser, which is an inverted cone over which a filter cloth is fixed. The feed was introduced at 1.5 m above the diffuser level. The air is passed through diffuser to generate the bubbles from surface of the filter cloth. The tailings were removed by a siphon arrangement, which maintained the interface level in the column. The ratio of diameter of diffuser and column is 0.81 13]. Initially 20 kg of coal sample was conditioned with required dosage of commercial diesel oil as collector. The additional water was added to bring down the solid concentration from 40% to 20% in the 200 litre capacity of the conditioner. From beginning of the experiment, compressed air is passed to the diffuser through rotameter before adding water in the column to prevent the entering of the water inside the diffuser. Superficial air velocity was maintained within range of 0.6-1.5 cm/sec. To run the equipment, first tap water was pumped by slurry sand pump with frother from dosing pump to generate the bubbles inside the column. Simultaneously

Table 1 : Size and Ash Distribution of Samples

Size,	Sudamudih coal		Bheltand coal		West Bokaro tailings	
Microns	Wt, %	Ash, %	Wt, %	Ash, %	Wt, %	Ash, %
+ 1000			2.4	25.4		
-1000 +500	14.5	24.5	12.1	27.2	32.2	34.2
-500 +420	13.5	27.7				
-500 +300			18.9	26.5		
-500 +210					34.4	31.2
-420 +210	29.9	29.1				
-300 +150			20.9	18.3		
-210 + 125	7.5	30.0				
-210 + 75					12.0	29.6
-150 + 75			12.4	13.8		
-125 + 75	8.6	34.2				
- 75 + 45	1.1	67.6	5.0	17.6	6.2	37.3
-45	24.9	36.6	28.3	18.0	15.2	46.2
Total	100.0	31.0	100.0	19.8	100.0	34.6

Table 2 : Results of Batch Flotation Tests of Sudamdih Coal

Sl. No.	Weight, %		Ash, %			Combustible
	Product	Tails	Product	Tails	Feed	Recovery, %
1	56.8	43.2	20.93	69.71	41.86	77.2
2	61.1	38.9	21.85	73.69	42.02	82.3
3	60.4	39.6	20.00	74.46	41.57	82.7
4	72.0	28.0	18.31	67.37	32.05	86.5

Table 3 : Results of Column Flotation Tests of Sudamdih Coal

Sl. No.	Weight, %		Ash, %			Combustible
	Product	Tails	Product	Tails	Feed	Recovery, %
1	48.0	52.0	11.5	50.8	31.9	62.4
2.	48.1	51.9	15.2	61.2	39.1	70.8
3.	56.3	43.7	12.5	61.9	34.1	74.8
4.	52.2	47.8	14.2	62.6	37.3	71.5
5.	17.1	82.9	7.1	42.5	36.3	25.0
6.	76.0	24.0	20.0	72.9	32.7	90.4
7.	61.3	38.7	21.2	79.8	43.8	86.0
8.	50.0	50.0	13.6	60.8	37.2	68.8
9,	59.6	40.4	17.3	69.4	38.3	79.9
10.	40.3	59.7	13.1	65.6	44.4	62.9
11.	85.3	14.7	11.2	70.5	30.3	86.3
12.	78.5	21.5	21.7	75.7	32.3	92.7
13.	62.2	37.8	17.1	63.8	34.7	79.4
14.	63.8	36.2	18.5	66.3	35.8	81.0
15.	29.6	70.4	10.1	48.4	37.1	42.3

Table 4 : Results of Batch Flotation Tests of Bhelatand Coal

Sl. No.	Weight, %		Ash, %			Combustible
	Product	Tails	Product	Tails	Feed	Recovery, %
1	10.4	89.6	6.1	23.54	21.70	12.5
2	36.5	63.5	7.0	31.90	22.80	44.0
3	68.2	31.8	9.5	50.10	22.40	68.2
4	76.2	23.8	10.5	59.40	22.10	87.0
5	85.0	15.0	12.6	68.20	20.30	94.5

Table 5 : Results of Column Flotation Tests of Bhelatand Coal

Sl. No.	Weight, %		Ash, %			Combustible
	Product	Tails	Product	Tails	Feed	Recovery, %
1	56.6	43.4	7.6	36.6	20.2	56.6
2	67.3	32.7	8.2	48.8	21.5	67.3
3	68.3	31.7	8.4	54.7	23.1	68.3
4	77.3	22.7	8.7	57.0	19.7	77.3
5	84.2	15.8	8.9	84.4	20.8	96.8

Table 6 : Results of Batch Flotation Tests of Recovery of Carbon Values from West Bokaro Washery Tailings

Sl. No.	Weight, %		Ash, %			Combustible
	Product	Tails	Product	Tails	Feed	Recovery, %
1	6.5	93.5	11.00	32.80	31.10	8.50
2	23.5	77.5	12.50	36.50	32.50	36.50
3	37.1	62.9	15.30	41.00	31.40	46.10
4	52.6	47.4	16.70	47.80	31.40	63.90
5	.68.7	31.3	20.40	59.30	32.50	81.00

Table 7 : Results of Column Flotation Tests of Recovery of Carbon Values from West Bokaro Washery Tailings

Sl. No.	Weight, %		Ash, %			Combustible
	Product	Tails	Product	Tails	Feed	Recovery, %
1	46.1	53.9	13.10	46.30	31.00	58.0
2	55.6	44.4	15.60	48.00	30.00	67.0
3	60.0	40.0	17.60	52.10	31.40	72.1
4	61.8	37.2	18.50	52.80	31.60	73.6
5	68.0	32.0	19.90	52.10	32.50	80.7

Table 8 : Results of Pilot Plant Column Flotation Tests of Recovery of Carbon Values from West Bokaro Washery Tailings

Sl. No.	Weight, %		Ash, %			Combustible
	Product	Tails	Product	Tails	Feed	Recovery, %
1	52.0	48.0	17.90	49.20	32.90	63.6
2	45.5	44.5	16.00	44.50	31.50	55.8
3	43.3	46.7	16.90	46.20	33.50	54.1
4	48.0	52.0	14.80	40.90	28.40	57.1
5	52.2	47.8	17.40	49.10	32.60	63.7
6	60.2	39.8	14.50	43.70	26.10	69.6
7	57.1	32.9	18.50	44.90	29.80	66.3
8	62.1	37.9	16.70	44.70	27.30	71.1
9	71.3	28.7	16.00	47.50	25.00	79.8

wash water was added through peristaltic variable speed pump to maintain the steady state in the system. Then the slurry was pumped by peristaltic pump at particular feed rate to the sand pump in place of tap water and the slurry flow rate from the outlet of sand pump was checked measuring the slurry rate by conventional method. The details set up and running of the flotation column is shown in Fig.2. The experiments were carried out by varying the different rate of feed slurry, air, wash water, collector and frother. In each case, when the steady state reached, the concentrate and tailings were collected separately for analysis. Then the samples were dried and weighed. The proximate analysis was carried out according to ASTM standard using Fisher Coal Analyser Model 490.

RESULTS AND DISCUSSION

Size and ash distribution of each samples were given in Table 1. In each case there is also substantial amount of +500 micron particles. The flotation was carried out at different dosages of reagents in both flotation cell and column. The experiments were also carried out at different operating variables i.e., air rate, feed rate, wash water rate and variable solid concentration of slurry in the flotation column. Some of the results of flotation cell and columns for different samples are given in Table 2-7. Typical results of pilot plant flotation column at West Bokaro coal washery is given in Table 8. The results of the batch flotation in cell

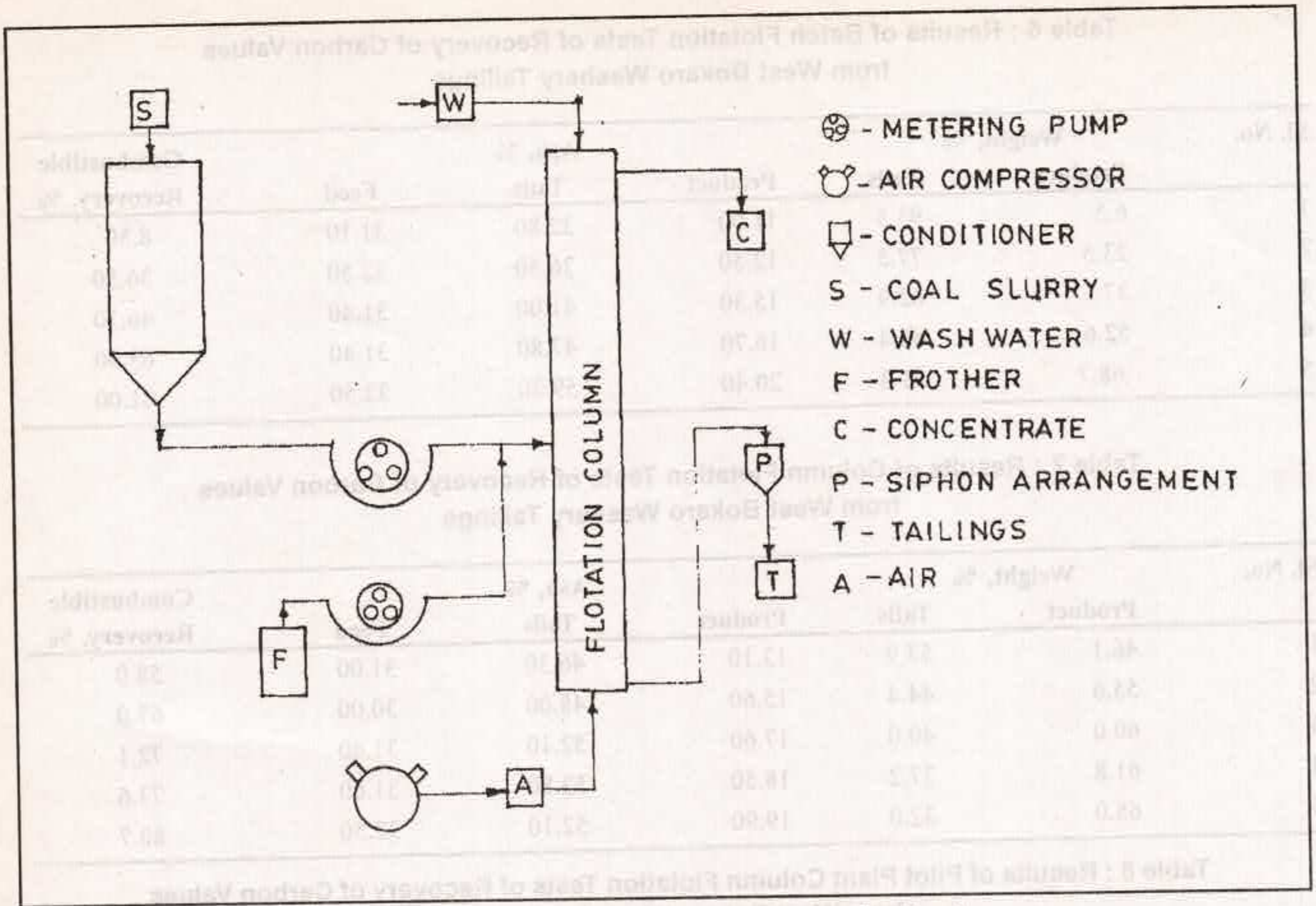

Fig. 1 : Schematic diagram of the whole set up of the flotation plant.

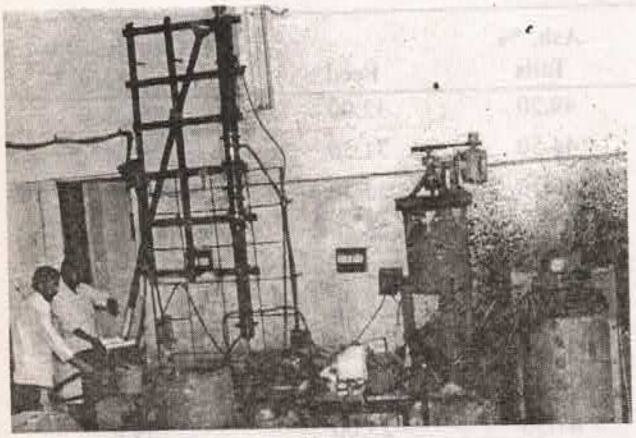

Fig. 2 : Experimental set up of flotation column.

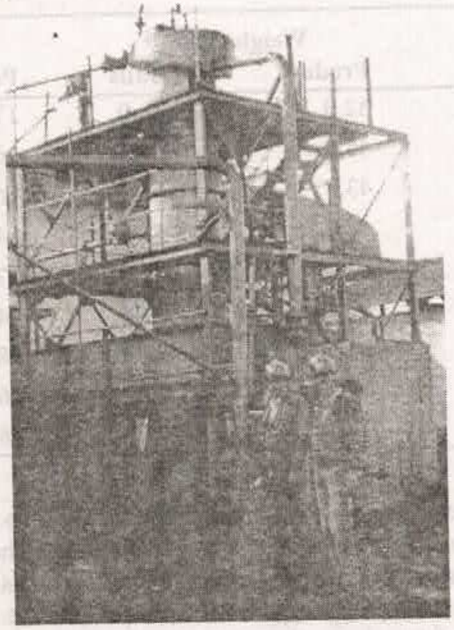

Fig. 3 : Pilot plant flotation column at West Bokaro coal washery.

and column flotation tests on Sudamudih coal are given in Table 2 & 3. It has been observed that the flotation column gives better results in comparison with flotation cell with respect to ash percentage of the concentrate and combustible recovery. Similar observation has been seen in Table 4-7 in case of Bheltand coal and West Bokaro washery tailings. The pilot plant results are given in Table 8. It can be concluded that almost the laboratory results could be reproduced in the pilot scale plant. As coal is low cost material, addition of cleaning stage will be expensive. For single stage flotation, the column will give better performance than flotation cell due to have extra cleaning zone. As all three samples contain substantial amount of +500 micron, so that maximum coarse particles were reported in tailings in the flotation column due to high residence time and high froth bed. If the sample does not contain +500 microns particles, the results of the flotation in the column will be excellent.

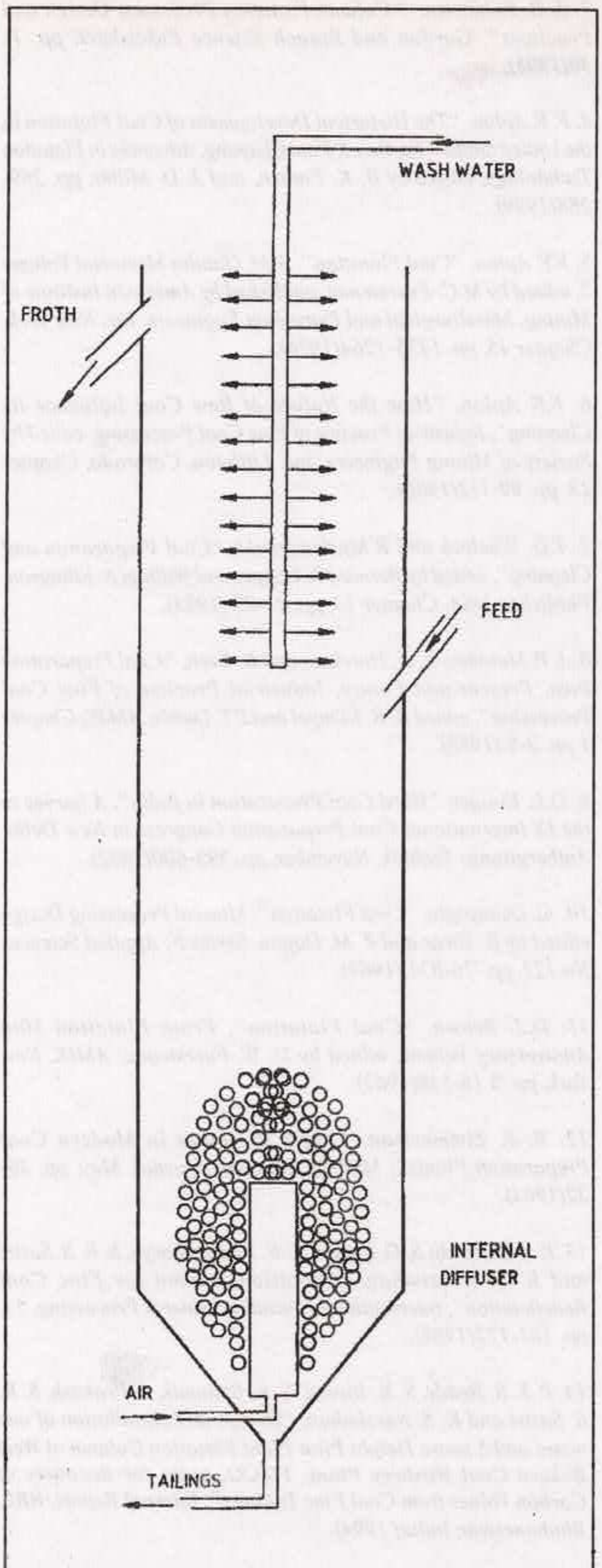

Fig. 4 : Pipe line wash water distribution in flotation column.

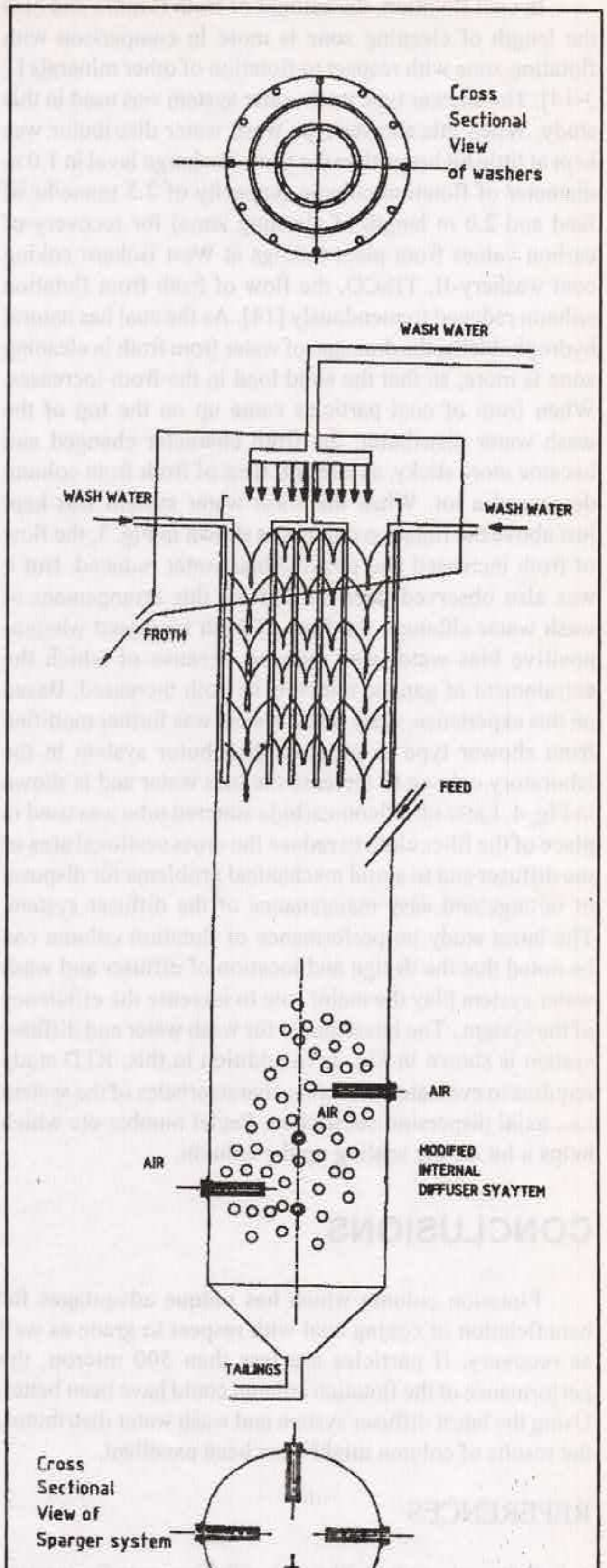

Fig. 5 : Latest design of wash water distribution and diffuser.

In coal flotation, the amount of froth is more and also the length of cleaning zone is more in comparison with flotation zone with respect to flotation of other minerals [13-14]. The shower type wash water system was used in this study. When this shower type wash water distributor was kept at little bit lower than the froth discharge level in 1.0 m diameter of flotation column (capacity of 2.5 tonne/hr of feed and 2.0 m length of cleaning zone) for recovery of carbon values from plant tailings at West Bokaro coking coal washery-II, TISCO, the flow of froth from flotation column reduced tremendously [14]. As the coal has natural hydrophobicity, the drainage of water from froth in cleaning zone is more, so that the solid load in the froth increases. When froth of coal particles came up on the top of the wash water distributor, the froth character changed and became more sticky, as a result, flow of froth from column decreased a lot. When the wash water system was kept just above the flotation column as shown in Fig. 3, the flow of froth increased and positive bias water reduced. But it was also observed later on that for this arrangement of wash water although the flow of froth increased whereas positive bias water was reduced because of which the entrainment of gangue minerals in froth increased. Based on this experience, wash water system was further modified from shower type to pipeline distributor system in the laboratory column to increase the bias water and is shown in Fig. 4. Later on silicon-carbide sintered tube was used in place of the filter cloth to reduce the cross sectional area of the diffuser and to avoid mechanical problems for disposal of tailings and easy maintenance of the diffuser system. The latest study on performance of flotation column can be noted that the design and location of diffuser and wash water system play the major role to increase the efficiency of the system . The latest model for wash water and diffuser system is shown in Fig. 5. In addition to this, RTD study requires to evaluate the mixing characteristics of the system i.e., axial dispersion coefficient, Peclet number etc which helps a lot during scaling up the column.

CONCLUSIONS

Flotation column which has unique advantages for beneficiation of coking coal with respect to grade as well as recovery. If particles are less than 500 micron, the performance of the flotation column could have been better. Using the latest diffuser system and wash water distributor, the results of column might have been excellent.

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ALLEVIATION OF THE ROCK BURST HAZARD IN COAL MINES USING THE ENERGY RATE CONCEPT

P. R. Sheorey*

इस आलेख में कोयला खदानों में चट्टान विस्फोटन के दो केस - पहला गिरिमिन्ट में तथा दूसरा चिनाकुरी में - का प्रस्तुतीकरण किया गया है। 3 डी मॉडेलिंग व्यवहृत इन दो केसों तथा चिनाकुरी की विभिन्न खानों के लेआउट के लिए ऊर्जा निष्कासन की दर का अभिकलन किया गया है। इससे यह निष्कर्ष निकलता है कि चट्टान विस्फोटन के नियंत्रण के लिए ईल्ड पिलर्स के साथ डिपलरिंग अन्य विधियों से बेहतर है।

INTRODUCTION

With the advent of advanced computer modelling techniques, rock burst control by mine layout design has become a reality using a new parameter, energy release rate (ERR), in S. African gold mines (Salamon, 1983). ERR is the strain energy released by mining a unit area of the seam or ore body. It has been shown from case studies that the number of rock bursts is non-linearly proportional to ERR, Fig. 1 (Ortlep, 1983).

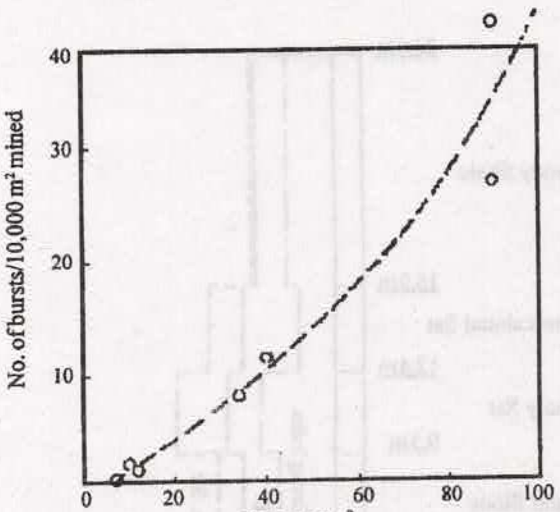

Fig. 1 : Variation of rock burst incidence with EER in S. African gold mines

Planning of a mine layout consists of minimising ERR by adopting the following strategies:

(a) Reduction in panel width
(b) Reduction in extraction height
(c) Filling or stowing

Another method gaining increasing popularity, in coal mines particularly, consists of leaving carefully designed yield pillars (Campoli et al. 1987).

CASE STUDIES OF ROCK BURSTS IN COAL MINES

Although rock bursts in coal mines of Ranigunj coalfield have been reported since the 1920's, scientifically studied cases are very few.

Only two known cases of rock burst, Chinakuri No. 1 mine and Girmint of ECL, which led to fatality or serious injury could be studied in detail. Of these, Koithee seam workings of Girmint, in which the rock burst occurred, are now closed. On the other hand Dishergarh seam at Chinakuri No. 1 continues to be exploited and in fact, longwall extraction with caving was planned until recently in this seam.

Girmint Colliery

Panel 26-K of Girmint, where one serious injury occurred due to a coal bump on 19.2.85, was probably the deepest case of depillaring with caving (depth 490 m). It had the following particulars:

Seam	Koithee
Panel	26-K
Average depth	490 m
Extraction height	4.57 m
Seam thickness	4.57 m
Gradient	1 in 8.5
Pillar size (c/c)	40 to 54 m
Roadway width	4.2 to 4.8 m
Roof	Coaly shale

*Central Mining Research Institute, Dhanbad-826 001

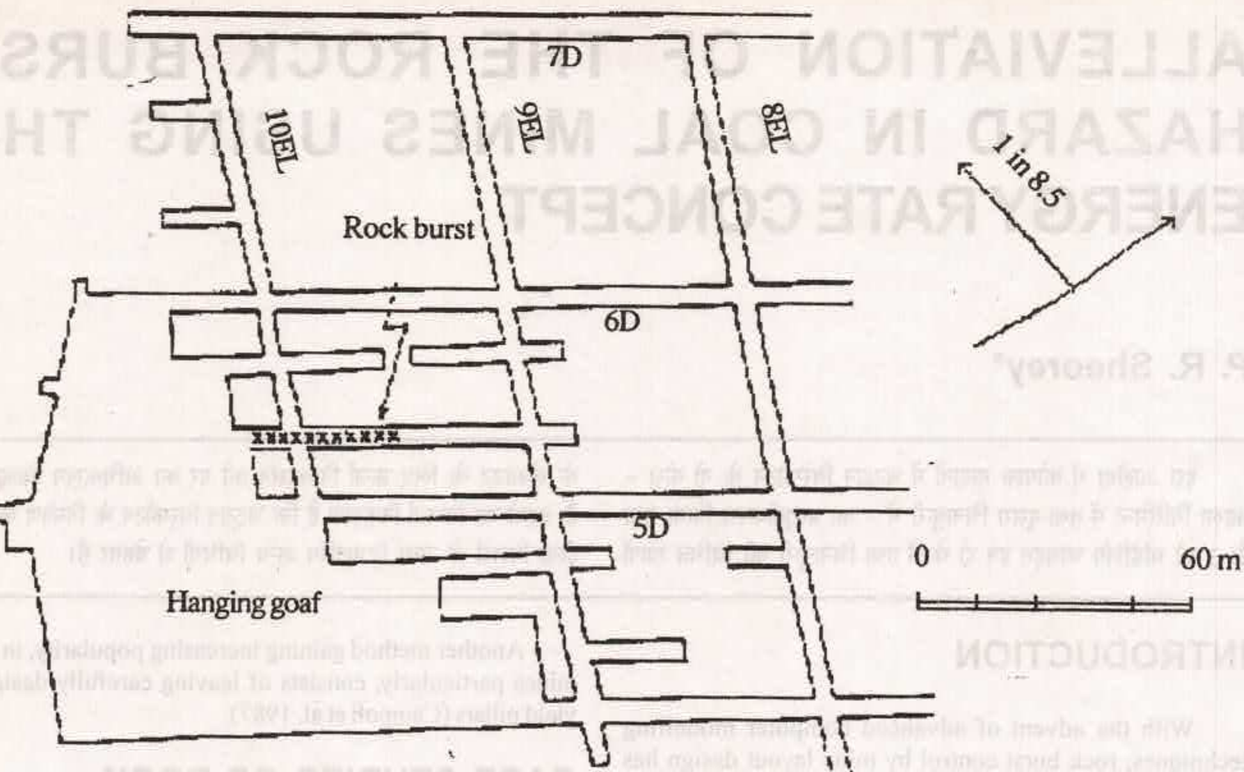

Fig. 2 : Location of rock burst at Girmint

The method of extraction consisted of conventional depillaring with ribs and slices and wooden supports. The site of the rock burst (bump), the position of the face etc. can be seen from Fig. 2. Only a little caving had taken place until that date.

Since CMRI was approached some time after the fact a physical picture of the incident was collected from the management. Considerable coal had dislodged from one side near the floor knocking a few props down, while the chocks remained standing. No roof fall at the accident site could be seen. This was thus a case of coal bump than of roof or floor burst.

The initial DGMS permission allowed two dip-rise splits in a pillar with ribs 2.5 m and slices 3.9 m wide. After the accident the permission was revised to level splits (splitting being now restricted to only one row of pillars) and a slice width of 3 m only.

CMRI investigations included determination of the energy index of the coal which was found to be 3.17, indicating moderate bump-proneness, and roof diamond drilling to obtain RQD and strength, which are shown plotted in Fig. 3. CMRI also recommended side rope stitching. Further extraction in the panel could be done without incident.

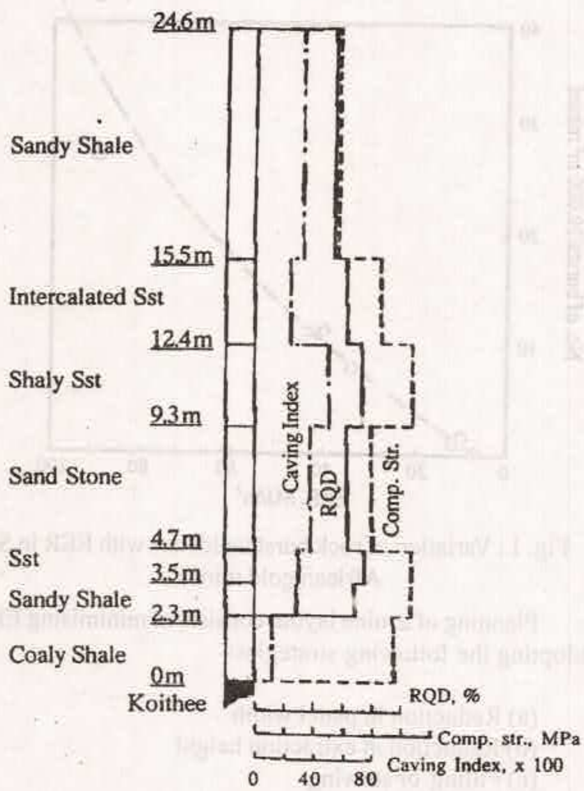

Fig. 3 : Roof characteristics at Girmint

The ERR values were obtained by 3-D BESOL modelling as

Average ERR	1.32×10^5 J/m²
Maximum ERR	1.50×10^5 J/m²

The CMRI cavability index (Sarkar and Singh, 1985) may be calculated here as a matter of interest from

$$I = \sigma_c \, t^{0.5} \, L^n / 5$$

where σ_c = compressive strength of a bed, kg/cm²
t = bed thickness, m
L = average core length, cm
n = 1.0 for RQD < 80 and 1.2 for RQD > 80

Chinakuri No. 1 Mine

The Panel in this mine, where a rock burst took place on 10.10.92 leading to 3 fatalities, had the following details:

Seam	Dishergarh
Panel	W-13
Average depth	600 m
Extraction height	2 m
Seam thickness	3.35 m
Gradient	1 in 4.5
Pillar size (c/c)	45 m (min.)
Roadway width	3.6 - 4 m
Roof strata	Coal & intercalation

The method of working was straight line depillaring (modified longwall, Barry face longwall) with stowing using TCR props and link bars and chain conveyor. Coal wining was done by drilling and blasting, each cut being 0.6 m. At the end of extraction of the first row of Pillars, 7 m wide ribs (as per permission) were left, the chain conveyor and supports were shifted to the outbye advance gallery and the second row extraction was started. When the average rib width was 8 m, (average 8.2 m to be exact) preparations for the new face were going on in 58R and a rock burst occurred (see Fig. 4) in this roadway.

Visual and Scientific observations

Underground inspection revealed the following:

(a) About *1.5* - 2 **in** of roof consisting of coal and shaly sandstone had collapsed in 58R dislodging all supports. Some falls in 1 WRL and 2 WRL ahead of the face had also taken place.

(b) Severe spalling, squeezing and extrusion of ribs had taken place.

(c) Warm water was dripping at the face, which was always dry.

Because of the roof fall and rib extrusion this was a case of roof burst as well as coal bump.

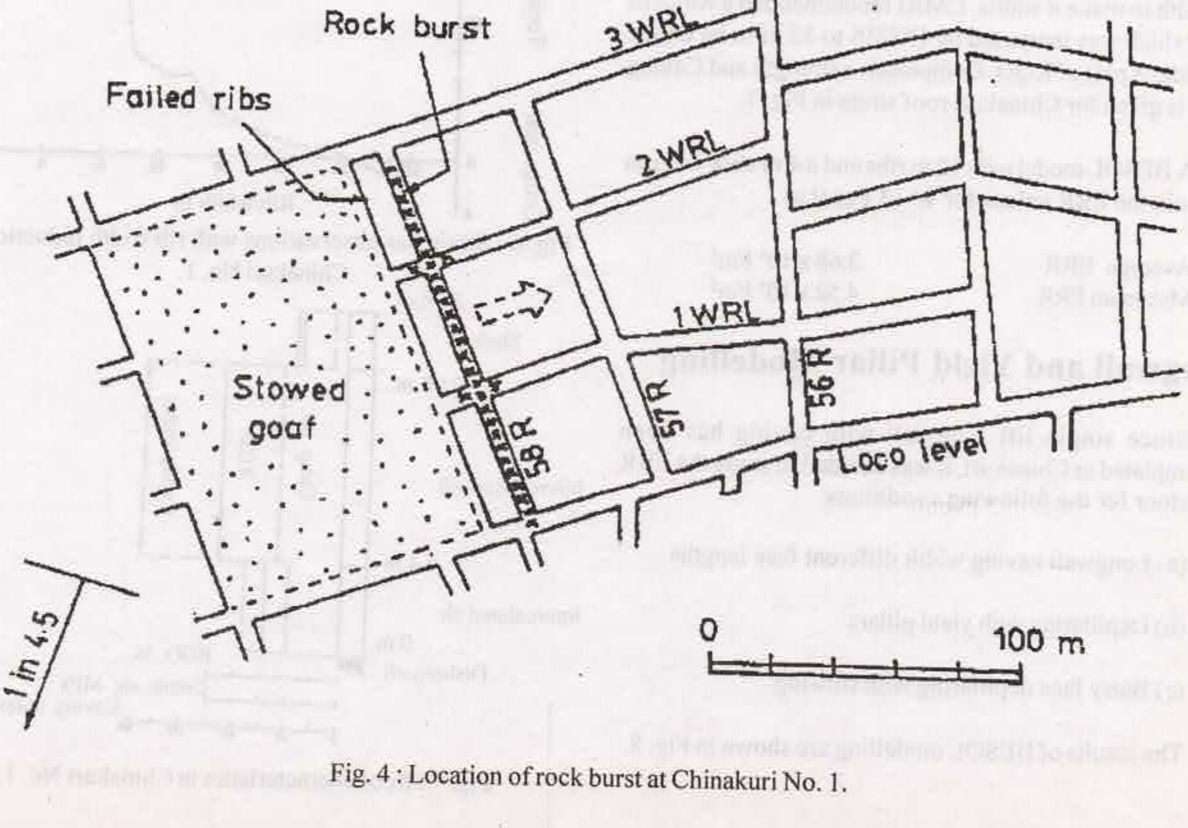

Fig. 4 : Location of rock burst at Chinakuri No. 1.

Scientific observations taken from the start of the Panel consisted of drilling yield tests at the face and vertical strain monitoring in ribs ahead of the face. Drilling yield test results are given in Fig. 5 and the last test on 9.10.92, when the rib width was 8.5 m indicated a rock burst.

Vertical rib strain was measured with simple mechanical strain bars installed ahead of the face. The strain changed from compression to tension as the Pillar width reduced and lastly to accelerated tension when it was 10 m or less (Fig. 6). These observations were typical of pillar failure and were valuable in that they gave the stable rib size as + 10 m.

A BESOL model of the situation gave the average rib load as 28 MPa while the rib strength was obtained from the CMRI Pillar strength formula as 27.1 MPa:

$$S = 0.27\,\sigma_c\,h^{-0.36} + [(h/250) + 1]\,.\,[(w_e/h) - 1] \quad \text{Mpa}$$

where σ_c = coal strength (35 MPa)
h = extraction height (2 m)
H = depth (600 m)
$w_e = 2w_1w_2/(w_1+w_2)$
w_1 = rib width (8 .2 m)
w_2 = rib length (40 m)

The safety factor of the rib was thus 0.97, indicating failure.

The strain bar observations and this pillar stability calculation afforded a simple and expeditious means of rock burst control in this situation and that was to increase the rib width to make it stable. CMRI recommended a width of 10 m which was increased by DGMS to 12 m to be on the safe side. A plot of RQD, Compressive strength and Caving index is given for Chinakuri roof strata in Fig. 7.

A BESOL model with 12 m ribs and a 4 m slice was run to obtain the ERR values for W-13 Panel as

Average ERR	3.68×10^5 J/m^2
Maximum ERR	4.58×10^5 J/m^2

Longwall and Yield Pillar Modelling

Since single lift longwall with caving has been contemplated at Chinakuri, it was decided to study the ERR behaviour for the following conditions:

(a) Longwall caving width different face lengths

(b) Depillaring with yield pillars

(c) Barry face depillaring with stowing

The results of BESOL modelling are shown in Fig. 8.

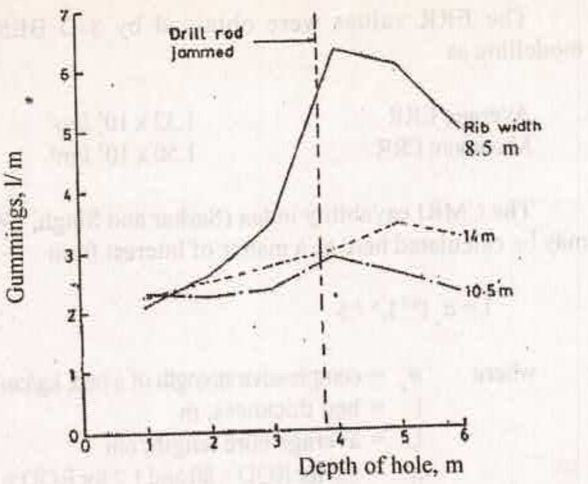

Fig. 5 : Drilling yield test results at Chinakuri No. 1.

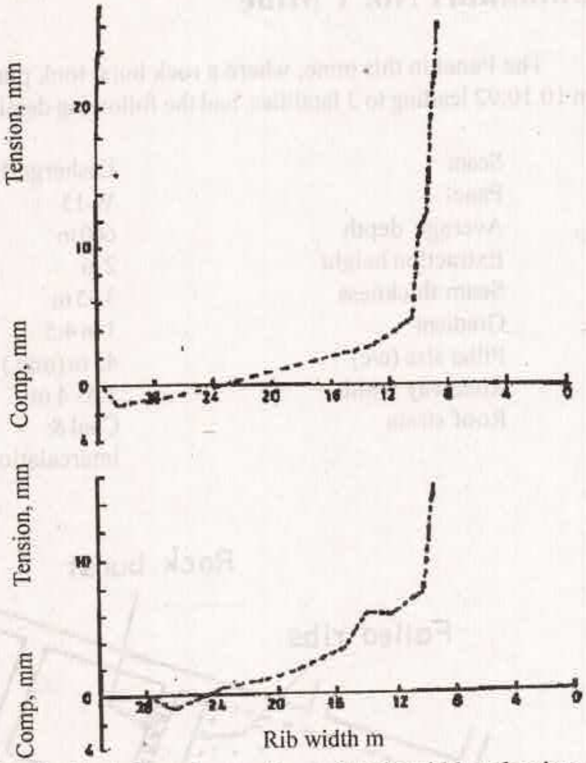

Fig. 6 : Strain bar observations with rib width reduction at Chinakuri No. 1.

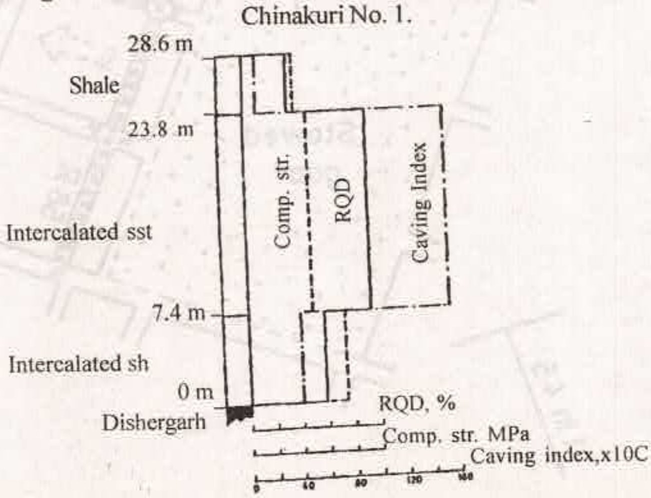

Fig. 7 : Roof characteristics at Chinakuri No. 1.

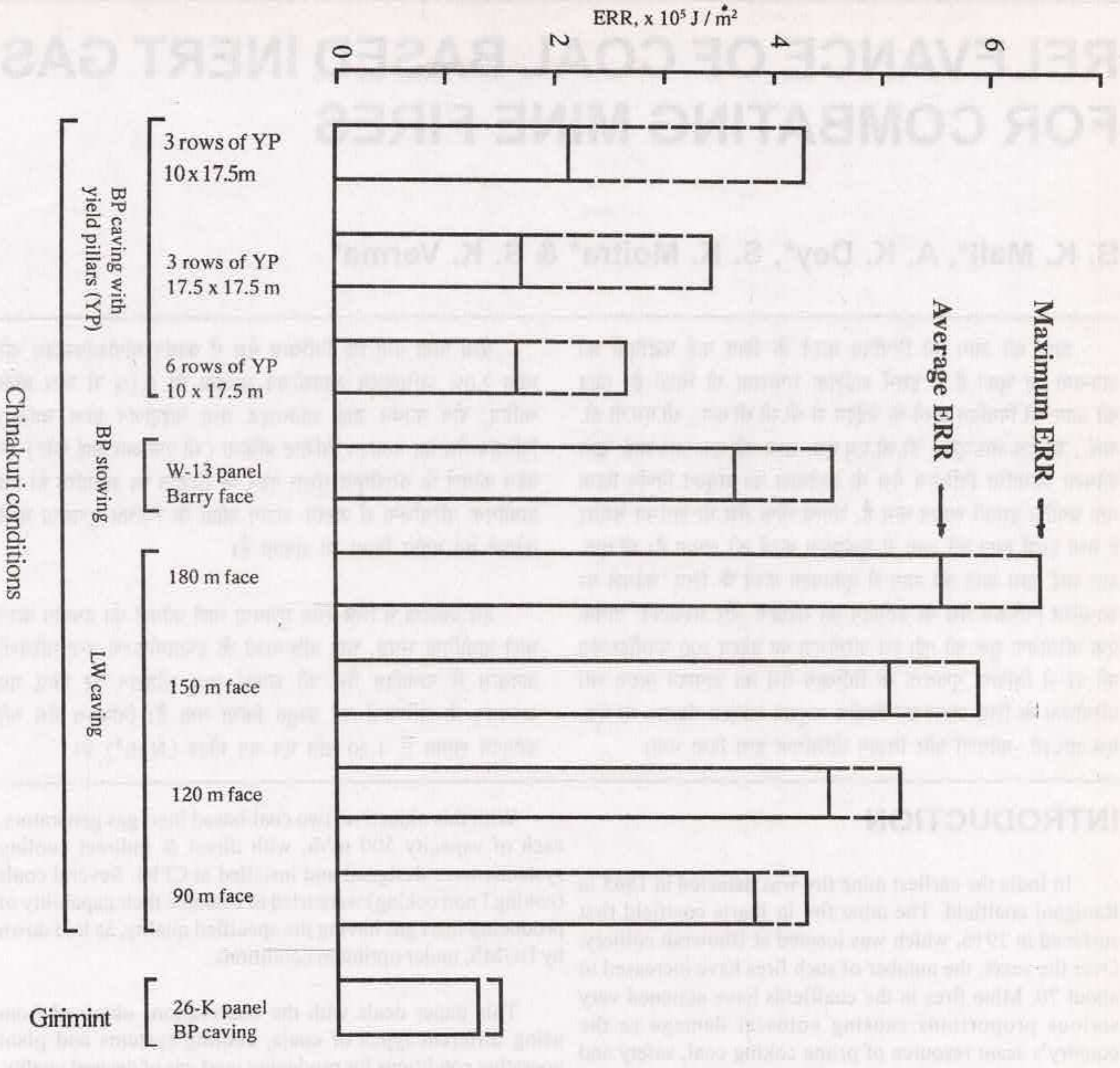

Fig. 8 : Comparison of ERR for different mining layouts.

CONCLUSION

It is readily seen from Fig. 8 that stowing is better than longwall caving and yield pillars are better than stowing. Yield Pillars, obviously, have to be designed for "stable" rather than sudden failure.

The ERR has so far been used only as a comparative index. The question arises is: is it possible to reduce ERR to a cut-off value at which rock bursts can be totally eliminated? Such a cut-off value must exist as is seen from Fig. 1. Since coal measures are roughly 10-20 times weaker than burst-prone rocks in gold mines, the ERR cut-off value in coal mines will be that much less.

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RELEVANCE OF COAL BASED INERT GAS FOR COMBATING MINE FIRES

B. K. Mall*, A. K. Dey*, S. K. Moitra* & S. K. Verma*

खान की आग को नियंत्रित करने के लिए कई पद्धतियों को अपनाया जा चुका है व इसमें आंशिक सफलता भी मिली है। खान की आग को नियंत्रित करने के उद्देश्य से बी.सी.सी.एल., सी.एम.पी.डी.आई., सी.एम.आर.एस., डी.जी.एम.एस. तथा सी.एफ.आर.आई. द्वारा कोयला आधारित निष्क्रिय गैस के इस्तेमाल का संयुक्त निर्णय लिया गया क्योंकि इसकी लागत कम है, घनत्व गोफ गैस के लगभग बराबर है तथा इसमें खान की आग से मुकाबला करने की क्षमता है। सी.एफ.आर.आई. द्वारा खान की आग से मुकाबला करने के लिए 'कोयले पर आधारित निष्क्रिय गैस के उत्पादन का संरेखण और संस्थापन' नामक एक परियोजना शुरू की गई। इस परियोजना का उद्देश्य 500 घनमीटर/घंटा की दर से विशिष्ट गुणवत्ता के निष्क्रिय गैस का उत्पादन करना था। परियोजना के लिए आवश्यक वित्तीय अनुदान कोयला मंत्रालय के एस.एस.आर.सी.-कोयला और विज्ञान प्रौद्यिगिक द्वारा दिया गया।

ऐसा पाया गया कि निष्क्रिय गैस में कार्बन मोनोऑक्साइड की मात्रा 2.0% अधिकतम आक्सीजन सान्द्रता पर 0.1% से कम होनी चाहिए, शेष कार्बन डाई आक्साइड तथा नाइट्रोजन होना चाहिए। निष्क्रिय गैस का उत्पादन विशिष्ट प्रक्रिया (सी.एफ.आर.आई पेटेंट) के तहत कोयले के तरलीकृत संस्तर दहन के सिद्धांत पर आधारित है। इस प्रायोगिक परियोजना में अलग-अलग स्रोतों के विभिन्न गुणवत्ता वाले कोयले का प्रयोग किया जा सकता है।

इस आलेख में भिन्न-भिन्न गुणवत्ता वाले कोयले का उपयोग करने वाले प्रायोगिक संयंत्र, चल प्रक्रियाओं के इष्टतमीकरण तथा परिवेशी तापक्रम में उत्पादित गैस की सफाई तथा शीतलन पर किए गए अध्ययन के परिणामों को प्रस्तुत किया गया है। निष्क्रिय गैस की उत्पादन लागत रु. 1.20 प्रति एन घन मीटर (N m^3) है।

INTRODUCTION

In India the earliest mine fire was detected in 1865 in Raniganj coalfield. The mine fire in Jharia coalfield first surfaced in 1916, which was located at Bhowrah colliery. Over the years, the number of such fires have increased to about 70. Mine fires in the coalfields have assumed very serious proportions causing colossal damage to the country's scant resource of prime coking coal, safety and economic health of mining operations. According to official estimates 45 million tonnes of prime coking coal have already been lost in Jharia coal field alone and devastated about 17.32 sq. km. of land. Moreover, nearly 2000 million tonnes of prime coking coal have been locked up by these mine fires. Several methods have been tried to control these fires and marginal success could be achieved so far.

In this context, a project entitled "Design and installation of two coal fired inert gas generators, for combating mine fires in Jharia coal field" was undertaken by CFRI under SSRC-S & T grant of Department of coal. The decision to use coal based inert gas was taken due to its cheap price, density nearly equal to that of the goaf gas and efficacy to combat mine fires. As per DGMS specifications inert gas should contain less than 0.1 % carbon monoxide at maximum oxygen concentration of 2.0 %, rest being carbon dioxide and nitrogen.

With this objective, two coal-based inert gas generators, each of capacity 500 m^3/h, with direct & indirect cooling systems, were designed and installed at CFRI. Several coals (coking I non coking) were tried to examine their capability of producing inert gas having the specified quality, as laid down by DGMS, under optimum conditions.

This paper deals with the observations obtained from using different types of coals, cooling systems and plant operating conditions for producing inert gas of desired quality.

INERT GASES FOR COMBATING MINE FIRES

It has been reported that 75 to 85% of mine fires all over the world are caused by spontaneous combustion of coal which can be well prevented by reducing oxygen concentration below 3% [1]. Significant advances have been made in recent years towards the control of mine fires by using fogged inert gas (a mixture of inert gas and water vapour droplets), dry inert gases (a mixture of inert gases and nitrogen) for extinguishing underground mine fires [2]. The use of inert gas for combating mine fires greatly reduces the time of recovery of a fire zone as compared to the old method of sealing a fire and waiting long enough for the fire zone to cool sufficiently before its reopening. Theoretically, any inert gas can be used for the purpose of combating mine

*Central Fuel Research Institute, P.O.-FRI, Dhanbad - 828 108

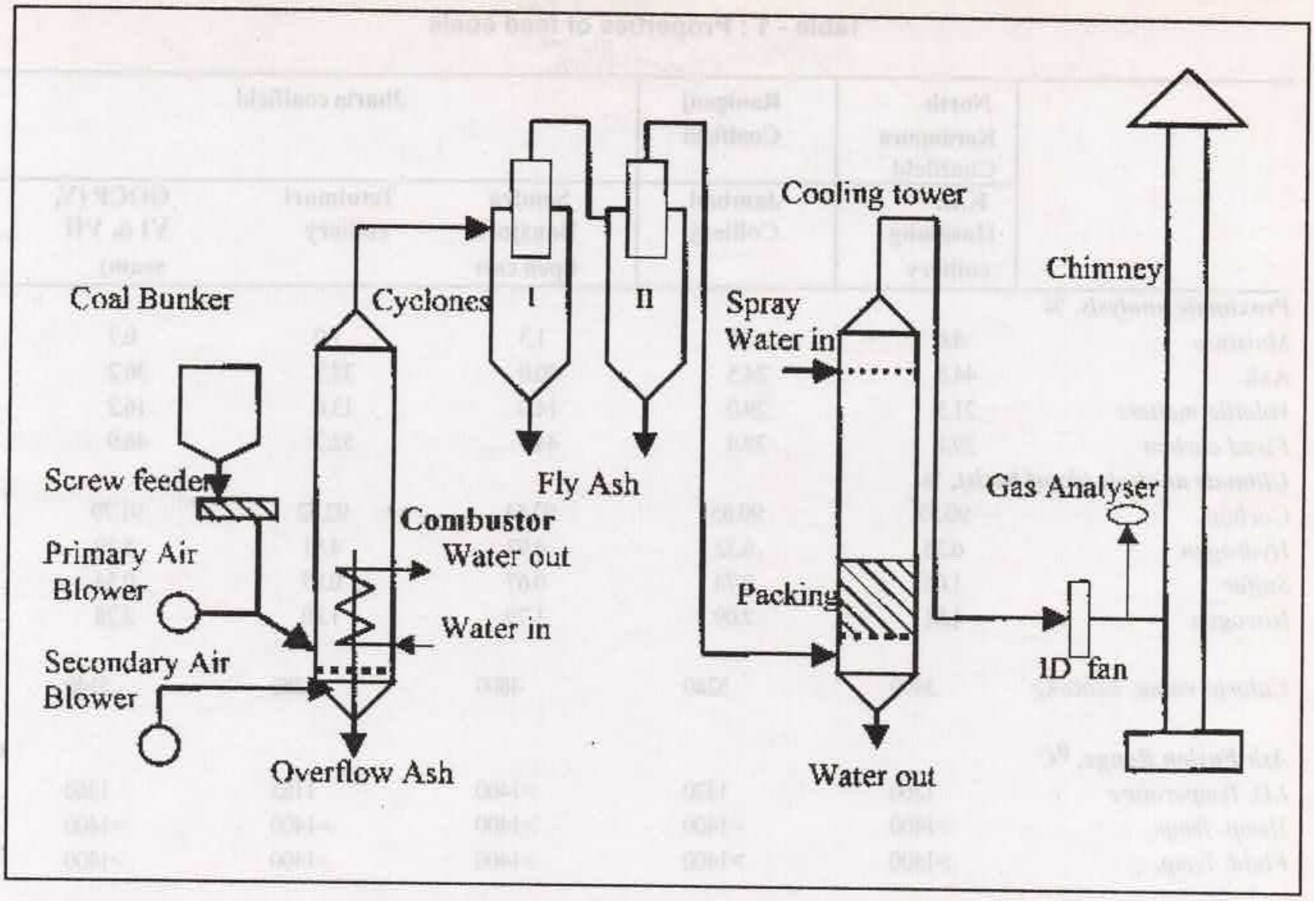

Fig. 1 : Process Flow Sheet.

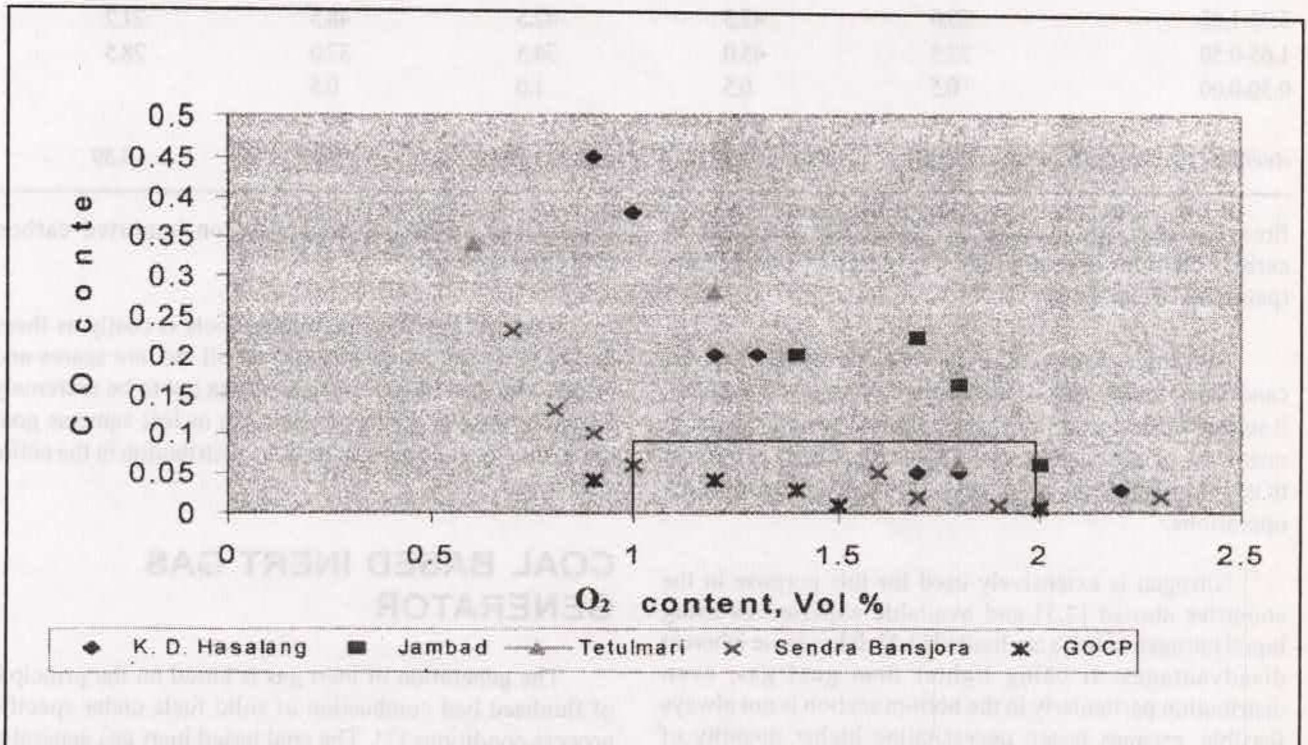

Fig. 2 :Effect of O_2 on CO.

Table - 1 : Properties of feed coals

	North Karanpura Coalfield	Raniganj Coalfield	Jharia coalfield		
	K.D. Hasalang colliery	Jambad Colliery	Sendra Bansjora Open cast	Tetulmari colliery	GOCP (V, VI & VII seam)
Proximate analysis, %					
Moisture	4.6	7.1	1.3	1.0	0.7
Ash	44.8	24.5	40.0	32.7	36.2
Volatile matters	21.5	29.0	14.0	13.6	16.2
Fixed carbon	29.1	39.4	44.7	52.7	46.9
Ultimate analysis (damf basis), %					
Carbon	90.55	90.85	92.61	92.52	91.79
Hydrogen	6.58	6.32	4.93	4.81	5.39
Sulfur	1.06	0.74	0.67	0.87	0.54
Nitrogen	1.81	2.09	1.79	1.80	2.28
Caloric value, kcals/kg	3490	5280	4800	5380	5140
Ash Fusion Range, ^0C					
I.D. Temperature	1260	1320	>1400	1185	1360
Hemp. Temp.	>1400	>1400	>1400	>1400	>1400
Fluid. Temp.	>1400	>1400	>1400	>1400	>1400
Particle Size distribution, wt. %					
Size range, mm					
6.00-3.33	24.0	12.0	22.0	14.0	47.0
3.33-1.65	53.0	42.5	42.5	48.5	21.2
1.65-0.50	22.5	45.0	34.5	37.0	28.5
0.50-0.00	0.5	0.5	1.0	0.5	
Average particle, size, mm	2.08	1.56	1.75	1.72	3.59

fires. However,., in practice the choice narrows down to carbon dioxide, nitrogen and flue gases of combustion (petroleum fuels/ coal).

Although carbon dioxide was one of the earliest candidates for the role of inertisation of mine atmosphere, it suffers from several drawbacks. It is not available in large quantities, stratifies along the floor forcing the fire to migrate to upper sections and is difficult to remove during recovery operations.

Nitrogen is extensively used for this purpose in the countries abroad [2,3] and available experiences using liquid nitrogen in India are limited [4,5]. It has some inherent disadvantages. It being lighter than goaf gas, even distribution particularly in the bottom section is not always feasible, escapes faster, necessitating higher quantity of gas input. Moreover, commercial production of nitrogen by liquefaction of air needs a very high level of capital investment and its production by PSA (pressure swing adsorption) method is dependent on imported carbon molecular sieve [6].

Inert gas based on petroleum fuels is costly as these fuels e.g. diesel, kerosene and fuel oil etc. are scares and costly. The coal based inert gas works out to be extremely cheap. Moreover, its density is more or less same as goaf gas in the fire zone permitting even distribution in the entire void area.

COAL BASED INERT GAS GENERATOR

The generation of inert gas is based on the principle of fluidised bed combustion of solid fuels under specific process conditions [7]. The coal based inert gas generator can take up widely different qualities of coal independent of their sources. The process flow sheet of the pilot plant is shown in Figure - 1.

Table - 2 : Result of typical tests with coals of different qualities

	North Karanpura Coalfield	Raniganj Coalfield	Jharia coalfield		
	K.D. Hasalang colliery	Jambad Colliery	Sendra Bansjora 0pen cast	Tetulmari colliery	GOCP (V, VI & VII seam
Feed size, mm	6.00	6.00	6.00	6.00	6.00
Feed rate, kg/h	90	60	55	50	80
Average bed temp., °C	910	900	900	900	900
Superficial fluidising velocity, m/s					
At 30^0C	0.46	0.42	0.35	0.37	0.40
At average bed temp.	1.81	1.65	1.35	1.43	1.55
Air to coal ratio, Nm^3/kg	4.63	6.30	5.66	6.69	5.64
Product inert gas analysis, vol. %					
Carbon dioxide	16.70	16.20	16.90	16.60	17.10
Oxygen	1.70	2.00	1.60	1.80	1.40
Carbon monoxide	0.05	0.06	0.05	0.06	0.03
Nitrogen	81.55	81.74	81.45	81.54	81.47
Heat input rate, kW	365.0.	368.3	306.9	312.7	478.1
Heat release rate, kW	357.7	357.3	294.6	297.1	456.6
Combustion efficiency, %	98.0	97.0	96.0	95.0	95.5

The combustor is a cylindrical vessel the bottom section of which is lined inside with a layer of firebricks. A distribution plate is fitted at the bottom of the combustor on which coal is fed through screw feeders and conveyed by the primary air. A cooling coil is placed inside the lined section through which water is circulated in order to control the reaction temperature The combustor is also provided with an overflow pipe centrally placed for the ash discharge as well as control of the bed height. Thermo-wells at different places and pressure tapping are also provided for the measurement of temperatures and pressure drop in the bed. The combustor is followed by cyclones (primary & secondary) for cleaning and cooling systems (indirect/direct) for the product gas. On-line oxygen and carbon monoxide analysers are also fitted before its exit to the atmosphere to monitor the quality of the gas continuously. In the first unit, for indirect cooling, two shell and tube (single pass) heat exchangers in series have been installed where hot gas is passed through tubes and cooling water through the shell. In the second unit, for direct cooling a partially packed spray tower has been used to cool the hot product inert gas to ambient temperature.

TEST PROCEDURE

The primary air through coal fee ' line and secondary air through distribution plate were pa sed to keep the bed (coal ash & lignite) fluidising. The bed was then ignited by addition of a small quantity of separately lighted sized charcoal. When the bed temperature reaches the ignition temperature of coal, feeding of coal (crushed and screened below 6 mm) was started and gradually raised to the desired level. The air and water rates were then set accordingly in order to attain optimum condition which is indicated by the temperature and pressure drop of the bed. After attaining the steady state conditions a test was conducted for period 4 to 5 hours for the data collection.

COALS TESTED AND THEIR OBSERVATIONS

Coals, both non-coking and coking with wide variation in ash content, (24 to 45%) and from different coalfields were tried. Their analyses and physical properties are presented in Table - 1. The details of optimized process variables, e.g. average bed temperature, bed height, coal feed rate, air rates, coal particle size and average composition of the product gas obtained are shown in Table - 2.

It was observed that at an average bed temperature of 920+/- 20°C and feed rates varying between 50 to 90 kg/h with non-coking coals from K. D. Hasalang colliery of North Karanpura field as well as Jambad colliery of Raniganj field and coking coals from (I) Sendra Bansjora open cast (VIII seam), (ii) Tetulmari colliery (II seam), (iii) GOCP coal (V, VI, VII seams) of Jharia field the inert gas containing CO less than 0.1% at 02 levels of 2% or below could be produced [8].

Carbon dioxide and nitrogen present in the inert gas were 16 to 17 % and 81 to 82% respectively. Other combustible gases e.g. hydrocarbons and hydrogen were absent. The combustion efficiency in all cases was found to be above 95%. A gradation in combustion efficiency obtainable with variation in rank of coal was observed. Higher the rank, lower was the combustion efficiency. The results suggest that within the range of the process variables employed the rank of coal played an important role in deciding combustion efficiency which in turn dictates the quality of inert gas.

The effect of residual oxygen content in the inert gas on carbon monoxide content has been shown in Figure - 2. Desired CO concentration was found well within the reach at 1-2% 02 in the inert gas with all coals tested in the pilot plant. The product gas was cleaned by two suitably designed cyclones (primary & secondary) and installed in series.

In the indirect cooling system the inert gas (350°C after exit from the secondary cyclone) could be cooled to 50°C. The critical temperature of the product gas is around 49°C below which the moisture (6 to 8% by wt.) gets condensed and causes corrosion in the tubes of the heat exchangers due to SO_2 (present in the inert gas) dissolved in the condensed moisture. Also fine dust (escaped from the cyclones) present in the gas (up to 40gm/m^3) gets accumulated at the mouth of the heat exchangers and gets stuck on the inner surface of the tube, which in the longer period (after 13 to 15 days) cause choking. Hence, heat exchangers need to be cleaned frequently.

In the direct cooling system the product inert gas could be cooled from 350°C to ambient temperature and also completely free from dust. However, cooling water in the gas cooler becomes acidic in nature and contaminated with dust. Therefore, fresh cooling water is needed continuously for the purpose. Only the bottom part of the gas cooler, which remains in contact with the acidic water, needs an acid resistant lining for protection from corrosion. For gas-water, resistance to heat transfer is offered mostly by the gas phase. So, for efficient cooling in case of high gas velocity, a packed bed is an obvious choice for this purpose. This in turn makes pressure drop a critical consideration. It has been estimated that a bed (packed with 25 mm raschig rings) 0.8 meter in diameter and 2 meters in height will be required to cool the gas to ambient temperature. But in this case the packed bed is liable to be choked due to dust load (40 gm/m^3 of the product gas) leading to gas channeling through the bed. So, the cooling tower is kept partially packed which causes less cost as well as low pressure drop i.e. less load on the fan. Cooling tower functions trouble free and needs no maintenance even after running for longer periods.

COST ESTIMATE

The cost estimate was made by the standard procedure and on the basis of two shifts, each of 8 hours duration and 300 working days per year. The cost estimate is given below:

		Rs. (Lakh)
Fixed Capital	:	22.80
Working Capital	:	2.22
Production Cost	:	22.20
General Expenses	:	6.47

The cost of production of coal based inert gas thus works out to be around Rs. 1.20.

CONCLUSIONS

The findings of the present study has established that the inert gas of desired quality can be produced cheaply by burning low grade coals by the technology developed at CFRI which has potentiality of combating mine fires successfully at the present juncture and also in future. The pilot plant is flexible in respect of quality and source of coal. The plant of this capacity is suitable for commercial exploitation hence needs no scaling up. The plant is extremely simple and cheap involving all indigenous equipment and instruments. Maintenance cost of the plant is very low. There is an urgent need for actual field trials with this inert gas generator in the selected mine under fire.

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PILLAR STRENGTH CONSIDERING IN SITU STRESSES

P.R. Sheorey*

खनन के पश्चात खनित क्षेत्र के चारों तरफ के शैल की विफलता की प्रक्रिया उस सकल तनाव पर निर्भर करती है जो खनन से पूर्व और खनन के बाद के तनाव के योग के बराबर होता है। कोयला खनन के दौरान दीर्घाओं के मध्य अथवा गोफों के मध्य छूटे पिलरों की विफलता के आकलन के लिए इस तनाव को जानना आवश्यक है। इस संदर्भ में पिलर की सामर्थ्य के आकलन के लिए एक समीकरण को विकसित किया गया था। इस आलेख में इस समीकरण को इण्डियन कोल माइन्स रेगुलेशन (1957) के संदर्भ में परखा गया है और अधिक सही आकलन के लिए इसे समुन्नत करने का प्रस्ताव रखा गया है।

INTRODUCTION

The failure process in rock around any excavation depends on the total stresses which are a sum of the pre-excavation or in situ stresses and excavation induced stresses. These stresses are matched against a suitable failure criterion to define failure or stability. Coal pillars, which occur between roadways or gob excavations, must also be assessed for stability using this fundamental procedure in rock mechanics. This procedure in the case of pillars need not be followed rigorously, and in fact, an indirect method can be adopted for defining pillar strength because such factors as the influence of width-to-height ratio (w/h) and size on pillar strength are more or less well established.

1. It should fit as many case studies (both failed and stable) as possible.

2. It should preferably be applicable to all practical ranges of w/h.

3. The effects of w/h, end constraint, and size should be incorporated in the equation. The equation should be amenable to changes for unusual and conditions (e.g., soft bands in the roof/floor).

4. A suitable coal strength parameter should be included in the formula. The uniaxial compressive strength of small specimens (1 inch cubes) is recommended, as the in situ large-scale strength is likely to be biased with cover depth (1)[2].

5. The influence of in situ stresses should be properly included.

A pillar strength equation was earlier developed based on these features (1). This paper examines the performance of this equation against case studies as well as against the Indian Coal Mines Regulations (1957) and suggests suitable refinement of the equation.

PERFORMANCE OF EARLIER FORMULA

The Central Mining Research Station (CMRS) formula reads as follows :

$$S = 0.27\,\sigma_c\,h^{-0.36} + [h/160\,\{(w/h) - 1\}] \qquad (1)$$

where S = pillar strength, MPa,
σ_c + strength of 1-in cubes of coal, MPa
h = extraction height, m
H = depth of rock cover, m
and w = pillar width, m

The first term on the right defines the effect of size on strength, and the second incorporates the influence of w/h, end constraint, and pre-excavation stresses jointly. The formula is thus essentially a triaxial strength relation, with the first term defining the uniaxial strength of a unit cube of side h, and the second defining the increase in strength due to triaxial effects when w/h>1 and decreases when w/h<1.

Performance of equation 1 along with some of the better known equations was studied (1) againtst 23 failed and 20 stable pillar cases (Tables 1-2) using the follwing two criteria specified by Salamon and Munro (2) :

1. In a plot of safety factors, the line of safety factor = 1 must pass through the middle of failed case data points.
2. All stable cases must have safety factors > 1.

* Central Mining Research Institute, Barwa Road, Dhanbad - 826 001

Table 1 : Failed pillar cases

	M	Seam	H, m	h, m	w, m	B, m	w/h	σ_c, MPa
1.	Amritnagar	Nega Jamehari	30	4.5	3.6	5.7	0.8	45
2.	Amritnagar	Nega Jamehari	30	6.0	3.6	5.4	0.6	45
3.	Begonia	Begonia	36	3.0	3.9	6.0	1.3	26
4.	Amlai	Burhar	30	5.4	4.5	4.5	0.83	25
5.	Sendra Bansjora	X	23	8.1	4.65	5.55	0.57	24
6.	W. Chirimiri	Main	90	3.75	5.4	6.0	1.44	45
7.	Birsingpur	Johilla Top	129	3.6	7.5	6.0	2.08	38
8.	Pure Kajora	Lower Kajora	54	3.6	5.4	6.0	1.5	33
9.	Pure Kajora	Lower Kajora	56	3.6	4.95	6.45	1.38	33
10.	Shankarpur	Jambad Bottom	42	4.8	4.5	4.5	0.94	47
11.	Ramnagar	Begunia	70	1.8	2.85	3.15	1.58	26
12.	Ramnagar	Begunia	51	1.8	3.0	3.6	1.67	26
13.	Kankanee	XIII	160	6.6	19.8	4.2	3.0	27
14.	Kankanee	XIV	140	8.4	18.6	5.4	2.2	25
15.	Jitpur	XIV	450	3.6	18 x 34.5	6	6.5	19
16.	Jitpur	XIV	450	3.6	10.5 x 12	6	3.08	19
17.	Jitpur	XIV	450	3.6	12 x 12	6	4.13	19
18.	Jitpur	XIV	450	3.6	18 x 21	6	5.33	19
19.	Jitpur	XIV	450	3.6	18 x 25.5	6	5.83	19
20.	Jitpur	XIV	450	3.6	15 x 63	6	6.67	19
21.	Jitpur	XIV	450	3.6	18 x 30	6	6.25	19
22.	Jitpur	XIV	450	3.6	18 x 30	6	6.25	19
23.	Jitpur	XIV	450	3.6	15 x 32	6	5.67	19

Note : In the case of rectangular pillars, the equivalent square pillar width was obtained from $2W_1W_2/(W_1 + W_2)$ where W_1, W_2 are the two sides of a rectangular pillar.

Table 2 : Stable pillar cases

	Mine	Seam	H, m	h, m	w, m	B, m	w/h	σ_c, MPa
1.	Bellampalli	Ross	36	3.0	5.4	6.0	1.8	48
2.	Nimcha	Nega	48	6.0	9.9	6.0	1.7	50
3.	Morgan Pit	Salarjung	270	3.0	8.1	3.6	2.7	46
4.	Ramnagar	Ramnagar	75	2.7	9.9	6.6	3.7	28
5.	Lacchipur	Lower Kajora	38	5.1	7.2	3.9	1.4	33
6.	N. Salanpur	X	30	5.1	9.0	6.0	1.8	21
7.	Bankola	Jambad Top	102	4.8	10.1	2.4	2.1	35
8.	Bankola	Jambad Top	75	3.0	6.3	4.2	2.1	35
9.	Sura kacchar	GI	106	3.5	16.0	4.0	4.6	29
10.	Lacchipur	Lower Kajora	38	5.1	18.3	4.2	3.6	33
11.	Sripur	Koithi	266	4.8	40.0	5.0	8.3	43
12.	E. Angarpathra	XII	30	2.1	6.0	6.0	2.9	19
13.	Kargali Incline	Kathara	36	3.6	9.3	5.7	2.6	40
14.	Jamadoba 6 & 7 Pits	XVI	80	2.1	5.8	5.5	2.9	29
15.	Topel	Singharan	85	1.8	7.0	3.9	3.9	41
16.	Jitpur	XIV	450	3.6	21 x 39	6.0	7.6	19
17.	Jitpur	XIV	450	3.6	18 x 48	6.0	7.3	19
18.	Jitpur	XIV	450	3.6	19.5 x 30	6.0	6.5	19
19.	Jitpur	XIV	450	3.6	18 x 31.5	6.0	6.4	19
20.	Jitpur	XIV	450	3.6	18 x 42	6.0	7.0	19

For estimating the safety factor, pillar load P was estimated from :

$$P = \gamma H [\{w+B\}/w]^2$$

where γ = unit rock pressure = 0.025 MPa/m
and B = roadway width, m.

Figure 1 shows that the two conditions of Salamon and Munro are well satisfied.

Since the publication of this equation, the author and his mining colleagues had occasion to use it in a variety of situations. It was seen that the equation underestimated pillar strength to some extent under shallow covers. It was also observed that pillar sizes increased rather slowly at depths greater than 300 m. The first discrepancy was noticed from its application to the Indian Coal Mines Regulations, which specify pillar sizes at various depths (Table 3). Generally, these regulations have given stable pillars except in very weak seams (σ_c < 20 MPa) or at greater depths (H > 400 m), when some side spalling has been evident. Figure 2 shows two example plots when the roadway widths are 3.0 and 4.8 m and the coal strength is σ_c = 15 MPa. The safety factor taken is 2.0 for long-term stability, which is the usual value taken in India.

MODIFICATION OF CMRS FORMULA

The general form from which equation 1 has been derived is :

$$S = \sigma_c h^{-\alpha} + am\gamma [d\sigma_x/d(w/h)].[(w/h)-1]$$

where α = exponent defining size effect,
a = triaxial factor
and m = horizontal to vertical in situ stress ratio

The differential coefficient defines the rate of change of the total horizontal stress σ_x with w/h and end constraint. For simplicity, it is assumed to be directly proportional to depth so that

$$S = \sigma_c h^{-\alpha} + am\gamma cH.[(w/h)-1] \qquad (2)$$

Thus, H is the average vertical stress and the in situ stress ratio m is assumed to be constant at all depths. This assumption is incorrect as it has been shown that m reduces with depth according to the general form (3)

$$m = p + q/H \qquad (3)$$

where p and q are empirical constants.

The constants p and q can vary considerably depending on local conditions. Unfortunately, inseam stress measurement data are very scantly, and hence the values of these constants have been estimated from a theory being

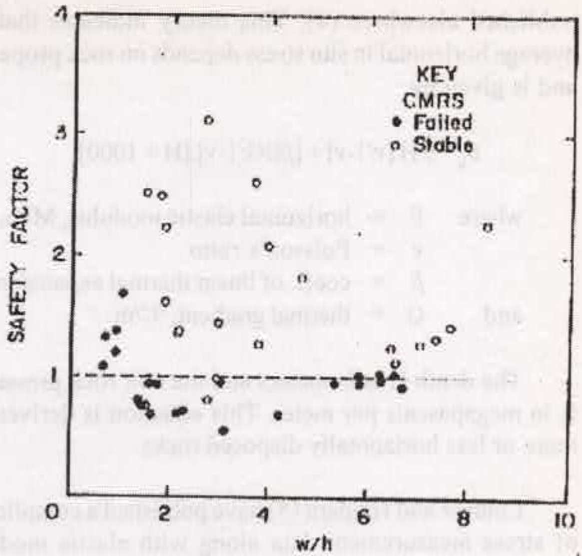

Fig. 1 : Performance of equation 1 against case studies.

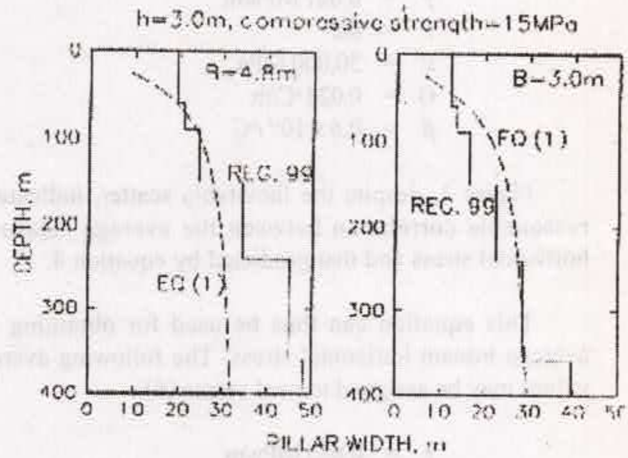

Fig. 2 : Pillar size as per equation 1 and Regulation 99.

Table 3 : Pillar sizes according to Regulation 99 of Indian Coal Mines Regulations, 1957 (h equal to or less than 3 m)

Depth of cover (H), m	Pillar size (centre to centre) for different roadway widths (B), m			
	B = 3.0	B = 3.6	B = 4.2	B = 4.8
<60 m	12	15	18	19.5
60 - 90	13.5	16.5	19.5	21
90 - 150	16.5	19.5	22.5	24.5
150 - 240	22.5	25.5	30.5	34.5
240 - 360	28.5	34.5	39.5	45
>360	39.5	42	45	48

published elsewhere (4). This theory indicates that the average horizontal in situ stress depends on rock properties and is given by

$$\sigma_h = \gamma H [\nu/1-\nu] + [\beta EG/1-\nu].[H + 1000] \qquad (4)$$

where E = horizontal elastic modulus, MPa,
ν = Poisson's ratio
β = coeff. of linear thermal expansion,
and G = thermal gradient, °C/m.

The depth H is in meters and the unit rock pressure γ is in megapascals per meter. This equation is derived for more or less horizontally disposed rocks.

Lindner and Halpern (5) have published a compilation of stress measurement data along with elastic modulus values. Equation 4 was checked for validity against these data using the following average values for crustal rocks:

γ = 0.027 MPa/m
ν = 0.2
E = 50,000 MPa
G = 0.024 °C/m
β = 0.6×10^{-5} /°C

Figure 3, despite the inevitable scatter, indicates a reasonable correlation between the average measured horizontal stress and that predicted by equation 4.

This equation can thus be used for obtaining the average inseam horizontal stress. The following average values may be assigned to coal seams (6) :

γ = 0.025 MPa/m
ν = 0.3
E = 3,000 MPa
G = 0.03 °C/m
β = 3×10^{-5} /°C

Dividing equation 4 by γH, the in situ stress ratio m was finally obtained as

$$m = 0.58 + 154/H$$

or

$$m = 0.6 + 150/H \text{ say} \qquad (5)$$

Substituting equation 5 in equation 2, the general equation becomes

$$S = \sigma_c h^{-\alpha} + ac\,\gamma H.[0.6 + (150/H)].[(w/h)-1]$$

$$S = \sigma_c h^{-\alpha} + C H.[0.6 + (150/H)].[(w/h)-1]$$

The value of $\alpha = 0.36$ as in equation 1 and the value of σ_c as the strength of 1-in cubes results in the equation in SI units :

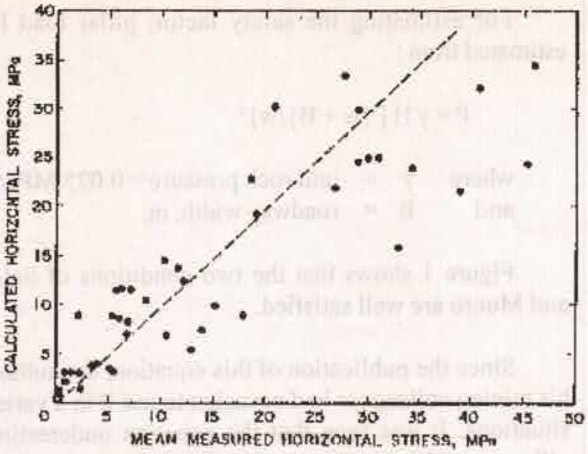

Fig. 3 : Comparison between the mean horizontal stress predicted by equation 4 and measured values.

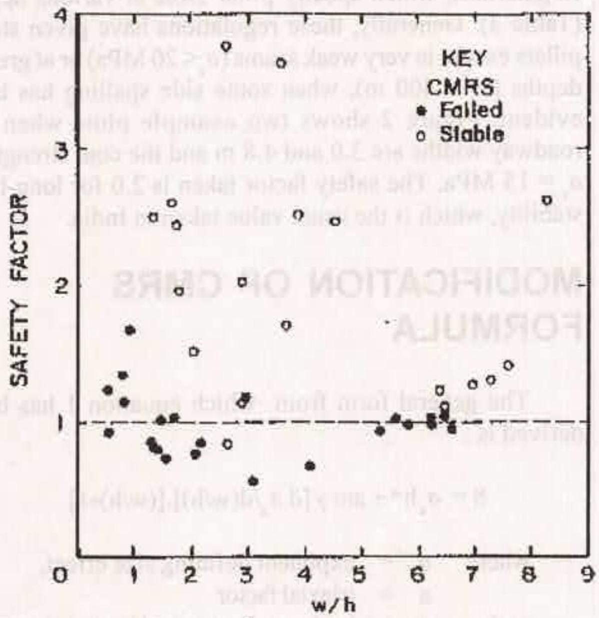

Fig. 4 : Performance of equation 6 against case studies.

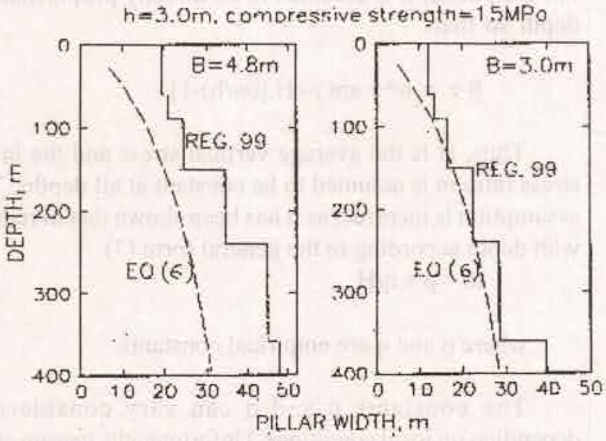

Fig. 5 : Pillar sizes according to equation 6 and Regulation 99.

$$S = 0.27\ \sigma_c h^{-0.36} + CH[0.6 + (150/H)].[(w/h)-1]\ MPa$$

The constant C was determined from the case studies of Tables 1 and 2 such that the two criteria of Salamon and Munro were satisfied approximately, as shown in Figure 4. The final pillar strength equation is obtained as :

$$S = 0.27\ \sigma_c h^{-0.36} + [(H/250) + 1] . [(w/h)-1]\ MPa \quad (6)$$

Figure 5 shows the performance of equation 6 against the pillar sizes of Regulation 99 for σ_c = 15 MPa. The improvement in the performance is apparent when compared with Figure 2.

GENERALIZATION FOR IN SITU STRESSES

Equation 6 can be generalized to include any arbitrary in situ stresses. Equation 2 can be rewritten as :

$$S = 0.27\ \sigma_c h^{-0.36} + m.\ ac\gamma.\ (\gamma H/\gamma)\ .\ [(w/h)-1]$$

or $S = 0.27\ \sigma_c h^{-0.36} + [\sigma_v/150\gamma].[p + (q/H)].[(w/h)-1]$

or $S = 0.27\ \sigma_c h^{-0.36} + [\sigma_v/3.75].[p + (q/H)].[(w/h)-1]\ MPa \quad (7)$

in which γH is replaced by the vertical stress σ_v since the two are not necessarily equal.

Equation 7 may be used for coal pillar design in unusual in situ stresses.

CONCLUSION

Equation 6 is proposed for pillar design in average coal measures. It was derived with the help of case studies, and its performance is checked against Regulation 99 of the Indian Coal Mines Regulations for the seam strength of 15 MPa, which is one of the lowest values in India. These regulations generally provide stable pillar sizes, although they are not entirely true to logic since the pillar sizes increase slowly down to the depth of 150 m, whereafter they increase rapidly.

Equation 7 is proposed for unusual in situ stresses in coalfields. The vertical stresses σ_v and the constants p and q have to be determined by stress measurement in such cases.

Equation 5, which is proposed for the inseam horizontal-to-vertical stress ratio, may be verified by stress measurement.

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DEVELOPMENT OF MOBILE DRILLING EQUIPMENT FOR MECHANIZED ROOF BOLTING OPERATIONS IN COAL MINES

Dr V. Venkateswarlu

भारत में रूफ बोल्टिंग के लोकप्रिय न हो पाने का प्रमुख कारण है पारम्परिक वेधन वेधन मशीनों से चाल में वेधन करने में आने वाली समस्याएं। उपलब्ध हाइड्रॉलिक ड्रिल अतिभार की समस्या से ग्रस्त हैं। इसे ध्यान में रखकर नेशनल इन्स्टीट्यूट आफ रॉक मैकेनिक ने विभिन्न प्रकार के हाइड्रॉलिक ड्रिलिंग-सह-बोल्टिंग यंत्रों को स्वदेशी तकनीक से विकसित किया है जिन्हें यांत्रिक खानों में आसानी से ले जाया जा सकता है। इनमें हैं - सेल्फ प्रोपेल्ड मोबाइल ड्रिलिंग मशीन, रोड हेडर माउंटेड ड्रिलिंग मास्ट और एस.डी.एल. माउंटेड ड्रिल मास्ट। इन तीनों का फील्ड परीक्षण एससीसीएल और डब्ल्यूसीएल की खानों में किया गया है। इनका व्यापक उपयोग द्रुत बोल्टिंग में सहायक है जिससे कोयला खानों की उत्पादकता में सुधार होगा।

INTRODUCTION

For improving the productivity in coal mines, it is imperative for the coal mining industry to introduce more and more mechanization. Roof bolting has been recognized as the most effective system of support. Though roof bolting has been introduced in several mines, drilling in the roof has been a major constraint for its large scale application in Indian coal mines. Particularly in mechanized workings, the progress of extraction is much below the desired level because the conventional supports and the manual bolting operations do not keep up pace with the drivage advance. Therefore, there is a need to develop indigenous equipment that would facilitate faster roof bolting operations in mechanized districts (1).

CONVENTIONAL DRILLING PRACTICES

Presently, drilling for roof bolting is done manually using hand-held electric coal drills. The workers resent drilling vertical holes in the roof with these machines as the dust falls directly on their heads. As these machines are not quite efficient, particularly in sandstone strata, the support operations are slow and time-taking. Therefore these systems do not provide immediate support close to the working face.

Imported hydraulic drills, such as the PTT-46 of UK, were tried earlier. This is a portable equipment, weighing less than 50 kg. The machine is mounted on a pusher leg. They were suitable for drilling in soft and moderately hard rocks with compressive strength up to 50 MPa. Tests were also carried out with Pegasus drill (Victor make) of UK. This is mast-mounted equipment. The drill machine moves along the vertical steel mast, and the drill could be operated from a safe distance. It had high torque of 135 N-m, and therefore, suitable for drilling in hard sandstone roofs. However, movement of the mast unit was found difficult in slushy or uneven floor, and shifting of the power pack was difficult due to its heavy weight.

Similar equipment was also developed indigenously, such as those of Shanark Co., and those fabricated by the Korea regional workshop in Chirimiri Area, SECL. However, all these systems suffer from two main drawbacks, namely that the mast unit is too heavy, and that it is very difficult to shift the power pack from one place to the other.

Light-weight, compressed air operated drill machines of Australian make are available. However, there has been a general resistance from the practising mining engineers to the introduction of this type of machines, because of the problems of introducing a new system such as the compressor and its air lines.

NIRM took the initiative to develop hydraulic drilling equipment under an S&T project funded by the Department of Coal (2). These include a mobile type, self-propelled drilling machine mounted on a jeep chassis, a system of twin-mast drilling equipment mounted on a road header,

*Scientist, National Institute of Rock Mechanics, Kolar Gold Fields

and a drill mast mounted on an SDL. These equipment are innovative in design, specifically suitable for Indian coal mining conditions (3).

MOBILE BOLTING EQUIPMENT

This is a fully self-contained and self propelled hydraulic drilling machine. In this, all the drilling components are placed on a single chassis, which is provided with traction for mobility (Fig. 1). From one standing position, it can drill four to five holes vertically in the roof, and the whole equipment can be easily taken from district to another. These features are not available in any other equipment manufactured in India or abroad.

The equipment overall height is 1.4 m, total length is 4.6 m, and the maximum width is 1.4 m. Total weight of the equipment with oil is about 800 kg. The equipment consists of a power pack, control console, and a mast unit attached to a hydraulic pusher arm, all mounted on a 4-wheel jeep chassis. A traction drive is provided for all four wheels. Hydraulic braking arrangement is provided, and the machine can cope up with slushy floors, steep gradients and underground coal mining conditions.

The electro-hydraulic **power pack** consists of an FLP electric motor, a hydraulic pump, and a hydraulic tank. To the 15/20 hp electric motor, compatible electrical starter, joint box and push button station have been connected with plug & socket joints. The motor runs the hydraulic gear pump, which in turn distributes the hydraulic oil to different hydraulic motors. High pressure hydraulic hoses with snap-on quick release couplings have been provided between the different units.

The **control panel** is located near the hydraulic tank. It consists of directional control valves for operating the pusher arm (swing arm arrangement), stelling jack, rotation motor and feed system. It also contains flow control valves for controlling the speeds of rotation and feed motors. The traction mechanism can be activated only from this control panel. A 5-core trailing cable receives power from the Gate End Box in the mine, and feeds to the FLP junction box through a plug-and-socket arrangement. The motor, the starter and the junction box are connected through a 3-core cable.

The **mast unit** is mounted on a hydraulic swing-arm arrangement ("Pusher Arm"). With a hydraulic extension of 0.7 m, it can swing in a horizontal arc of 110°. The drill mast is a structural steel unit welded with guides for travel of the rotation motor. It can be folded back on to the chassis when not in use, and can be lifted up for drilling position. Its closed height is 2.3 m, and open height 3.1 m. It can drill vertical holes in the roof, to a maximum depth of 1.65 m.

Danfoss make **hydraulic motors** have been provided for all the drilling units. The drilling operation is through a rotation motor coupled to a drill chuck, mounted on a drive sprocket and a double length chain feed. The feed motor is fixed at the bottom of the mast, and runs on chain/sprocket. Spikes are fitted at both ends for spragging the mast firmly in drilling position. The hydraulic motor of the drill rotation unit runs at 250 to 500 rpm, with a torque of 120 N-m. The equipment uses the conventional drill rods and bits.

The equipment is mounted on a 4-wheel drive **jeep chassis**, consisting of four brake hubs (2-differential, 2-axles), and clutch and pedal operated brake system. Steering

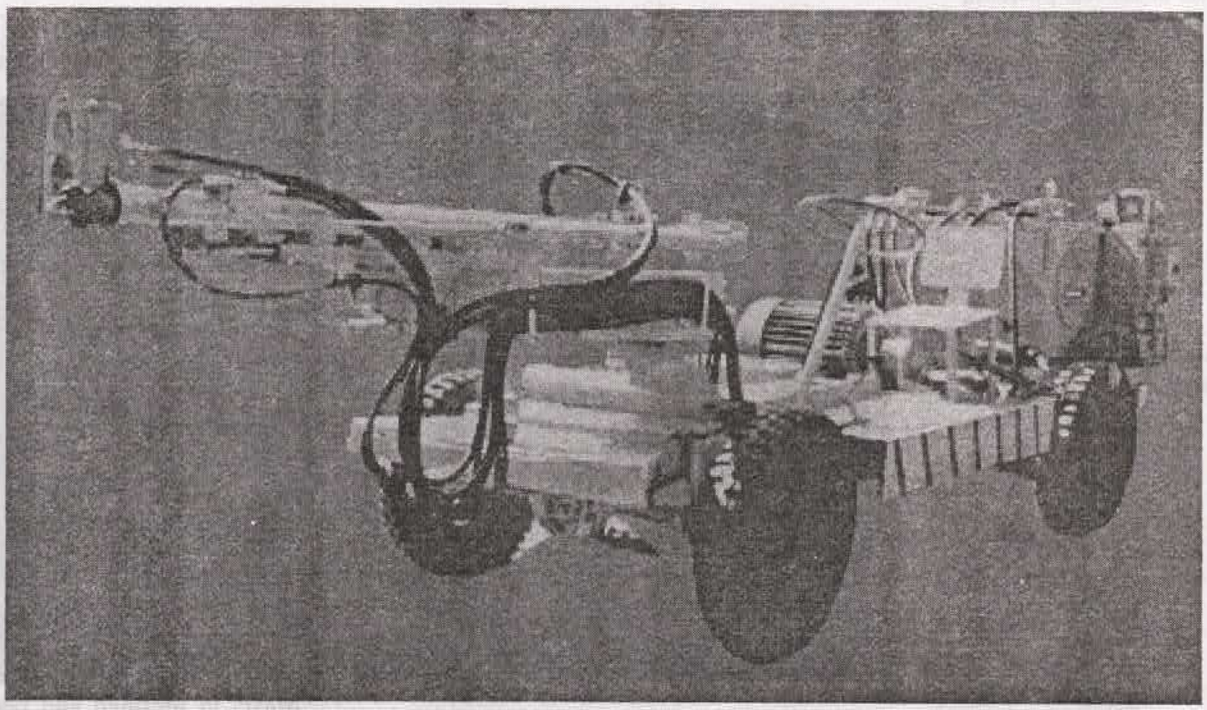
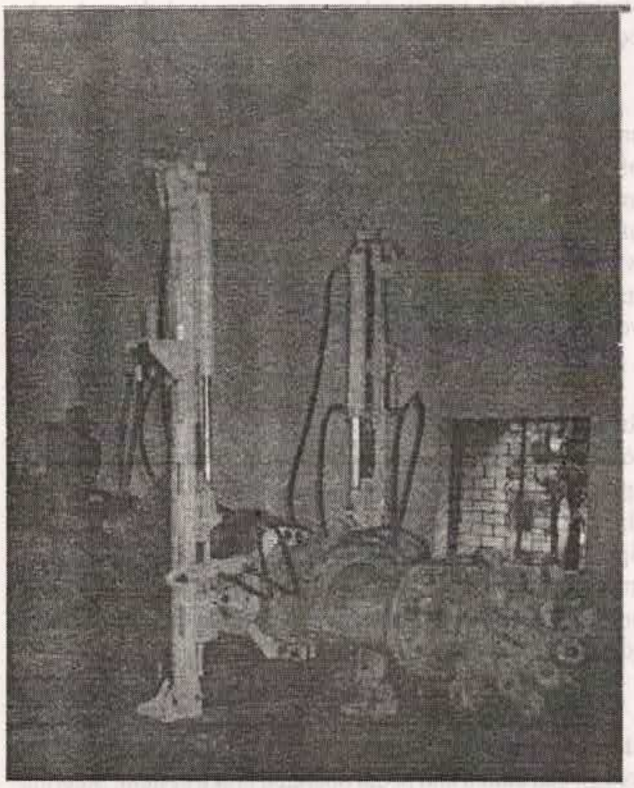


Fig. 1 : Jeep-chassis mounted mobile drilling machine

system is provided for the two front wheels. Danfoss make hydraulic motor is provided to drive the traction gear box. The drive motor is provided with an independent hydraulic braking valve. Each wheel is provided with brakes, which can be actuated from a centralized position within easy reach of the operator. Positive hydraulic brakes are applied all the while when the power pack is in operation; but the drilling units are independent of the traction mechanism.



Masts in vertical position,

The Directorate General of Mines Safety, Dhanbad, had approved all the electrical units of the equipment, and had given permission for the experimental field trials. The trials were carried out at GDK-10 incline Ramagundam division of Singareni Collieries Co Ltd. The equipment has good mobility and maneuverability, but has the imitation of working in gradients steeper than 1 in 6. The equipment has now been deployed at Saoner mine, WCL.

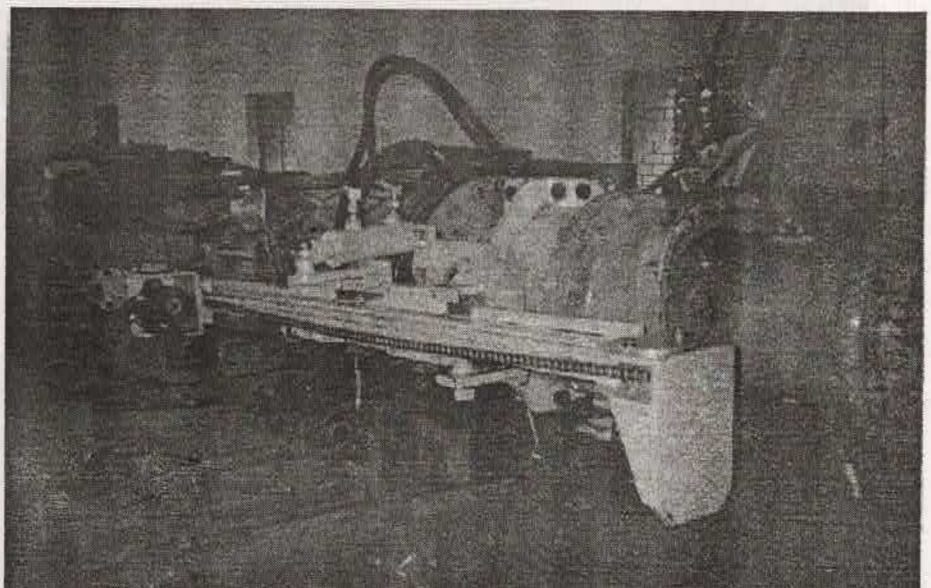
ROAD HEADER MOUNTED BOLTING EQUIPMENT

In longwall gate road drivages, bolting operations using the hand-held electric coal drills do not match the fast rate of cutting by the road heading machines. Special type road headers fitted with a separate boom for drilling are available in world market, but they are very expensive. In Australia, individual hydraulic drilling units are mounted on either side of the road header. This type arrangement was successful in achieving high bolt installation rates.

Similar type of road header mounted drilling arrangement has been developed in India too. The equipment consists basically of two hydraulic drilling masts, along with a bracket type clamping device. With the help of the clamping device, the two masts are mounted on either side of the cutter boom of a road header. Additional hydraulic jacks are provided for positioning of the masts in a line by tilting the masts for drilling inclined holes on both sides of the boom (Fig. 2). With this arrangement, 4 to 6 bolts can be installed from one fixed position of the road header boom. The hydraulic power required for the drilling operations is drawn from the road header itself.

Each drill mast operates independently. The mast length is 2.2 m, and it has a slide of 0.6 m. When placed in

Fig. 2 :Road header mounted drill masts;



Masts in inclined and resting positions

the normal position, they can drill vertical holes at 1 to 1.2 m apart. The masts can swing out through an arc of 45° for drilling inclined holes in the same line. When not in use, the masts can be folded alongside the boom.

The control panel consists of a manifold block with direction control valves for rotation motor, feed motor, tilt cylinder and jack-up cylinder functions. It is equipped with a selector valve which diverts the operational functions to either Mast-1 or to Mast-2 via the two logic cartridge blocks. When placed in the neutral position, the flow from the main manifold is diverted to the road header functions.

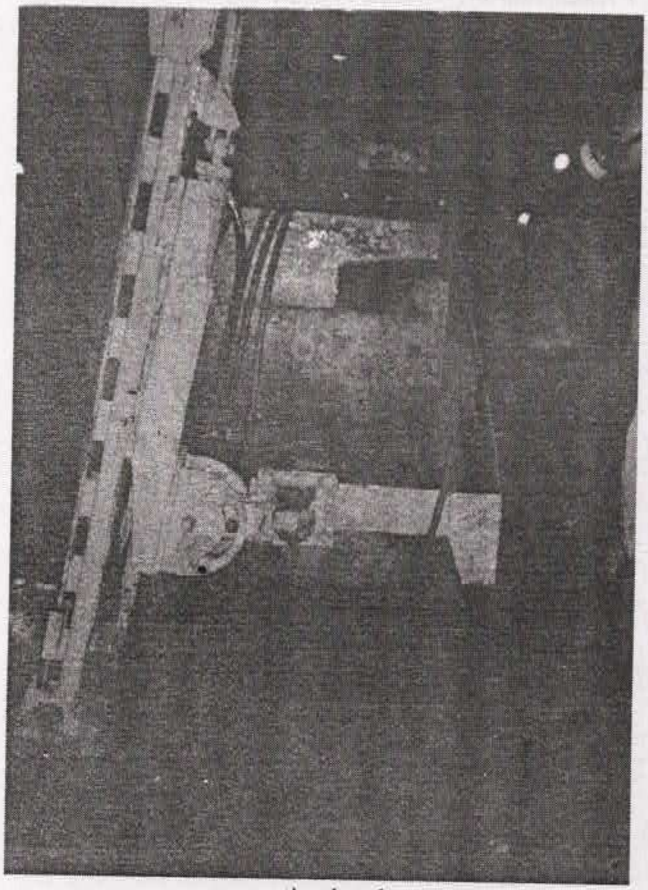
The hydraulic power is tapped at the chain conveyor function of the road header. It is fed to a main manifold block consisting of a logic cartridge valve, a pressure relief valve and two flow control valves. They in turn effect the drill mast functions and then feed back to the two return lines. All the hydraulic hoses are neatly tucked at appropriate places. To avoid clustering of the hoses, steel lines have been used on the boom along its length.

This type of twin mast arrangement on the road header would increase the speed of roof bolting in the longwall gate roads, and thus help in faster roadway development. This equipment has been mounted on a Dosco LH-100 road header. The equipment was demonstrated at GDK 10 A incline of Ramagundam division, SCCL. However, it was observed that the drill masts were coming in the way of the cutting operations of the road header boom. Further modifications are necessary in this system.

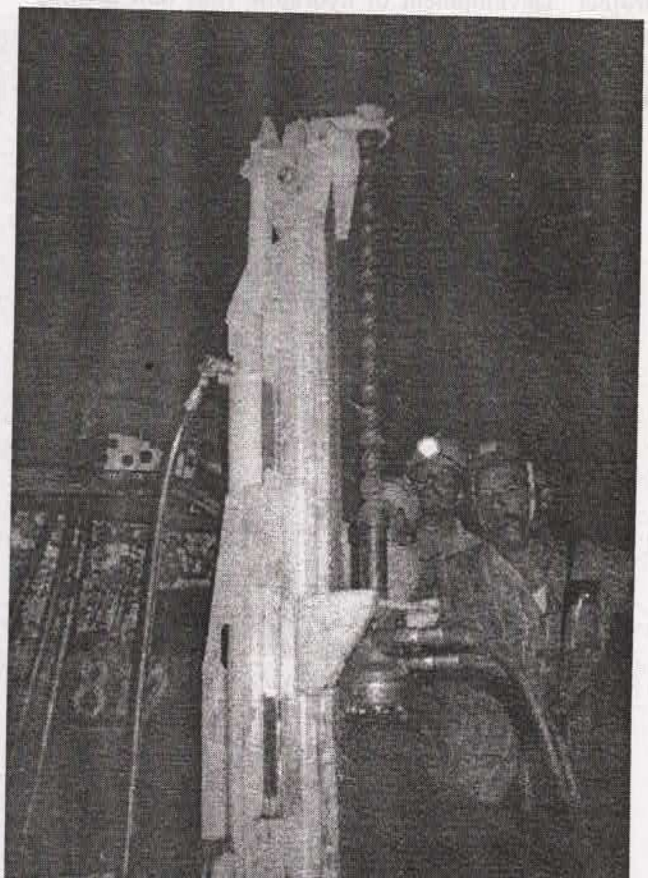
SDL MOUNTED BOLTING EQUIPMENT

The productivity in mechanized depillaring districts could be further improved by the use of hydraulic roof bolting equipment for drilling and fixing the bolts. Such systems could be directly mounted on the SDLs/LHDs, as there would be sufficient idle time available for these equipment. Accordingly, a hydraulic drill mast unit compatible with an SDL has been developed (Fig. 3). There is a clamping arrangement for fixing the mast unit on the SDL, and there is also an arrangement for swinging it in and out.

The mast unit is provided with a jacking arrangement, and drill and rotation motors. Danfoss motors have been provided for all the hydraulic operations. This equipment has been fitted on to an Eimco-Elecon SDL-625 at the rear side (front side is for the bucket loading operations). It draws the required hydraulic power from the additional banks provided on the SDL. The equipment underwent field



mounting bracket



drilling in progress

Fig. 3 : SDL mounted drilling equipment;

trials first at Bandewara colliery in Rajur Sub-area of Wani Area, WCL, and later at Ballarpur 3&4 Pits, Ballarpur Area, WCL.

CONCLUSIONS

All the above hydraulically operated roof bolting equipment have been developed indigenously to the specific needs of the industry. These equipment fulfill a long felt need for a mechanized system of drilling for faster and easier roof bolting operations. The different types of equipment cater to the support needs of different mining situations, such as in depillaring and other extraction areas where mechanized loading is being practised and in longwall gate roads.

It is hoped that the successful application of these systems would enable faster support operations in mechanized workings, and increase the productivity form underground coal mines. NIRM had taken the initiative and developed the above proto-types. The industry may now follow up, and further develop the items as required by them.

ACKNOWLEDGEMENT

The above work was carried out as part of an S&T project "Development of hydraulic roof bolt drilling equipment, and drilling accessories for mechanized bord and pillar workings and fast roadway drivages", funded by the Department of Coal, Govt of India. The officers of CMPDI, Ranchi, carefully monitored the progress of the project. The main idea behind the development of the items is of Dr NM Raju, ex-Director, NIRM. M/s Arrow Metalspin Ltd, with the active participation of their Manager, Mr RJ Bastian, fabricated these equipment. The management of GDK-10 and GDK-10A inclines, the General Manager and other officers of RG-2 Area, SCCL, and the management and E&M engineers of Ballarpur 3&4 Pits, and of Rajur colliery (Wani Area), WCL, extended us the facilities to carry out the field investigations.

I thank all the above for their contributions to the success of this project. I also thank our Director, Prof RN Gupta, for his permission to publish this work.

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DEVELOPMENT OF MODELS FOR THE PREDICTION OF SINGLE AND MULTI-SEAM SUBSIDENCE

P. R. Sheorey, J. P. Loui, K. B. Singh and S. K. Singh*

इस आलेख में एकल एवं बहुल कोयला संस्तरों से निष्कर्षण वाले क्षेत्रों के ऊपर सम्पूर्ण भू-धसान के अनुमान के लिए विकसित विभिन्न मॉडलों को प्रस्तुत किया गया है। ऐसा पाया गया है कि अभंजित और अर्धसित शैल (एकल सीम) और पहले से उपस्थित धंसित गोफ (बहुल सीम) के नीचे से निष्कर्षण में अन्तर होता है और ढलान व तनाव भी अधिक होता है। फलस्वरूप खनन की इन दो स्थितियों के लिए अलग-अलग समीकरण और विनिर्देश दिए गए हैं। यह देखते हुए कि असतत् भू-धसान से अधिक नुक्सान होता है, इसके लिए सीमा भी निधारित की गई है।

एकल और बहुल सीम के लिए वास्तविक स्थितियों में अध्ययन किया गया है और प्राप्त परिणामों की विवेचना की गई है।

INTRODUCTION

In various Indian coalfields, the undergound coal mining has been carried out mostly at shallow or moderate depths. Problems due to surface subsidence have been a common feature in all the coalfields of India. Prior to any mining activity undergound, it is necessary to estimate the resulting surface subsidence and strain values which may cause damages to surface structures. Under shallow covers, discontinuous subsidence has also been found to take place which can cause more devastating surface damage.

In most of the coalfields multiple seams are extracted, and at times, extraction takes place below overlying goaf(s). The subsidence behaviour is found to be different when the extraction takes place below fractured rock.

Bord and pillar mining which is the principal method of extraction of coal in India mostly results in irregular panels. The basic terms used in subsidence engineering like width-to-depth ratio, critical width, subsidence profile, angle of draw etc. assume a plane strain condition and are thus difficult to correlate with an irregular extraction panel geometry. Therefore, a three dimensional and more comprehensive subsidence prediction model has been developed using influence function method. The influence function method has been proved to be more versatile while dealing with irregular workings and, with the use of a computer, this method can be applied to a gamut of underground mining situations.

This paper describes the development of norms to predict the important parameters which define surface subsidence. Also explained is the development of a three dimensional subsidence prediction tool using the influence function method.

DISCONTINUOUS SUBSIDENCE

Subsidence is termed as discontinuous when it occurs in shear steps and when the subsidence trough or profile cannot be represented by a continuous function.

Nomenclature

a	subsidence factor
A_o	area of influence circle
d	distance of extraction element centroid from starting face line
d_b	distance of extraction element to nearest boundary
d_{max}	distance between starting and ending face line
e	extraction expressed as percentage or fraction)
$E_{(-)}$	Maximum compressive strain
$E_{(+)}$	Maximum tensile strain
G	Maximum slope
H	depth of cover
K_1, K_2, K_3	constants in equations for maximum slope, compressive strain and tensile strain respectively
K_z	influence function for subsidence
L	width of panel
NEW	non-effective width-to-depth ratio
r	radial distance of sector centroid from cen ter of influence circle
R	radius of influence circle
S	maximum subsidence
S_{max}	maximum possible subsidence (when the panel width is critical or super critical)
T	parting thickness between seems

*Central Mine Research Institute, Barwa Road, Dhanbad - 826 001, Bihar, India.

The surface damage due to this type of subsidence is particularly severe. Depth of cover, working height and percentage of extraction have been identified as the three major parameters which may decide the nature of subsidence (continuous or discontinuous). A line of demarcation between continuous and discontinuous subsidence cases is shown in Figure 1. This line has the equation (depth H and working height h in meters and extraction e in percent)

$$H/h.e = 0.3$$

which covers both single and multi-seam mining conditions. The condition for discontinuous subsidence to occur is thus

$$H/h.e < 0.3 \tag{1}$$

NON-EFFECTIVE WIDTH

The non-effective width is the maximum width of extraction upto which no significant symptom of subsidence occurs at the surface and is normally expressed as a ratio of this maximum width of extraction to the depth of cover. The non-effecive width to depth ratio is abbreviated as *NEW*.

For single seam cases, *NEW* has been found to vary between 0.3 and 0.8 depending on the strata competence. However, in multi-seam cases the *NEW* is found to vary between 0.23 and 0.37.

Several attempts were made to correlate *NEW* with strata characteristics but in the absence of complete geomechanical data for the entire strata sections, all such attempts led to poor correlations which were not statistically significant. Consequently, *NEW* was assigned only approximate values in different parts of the country depending on observation and experience.

SUBSIDENCE FACTOR

In order to eliminate the variation in *NEW* the maximum subsidence has been plotted versus $x = L/H/NEW$ as shown in Figures 2 and 3 for single and multi-seam extractions. Only those cases in which the *NEW* was actually measured have been selected for these two plots. A hyperbolic tangent function has been chosen and the general equation which would fit these plots has been chosen in the form

$$S = c_1 .h.e\left[1 - \frac{\tanh c_2 (x - c_3)}{\tanh c_2 (1 - c_3)}\right] \tag{2}$$

where c_1, c_2 and c_3 are constants. During this exercise

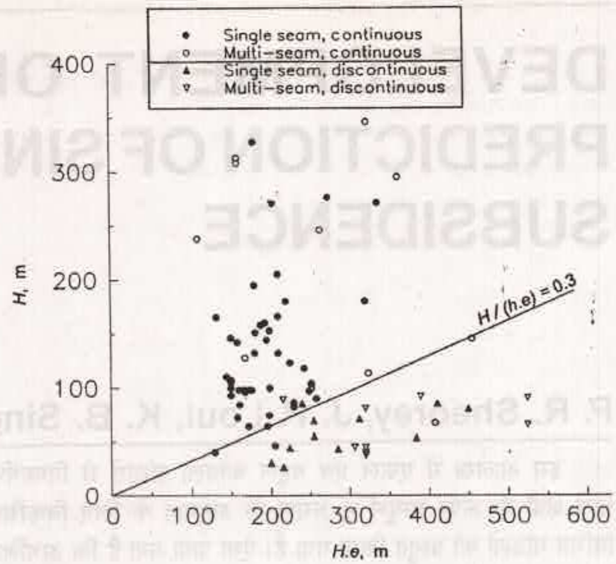

Fig. 1 : Condition defining discontinuous subsidence

it was seen that several cases from central India (SECL mines) gave the subsidence factor ($S_{max}/(h.e)$) as 0.3-0.35. Such unusually low values have not been reported from anywhere in the world. In addition, while fitting the influence function model to many single seam cases, higher values of the subsidence factor were really required. These cases of low subsidence factor, therefore, had to be regretfully rejected until such time that the reasons for such low subsidence could be established. Also eliminated from these plots are the cases of discontinuous subsidence.

It may be seen from Figures 2 and 3 that the subsidence factor increases from 0.69 for single seam cases to 0.81 for multi-seam cases.

Equation (2) has been non-linearly regressed employing the "search method" of solution which would satisfy the law of least squares. This led to the following two relations

single seam cases

$$S = 0.33[1 + 1.1\tanh(1.4(x - 1.8))] \tag{3}$$

multi-seam cases

$$S \doteq 0.12[1 + 5.9\tanh(0.38(x - 1.45))] \tag{4}$$

A word of caution regarding equation (4) is necessary. It may be noted that in multi-seam bord and pillar extractions the overlying goaf(s) may or may not overlap the extraction panel and also can have widely different and irregular shapes. Equation (4) therefore is only approximate and it will be always prudent to use the influence function method for assessing the complete subsidence in a given multi-seam situation.

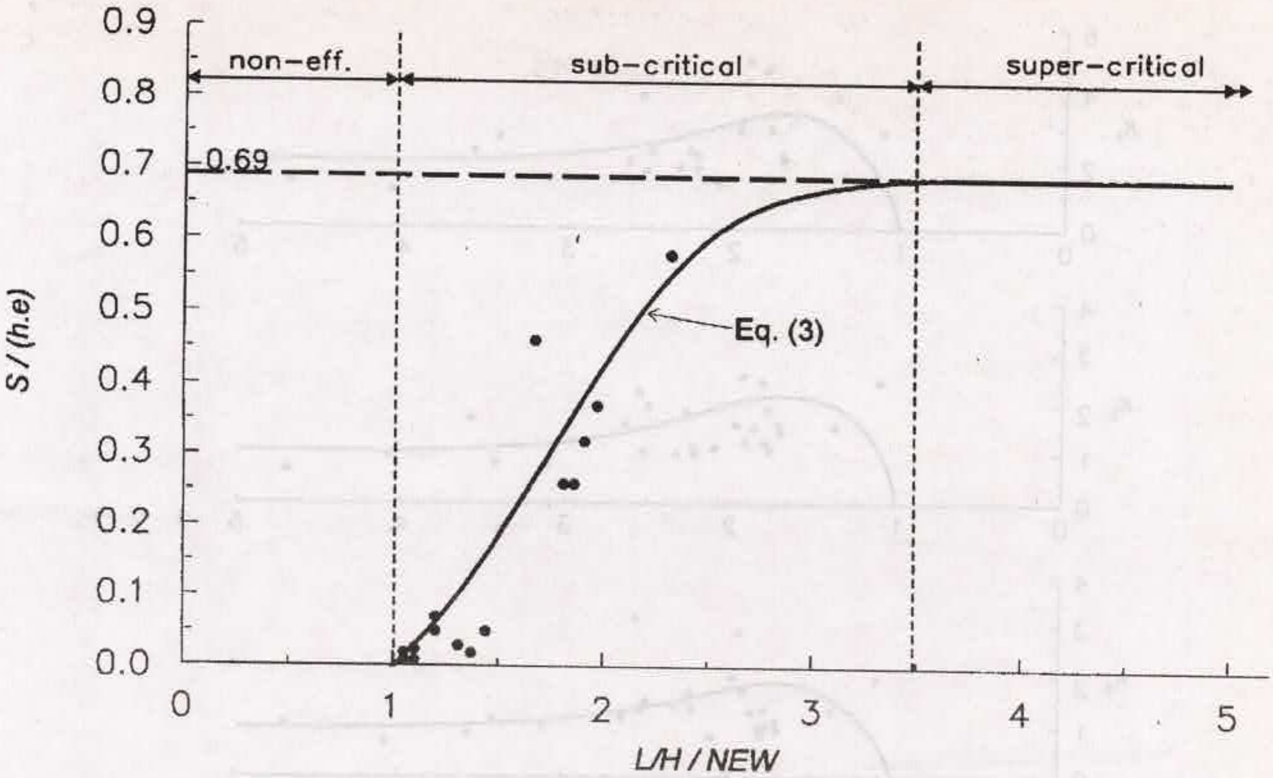

Fig. 2 : Maximum subsidence for single seam cases

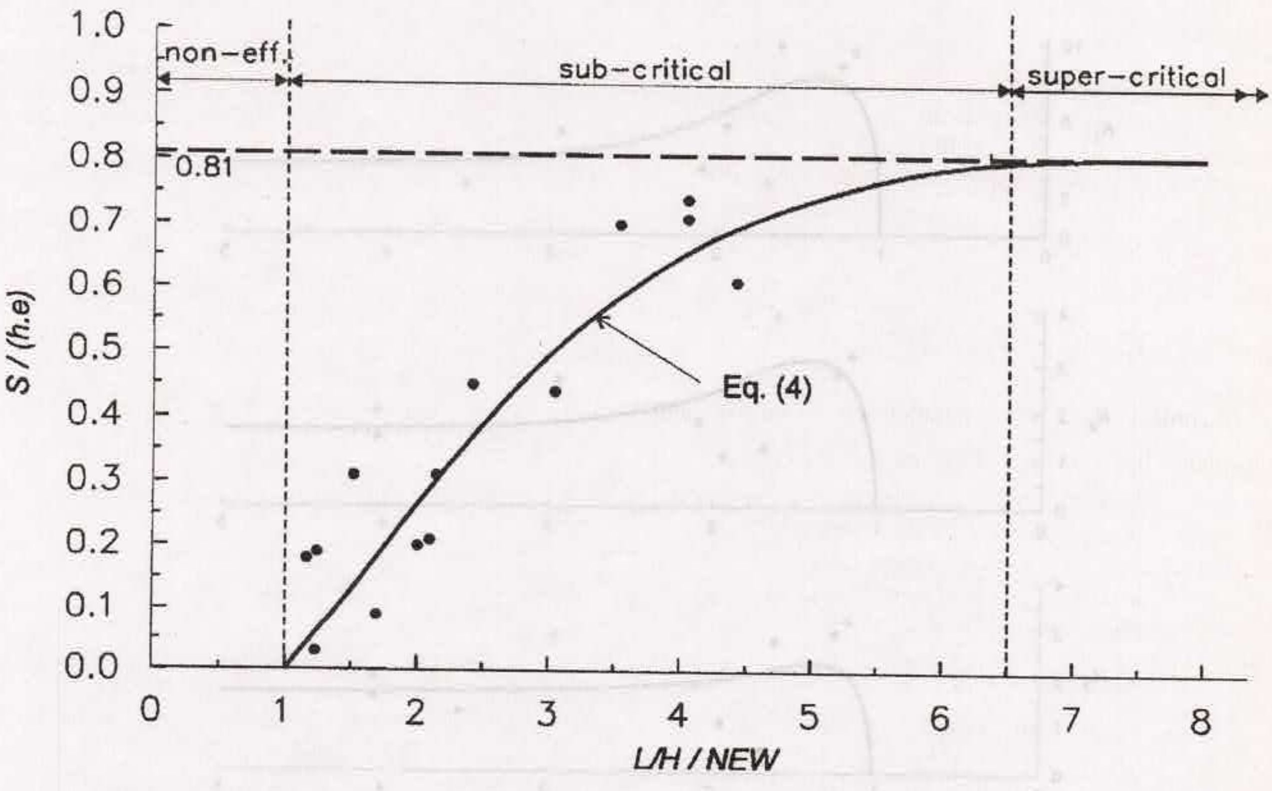

Fig. 3 : Maximum subsidence for multi-seam cases

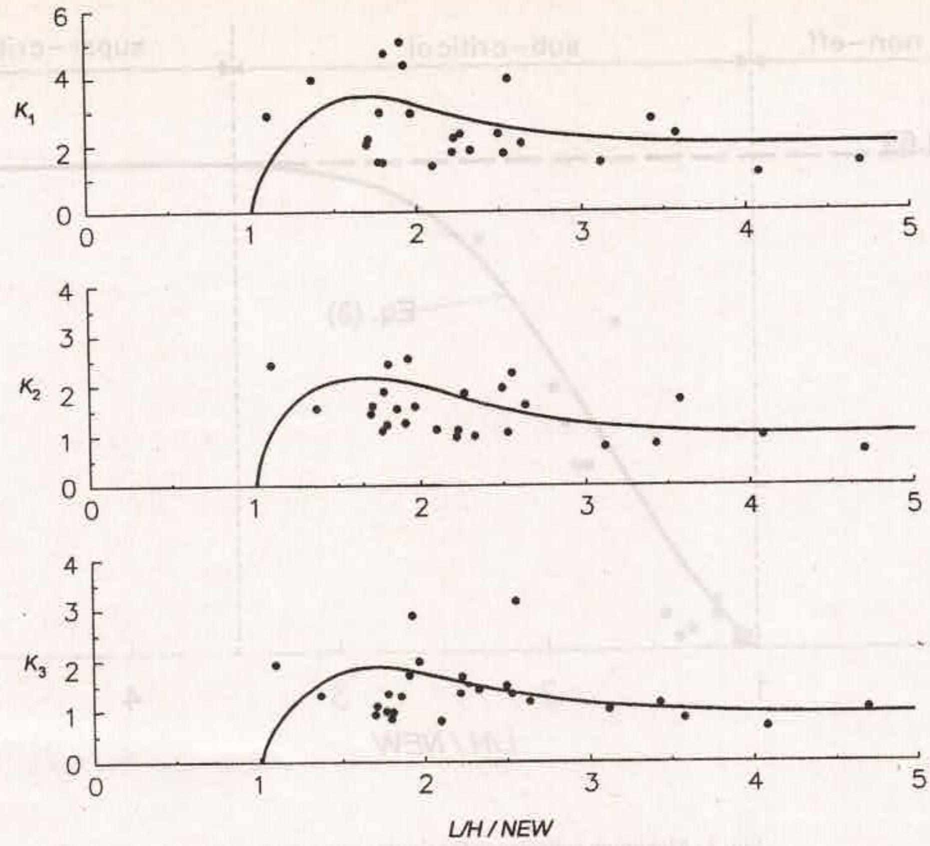

Fig. 4 : Reference graph for estimation of K_1, K_2, K_3 for single seam cases

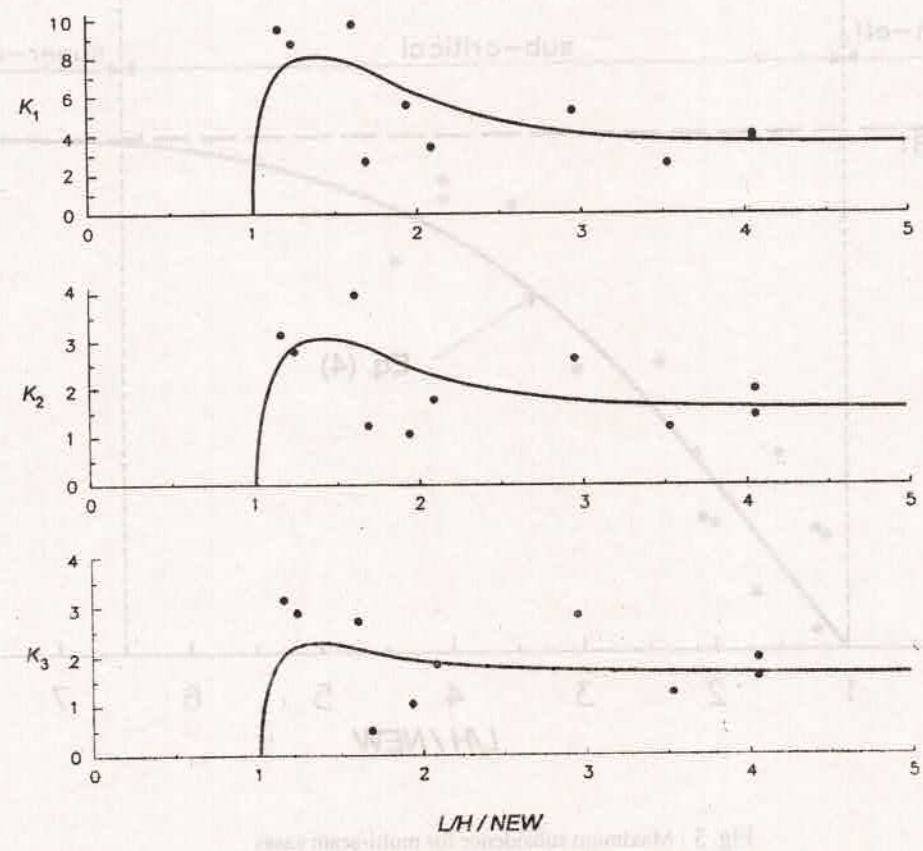

Fig. 5 : Reference graph for estimation of K_1, K_2, K_3 for multi-seam cases

The critical width of extraction is found to be $3.5 \times NEW$ in single seam cases while in multi-seam cases it is found to be 6.5-7 times the *NEW*. This clearly happens because of a fall in the *NEW* value in multi-seam cases.

MAXIMUM SLOPE AND STRAINS

Standard relationships for maximum slope and strains in relation to the maximum subsidence are given in the Subsidence Engineers' Handbook[1] which are as follows.

$$G = K_1 \frac{S}{H}$$

$$E_{(-)} = K_2 \frac{S}{H}$$

$$E_{(+)} = K_3 \frac{S}{H}$$

From these relations the constants K_1, K_2 and K_3 can be calculated from subsidence, slope and strain measurements at the surface. Figures 4 and 5 are plotted with the variable *L/H/NEW* as the abscissa. The two plots are given separately for single and multi-seam cases. In Figures 4 and 5 the curves are hand drawn and have not been obtained as best fits from regression analysis.

A MODIFIED INFLUENCE FUNCTION METHOD

Of all the different tools available for subsidence modelling the influence function method with suitable modifications appears to be a powerful method for complete subsidence prediction for all shapes of extraction panels. This method, in its classical form, consists of laying a surface grid of square elements overlapping the panel up to the draw limits. An influence circle, subdivided into a sufficient number of rings and sectors, gives the subsidence at the grid point which is the sum of the weighting factors of all the sectors falling within the extracted area. The weighting factors are derived from an axisymmetric influence function of bell shape covering the influence circle. The classical method, however, requires some modifications to suit the observed subsidence behaviour.

Modification of the Method

The following influence function

$$K_z = \frac{0.5352}{R^2}\left(1 + \cos\frac{\pi \cdot r}{R}\right) \qquad (5)$$

was found to be suitable while simulating surface subsidence. An influence function can have various forms, since the method is really empirical and the choice of a function is arbitrary, but it must satisfy the condition

$$\iint_{A_o} K_z dA = 1 \qquad (6)$$

where A_o is the area of the influence circle.

The conventional method as described earlier gives rise to much higher subsidence at the edges of a panel, as reported earlier [2,3]. This was also experienced by the authors of this paper. Ren *et al.* [2] have suggested drawing an artificial boundary some distance inside the panel on all sides and treating this new boundary as the extraction panel. Liao [3] has suggested fuzzifying the influence function closer to the panel edges.

Besides the correction to the influence function for panel edges, another correction required was to account for the asymmetry of the subsidence trough which is invariably found in Indian conditions. The point of maximum subsidence always occurs closer to the start line of the panel.

The correction function (weighting function) has been suitably designed as

$$W_z = 0.5 \tanh\left(\frac{5d_b}{1.5NEWh} - 2.4\right) + 0.5 \qquad (7)$$

where d_b is the distance of the extraction element to the nearest boundary of the extracted area.

The correction function for asymmetry has been designed as follows

$$Q_z = 0.9 - 0.1 \tanh\left[0.5\left(d - 0.4d_{\max}\right)\right] \qquad (8)$$

where d is the distance of the extraction element centroid from the starting face line and d_{max} is the distance between starting and ending face line.

We thus have the modified influence function using equations (7) and (8) as

$$dV_z^{/} = Q_z W_z dV_z$$

Influence of Overlying Goaf

As mentioned earlier, overlying goaf(s) influence the surface subsidence, changing the subsidence factor as well as the maximum slope and strains. After some hit and miss

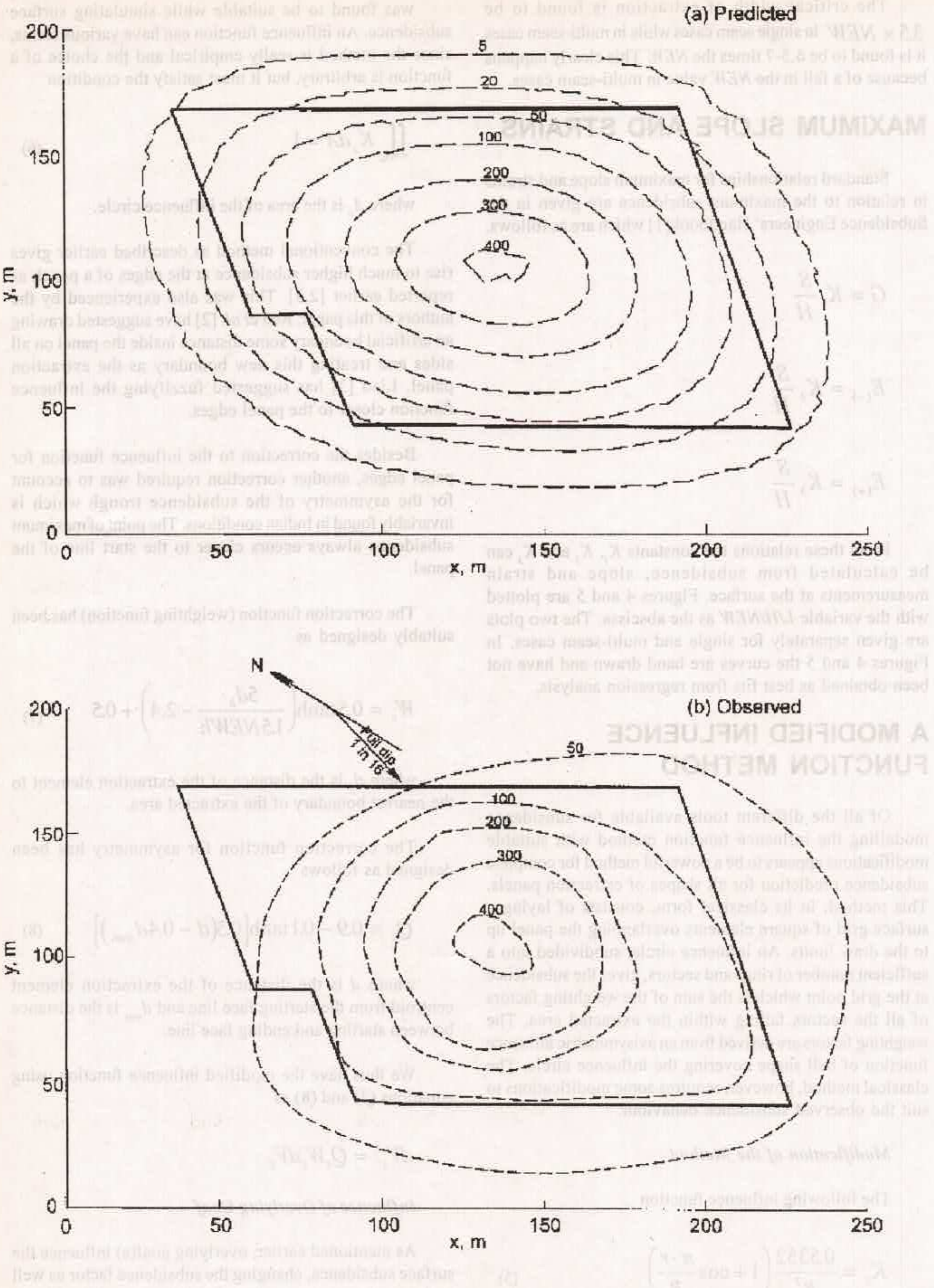

Fig. 6 : Predicted and observed subsidence contours at Jaykay Nagar colliery, ECL (single seam case)

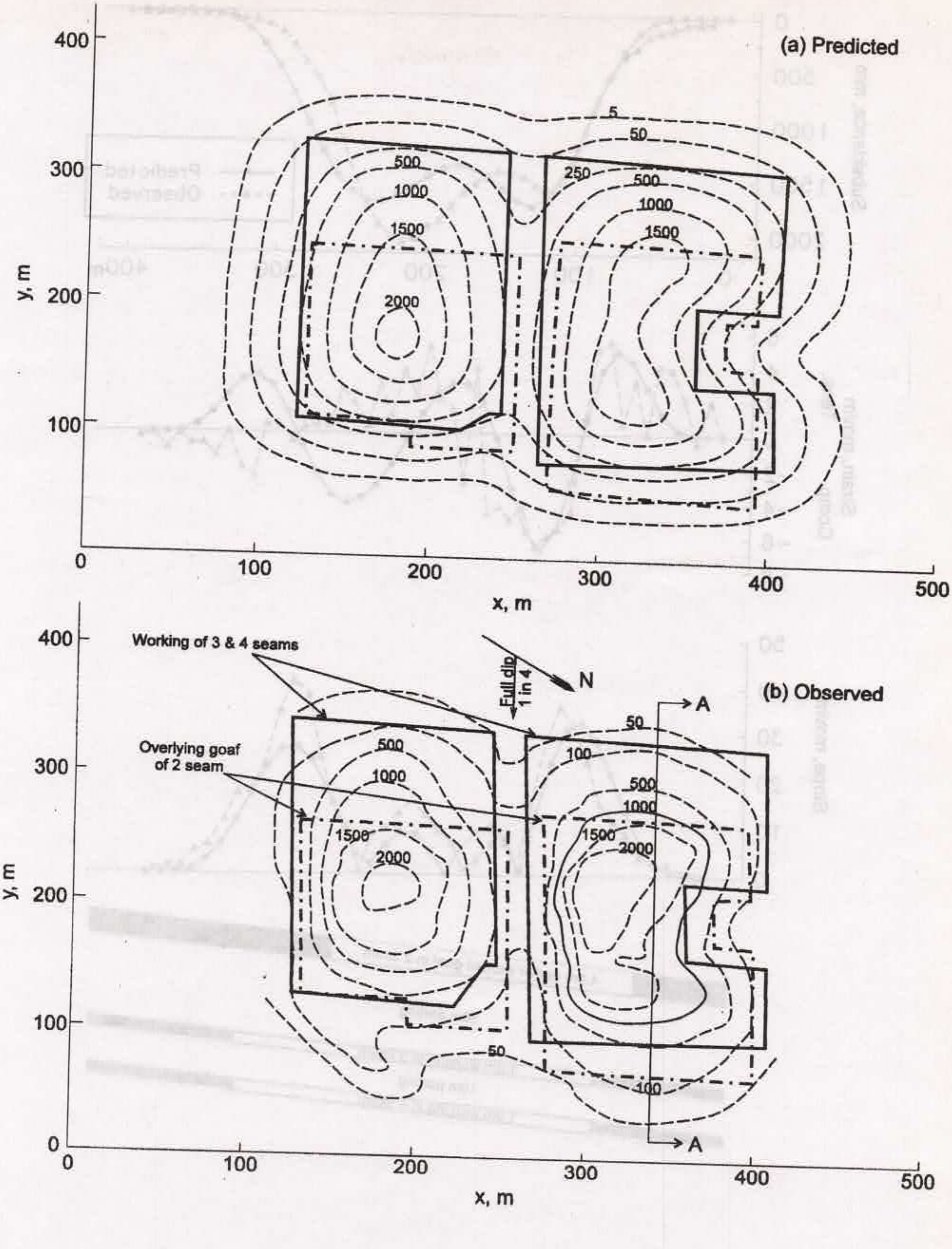

Fig. 7 : Predicted and observed subsidence contours at RK-8 incline, SCCL (multi-seam case)

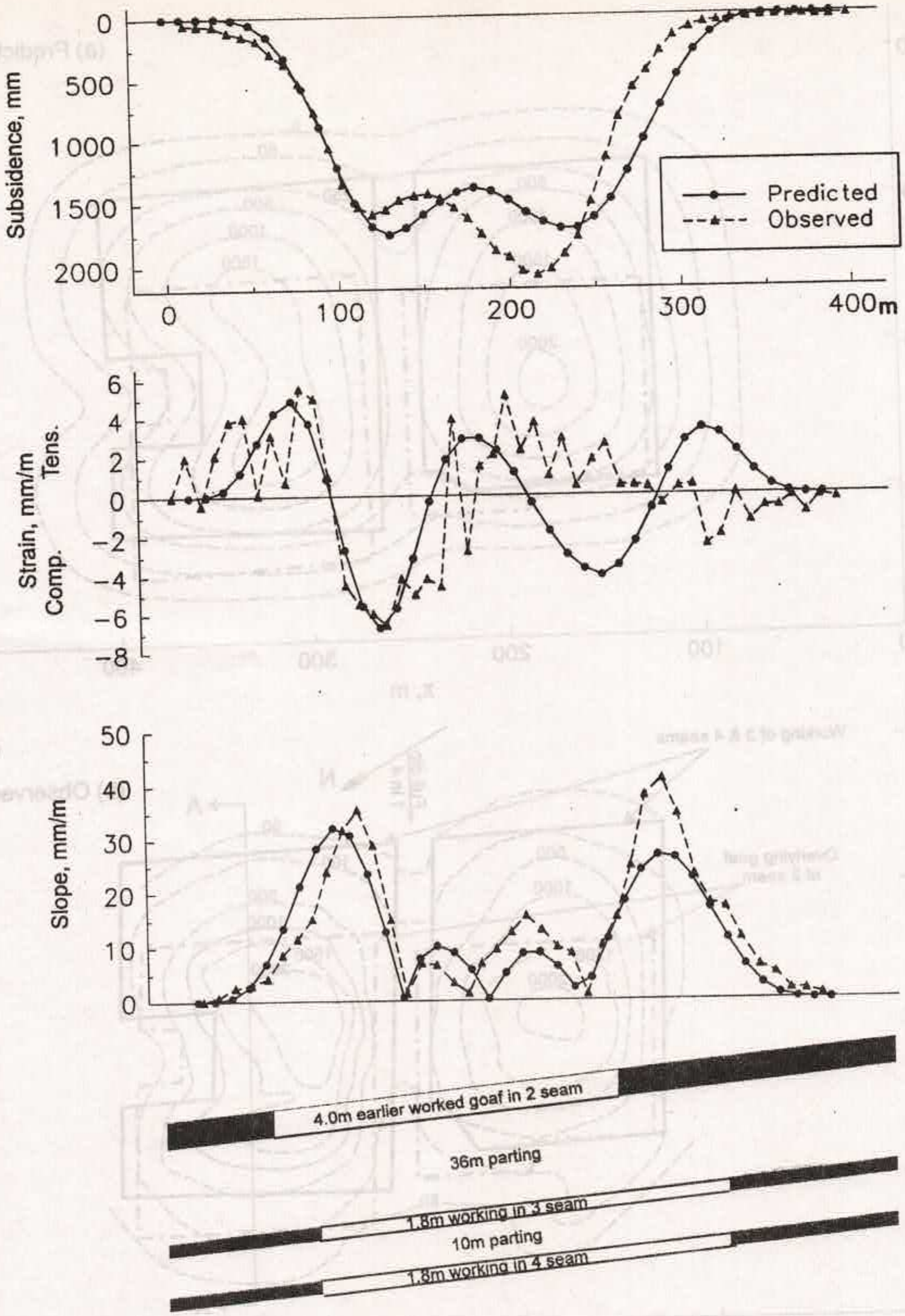

Fig. 8 : Subsidence, slope and strain profiles along section A-A in Figure 7 at RK-8 incline (multi-seam case)

trials it was found that a 40% reduction in the original *NEW* value along with a 17% increase in the subsidence factor enables prediction of multi-seam subsidence to a reasonable accuracy. Since the overlying goaf may or may not completely overlap the extraction panel, these changes in *NEW* and subsidence factor would be applicable only for those elements falling within the goaf area. Moreover, since the parting between the extraction panel and the overlying goaf is intact, the final evaluation is done by taking a weighted average as

$$NEW' = \frac{0.6NEW(H-T) + NEW(T)}{H} \quad (9)$$

$$a' = \frac{1.17a(H-T) + a(T)}{H}$$

where T is the parting between the extraction and overlying goaf.

COMPARISON BETWEEN PREDICTIONS AND ACTUAL OBSERVATIONS

Subsidence Contours

A computer program has been developed for this modified influence function method both for single and multi-seam extraction. To check the validity of the method 8 single seam cases and 5 multi-seam cases were taken. The comparison between the predicted and observed subsidence values was found to be satisfactory. Only one single seam case and one multi-seam case are shown as examples in this paper. For sufficient clarity, the predicted and observed values have been compared as contours on panel plans.

Jaykay Nagar (single seam case): Plan of the bord and pillar panel worked out at Jaykay Nagar colliery, ECL is shown in Figure 6. This panel occur at an average depth of 141m. The height of extraction for this panel was 2.1m, the extraction being 75%. Field observations gave a non-effective width-to-depth ratio of 0.48 and an angle of draw of 17°. The predicted and observed subsidence contours are presented in Figures 6a and 6b respectively

RK-8 incline (multi-seam case): Two bord and pillar panels were extracted side by side as shown in Figure 7. The simultaneous workings of 3 and 4 seams in these two panels are at an average depth of 100m from the surface. The overlying goafs in 2 seam occur at a parting of 35.6m from the 3 seam and their boundary is shown in Figure 7. 3 and 4 seams were simultaneously worked out to a combined height of 3.6m with an extraction of 90%. The observed angle of draw is 25° and *NEW* is 0.51. Figure 7a and 7b show the predicted and observed subsidence contours respectively for this case.

Slope and Strain

One case for multi-seam subsidence (RK-8 incline, SCCL) illustrates the comparison between observed slope and strain variation as shown in Figure 8. The subsidence, strain and slope profile is plotted along the section A-A marked in Figure 7.

DISCUSSION AND CONCLUSION

The line demarcating the limit of discontinuous subsidence has been given for single seam as well as multi-seam cases, but it is felt that the thickness of unfractured rock in relation to the total depth in multi-seam cases will probably make some difference to this limit.

A definite increase in the subsidence factor and also slope and strains is noticeable while extraction below goaf(s) is carried out.

Norms for assessing the maximum slope and strains for single seam and multi-seam subsidence have been established separately.

The influence function method with suitable modifications, as mentioned in this paper, for subsidence asymmetry, edge effect and overlying goaf(s) is found to be reasonable for all panel shapes and sizes.

The influence function method as described in this paper is for horizontal or near horizontal seams and when the gradient starts influencing subsidence, the predictions do not agree very well with the observations. As an example, this can be seen from the case of RK-8 incline (Figure 6b) where the gradient is 1 in 4. The authors intend to further the work and develop a suitable model for inclined seams.

The occurrence of unusually low subsidence in some of the SECL mines cannot be explained properly and a detailed investigation for this specific area may be necessary. Such low subsidence may be related to several causative factors such as bulking factor upon caving, *in situ* stress field *etc*. Some studies may be done in the future to identify the factors responsible for such low subsidence.

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DEVELOPMENT OF A MECHANISED SPRAYING DEVICE FOR SPRAYING FIRE PROTECTIVE COATING MAERIAL IN THE COAL BENCHES OF OPENCAST MINES FOR PREVENTING SPONTANEOUS COMBUSTION

R. V. K. Singh* and V. K. Singh*

कोयला खानों में आग लगने का मुख्य कारण कोयले का स्वत: दहन है। कोयले में आग, विशेषकर खुली मुख खदानों में लगी आग, से न केवल राष्ट्रीय सम्पति की अपूरणीय क्षति होती है वरन् सतह पर संरचनाओं को भी नुकसान पहुँचता है एवं पर्यावरण भी प्रदूषित होता है। खुली मुख खानों की कोयला बेन्चों में अग्नि/स्वत: दहन की समस्या बहुत गंभीर है। वर्तमान समय में भारतीय खानों से प्राप्त कोयले के कुल उत्पादन का 75 प्रतिशत कोयला खुली मुख खदानों से प्राप्त किया जा रहा है। तदनुसार खुली मुख खदानों की कोयला बेन्चों में स्वत: दहन को रोकने के लिए सेन्ट्रल माइन रिसर्च इन्स्टीच्यूट, धनबाद तथा मेसर्स सिग्नुम फायर प्रोटक्सन (इण्डिया) लि. नागपुर ने भारत सरकार के कोयला मंत्रालय द्वारा वित्त पोषित विज्ञान एवं प्रौद्योगिकी परियोजना के अन्तर्गत संयुक्त रूप से अग्नि विरोधी विलेपन सामग्री के छिड़काव उपकरण को विकसित किया है। इस आलेख का उद्देश्य यंत्रीकृत छिड़काव उपकरण तथा खुली मुख कोयला खदानों की बेन्चों में अग्निरोधी विलेपन सामग्री के छिड़काव में इस उपकरण के प्रयोग के बारे में विस्तृत जानकारी देना है।

INTRODUCTION

The problem of spontaneous combustion/heating is very common in coal mines of different countries. Coal industry is facing a lot of problems due to occurrence of spontaneous combustion in the mines. The exact mechanism of spontaneous combustion of coal is not yet fully established. However, most of the researchers agree that reaction of oxygen with coal is a surface phenomena which is believed to start at -80 ^{0}C, as soon as the coal is subjected to atmosphere. Every coal has its own incubation period. Coal gets oxidised within the incubation period. It has been found that about *75%* of fires occur in coal mines due to spontaneous combustion. At present over *75%* of the production of coal is being carried out by open cast mining and most of the coal seams are thick and highly susceptible to spontaneous combustion.

In open cast mines, the main factors attributable to bench fires due to spontaneous combustion are:

- presence of micro and macro cracks in the bench walls which provide air entry within,
- long exposure of the bench walls to the open atmosphere,
- accumulation of loose coal at the bench floor.

After exposure of the coal benches of open cast mines, it is essential to apply suitable fire protective coating material over the coal surface to cut off the contact of air with the coal surface for preventing spontaneous combustion. With this idea in view, a suitable fire protective coating material and a small spraying system were developed earlier under S & T project funded by Ministry of Coal, Govt. of India [1] and [2]. The earlier developed spraying system was successfully utilised during field application in Karkatta OCP, Dakra (N.K. Area, CCL) and Jagannath OCP, Talcher (Jagannath Area, MCL [3] and Jhingurdah OCP, Singrauli, NCL [4] for spraying the fire protective coating in the benches of open cast coal mines. But the capacity of spraying the material was maximum up to 10 m and discharge of the material was *5 litre/mm.* As that small spraying system was in-effective for height more than 10 m but in actual practice some of the benches are more than 10 m high, where we feel difficulty for spraying the fire protective coating and we found that the uncovered portion more than 10 m high caught fire within the incubation period. Thus it is felt that there is an urgent need for development of a mechanised spraying device for spraying the fire protective coating material to cover about 20 meters height in large open cast mines so as to prevent spontaneous

* Scientists, Mine Fire Section, Central Mining Research Institute, Barwa Road, Dhanbad - 826 001, Bihar, India.

combustion. Accordingly, for fulfillment of the need of the mining industry a mechanised spraying device has been developed for spraying the fire protective coating material for industrial application in the coal benches of large opencast mines jointly by CMRI, Dhanbad and M/s Signum Fire Protection (India) Pvt. Ltd., Nagpur under S & T grant project funded by Ministry of Coal Govt. of India *[5]*.

LITERATURE SURVEY

After carrying out literature survey and Patent search, it has been found that no mechanised spraying device is available elsewhere in the world for spraying the fire protective coating material in the coal bench of open cast mines up to 20 meters height. Earlier CMRI developed a small, simple and easy spraying system for spraying the fire protective coating material in the coal benches of open cast mines effective up to maximum 10 meters height under S & T Project. In this spraying system, compressed air was being used and maximum throw obtained was 10 meters high and discharge of the fire protective coating material was 5 liters per minutes. This system has been successfully applied for spraying the fire protective coating during field trial in two Indian opencast coal mines viz. Karkatta open cast project, Dakra (CCL) and jagannath open cast project, Talcher (MCL). Later on in another project, the same system has been utilised for spraying fire protective coating in Jhingurdah OCP, Singrauli (NCL), India. It has been observed that the uncovered portions of the coal bench more than 10 meters height caught fire within the incubation period whereas the covered portion remained safe from occurrences of spontaneous combustion/fire.

MECHANISM OF OXIDATION IN COAL BENCHES OF OPENCAST MINES

Coal interacts with oxygen of air at all temperatures. Higher is the temperature, greater is the rate of oxygen interaction with coal. At a lower temperature (40 °C and below) the interaction are mainly physical adsorption and chemisorption of oxygen with the formation of surface oxycomplex, with very low evolution of gaseous products. With rise in temperature (between 40 OC -70 OC) the surface oxycompounds break down, resulting in more rapid rate of oxidation, leading at more evolution of gaseous products and heat.

Due to use of heavy explosives, movement of heavy earth moving machines and transport vehicles on coal benches as well as relative movements of strata and deposits, fractures and micro fractures are created in the coal massive of the bench. The process results in increased overall permeability. Very close to the surface of top and side wall (less than 0.25 m) of the bench are particularly reckoned for susceptibility to oxygen due to abundance of both coal fines in clear cracks and the air supply. Although the rate of oxidation is high, the initial temperature rise is not much due to heat dissipation by wind current. Beyond half metre or so from the surface, interaction of oxygen with coal that has been powdered in the cracks may lead to profuse heat build up due to adverse heat transfer conditions. This nucleus zone is the most vulnerable one during the early periods of heating and threshold temperature is likely to reach earlier. The heat from this zone is, however, conducted in all directions as per thermal gradient created and the transfer of heat is determined by the surroundings temperature, wind current condition and free moisture of coal apart from thermal conductivity [6].

Besides this, it has also been observed that fine grained loose coals accumulated near the toe of high wall side of bench catch fire first and then the coal face catches fire by induction.

PREVENTION OF AIR INGRESS THROUGH BENCH WALLS

In the process of winning of coal by blasting, some portion of the bench walls generally develop cracks. Due to presence of macro and micro cracks, air enters into the cracked holes which leads to coal oxidation and heat build up. To render the bench walls impermeable to air, suitable fire protective coating should be applied over the exposed coal bench. Since vulnerable sites in the coal benches are only up to a meter deep from the surface, it may be useful to spray fire protective coating material under suitable pressure so that the material could fill up the cracks up to a certain depth, in the holes present in the coal massive.

DEVELOPMENT OF MECHANISED SPRAYING DEVICE

During the development of the mechanised spraying device, various designs and drawings were discussed and thoroughly scrutinised to fulfill the need of the mining industry. Out of these, one sketch was selected for fabrication and development of the spraying device. The details of the mechanised spraying device is as follows [5].

The mechanised spraying device comprises of a storage vessel, for storing fire protective coating material and another storage vessel for storage of raw water connected to a diesel operated engine driving the equipment through which the fire protective coating is delivered to the spray nozzle. Both the storage vessels are connected to a high pressure output equipment through valves for changing over of operation from spray cycle to cleaning cycle and vice versa. The equipment has suction port connected from the storage vessels, and delivery port connected to delivery line. The delivery line is further connected to a valve which facilitates control of fire protective material to the spray nozzle. The valve is connected to the spraying device, mounted on a stand through flexible hose. The whole system is mounted on a trolley which can be driven by suitable vehicle. The materials~ storage vessel and the water storage vessel are provided with a drain valve and two ports.

The design of the spraying device has been develop for the purpose to throw fire protective coating materials up to 20 meters height over the exposed coal surface for preventing spontaneous combustion. The coating material is sucked directly from the material containers, through flexible hose to the material storage vessel and controlled through flow valve from material vessel or material containers as deemed suitable at specific time. After completion of spraying job, the raw water vessel connected through a pipe line to the suction equipment through a valve. When the valve opens the suction line operates from raw water storage vessel and closes the suction line from material storage vessel. Cleaning liquid is sucked through the raw water vessel and delivered by the system equipment, through the delivery line to the spraying device and drained out. The valve is completely closed to the delivery line during the cleaning cycle. The valve is also completely closed to the suction line from material storage tank during cleaning cycle. The bye passed material in material storage vessel can be sprayed through the system or drained to empty material containers again if desired at the site (Figure 1).

LABORATORY SCALE STUDIES

For carrying out studies in laboratory a post of 20 m height was erected. After that several laboratory scale studies were conducted with different types & sizes of fabricated nozzles with the developed spraying system.

(1) In this mechanised spraying device, the equipment specially fabricated for this fire protective coating (cationic bitumen emulsion based) is fitted with suction and delivery pipe attached with different sizes of fabricated spraying nozzle tips of various Internal dia. one by one. After carrying out experimental work, a throw up to a height of 20 meters could not be achieved through any of the spraying device tips fitted during the trials. Maximum throw obtained was up to 16 meters height.

(2) In the same mechanised spraying device, the equipment was fitted with another pipe attached with fabricated nozzles one by one. After carrying out experiment work with another different types of spraying nozzle tips of various internal dia. one by one, a throw of 20 meters height with sufficient discharge of the fire protective coating material was achieved.

Finally after carrying out a series of studies on laboratory scale and further analysis of the feedback data, a mechanised spraying device has been developed. The results were established by conducting several repeat runs with the above developed spraying device.

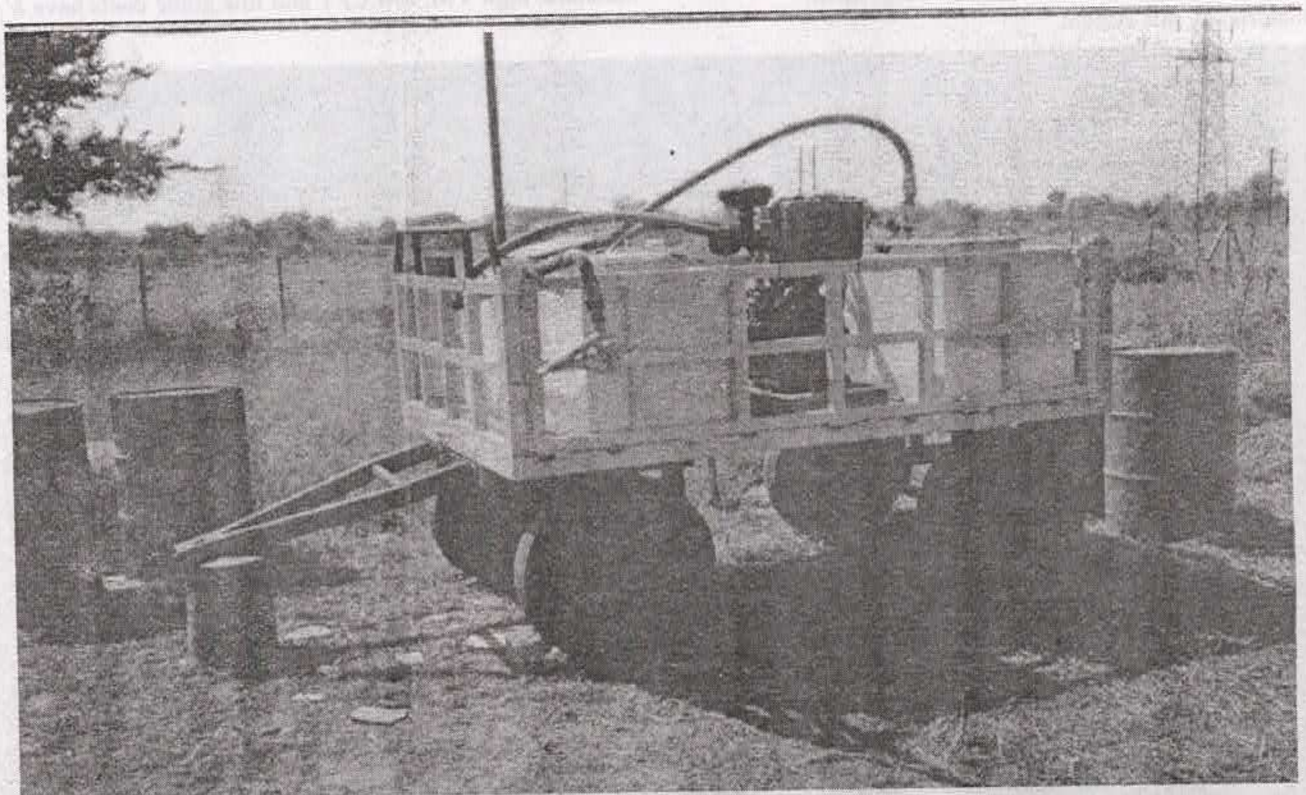

Fig. 1 : A mechanised spraying device mounted on trolley

FIELD TRIAL

CMPDIL, Ranchi suggested Jhingurdah open cast project, Singrauli, NCL as a site for field trial. Jhingurdah open cast project has a large number of fire problems in the coal benches due to low incubation period, low CPT value and high spontaneous heating susceptible characteristic of coal. Accordingly, we visited Jhingurdah OCP for the selection of site for testing the field effectivity of the developed mechanised spraying device (Figure 2). After selection of site, the exposed benches were monitored by infra-red thermometer and the fire protective coating material was sprayed by the developed mechanised spraying device in the 708 face of Jhingurdah top seam (Figure 3).

The details of the field trial is as follows:

Name of the Mine	: Jhingurdah opencast project, Singrauli, NCL
Seam	: Jhingurdah Top Seam
Bench height	: 19 m
Location	: 708 face
Area sprayed	: 90 m x 19 m
Bench height covered	: 19 m
Spraying time	: Two hours including spraying of the material, shifting of machine & refilling of fire protective coating material into the material storage vessel.
Consumption of material by this system	: 0.80 to 0.90 Kg./sq. m.

COAL CHARACTERISTICS OF JHINGURDAH OCP, SINGRAULI, NCL, INDIA

Jhingurdah seam of Jhingurdah open cast project, Singrauli, belongs to Raniganj Measure. It is one of the thickest seam in the world with thickness measuring up to 138 m. The seam is spread North wise in a gradient 1 in 6 and terminates abruptly by an up throw fault. The coal is won by blasting. The blasted coal is removed with large mechanised shovel which also fills the dumper for subsequent transport. Coal is of low grade (E) and is mainly supplied to the power houses. For quite a long time almost very soon after the start of the mine, the problem of spontaneous heating in the coal benches is creating lots of problem to the colliery authority for winning of coal. Jhingurdah top seam coal is of high moisture low grade as observed from its proximate analysis results.

The coal of Jhingurdah top seam showed very high susceptibility to spontaneous heating on the basis of the crossing point and ignition point temperature values. The high moisture, high VM content of the Jhingurdah seam coal may be attributed to the high oxidation and proneness to spontaneous heating in view of increased oxygen avidity. In fact it has been reported by the colliery authorities that the incubation period of the coal is found to be four weeks in the coal benches and about two weeks in the loose coals. From practical experiences it has been established that high moisture, high VM, low CPT and low grade coals have a

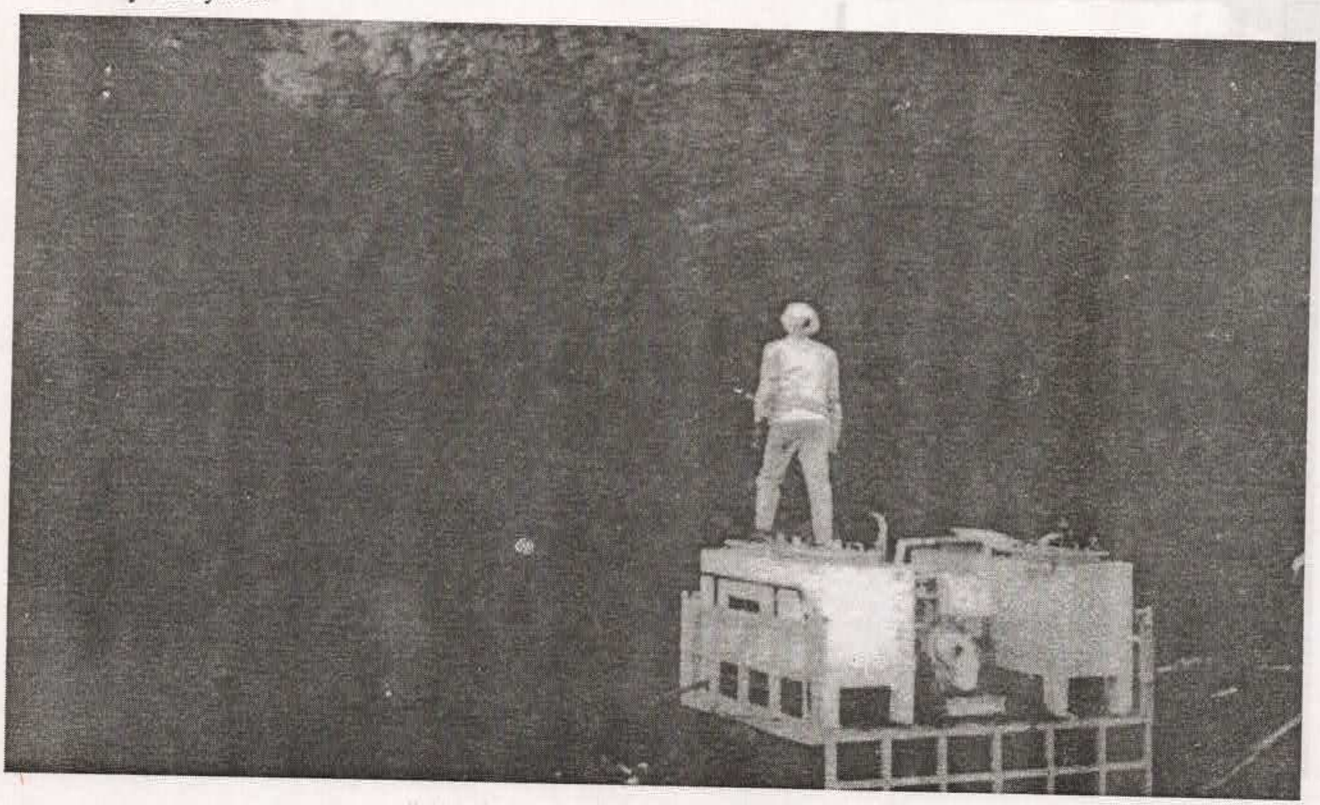

Fig. 3 : Spraying of fire protective coating at 20 m height of coal bench using spraying device in Jhingurdah OCP, Singrauli, NCL.

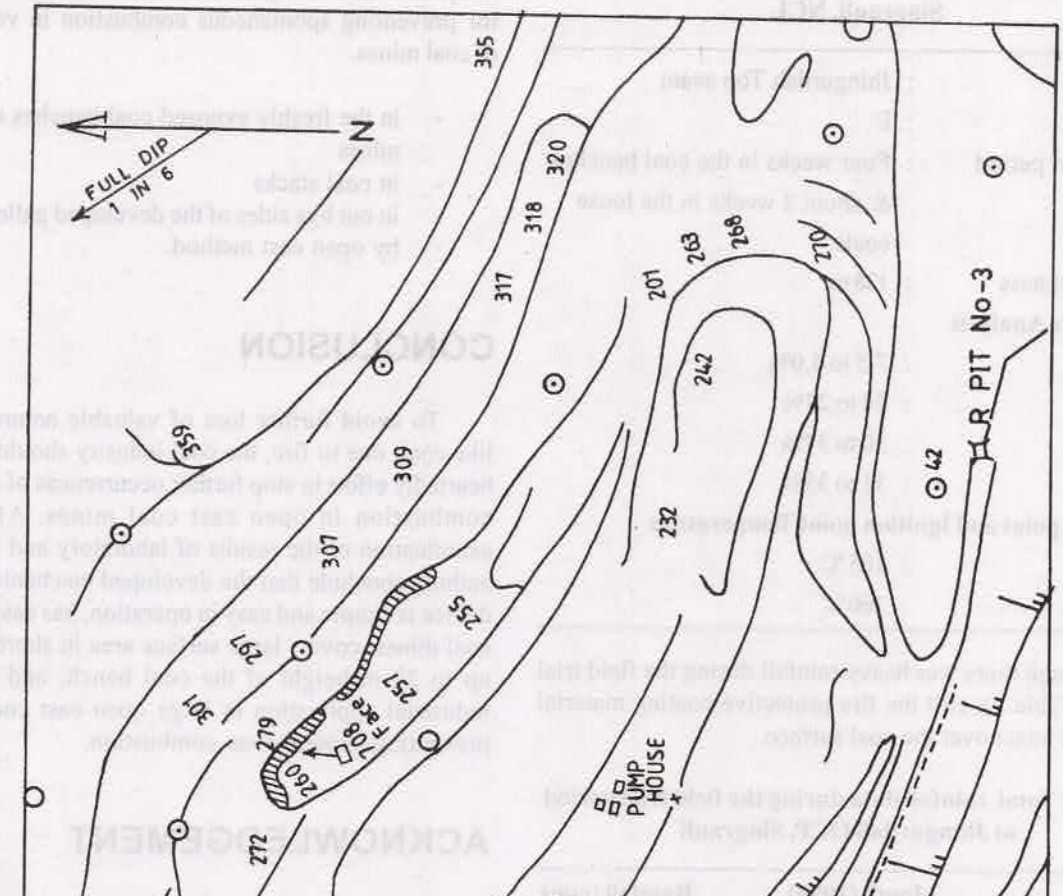

Fig. 2 : Field trial site at Jhingurdah OCP, Singrauli, NCL. (Not to scale)

greater affinity for oxygen absorption and hence more susceptible to spontaneous combustion. The details of coal characteristics are stated in Table 1.

Table 1 : Coal characteristics of Jhingurdah top seam, Singrauli, NCL

Seam	: Jhingurdah Top seam
Grade	: E
Incubation period	: Four weeks in the coal benches & about 2 weeks in the loose coals.
Seam thickness	: 138m
Proximate Analysis	
Moisture	: 7.5 to 8.0%
V.M.	: 24 to 27%
Ash	: 30 to 37%
F.C.	: 31 to 35%
Crossing point and Ignition point Temperature	
C.P.T.	: 105 °C
I.P.T.	: 160 °C

Though there was heavy rainfall during the field trial period (Table 2), still the fire protective coating material remained intact over the coal surface.

Table 2 : Total rainfall data during the field trial period at Jhingurdah OCP, Singrauli

Sl. No.	Month (1999)	Rainfall (mm)
1	May	16.00
2	June	276.90
3	July	539.40
4	August	274.90

RESULTS AND DISCUSSION

After carrying out field trial of the effectivity of the developed mechanised spraying device in Jhingurdah OCP, Singrauli, NCL. It has been observed that fire protective coating material has been successfully sprayed on the coal benches uniformly up to 20 m height of the coal bench. The exposed coal benches of 19 m height was covered by fire protective coating material fully and after the lapse of the incubation period of the coal benches of Jhingurdah OCP (five weeks) temperature was measured by Chino Infrared thermometer and found that temperature in the applied coated zone varied from 40.5 to 42.0 °C in the coated sites and ambient temperature was 38.5 °C. Temperature of the applied coated zone was near ambient. There was no indication of temperature rise in the coated zone.

APPLICATION

The developed mechanised spraying device will be very much useful as routine application for spraying the fire protective coating material in the exposed coal surfaces for preventing spontaneous combustion in various areas of coal mines.

- in the freshly exposed coal benches of open cast mines
- in coal stacks
- in out bye sides of the developed galleries worked by open cast method.

CONCLUSION

To avoid further loss of valuable natural resources like coal, due to fire, the coal industry should give whole heartedly effort to stop further occurrences of spontaneous combustion in open cast coal mines. After careful examination of the results of laboratory and field studies authors conclude that the developed mechanised spraying device is simple and easy in operation, has easy mobility in coal mines, covers large surface area in short time, throw up to 20 m height of the coal bench, and suitable for industrial application in large open cast coal mines for preventing spontaneous combustion.

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ROLE OF MATERIALS ENGINEERING TOWARDS CONTROLLING THE WORKING EFFICIENCY OF COAL MINING AND PROCESSING INDUSTRIES

A.K. Jha, S. Das, O.P. Modi, B.K. Prasad, R. Dasgupta and A.H. Yegneswaran

इस आलेख में कोयला खनन और प्रॉसेसिंग मशीनों के इंजीनियरिंग पुर्जों की कार्यकुशलता और जीवन पर पदार्थ संबंधित प्राचलों के प्रभाव की चर्चा की गई है। इस विषय पर विभिन्न परियोजनाओं पर शोध और विकास कार्य करते हुए प्राप्त अनुभवों से निष्कषों को बल मिलता है।

ऐसा कहा गया है कि गुण और कार्यकुशलता का सर्वश्रेष्ठ संबंध पदार्थ की सूक्ष्मसंरचना और प्रॉसेसिंग प्राचलों के इष्टतमीकरण से प्राप्त होगा। अनुभवों से पाया गया है कि लक्ष्य प्राप्त करने हेतु सावधानी से विचार करना आवश्यक है अन्यथा सम्पूर्ण प्रयास बेकार हो सकता है। रीजनल रिसर्च लैबोरेटरी, भोपाल इसके अध्ययन के लिए सक्षम है।

INTRODUCTION

The important role played by coal as a primary source of energy in the country is beyond any doubt. However in the present era of quality and cost consciousness, the coal mining and processing industries, like other sectors, faces now a days the challenge to minimize the cost of production at the first instance. One of the ways to attain this could be to make most effective utilization of manpower and machinery system. Down time costs quite heavily in this case in view of long time needed for the replacement of failed components in various machines. Obviously, measures taken to improve the working efficiency and prolong the life span of the machinery components could lead to at least one step forward towards attaining the goal.

In view of the above, an attempt has been made in this paper to analyze the nature of the problem and suggest possible remedial measures to minimize the extent and severity of the problems. The views putforth have further been substantiated through appropriate case studies.

NATURE OF THE PROBLEM AND REMEDIAL MEASURES

Failure of majority of mining and mineral processing machinery components is through wear and tear. Table 1 represents some of the components along with the nature of the wear experienced by them. Wear is an unwanted but inevitable removal of material due to relative motion between two interacting surfaces under pressure. As the definition indicates, wear related failure of materials is a surface phenomenon. Accordingly, surface characteristics and conditions of working greatly influence the overall performance of components. The aspect becomes of great concern for components working in dusty environments (such as during mining, crushing and grinding operations) or in corrosive environments (e.g. in operations like stowing, pumping, coal washing etc.). In practical situations, a blend of corrosive and dusty environments is generally encountered by the components during operation.

Accordingly, a number of material and working condition related parameters come into picture when the question of their effect on the working efficiency of components arises. In fact, a precise assessment of the performance level of a component becomes difficult, in spite of available related scientific information. This is because of the complexity of the processes occurring in actual working conditions. However, the problem could be reduced considerably through a systematic study of various factors responsible for a specific level of performance under various conditions on a laboratory scale followed by performance evaluation under actual working conditions.

Development of any engineering component involves design of material and product. Both the aspects are interrelated and strongly influence the overall performance of the product(s). In general, aspects pertaining to component design are paid primary attention. On the contrary, parameters relating to material design are not considered seriously due to a lack of understanding and complexity involved therein. In fact, responsibility goes primarily to the material at the first instance in the event of a component failure in practice. It may be noted that design of a component normally does not change much, once finalized, whereas there always lies ample scope to modify the materials design with a view to improve the performance of the component.

Regional Research Laboratory (CSIR), Bhopal - 462 026

Table 1 : Various mining machinery components and associated mode(s) of wear causing failure

Components	Nature of Wear
Sprockets, gears, shafts, axles, pinion , teeth, concrete and pan mixer wear plate, ropeway, car rails, wheels, points and crossings, crane wheels, ropeway trolley wheels, bush etc.	Rolling and Sliding
Crane wheels, gears, shafts, brake shoes, rail ends, conveyor bucket lips, AFC pans, shear blades, shovels, dredger and elevator bucket lips, rock drills, impellers, excavator buckets, bucket teeth, cams, fan blades, exhaust blades, craper bars, mill hammers, pulverisers, grinder rings, crusher hammer, roll crusher, track shoes, picks of coal cutter, digger teeth, valve seat, dredger buckets, rail road trucks, conveyor screws, boring tools, impellers, mill hammers, mixer blades, digger teeth, hoppers, conveyor and screws, coke chutes, grinders, pump, impellers etc.	Abrasive Wear
High pressure valves, turbine blades and valve seat	Erosive Wear

Note : Corrosive wear also occurs along with the mentioned modes of wear wherever the environment contains (corrosive) liquid.

In the subsequent text, the role of material related parameters on the wear performance on a laboratory scale and their correlation with field performance would be discussed with the help of typical case studies.

Factors controlling the wear performance of components can be related to material and service conditions. Material related aspects include shape, size, hardness, mode of distribution and content of various phases in the material microstructure. The parameters can be controlled through compositional modifications and adopting appropriate processing cycles/ parameters to suit specific service conditions. In this context, it is imperative to note that a proper optimization of the parameters needs to be carried in order to realize best performance of components in practice. In fact, specific material properties emanate from appropriate optimization of mentioned microstructural factors. No direct correlation between mechanical and wear properties exists in reality in spite of the general understanding that, harder and stronger the material, better the wear response. Further, the wear performance is greatly dependent on working conditions to a great extent. Accordingly, the same material can work differently in various service conditions. Another important aspect of components for wear related applications is that their surface characteristics play the principal role in controlling their performance and bulk properties bear rather secondary importance. Accordingly, modification of surface characteristics has nowadays become an important tool towards the realization of good service life of components.

An appraisal of the above discussion suggest that there exist several factors which greatly control the overall performance of a component. Optimization of various microstructural parameters could lead to best performance of components in specific applications.

Regional Research Laboratory (CSIR), Bhopal has been carrying out extensive R&D work towards the development and characterization of different material systems and components thereof to suit different engineering applications. They include zinc alloy bearings, aluminium alloy composites for mineral processing machineries and automobiles, steels and their processing

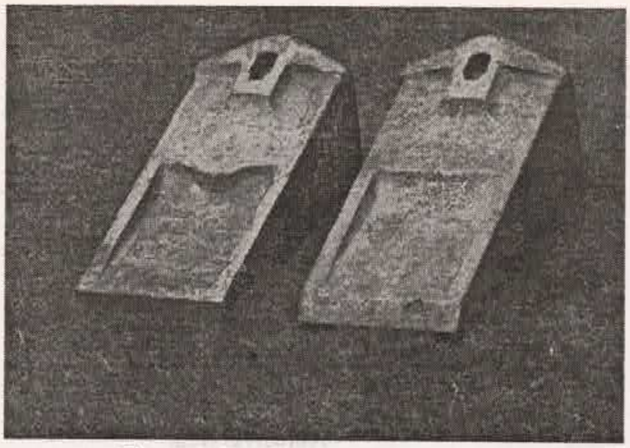


Figure 1 : Tooth Points of P&H Shovel

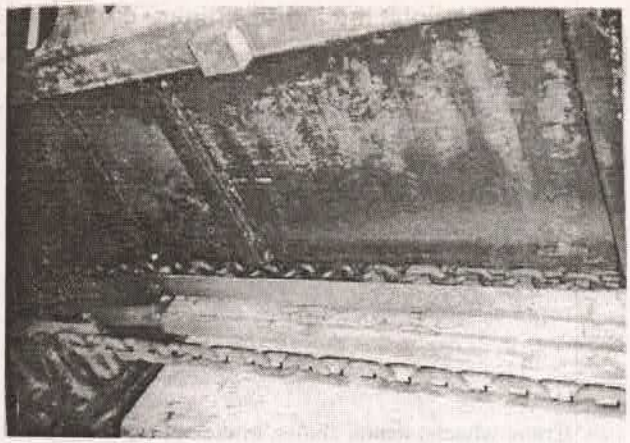


Figure 2 : AFC Pan Assembly

for preparing improved versions of mining machinery components and surface modification of steels and components. The developments have been undertaken under various sponsored projects to bridge the gap between the present practice and latest scientific information available. The studies have also enabled to translate the outcome generated through laboratory scale investigations into actual practice through the development of typical prototypes followed by their performance evaluation in field as discussed subsequently.

One of the R&D activities sponsored by Ministry of Coal was pertaining to the development of improved versions of tooth points (Fig. 1) and AFC pans (Fig. 2). The study was initiated with the analysis of the material composition and mechanical properties of the existing components. This was followed by the synthesis and processing of several modified compositions of the material system and characterization of the compositions for their relevant engineering properties on a laboratory scale. The laboratory scale investigations indicated considerable scope to (a) improve the performance of the existing material composition through appropriate processing cycles and parameters and (b) develop improved performance components through compositional and microstructural modifications.

Based on the outcome of the studies, prototype components (tooth points and AFC pans) were prepared using optimized material composition and processing (e.g. heat treatment/ surface treatment) parameters.

In the case of tooth points, the existing components after treatment as well as the ones developed using modified material compositions and processing (heat-treatment) cycles performed considerably better than the existing components. Further, the tooth points developed by RRL, Bhopal performed comparable to the imported ones at one of the mine sites (Rajrappa and Gevra) of Coal India Ltd., while they cost nearly one-third of the latter.

In the case of AFC pans, a variety of hardfacing materials were developed and characterized by RRL, Bhopal. One of the best ones was used for the surface treatment of the existing AFC pans and tested for their field performance at Munidih Colliery, Dhanbad. The study showed that same level of performance can be realised by treating the AFC pans with a material with ~60% cost of the conventionally used ones.

Material treatment in this study involved bulk treatment such as heat treatment and surface treatments. In the case of bulk treatment, properties of the entire material undergo a change. On the contrary, surface treatment involves modification in the (surface) characteristics only up to a thickness of ~0.01-2 mm of a component, bulk characteristics remaining practically unchanged. Accordingly, the (surface) modification treatment has the potential to become a cost and energy effective proposition to develop components with much improved performance. The treatment is especially suitable for conditions such as wear wherein surface properties mainly control the performance of a component.

In another investigation, attention was focused to develop better substitutes to bronze (copper-based alloy) components used in heavy machineries. An alternate alloy system was developed and characterized on a laboratory scale to assess its working capability in place of bronze components. Typical bushes (Fig. 3) were fabricated and

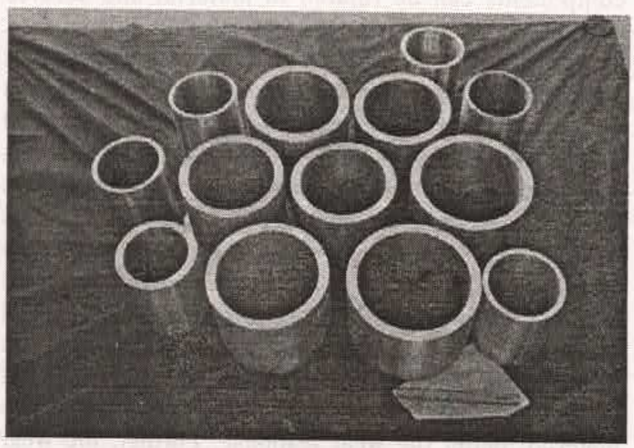


Figure 3 : Bushes of P & H Shovel

Table 2 : A comparison of the cost and the range of physical and mechanical properties of the developed bush material system with the conventional gun metal/bronzes

Sr. No.	Property	Developed Material System	Gun Metals (Leaded-tin bronzes)
1.	Cost, Rs. per Kg	150	250
2.	Melting Range, °C	375-500	850-950
3.	Density, gm/cc	4.5-6.0	8.5-9.5
4.	Thermal Conductivity, W/m°C	115-125	46-60
5.	Sp. heat, KJ/Kg/K	450-525	376-386
6.	Coefficient of Linear Expansion, 10^{-6}	24-26	17-18
7.	Energy Required for Melting, KWH/Ton	150	225
8.	Tensile Strength, MPa	275-440	200-250
9.	Yield Strength, MPa	210-370	120-130
10.	Elongation, %	3-12	5-15
11.	Young's Modulus, $x10^3$ MPa	75-85	75-90
12.	Hardness, BHN	85-120	60-80

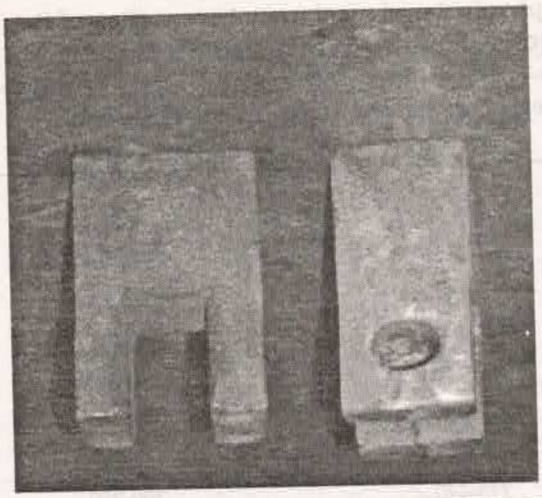
Table 3 : Properties of the composite and cast iron

S. No.	Properties	Al-Si-SiC Composite	Cast Iron
1.	Density (g/cc)	2.8	7.2
2.	Thermal Conductivity (W/m. K)	182	47
3.	Specific Heat (J/kg. K)	837	402
4.	Thermal Expansion Coefficient, $x10^{-6}$	17.4	12.2
5.	Tensile Strength (Mpa)	240	250
6.	Hardness (HV)	140	200

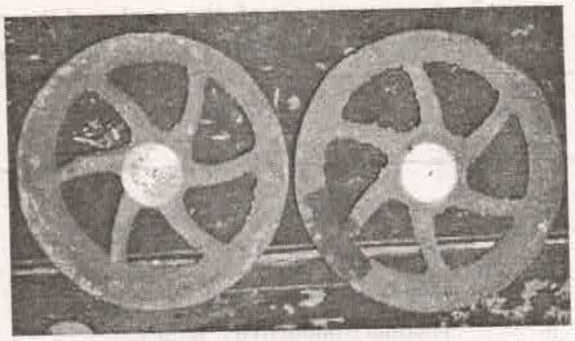
tried out at one of the mine sites (at Gevra) of Coal India Ltd. The components having identical performance level cost ~ 60% of that of the ones fabricated with conventional bronzes. In another similar application at Hindustan Zinc Ltd. (Agucha Mines), the new material components offered double the life span of the conventional (bronze) ones (while costing ~ 60% of the latter). Table II shows comparative properties of both of the alloy systems.

Overall performance of cast components greatly depends on its quality such as dimensional accuracy and level of porosity within. Castings from which the components are given the final shape need to be sound enough otherwise, no matter how appropriate remaining processing steps/ cycles/ parameters are, the components are bound to perform poorly in service. Under an R&D project sponsored to RRL, Bhopal by Singreni Collieries Company Ltd., Kothagudem, it was shown that there exists tremendous scope to significantly reduce the level of porosity and improve the yield of castings (Fig. 4) by adopting simple but logical modifications in the mould design and other related methodologies. The outcome was lively demonstrated on the shop floor at Kothagudem in association with the working personnel. However, this needs proper education of the working personnel which was materialized through delivering a series of technical talks to them and discussion thereof on related issues.

Aluminium alloy hard particle composites are another important development at RRL, Bhopal which has great potential for use in mining industries. The laboratory has synthesized a variety of composite materials by selecting various matrix alloys and reinforcements and characterized for their relevant engineering properties. Table III shows a comparison of various properties of a typical aluminium alloy composite vis-a-vis a conventional cast iron used for fabricating automobile brake drums. On the basis of encouraging observations made during the investigations, typical engineering components (Fig. 5) for automobile and mineral processing machineries have been fabricated. Performance evaluation of the developed automobile component such as brake drum (Fig. 5a) has shown significantly improved braking efficiency and less frictional heating than that of the conventional cast iron component



Coal Mill Hammers

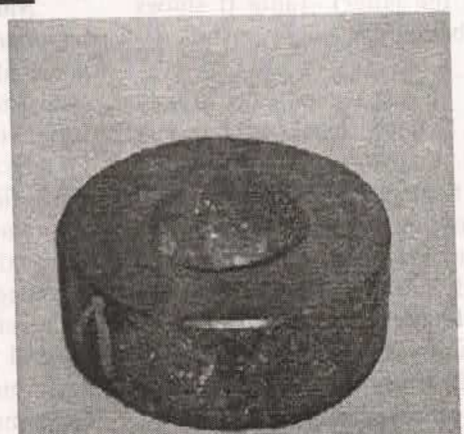


Tub Wheels

Tooth Point of Dragline

Adapter of Shovel

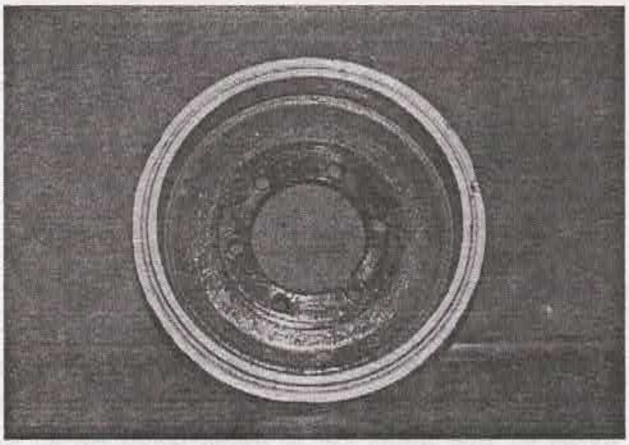
Tooth Point of Shovel



Bush
Casting

Machined Component

Figure 4 : Typical components studied for improvement in casting quality and component property


(a) Automobile brake drum

(b) Refrax Apex Insert of Cyclone

Figure 5 : Typical components developed using aluminium alloy composites

in actual working conditions. Components of mineral processing equipment such as refrex apex insert of a cyclone (Fig. 5b) have also been developed using the composite materials. Their preliminary trials have shown encouraging results.

CONCLUDING REMARKS

There exist several material and working condition related parameters which control the working life of components in various coal mining and processing machineries. The view has further been substantiated through typical examples

A systematic understanding and analysis of the same could enable to find out cost and energy effective solutions to various material related problems. This could ultimately lead to the development of engineering components with significantly improved performance/ life. In view of the expertise developed and experience gained in this area, RRL Bhopal could be helpful to effectively resolve the issues addressed in this study under sponsored R&D projects.

ACKNOWLEDGEMENT

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MOLTEN CARBONATE FUEL CELL OPERATION WITH COAL GAS

Sanjay Bali*, Vidya S Batra* and Ajay Mathur*

कोल गैस में ऊर्जा उत्पादन हेतु ''इंटिग्रेटेड गैसीफायर मोल्टेन कार्बोनेट फ्यूएल सेल'' एक आकर्षक रास्ता दिखाता है। यह उच्च दक्षता एवं पर्यावरणीय मित्र है। इस आलेख में TERI के मॉल्टेन कार्बोनेट फ्यूएल सेल कार्यक्रम का वर्णन किया गया है। इसका प्रारंभ सी. एम. पी. डी. आई. से प्राप्त धन द्वारा किया गया था। हाइड्रोजन ईंधन एवं अनुकरणीय कोल गैस पर काम्पोनेन्ट विकास एवं सेल टेस्टिंग के परिणामों का वर्णन किया गया है। अपेक्षित विशेषताओं वाले संघटकों की प्राप्ति संसाधन दशा के समुचित नियंत्रण द्वारा की जा सकती है। लघु अन्तर को छोड़कर कोल गैस के साथ सेल का निष्पादन परम्परागत ईंधन के साथ निष्पादन से तुलनीय है।

INTRODUCTION

Fuel cell *is* an attractive power generating device, which is highly efficient and environmentally friendly. It works on the principle of converting oxidation energy of a fuel into electricity and heat via an electrochemical reaction. There are different types of fuel cells such as Alkaline Fuel Cell (AFC), Polymer Electrolyte Membrane Fuel Cell (PEMFC), Phosphoric Acid Fuel Cell (PAFC), Molten Carbonate Fuel Cell (MCFC) and Solid Oxide Fuel Cell (SOFC). AFCs and PEMFCs are low temperature fuel cells being developed for transportation applications. PAFC operates at 200°C and is suited for stationary applications. High temperature fuel cells like MCFC and SOFC are also suitable for stationary applications and are more efficient than PAFCs [1].

Molten carbonate fuel cell operates at 650 °C. It has the advantage of being able to use coal gas as fuel, which makes it suitable for cogeneration. The fuel H_2, along with CO_2 is fed to the anode while the oxidant and CO_2 are fed to the cathode (Figure 1). The electrodes (nickel + aluminium anode and lithiated nickel oxide cathode) are separated by a molten carbonate electrolyte, retained in a porous lithium aluminate matrix, which provides carbonate ions for the reactions. The components are housed in stainless steel end plates. At the anode the fuel reacts with carbonate ions to form water, carbon dioxide and free electrons. At the cathode the oxygen reacts with carbon dioxide and electrons to form carbonate ions. The electrons are transported through the external circuit and the carbonate ions are transported through the electrolyte. The electrode reactions are shown below. Figure 1 represents a monocell; such cells are stacked up in series to obtain the desired power output.

$H_2 + CO_3^{2-} \rightarrow H_2 0 + CO_2 + 2e$ Anode reaction
$20_2 + CO_2 + 2e \rightarrow CO_3^{2-}$ Cathode reaction
$H_2 + 20_2 \rightarrow H_2 0$ Overall reaction

When coal gas is used as fuel, the CO gets converted to H_2 via the water gas shift reaction[2].

$$CO + H_2 0 \rightarrow CO_2 + H_2$$

Compared to other technologies for obtaining electricity from coal such Pressurised Fluidized Bed Combustor Combined Cycle (PFBCC) and Integrated Gasification Combined Cycle (IGCC), the Integrated Gasifier Molten Carbonate Fuel Cell (IG-MCFC) offers higher efficiency and has tremendous potential in a coal based power deficit country like India.

MCFC work at TERI was initiated with support from CMPDIL with the view of assessing and demonstrating MCFC technology using Indian coal. In this paper, the results from the study are described.

COMPONENT DEVELOPMENT

Electrodes

The characteristics of the porous nickel electrodes are different for cathode and anode. The anode has a porosity

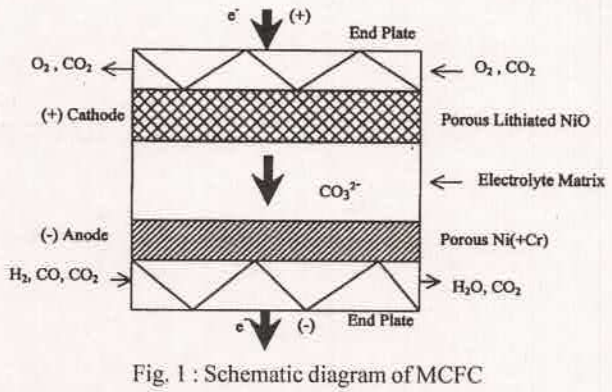

Fig. 1 : Schematic diagram of MCFC

*Tata Energy Research Institute, Habitat Place, Lodhi Road, New Delhi 110003

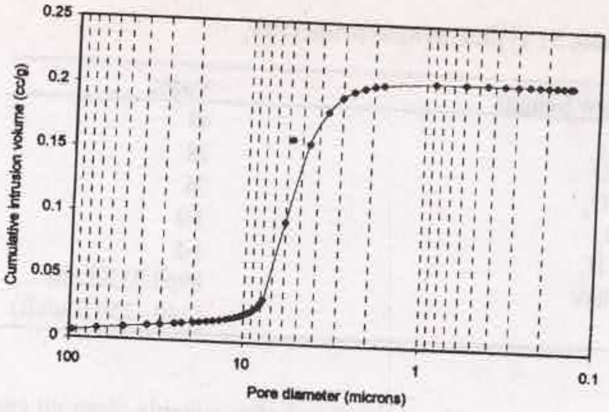

Fig. 2 : Pore distribution of anode.

of 5 5-70 vol% and a mean pore diameter of 3-6 .um while the cathode has porosity of 70-80% and 7-12 micron mean diameter [3], [4]. The nickel cathode forms lithiated nickel oxide insitu during cell heating.

The development efforts were aimed at achieving these values. The electrodes were prepared by tape casting and sintering.

The appropriate slurry preparation and sintering conditions were identified to obtain optimum aluminium incorporated nickel anode and nickel cathode. The characteristics of anode and cathode are shown in Figure 2, Table 1 and Figure 3, Table 2 respectively.

Table 1 : Anode porosity data

Total intrusion volume	0.2067 cc/gm
Total pore area	0.160 sq. mg
Median pore diameter	5.81 microns
Bulk density	2.9560 g/cc
Porosity (assuming 8.9g/cc as density of zero pore sample)	66%

Table 2 : Cathode porosity data

Total intrusion volume	0.47 10 cc/gm
Total pore area	0.162 sq. m/g
Median pore diameter	11.4284microns
Bulk density	1 .6289 g/cc
Porosity (assuming 8.9g/cc as density of zero pore sample)	81.6%

Matrix

The gamma lithium aluminate support which retains the molten carbonate was prepared by tape casting and a suitable slurry formulation was developed. Since the matrix was being used in the green form in close contact with hygroscopic carbonate electrolyte, a non-aqueous system was selected.

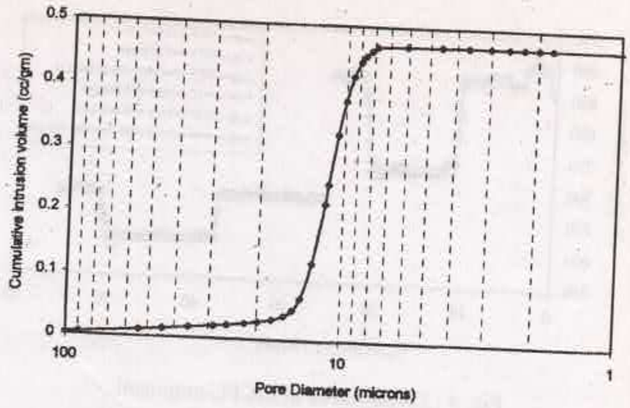

Fig. 3 : Pore distribution of cathode.

Electrolyte

The electrolyte loading was attempted in different ways during cell testing. In the initial cell tests, the lithium and potassium carbonate powders were mixed in the appropriate ratio and used in the powder form. However, to facilitate the assembly of the cell, the electrolyte was preferred in the form of a tape. Therefore trials were made to obtain a suitable slurry for tape casting using non-aqueous binders.

CELL TESTING

Several cell tests were performed with different electrode combinations, electrolyte loading and with different fuels (hydrogen and coal gas). The assembly and testing procedure were same as described in reference [5]. The coal gas compositions that were tested are shown in Table 3. These compositions were simulated in the lab and cell performances obtained are shown in Table 4. The cell performance was also monitored by changing the fuel feed from hydrogen to coal gas. The change led to a voltage drop of -150 mV. However, the original voltage could be recovered by changing the feed back to hydrogen even after -10 hours. This is shown in Figure 4.

The first field demonstration of this concept was carried out at Louisiana Gasification Technology Incorporated, USA using a Energy Research Corporation, USA's 20 kW MCFC stack. The typical gas composition is shown in Table 5. The coal gas after cleaning and humidification was fed to the fuel cell. The stack generated up to 21 kW and post test analysis revealed no apparent effects of contaminants.

CONCLUSION

The components for MCFC can be fabricated with the required properties by suitably controlling the processing conditions. The cells tested with these components gave satisfactory results. The tests with coal gas demonstrate

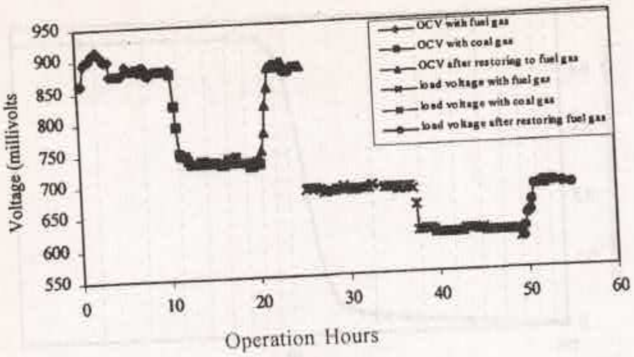

Fig. 4 : Performance of MCFC monocell with different fuel gases.

Table 3 : Composition (vol%) from BHELs coal gasifier using different varieties of Indian coal

	Gas composition (vol %) dry basis						
Gas/coal type	CO_2	CO	H_2	CH_4	N_2	02	HCV (kcal NM-3)
Coal typeI	13.4	11.2	12.8	0.9	61.50	0.2	813.6
Coal type II	14.0	10.2	12.87	0.87	62.06	-	782.6
Coal type III	13.5	12.1	12.58	1.05	60.77	-	848.4

Table 4 : Open circuit voltage and operation hours of MCFC running on simulated coal gas

Coal type/variable	Coal type I	Coal type II	Coal type III
Open circuit voltage (mV)	723	730	750
Operation hours	8	8	8

Table 5 : Typical syngas properties

Raw Syngas	Vol%
H_2	43
CO	28
CO_2	26
N_2	1-2
CH_4	1-2
HHV	8960-9330kJ/m^3 (240 - 250 Btu/cft)

that Indian coals, gasified and after suitable clean up can be used effectively as fuel for MCFC. This concept has also been tested in the field with satisfactory performance.

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May - Aug., 2000

RECLAMATION OF ASH POND OF NEYVELI LIGNITE CORPORATION, NEYVELI, INDIA

Selvam, A.* and A. Mahadevan**

जैवीय उर्वरक (Glomus mosseae, Azospirillum, Azotobacter और Phosphobacteria) की मदद से थर्मल पावर स्टेशन-I के समीप पाँच एकड़ क्षेत्र में फैले हुए अपमार्जित राखयुक्त तालाब को सफलतावूर्वक कृषि योग्य बनाया गया। Achras sapota, Anacardium occidentale, Annona squamosa, Artocarpus integrifolia, Citrus limonum, Emblica officinalis, Mangifera indica, Murraya koenigii, Psidium guajava, Punica granatum, Tamarindus indica, Acacia leucophloea, Albizia lebbeck, Azadirachta indica, Casuarina equisetifolia, Bombax ceiba, Dendrocalamus strictus, Eucalyptus spp., Pongamia pinnata और Tectona grandis आदि को राखयुक्त तालाब के क्षेत्र में रोपा गया और उनमें खाद डाली गयी। समुचित नियंत्रण द्वारा उनका रख-रखाव किया गया एवं एक वर्ष की अवधि के लिए उपज की वृद्धि का प्रबोधन किया गया। मूल गोलाकार क्षेत्र में वेसीकुलर-आर्बुसकुलर माइकोरीजल(VAM)के उपनिवेष्ठन का % तथा बहुलसंख्यक(VAM) बीजाणु का निश्चित रूप से वृद्धि के साथ संबंध स्थापित किया गया। बागान में माइकोरीजल संचारण से बारह महीने के उपरान्त पौधों में 34% तक की वृद्धि पाई गई।

INTRODUCTION

In India, out of a total land area of 328.8 mha, coal occurs over an area of about 2.13 mha. Conceding that degradation of land is an inevitable consequence of mining for minerals, damage over 0.36 mha i.e. 0.1% of the total land surface can, at the most, be attributed to past coal mining activities. As per requirements, by 2000 AD, the country may produce 400 million tons of coal of which about 275 million tons would be by opencast mining alone. It is estimated that every million ton of coal extracted through application of open cast mine damages a surface area of about 4 ha. With the level of production planned for the future, the coal industry accounts for rendering biologically unproductive an area of about 500 hectares a year today which will rise to no less than 1400 ha a year by 2000 Ad (1).

Fly ash, the principal by-product of coal-fired thermal power station, is the residue resulting from the combustion of pulverised fuels. This material is mainly piped as slurry of ash and water to storage ponds located near power stations. These ponds are referred to 'ash ponds'. Once the ponds are filled with slurry, they are abandoned and the area becomes unproductive.

Establishment of vegetation on abandoned fly ash ponds serves a variety of functions, including stabilizing the ash against wind and water erosion, providing shelter and habitat for wildlife and providing aesthetically pleasing landscape. However, revegetation of these sites often proceeds slowly due to the presence of physical and/or chemical conditions in the ash that are deleterious to the survival and growth of plants [2.3, 4, 5, 6, 7].

The formation of mycorrhizae is considered essential for the survival and growth of majority of plant species in natural ecosystems. VA mycorrhiza is estimated to form association with almost all angiospermous plants {8}. Their role in enhancing water and nutrient uptake, especially P, Zn and Cu is well known. Success or failure of plant species on mine sites depends on the presence of suitable and viable mycorrhizal inoculum. They also help in the establishment, survival and growth of sand dune colonizing plants {9}.

MATERIALS AND METHODS

Twenty plant species were selected to plant the ash pond area. Of these, 10 plant species belong to orchard species and the other 10, forest tree species. Uniform sized plants of each species were planted in 2' x 2' x 2' pits. In order to give anchorage to the plants, the pits were filled with red soil, tank silt and press mud, a waste from sugar mills in the ratio of 1:1:1 (v/v/v).

Plants were inoculated with an endomycorrhizal fungus Glomus mosseae, which was isolated from the ash pond (Fig. 1). All the plants were inoculated with a mixture of bacterial biofertilizers (Azospirillum, Azotobacter and phosphobacteria) irrespective of mycorrhizal inoculation. Six replicated were maintained in each plant species. The chemical constituents of ash pond soil are presented in Table 1.

*Centre for Advanced Studies in Botany, University of Madras, Guindy Campus, Chennai - 600 025
** Centre of Theoretical Biology, No. 40, I Main Road, Indira Nagar, Chennai - 600 020

Table 1 : Chemical characteristics of ash pond soil

Parameter	Ash pond soil
Loss of ignition %	13.06
Silica as SiO_2%	70.93
Alumina as Al_2O_3%	2.81
Iron as $Fe_2O_{3\%}$	6.4
Calcium as CaO%	5.6
Magnesium as MgO%	1.2
Sulphates as SO_4	Traces

Height of plants, mycorrhizal colonization and number of VAM spores in the rhizosphere soil were recorded for a period of one year at 3 month intervals. Mycorrhizal colonization was estimated using the clearing and staining method of Phillips and Hayman {10} and the mycorrhizal spores were isolated using wet sieving and decanting method of Gerdemann and Nicolson {11}.

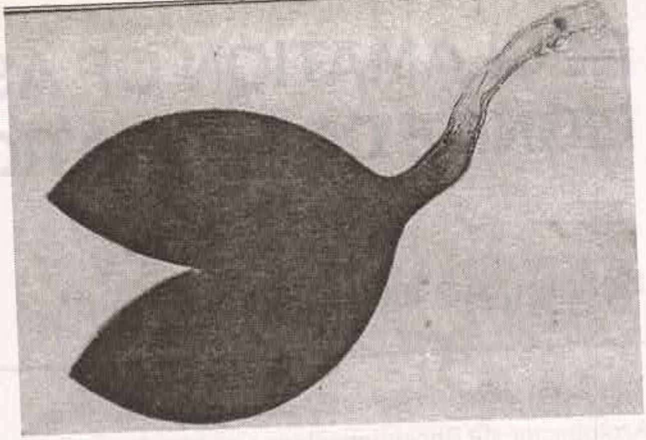

Fig. 1 : Spore of *Glomus mosseae* x 510

RESULTS

Increased colonization due to mycorrhizal application resulted in the increased growth of plants. Height of plants after 3 months of planting and mycorrhizal inoculation, are presented in Table 2 and 3.

Table 2 : Height, mycorrhizal colonization and number of VAM spores in the rhizosphere soils of plants grown in ash pond soil after 3 months of planting.

Plant species	Height of plants (cm)		VAM colonization(%)		Spores/100g soil	
	Control	VAM	Control	VAM	Control	VAM
Achras sapota	54	60 (11)	5	19	16	34
Anacardium occidentale	20	22 (10)	6	14	15	31
Annona squamosa	32	39 (9)	5	21	15	36
Artocarpus integrifolia	38	41 (8)	6	12	13	27
Citrus Limonum	22	25 (14)	2	12	14	22
Emblica officinalis	38	45 (18)	10	26	18	38
Mangifera indica	46	51 (11)	7	16	11	26
Murraya konigii	5	6 (20)	6	10	11	20
Psidium guajava	17	24 (41)	4	14	14	34
Punica granatum	16	19 (19)	5	16	12	26

Values in parenthesis indicate the per cent increase in height of mycorrhizal inoculated plants over uninoculated plants.

Table 3 : Height, mycorrhizal colonization and number of VAM spores in the rhizosphere soils of plants grown in ash pond soil after 3 months of planting.

Plant species	Height of plants (cm)		VAM colonization(%)		Spores/100g soil	
	Control	VAM	Control	VAM	Control	VAM
Tamarindus indica	41	45 (10)	12	21	18	33
Acacia leucophloea	33	40 (21)	14	31	16	39
Albizia lebbeck	63	68 (8)	10	21	15	23
Azadirachta indica	45	56 (24)	12	24	23	44
Casuarina equisetifolia	17	19 (12)	9	33	28	53
Bombax ceiba	50	58 (16)	11	20	20	30
Dendrocalamus strictus	66	76 (16)	13	32	34	53
Eucalyptus spp	72	81 (13)	10	27	22	40
Pongamia pinnata	32	35 (9)	7	20	20	36
Tectona grandis	11	15 (36)	11	30	26	42

Values in parenthesis indicate the per cent increase in height of mycorrhizal inoculated plants over uninoculated plants.

Table 4 : Height, mycorrhizal colonization and number of VAM spores in the rhizosphere soils of plants grown in ash pond soil after 6 months of planting.

Plant species	Height of plants (cm)		VAM colonization(%)		Spores/100g soil	
	Control	VAM	Control	VAM	Control	VAM
Achras sapota	58	68(17)	8	28	18	47
Anacardium occidentale	27	31(15)	11	23	20	46
Annona squamosa	35	42(17)	12	30	23	50
Artocarpus integrifolia	46	51(13)	10	21	17	43
Citrus Limonum	26	31(19)	5	19	15	31
Emblica officinalis	111	134(21)	16	38	23	54
Mangifera indica	56	67(20)	8	26	15	38
Murraya konigii	13	17(31)	8	18	14	34
Psidium guajava	28	40(43)	10	23	20	49
Punica granatum	28	34(21)	10	24	18	37

Values in parenthesis indicate the per cent increase in height of mycorrhizal inoculated plants over uninoculated plants.

Table 5 : Height, mycorrhizal colonization and number of VAM spores in the rhizosphere soils of plants grown in ash pond soil after 6 months of planting.

Plant species	Height of plants (cm)		VAM colonization(%)		Spores/100g soil	
	Control	VAM	Control	VAM	Control	VAM
Tamarindus indica	49	60(22)	17	33	30	51
Acacia leucophloea	92	100(9)	23	49	29	56
Albizia lebbeck	124	150(21)	18	37	28	47
Azadirachta indica	94	105(12)	20	39	40	68
Casuarina equisetifolia	38	55(45)	20	47	44	78
Bombax ceiba	117	142(21)	18	34	36	57
Dendrocalamus strictus	94	108(15)	22	56	51	88
Eucalyptus spp	119	144(21)	19	45	34	63
Pongamia pinnata	48	58(21)	15	36	31	56
Tectona grandis	34	49(44)	20	43	37	69

Values in parenthesis indicate the per cent increase in height of mycorrhizal inoculated plants over uninoculated plants.

The percent increase in height of mycorrhizal inoculated plants over control plants ranged from 8%-41%. Inoculated *Psidium guajava* and *Tectona grandis* had 36% and 41% increased height over their respective control plants.

Mycorrhizal colonization ranged from 2%-10% and 10%-26% in the uninoculated and mycorrhizal inoculated plants, respectively in orchard plant species (Table 2). In the forest tree species, VAM colonization ranged from 7%-14% and 20%-33% in the uninoculated and mycorrhizal inoculated plants, respectively (Table 3). Colonization increased due to the inoculation of mycorrhizal fungus.

Number of VA mycorrhizal spores in the rhizosphere soils also increased due to the application of G. mosseae. Number of spores isolated form 100 g of soil was 11-18 and 20-38 in control and VAM applied plants respectively in orchard plant species and in forest plant species it ranged from 15-34 and 23-53 in the control and VAM inoculated plants, respectively.

After 6 months of planting, per cent increase in the height of mycorrhizal inoculated plants over control plants ranged from 9%-45%. In the orchard plant species, *Emblica officinalis* developed good growth (111-134 cm height) followed by other plants (Table 4). *Citrum limonum* had the lowest height. In forest tree species except *Tectona grandis*, *Casuarina equisetifolia* and *Pongamia pinnata* all other plants displayed excellent growth (Table 5) . .

Mycorrhizal colonization ranged from 5%-16% and 18%-38% in control and VAM inoculated plants respectively in orchard plants species (Table 4). In forest plant species, colonization ranged from 15%-23% and 33%-56% in control and VAM inoculated plants. Number of spores ranged from 14-51 and 31-88 in control and mycorrhiza inoculated plants respectively.

Inoculation of *G. mosseae* on to plants grown in ash pond soil increased the growth of plants. After 9 months of planting and mycorrhizal inoculation, mycorrhizal colonization ranged from 23%-49% in orchard plant

Table 6 : Height, mycorrhizal colonization and number of VAM spores in the rhizosphere soils of plants grown in ash pond soil after 9 months of planting.

Plant species	Height of plants (cm)		VAM colonization(%)		Spores/100g soil	
	Control	VAM	Control	VAM	Control	VAM
Achras sapota	70	90(29)	10	35	20	55
Anacardium occidentale	58	73(26)	15	30	23	60
Annona squamosa	85	110(29)	18	40	27	65
Artocarpus integrifolia	98	131(34)	14	30	22	62
Citrus Limonum	43	51(19)	7	23	17	40
Emblica officinalis	219	279(27)	19	49	29	65
Mangifera indica	63	83(32)	11	32	19	50
Murraya konigii	39	47(21)	9	24	17	43
Psidium guajava	127	163(28)	15	30	24	60
Punica granatum	48	64(33)	13	31	21	53

Values in parenthesis indicate the per cent increase in height of mycorrhizal inoculated plants over uninoculated plants.

Table 7 : Height, mycorrhizal colonization and number of VAM spores in the rhizosphere soils of plants grown in ash pond soil after 9 months of planting.

Plant species	Height of plants (cm)		VAM colonization(%)		Spores/100g soil	
	Control	VAM	Control	VAM	Control	VAM
Tamarindus indica	74	89 (20)	21	42	40	80
Acacia leucophloea	199	244(23)	30	60	40	95
Albizia lebbeck	366	433(18)	23	47	39	75
Azadirachta indica	216	262(21)	25	50	56	90
Casuarina equisetifolia	271	342(26)	31	59	55	106
Bombax ceiba	197	242(23)	27	46	49	86
Dendrocalamus strictus	291	365(25)	34	70	63	125
Eucalyptus spp	330	426 (29)	31	61	44	98
Pongamia pinnata	128	153(20)	24	47	45	81
Tectona grandis	156	196(26)	26	52	47	96

Values in parenthesis indicate the per cent increase in height of mycorrhizal inoculated plants over uninoculated plants.

species and 42%-70% in forest tree species (Table 6-7). In the uninoculated plants, per cent mycorrhizal colonization ranged from 7%-19% in the orchard plant species and 21%-34% in forest tree species. Colonization was higher in forest tree species than in orchard plant species, in both mycorrhizal inoculated and uninoculated plants.

Per cent mycorrhizal colonization was positively correlated with the growth of plants. The height of plants increased due to mycorrhizal inoculation up to 27% in orchard plant species 29% in forest tree species. VAM spore production increased in forest species when compared with orchard plant species. It was due to the size of plant root. In forest plant species, all the plants showed good growth when compared to orchard plants. In the orchard plant species, *Emblica officinalis* responded very well compared with other plant species.

After one year of planting, mycorrhizal inoculation markedly enhanced the growth of plants (Fig. 2-5). Mycorrhizal colonization ranged from 11%-42% and 30%-72% in the control and inoculated plants, respectively. The range of colonization varied from 11%-28% and 30%-62% in the control and mycorrhizal inoculated plants respectively, in orchard plant species (Table 8). In forest tree species, the VA mycorrhizal colonization ranged from 22%-42% and 50%-72% in control and mycorrhizal inoculated plants species (Table 9). Spore counts ranged from 24-82 and 51-136 in the control and *G. mosseae* inoculated plants, respectively. Per cent increase in height of mycorrhizal inoculated plants over control plants was 19%-33% (Table 8-9).

Mycorrhizal inoculation influenced plant growth. Forest plant species are more suitable than orchard plant species for vegetating ash pond (Fig. 6-7).

DISCUSSION

Ash pond soils have very low water holding capacity when compared to the red soil. One of our experiments

Table 8 : Height, mycorrhizal colonization and number of VAM spores in the rhizosphere soils of plants grown in ash pond soil after 12 months of planting.

Plant species	Height of plants (cm)		VAM colonization(%)		Spores/100g soil	
	Control	VAM	Control	VAM	Control	VAM
Achras sapota	78	101(30)	16	49	31	66
Anacardium occidentale	71	94(32)	18	41	30	71
Annona squamosa	96	123(28)	24	48	33	78
Artocarpus integrifolia	130	173(33)	17	36	28	70
Citrus Limonum	49	59(20)	11	30	24	51
Emblica officinalis	228	295(29)	28	62	45	72
Mangifera indica	81	105(90)	17	40	28	58
Murraya konigii	53	64(20)	12	30	25	51
Psidium guajava	146	191(31)	19	43	43	74
Punica granatum	57	74(30)	17	40	32	60

Values in parenthesis indicate the per cent increase in height of mycorrhizal inoculated plants over uninoculated plants.

Table 9 : Height, mycorrhizal colonization and number of VAM spores in the rhizosphere soils of plants grown in ash pond soil after 12 months of planting.

Plant species	Height of plants (cm)		VAM colonization(%)		Spores/100g soil	
	Control	VAM	Control	VAM	Control	VAM
Tamarindus indica	89	108(21)	22	51	46	92
Acacia leucophloea	220	268(22)	34	66	52	104
Albizia lebbeck	380	452(19)	26	53	50	68
Azadirachta indica	232	285(23)	28	58	69	95
Casuarina equisetifolia	292	374(28)	39	66	71	113
Bombax ceiba	211	255(21)	34	50	58	94
Dendrocalamus strictus	319	418(31)	42	72	82	136
Eucalyptus spp	356	463(30)	38	68	63	119
Pongamia pinnata	140	172(23)	28	52	59	92
Tectona grandis	171	219(28)	30	56	62	120

Values in parenthesis indicate the per cent increase in height of mycorrhizal inoculated plants over uninoculated plants.

indicated that ash pond soil retained only 18% of the water added in a simulation column and red soil retained about 27% of the water. One of the additions in the pits, press mud retained about 70% of the water added to the column . Press mud is rich in organic carbon (11%) and contains a moderate level of N (85 kg/acre). The N content in the press mud, red soil and tank silt alleviated the very low level of N content in the ash pond soil. Generally ash is low in N due to volatilization during combustion [12].

Though there are generally high concentrations of P present in ash compared with soil [13], it is not in a form readily available to plants, presumably due to interaction with Al and Fe present in acidic ash and in case of highly alkaline ash, with Ca [3, 14, 15]. Mycorrhizal fungi are known to improve the nutrient status, especially P of Plants, that support plant growth to a large extent. Mycorrhizal colonization and spore prevalence in the rhizosphere soils were positively correlated with the growth of plants. The mycorrhizal hyphae have the ability to act as a substitute for a more extensive root system.

Development of vigorous microbial community can increase the suitability of the ash as a substrate for plant growth [16]. Addition of nitrogen fixing bacteria *Azospirillum* and *Azotobacter* and P mobilizing phosphobacteria compensated the N and P requirements of plants and enhanced the mycorrhizal activity.

Some fly ashes also exhibit pozzolanic properties i.e. they can react with water in the presence of lime to form cement, which can result in reduced infiltration and root penetration in ash deposits and ash amended soils [1, 3, 15,]. But in weathered or lagooned ashes, this property is substantially reduced and can affect the root development only in the deeper layers. Stem and foliar development can indirectly indicate the development of root system and its function.

Since bacterial biofertilizers are added to all the plants, increased growth due to mycorrhizal fungus *Glomus mosseae* inoculated plants can be attributed to the

Fig. 2 : Effect of *G. mosseae* on growth of Artocarpus integrifolia plants

Fig. 3 : Effect of *G. masseae* on growth of Punica granatum in ash pond soil

Fig. 4 : Effect of G. *mosseae* on growth of Anacardium occidentale plants

Fig. 5 : Effect of G. *mosseae* on growth of Mangifera indica plants

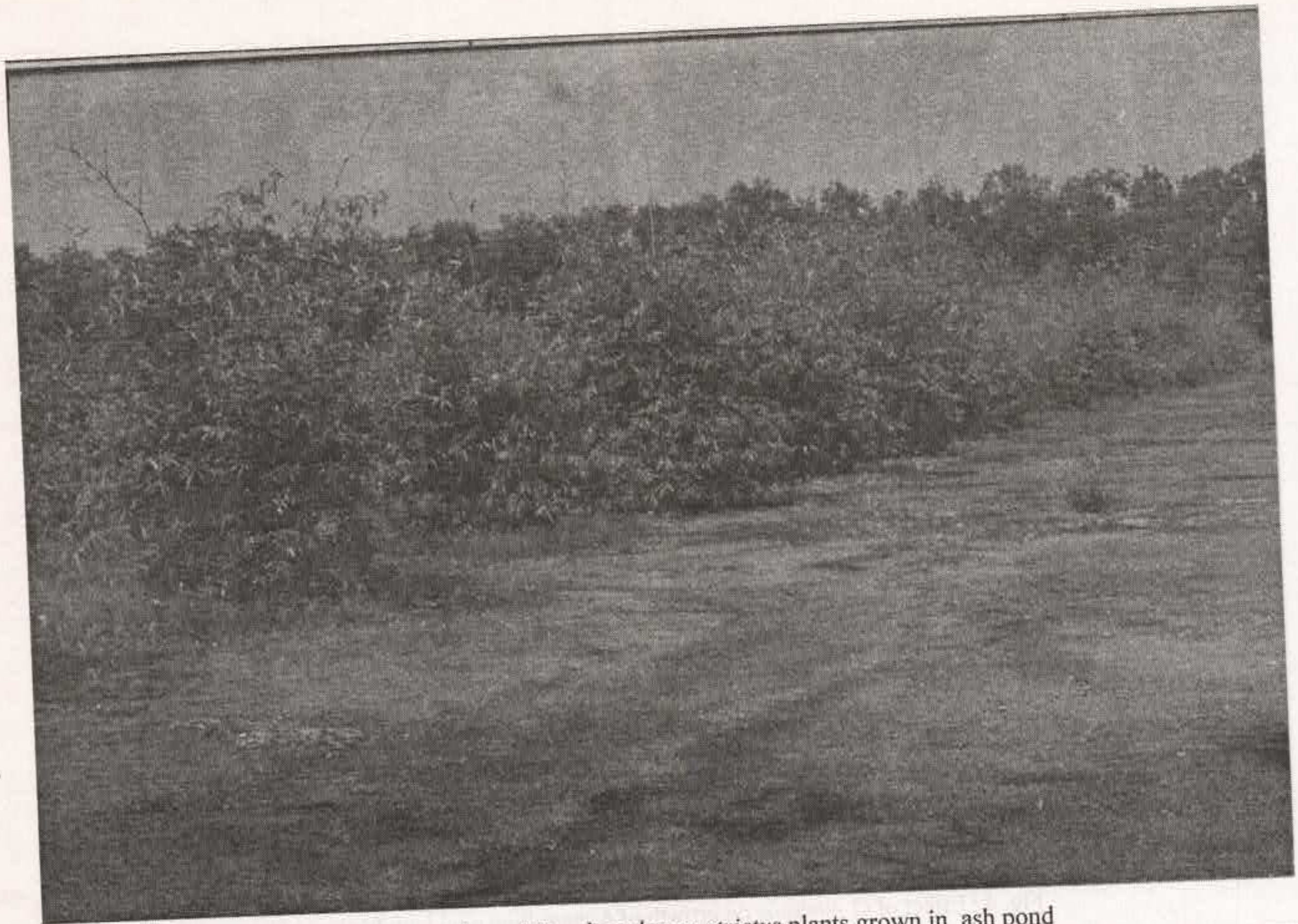

Fig. 6 : An overview of *Dendrocalamus strictus* plants grown in ash pond

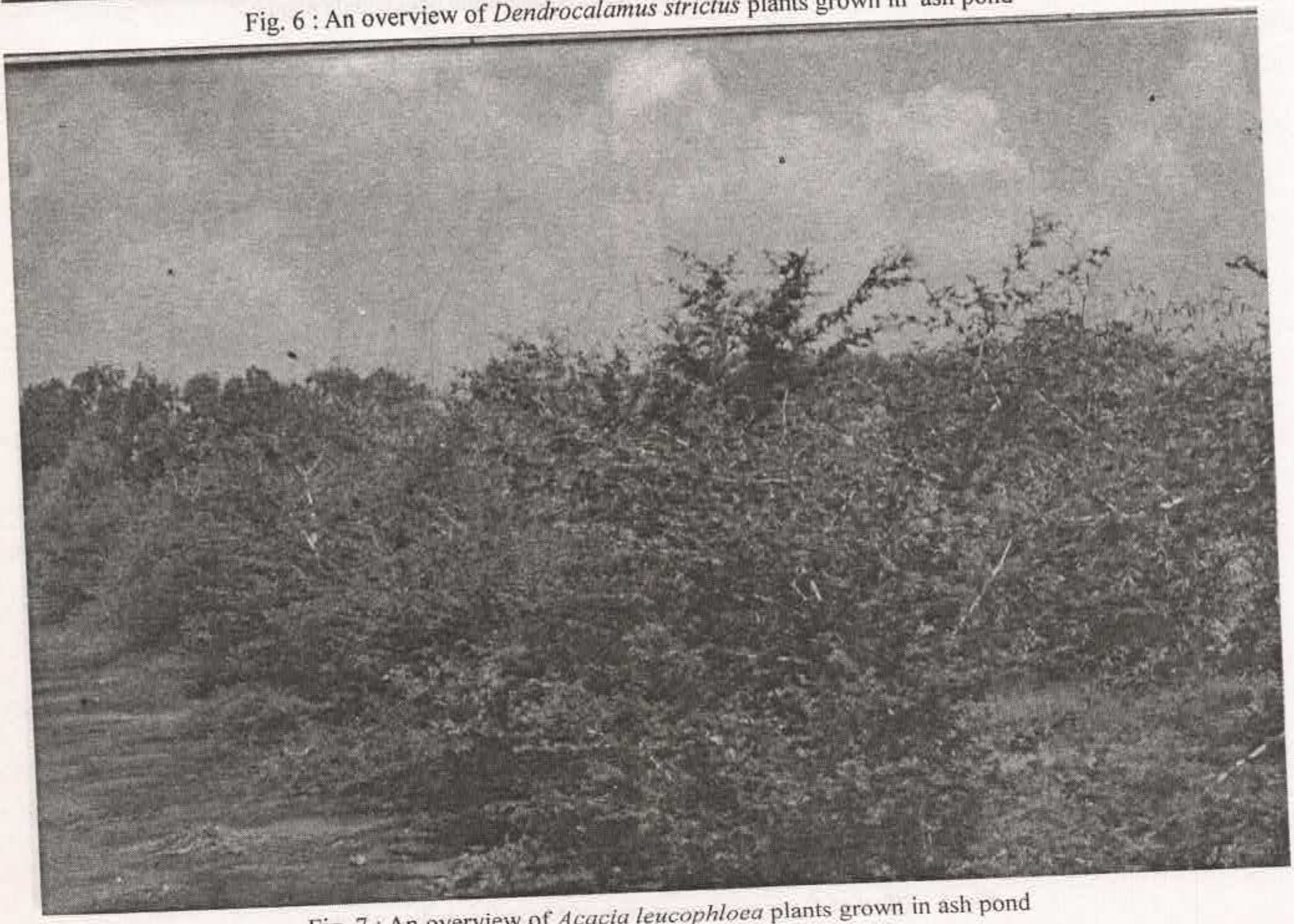

Fig. 7 : An overview of *Acacia leucophloea* plants grown in ash pond

mycorrhizal symbiosis. Clearly biofertilizers can be successfully used to reclaim ash pond soils.

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SOUND ATTENUATION THROUGH TREES : MEASUREMENT AND MODELLING

A. K. Pal*, V. Kumar** and N. C. Saxena***

इस आलेख में झरिया एवं रानीगंज के आठ वानस्पतिक स्थलों (प्लान्टेशन साईट) पर ध्वनि क्षीणता अन्वेषण के परिणामों का चित्रण है। वाइड बैण्ड मोड पर संचालित बी. एण्ड के. प्रकार के 4224 ध्वनि (शोर) स्रोतो से यह पाया गया कि हरित क्षेत्र (ग्रीन बेल्ट) से 50 मीटर गहराई में Leq के लिए झरिया कोयला क्षेत्र में 20.3 से 23.9 dB(A) तथा रानीगंज कोयला क्षेत्र में 20.1 से 24.0 dB(A) ध्वनिक्षीणता है। झरिया कोयला क्षेत्र तथा रानीगंज कोयला क्षेत्र में हरित क्षेत्र के समीप अधिकतम ध्वनिक्षीणता क्रमशः मात्र 3.8 से 5.7 तथा 5.8 से 8.1 dB(A) थी। उच्चतर फ्रिक्वेन्सी 250 Hz के लिए अत्यधिक ध्वनिक्षीणता कम फ्रिक्वेन्सी (150 Hz से अधिक या बराबर) की तुलना में अधिक 5 dB(A) थी। कोयला खनन परिसरों में विभिन्न स्थानों के लिए हरित क्षेत्र की न्यूनतम वांछित मुटाई की गणना करने हेतु हरित क्षेत्र की विभिन्न गहराइयों में संपूर्ण ध्वनिक्षीणता की औसत प्रवृति का मूल्यांकन किया गया। अत्यधिक ध्वनिक्षीणता के लिए वृक्षारोपण स्थलों की विशिष्टता तथा मापी गई ध्वनिक्षीणता के आधार पर मल्टीवेरीएबुल लीनियर संबंध स्थापित किया गया है।

INTRODUCTION

Among the environmental pollutants noise is considered to be a cause of the widespread occupational and community health problems in mining complexes. Adequate control measures have been taken in different countries, which mainly emphasize engineering control for major noise sources. Applicability of these measures is only possible in case of new mines or overall reconstruction of old mines. Since most of the Indian coalfields consist of a mix of old and new mines, it has so far not been possible to integrate the latest noise control technologies in mine planning.

The recent trend is to develop green belts in and around the mining and urban areas to minimise ambient air and noise pollution. It has been regarded as one of the cheapest method of pollution control in the developing countries. In managing noise, sound propagation in the environment has attracted attention of acousticians and environmentalists. Research in this field is directed towards absorption of acoustic energy by the different structures in a plant community (Martens, 1981). Plant leafs absorb acoustic energy by transferring the kinetic energy of the vibrating air molecules in a sound field to the vibration pattern of the leafs. Therefore vibration energy is withdrawn from the acoustic field and part of this energy is lost by transfer to heat since in a vibrating plant leaf friction occurs (Martens et al., 1982). Some pilot studies with Laser-Dopplex-Vibrometer System showed that when a sound field is present, vibration patterns exist over the leaf structures depending on the structure of the leaf. Vibration velocities at 100 dB sound pressure level are between 10^{-5} and $3x10^{-4}$ m/s. However, the exact mechanism of reflection, diffraction and absorption of sound waves around deciduous plant leaves is presently poorly understood (Martens et al., 1981).

The importance of different ground effects encountered in well developed green cover due to formation of highly porous humus layer with lower flow resistivities is noticeable (Piercy et. al., 1979 and Aylor, 1972). Ground effect produces excess attenuation below 500 Hz. Excess attenuation at or above 1 KHz has been attributed to both scattering by trunks and branches and scattering and absorption by foliage [Aylor (1972), Empleton (1963), Martens and Michelsen (1981), and Fricke (1984)].

This paper highlights the results of the systematic noise attenuation studies at eight plantation sites of Jharia and Raniganj coalfields (JCF & RCF respectively) [Saxena et al. (1999)].

CHARACTERISATION OF PLANTATION SITES

Eight plantation sites, four each from JCF and RCF were selected keeping in view of the following.

* Already existing, well developed plantation sites.
* Accessibility from the point of view of studies.
* A minimum width of 50 m.

* Assistant Professdor, Centre of Mining Engineering, Indian School of Mines, Dhanbad-826 004.
** Sr. Research Fellow, Centre of Mining Engineering, Indian School of Mines, Dhanbad-826 004.
*** Professor, CME & Dean, Indian School of Mines, Dhanbad-826 004.

Methodology

Standard quadrate method was used. Two to four quadrates of 10 m x 10 m were considered. The following parameters were observed.

* Aerial height (AH)
* Diameter at breast height (DBH)
* Branchless lower trunk (BLT)
* Canopy branch cover (CBC)
* Vegetation density quantification through vertical light penetration and horizontal light penetration.

NOISE ATTENUATION STUDY

Instruments used

For noise monitoring, B & K Modular Precision Sound Level Meter (Type 2231) and B & K Noise Analyser (Type 2260) were used. Wide band mode of B & K sound source (Type 4224) was used as noise source.

Noise level parameters monitored

MaxL — Maximum Root Mean Square (RMS) level which indicates the sound pressure level (SPL) exceeding 10 percent of the monitoring time.

Leq — The energy equivalent sound level which equals the constant SPL whose acoustic energy is equivalent to the acoustic energy of a fluctuating sound over some time interval.

Monitoring procedure

A systematic noise attenuation study at different plantation sites of JCF & RCF was conducted using B & K sound source (Type 4224) during 1997—98 (monsoon season). Green belt upto 50 m width was considered for monitoring. Ambient sound level in terms of MaxL and Leq was determined at the boarder line, 10 m, 20 m, 30 m, 40 m and 50 m within the green belts. With noise source (placed on ground at about 10 m far off from the border line) in operation noise level parameters were monitored in a similar fashion and average noise levels were determined. The noise monitoring instrument was set on a tripod at 1.5 m height from the ground.

Frequency spectrum analysis (1/1 octave band) was conducted to evaluate frequencywise attenuation at different distances from the source in the green belts.

Meteorological parameters, i.e., wind speed and direction, humidity and temperature were also monitored.

RESULTS AND OBSERVATIONS

Green belt characterisation

From the plantation composition, characteristics, etc. of the sites of JCF and RCF, it was observed that Loudoha forest, Jhanjhra incline mine and Bera plantation sites had more diversified species. In terms of average density (plants/ha) Loudoha College site had the maximum. Average spacing between consecutive plants varied from 1.1 m x 1.0 m to 3.0 m x 1.5 m. Average canopy cover at different plantation sites also varied with respect to different species (Pal et al. 1998).

The ground cover of the sites was studied and dominant species were modified. Vertical light penetration at 1.5 m above the ground and horizontal light penetration were observed as per standard procedure.

Noise attenuation

The total noise attenuation for MaxL and Leq levels in Wide band mode of B&K type 4224 noise source increased steadily with the increase of the width of green belt at all the plantation sites of JCF (Table 3) and was maximum at 50 m width of the green belt as shown in Table 1.

Table 1 : Total noise attenuation at 50. width in JCF

Plantation site	Wide band MaxL dB(A)	Wide band Leq dB(A)
Bera	25.2	25.6
Simlabahal	24.8	23.9
Putki Balihari	21.0	20.3
Kustore	21.0	22.7

Similar total noise attenuation pattern was observed for all plantation sites of RCF (Table 4). Maximum attenuation for both MaxL and Leq was at 50 m depth of the green belt as shown in Table 2.

Table 2 : Total noise attenuation at 50. width in RCF

Plantation site	Wide band MaxL dB(A)	Wide band Leq dB(A)
Loudoha college	21.1	23.2
Loudoha forest	21.1	22.4
Jhanjhra project	20.9	20.1
Jhanjhra incline	24.9	24.0

The noise attenuation in green belt was due to the combined effect of geometric spreading, plantation, ground cover and meteorological effect. Out of the these, the first three are supposed to contribute maximum to noise attenuation, whereas the noise attenuation due to meteorological effect is supposed to be negligible. The

Table 3 : Average total noise attenuation characteristics in different plantation sites of Jharia Coalfield (monsoon season)

Noise Source : B & K Type 4224 (wide band mode)

Locations	Sound attenuation in dB(A)							
	Bera		Simlabahal		Putki Balihari		Kustore	
	MaxL	Leq	MaxL	Leq	MaxL	Leq	MaxL	Leq
At border Line	0.0 (90.1)	0.0 (89.5)	0.0 (87.9)	0.0 (86.3)	0.0 (86.2)	0.0 (85.7)	0.0 (88.4)	0.0 (87.9)
At 10 m	5.7 -0.70	4.9 -0.62	7.0 -0.91	6.0 -0.83	4.2 -0.56	4.2 -0.55	4.9 -0.61	4.3 -0.57
At 20 m	9.7 -110	8.9 -1.06	13.4 -1.13	11.3 -1.18	8.5 -1.03	8.8 -0.99	8.3 -0.98	8.7 -0.99
At 30 m	13.7 -1.10	12.9 -1.12	18.8 -1.37	14.3 -1.25	11.9 -1.19	12.1 -1.18	11.6 -1.16	11.8 -1.15
At 40 m	18.2 -1.27	17.7 -1.31	20.8 -1.44	18.8 -1.40	16.9 -1.35	17.1 -1.30	17.2 -1.31	16.8 -1.27
At 50 m	21.7 -1.40	22.1 -1.57	24.8 -1.60	23.9 -1.55	21.0 -1.40	20.3 -1.36	21.0 -1.40	22.7 -1.56

Figure within ( ) indicates average observed noise level.

Table 4 : Average total noise attenuation characteristics in different plantation sites of Raniganj Coalfield (monsoon season)

Noise Source : B & K Type 4224 (wide band mode)

Locations	Sound attenuation in dB(A)							
	Laudoha College		Loudoha Forest		Jhanjhra Project		Jhanjhra Incline	
	MaxL	Leq	MaxL	Leq	MaxL	Leq	MaxL	Leq
At border Line	0.0 (89.3)	0.0 (88.7)	0.0 (86.3)	0.0 (85.8)	0.0 (86.0)	0.0 (85.4)	0.0 (87.8)	0.0 (82.2)
At 10 m	3.7 -0.86	3.6 -0.87	4.4 -0.86	4.4 -0.87	2.8 -0.58	2.8 -0.57	7.1 -1.01	6.1 -0.84
At 20 m	8.7 -1.11	8.7 -1.12	8.7 -1.11	8.9 -1.13	6.8 -0.99	7.0 -1.00	13.5 -1.14	11.4 -1.13
At 30 m	13.1 -1.11	13.2 -1.12	12.0 -1.10	12.2 -1.11	11.9 -1.12	11.9 -1.13	18.9 -1.30	14.4 -1.12
At 40 m	17.2 -1.30	18.5 -1.32	17.0 -1.30	17.1 -1.31	17.8 -1.32	16.8 -1.30	20.9 -1.40	18.9 -1.31
At 50 m	21.1 -1.34	23.2 -1.60	21.1 -1.36	22.4 -1.55	20.9 -1.48	20.1 -1.38	24.9 -1.59	24.0 -1.60

Figure within ( ) indicates average observed noise level.

ground effect, generally speaking, includes reflecting, scattering and absorption of sound waves.

Noise attenuation in open spaces with more or less similar ground cover and meteorological situation was monitored and it was used to evaluate excess noise attenuation exclusively by the green belts for all the plantation sites of JCF and RCF.

For all the plantation sites, maximum excess noise attenuation for both MaxL and Leq was at 50 m. depth of the green belt as shown in Table 5. The average excess noise attenuation is due to the combined effect of the characteristics of the green belts. However, the contribution of aerial height on excess attenuation seems to be negligible.

Frequency spectrum analysis

Frequency spectrum analysis for all the eight plantation sites, of JCF & RCF showed that the dominant frequency range at border line was 125 Hz to 1 KHz for Wide band mode. The total attenuation for all the sites increased with the width of the green belt in most of the frequencies. The maximum total attenuation at 50 m. width for certain frequencies was as shown in Table 6.

Table 5 : Excess noise attenuation

Plantation site	Wide band MaxL dB(A)	Wide band Leq dB(A)
1. Jharia coalfield		
Bera	4.8	4.4
Simlabahal	6.1	5.2
Putki Balihari	4.8	3.8
Kustore	4.4	5.7
2. Raniganj coalfield		
Loudoha forest	8.3	8.1
Jhanjhra incline	6.2	5.8
Loudoha college	6.6	6.1
Jhanjhra project	6.3	6.2

Frequency spectrum values in open space due to geometric spreading and ground cover were used to evaluate frequency wise excess noise attenuation exclusively by the green belt for all the plantation sites of JCF and RCF.

For all the cases, the excess attenuation was found to be varying with respect to different frequencies. For lower frequencies, i.e., upto 125 Hz, the excess attenuation was relatively less [within 3-4 dB]. The maximum excess attenuation for certain higher frequencies was as shown in Table 7.

Statistical Relationship

Excess noise attenuation values at 50 m. depth of eight plantation sites of JCF and RCF using B & K Type 4224 noise source (Wide band mode) were used to develop a statistical relationship through SPSS package as outlined below.

Dependent variable

X_1 = Excess noise attenuation (Leq) in dB(A)

Independent variables

X_2 = Average density (number of plants/ha)

X_3 = Average aerial height (m)

X_4 = Average canopy branch cover (m)

X_5 = Average diameter of the trunk at breast height (mm)

X_6 = Vertical light penetration

X_7 = Horizontal light penetration

Both vertical and horizontal light penetration were considered to assess canopy and branching coverage of the green belt. Low values of vertical and horizontal light penetration represent more dense green belt. For statistical analysis, suitable weightages were considered for these two parameters [Pal et al. (2000)].

Following multivariable linear relationship was found

$$X_1 = -0.8612 + 3.5296 \times 10^{-4} X_2 - 0.0016 X_3 + 0.0689 X_4 + 0.3196 X_5 + 2.2231 X_6 + 1.8123 X_7 \qquad (1)$$

The correlation coefficient (0.96) and coefficient of determination (0.92) were highly significant. Standard error estimate and analysis of variance justified the significance of the relationship.

Equation (1) indicates that all the independent variables (except X_3) contribute directly towards the excess noise attenuation. Average diameter of trunk at breast height (X_5) seems to contribute maximum (1.5 to 4.7 dB(A) in the entire range of monitored values. This was due to the scattering and reflection of sound waves. Average aerial height (X_3) is not contributing to overall excess attenuation. This was due to the location of the noise source (on the ground) and the monitoring instrument (at 1.5 m height above surface). Inspite of this, aerial height is found contributing negative effect, may be due to back reflection and scattering of some sound waves directed towards higher altitude.

Table 6 : Maximum total noise attenuation at 50 m width for certain frequencies for Wide band mode of noise source

Jharia Coalfield

Bera	Putki Balihari	Kustore	Simlabahal
500 Hz—23.1 dB	500 Hz—20.3 dB	63 Hz—20.3 dB	63 Hz—21.0 dB
1 KHz—20.8 dB	1KHz—20.3dB	500 Hz—20.3 dB	125 Hz—20.6 dB
16 KH_z - 21.1 dB	8 KHz—22.5 dB	2 KHz-20.0 dB	500 Hz—21.6 dB
	16 KHz—24.4 dB	8 kHz - 21.9 dB	8 KHz - 20.7 dB
		16 KHz—23.0 dB	16 KHz—23.5 dB

Raniganj Coalfield

Loudoha college	Loudoha forest	Jhanjhra project	Jhanjhra incline
125 Hz—23.0 dB	500 Hz—20.2 dB	63 Hz—20.5 dB	63 Hz—21.3 dB
2 KHz—26.5 dB	1 KHz—20.2 dB	125 Hz—20.3 dB	500 Hz—21.9 dB
4 KHz—26.0 dB	8 KHz—22.4 dB	250 Hz—20.6 dB	8 KHz—21.0 dB
8 KHz—24.3 dB	16 KHz—24.3 dB		16 KHz—23.5 dB

Table 7 : Maximum excess attenuation for certain frequencies

Wide band mode

Jharia Coalfield

Bera	Putki Balihari	Kustore	Simlabahal
500 Hz—6.9 dB	4 KHz—6.4 dB	4 KHz—7.3 dB	4 KHz—5.6 dB
1 KHz—5.4 dB	8 KHz—7.9 dB	8 KHz - 7.4 dB	8 KHz—6.1dB
	16 KHz—8.4 dB	16 KHz—7.1 dB	16 KHz—7.4 dB

Raniganj coalfield

Jhanjhra project	Jhanjhra incline	Loudoha college	Loudoha forest
250 Hz—5.3 dB	4 KHz—5.9 dB	2 KHz—12.2 dB	4 KHz—6.3 dB
4 KHz—7.9 dB	8 KHz—6.3 dB	4 KHz—12.2 do	8 KHz—7.8 dB
	16 KHz—7.7 dB	8 KHz—9.4 dB	16 KHz-8.3 dB

The excess noise attenuation due to the green belt alone as evaluated using equation (1) was as shown in Table 8. These computed values are close to the actual excess attenuation as observed in the field investigations.

DISCUSSIONS AND CONCLUSIONS

Noise attenuation studies at various plantation sites of JCF & RCF during monsoon season indicated the following range of total and excess attenuation at 50 m. width.

Total attenuation (Leq) : RCF : 20.1—24.0 dB(A).
JCF : 20.3—23.9 dB(A).

Excess attenuation (Leq) : RCF : 5.8—8.1 dB(A).
JCF : 3.8—5.7 dB(A).

This was due to the diverse characteristics of the green belt.

Figures 1 and 2 show the summarised average noise attenuation (both total and excess, Leq) at the plantation sites of JCF and RCF. The total attenuation at different plantation sites of JCF increased linearly with the depth of the green belt, whereas the increasing trend of excess attenuation is less with the depth of the green belt. It is imperative to conclude that the main contributor to higher total attenuation at higher depth is the effect of geometrical spreading. More or less a similar trend was observed for the plantation sites of RCF (Fig.2).

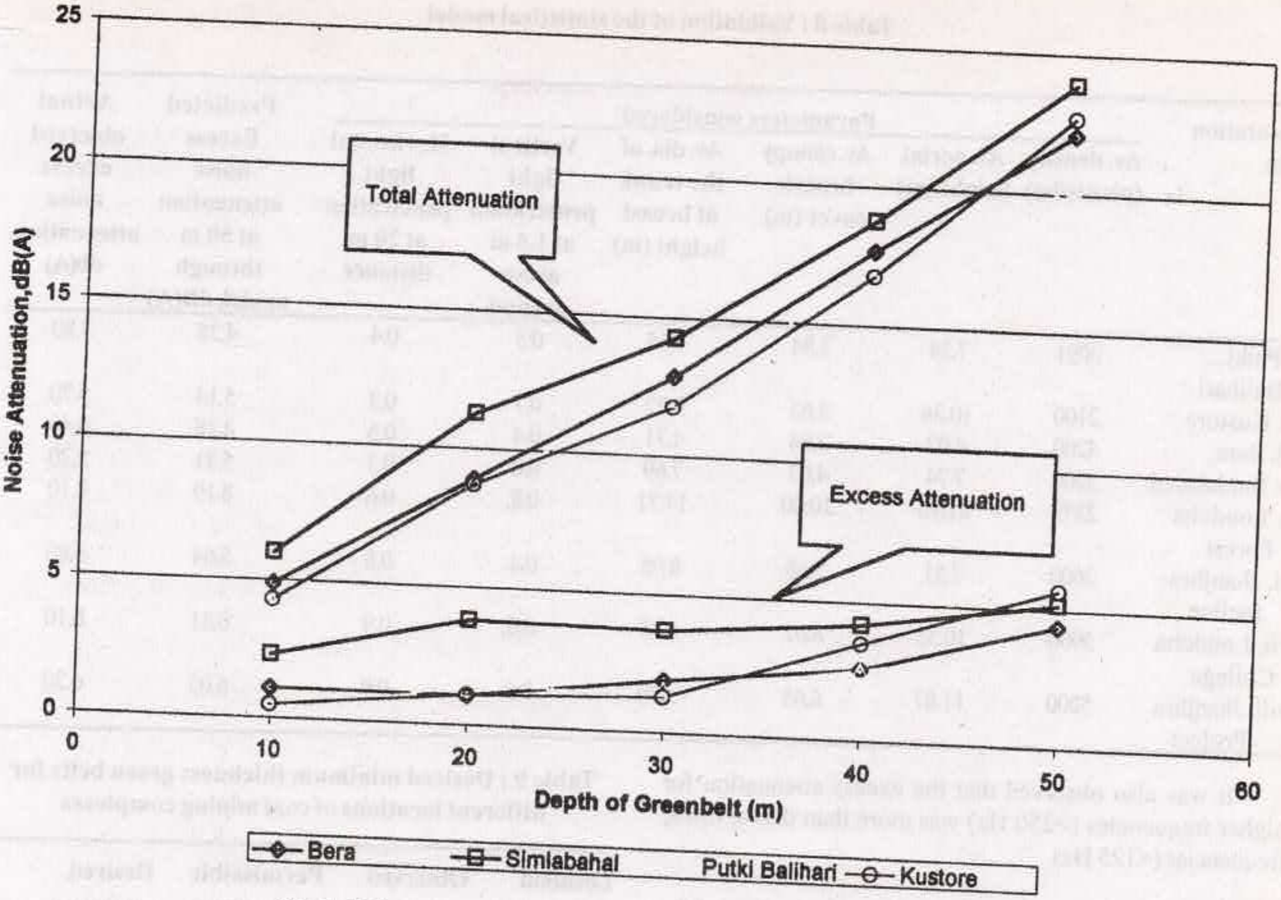

Fig. 1 : Noise attenuation at different plantation sites of JCF (Monsoon)
B & K Noise Source (Wide Band)

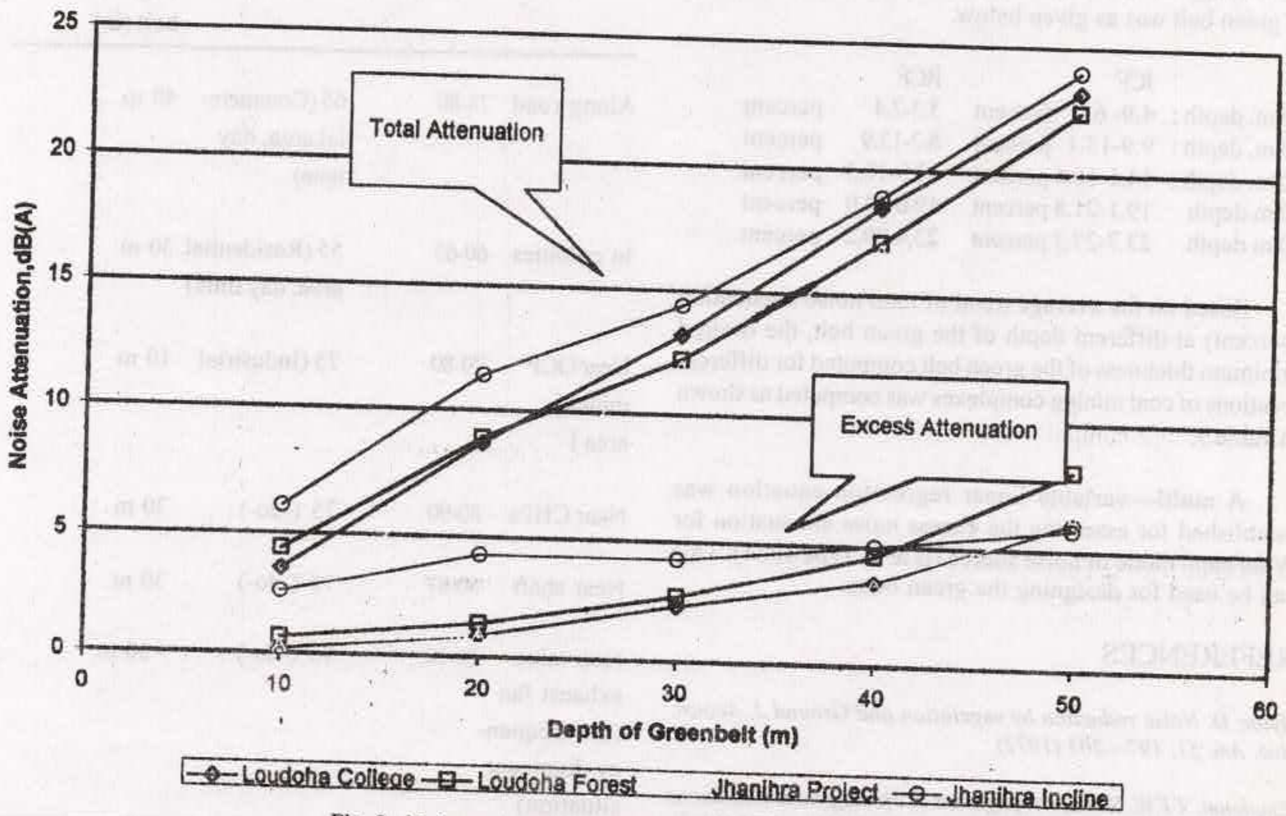

Fig. 2 : Noise attenuation at different plantation sites of RCF (Monsoon)
B & K Noise Source (Wide Band)

Table 8 : Validation of the statistical model

Plantation site	Parameters considered: Av. density (plants/ha)	Av. aerial height (m)	Av. canopy branch cover (m)	Av. dia. of the trunk at breast height (m)	Vertical light penetration at 1.5 m above ground	Horizontal light penetration at 20 m distance	Predicted Excess noise attenuation at 50 m through model, dB(A)	Actual observed excess noise attenuation, db(A)
i. Putki Balihari	2091	7.34	3.84	7.55	0.5	0.4	4.38	3.80
ii. Kustore	2100	10.36	5.65	8.72	0.7	0.3	5.14	5.70
iii. Bera	4200	4.02	2.68	4.71	0.4	0.6	4.28	4.40
iv. Simlabahal	3200	7.74	4.03	7.69	0.8	0.3	5.31	5.20
v. Loudoha Forest	2372	21.76	10.00	14.71	0.8	0.6	8.19	8.10
vi. Jhanjhra Incline	2600	7.51	5.68	8.98	0.4	0.8	5.64	5.80
vii. Loudoha College	5900	10.35	8.61	4.85	0.6	0.9	6.31	6.10
viii. Jhanjhra Project	5800	11.87	6.65	5.20	0.5	0.9	6.03	6.20

It was also observed that the excess attenuation for higher frequencies (>250 Hz) was more than that at lower frequencies (<125 Hz).

From 82 to 89 dB(A) Leq levels at border line of the green belt due to B & K noise source (Wide band mode), the average trend of total noise attenuation with the depth of green belt was as given below.

	JCF	RCF
10 m. depth :	4.9- 6.9 percent	3.3-7.4 percent
20 m. depth :	9.9-13.1 percent	8.2-13.9 percent
30 m. depth :	14.1-16.6 percent	13.9-17.5 percent
40 m depth	19.1-21.8 percent	19.6-23.0 percent
50 m depth	23.7-27.7 percent	23.4-29.2 percent

Based on the average trend of total noise attenuation (percent) at different depth of the green belt, the desired minimum thickness of the green belt computed for different locations of coal mining complexes was computed as shown in Table 9.

A multi—variable linear regression equation was established for assessing the excess noise attenuation for Wide band mode of noise source (B & K Type 4224). This can be used for designing the green belts.

Table 9 : Desired minimum thickness green belts for different locations of coal mining complexes

Location	Observed noise level (Leq), dB(A)	Permissible norm, dB(A)	Desired minimum thickness of green belt (in)
Along road	75-80	65 (Commercial area, day time)	40 m
In colonies	60-65	55 (Residential area, day time)	30 m
Near OCP mines area)	70-80	75 (Industrial	10 m
Near CHPs	80-90	75 (-do-)	30 m
Near shaft	80-87	75 (-do-)	30 m
Near mine exhaust fan (low frequency dominant situation)	85-92	75 (-do-)	>50 m

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FOREST ECOLOGICAL STUDIES OF COAL MINES AREAS OF CHARHI AND KUJU COAL REGION

A.N Prasad*, Binay Kumar Singh* and M. K. Dangi*

चरही तथा कुजु कोयला बेल्ट के विभिन्न क्षेत्रों के पेड़-पौधों का पारिस्थितिकी अध्ययन किया गया है। सभी चुने गए क्षेत्रों से कुल मिलाकर 56 प्रकार के पेड़ों की प्रजातियों को चिह्नित किया गया है। इन विभिन्न पेड़ों की प्रजातियों के प्रतिशत बारम्बारता, सघनता और प्रचुरता का परिकलन किया गया है। जाँच किए गए प्राकृतिक रूप से बढ़ने वाले प्रमुख जंगली पौधों को कोयला खनन क्षेत्रों की अवक्रमिक भूमि पर वानिकीकरण हेतु चुना जा सकता है।

INTRODUCTION

India is a vast country with diversified soil and climatic conditions. Forest plays important role in ecological balance. They regulate water supplies and reduce the incidence of floods and droughts by absorbing rain water which would otherwise, on unprotected land, run-off at a faster fertile soil. Forest have a beneficial role on climate and act as wind breaks and shelter belts.

Due to increase in population, industrialization and large demand for fire wood by rural population have compounded the deforestation problem. The demand for land for various purposes is on the increase. Agricultural and forest land cannot be spared for obvious reasons like continued self-sufficiency for food and long term ecological security. But for the development of country, industries play very important role and mining is a most important industry next only to agriculture.

Hazaribag district consists mainly coal mining areas and comes under a separate Bihar, Orissa and West Bengal geo-botanical region. The following physical divisions have been recognized:

(A) Southern highlands : This is a high land tract which is the southern part of this region. The hills are an extension of the Deccan plateau — Chotanagpur plateau. The rocks belong to the Archean — Dharwar era, including the coal bearing Hazaribag, Dhanbad, Giridih tract, mineral bearing Singhbum tracts and Mica bearing Koderma and Giridih tract.

(B) Ganga plain : The river Ganga flows across central Bihar and then diverges into a series of distributaries to from the famous Sunderbans delta.

(C) Coast: This region comprises of the coastal areas of Orissa and delta of the river Mahanadi. It is a tract of coastal sand of varying width broken by cliffs and lowhills.

Number of studies and research projects have been undertaken by various agencies on reclamation of coal mines areas. But all these studies restricted to revegetation of degraded land without knowing natural ecology of surrounding areas. With this view , the present investigation has been planed to know the natural growing trees plants species of surrounding coal mines areas of different types.

MATERIALS AND METHODS

In the present investigation, the following spots were selected:

(1) First type - Where open cast coal mining is operating:

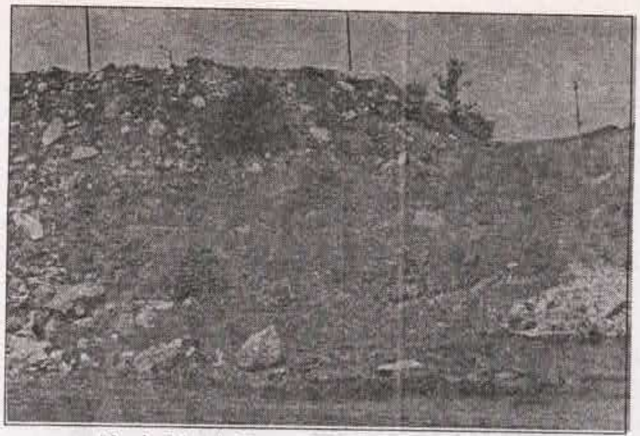
(a) Tapin North open cast coal mines areas,

(b) Tapin South open cast coal mines areas,

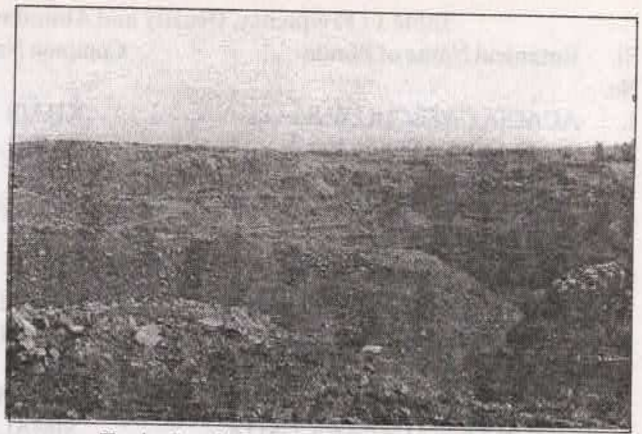
(c) Parez East open cast coal mines areas, and

(d) Kuju open cast coal mines areas.

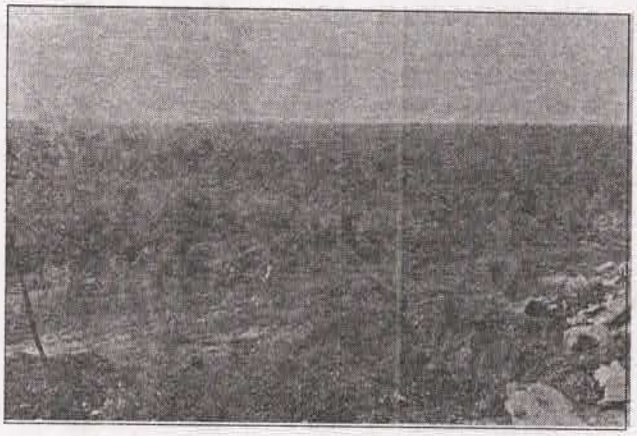
*University Department of Botany, Vinoba Bhave University, Hazaribag — 825 301 (Bihar)



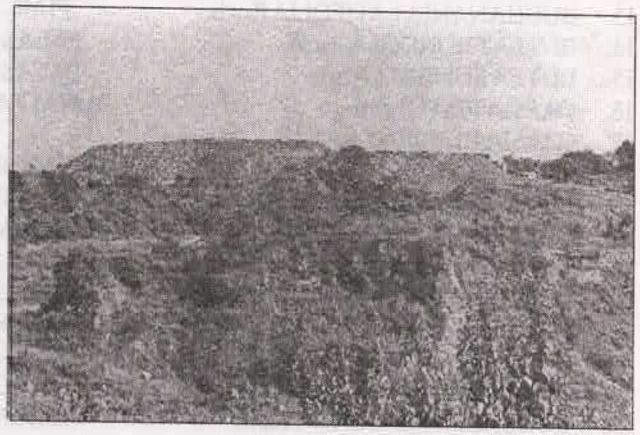
Tapin North Open Cast Coal Mines Area



Tapin South Open Cast Coal Mines Area



Parez East Open Cast Coal Mines Area



Kuju Open Cast Coal Mines Area

(2) Second type - Where coal is reported but mining is not going on:

(a) Parez West project areas

(3) Third type - Green forest near coal mines areas:

(a) Parez Green forest area

(4) Fourth type - Undisturbed forest lands:

(a) Charhi Forest areas and

(b) Canary Forest areas.

The frequency, density and abundance of forest trees surrounding all the selected open cast coal mines areas and undisturbed forest areas were studied by taking observation of five quadrats. Each quadrats size was 30 x 30 m^2. The frequency, density and abundance were calculated by the following formula:

$$\text{Frequency Percentage} = \frac{\text{Total number of quadrats in which species occurred}}{\text{Total no. of quadrats studied}} \times 100$$

$$\text{Density} = \frac{\text{Total number of individuals of the species}}{\text{Total number of quadrats studied}}$$

$$\text{Abundance} = \frac{\text{Total number of the individuals of the species}}{\text{Total no. of quadrats in which species occurred}}$$

After calculation of frequency, density and abundance of all above mentioned coal mines forest areas, their mean value were taken in the Table No.- 1.

RESULTS AND DISCUSSION

Ecological investigations were carried out of above mentioned all spots. After detail work, 56 trees plants were identified (Table 1). The tree plants were identified with the help of standard literature (Haines, 1910, 192*1*, *1925;* Randhawa, *1965;* Santapau, 1967; Sharma et al., 1993). The density, frequency and abundance of all trees plants (56) were calculated.

Table 1 : Frequency, Density and Abundance of Tree Plants of coal mines forest areas

Sl. No.	Botanical Name of Plants	Common Name	Family	Mean Density	Mean Abundance	Mean Frequency%
1.	ACACIA CATECHU Willd.	KHAIR	MIMOSACEAE	0.8	2.3	27
2.	ACACIA NILOTICA Benth.	BABUL	MIMOSACEAE	0.6	1.0	66
3.	ACACIA SP.	-	MIMOSACEAE	1.0	2.5	40
4.	ADINA CORDIFOLIA Hock.	KARAM	RUBIACEAE	0.2	1.0	25
5.	AEGLE MARMELOS Correa.	BEL	RUTACEAE	1.6	2.6	40
6.	ALBIZIA LEBBECK Benth.	SIRIS	MIMOSACEAE	1.0	2.5	60
7.	ANOGEISSUS LATIFOLIA Wall.	DHAUNTA	COMBRETACEAE	1.4	2.5	62
8.	ANTHOCEPPHALUS KADAMBA Miq.	KADAMB	RUBIACEAE	0.2	1.0	20
9.	AZADIRACHTA INDICA A. JUSS.	NEEM	MELIACEAE	0.3	1.0	30
10.	BAHUINIA VARIEGATA L.	KACHNAR	CAESALPINIACEAE	1.5	2.1	60
11.	BOMBAX MALABARICUM DC.	SIMAL	MALVACEAE	0.9	2.2	46
12.	BOSWELLIA SERRATA Roxb.	SALAI	BURSERCEAE	1.1	2.7	65
13.	BUCHANANIA LATIFOLIA Roxb.	PIAR	ANACARDIACEAE	2.2	5.1	50
14.	BUTEA FRONDOSA Roxb.	PALAS	PAPILIONACEAE	12.5	4.5	80
15.	BUTEA SUPERBA Roxb.	PALAS	PAPILIONACEAE	1.4	3.5	40
16.	CASSIA FISTULA L.	AMALTAS	CAESALPINIACEAE	0.4	2.3	58
17.	CASSIA SIAMEA Lam.	-	CAESALPINIACEAE	0.4	2.0	20
18.	CARISSA SP.	KARAUNDA	APOCYNACEAE	7.0	10.3	70
19.	CROTON OBLONGIFOLIUS Roxb.	PUTRI	EUPHORBIACEAE	4.0	7.6	75
20.	DALBERGIA SISSOO Roxb.	SHISAM	PAPILIONACEAE	2.4	6.0	60
21.	DIOSPYROS MELANOXYLON Roxb.	KENDU	EBENACEAE	8.4	10.6	84
22.	ELAEODENDRON GALUCUM Pers.	RATANGURA	CELASTRACEAE	8.0	8.0	90
23.	EMBLICA OFFICINALIS Gaertn.	AMLA	EUPHORBIACEAE	0.4	0.6	20
24.	EUCALYPTUS CITRIODORA Hook.	SAFEDA	MYRTACEAE	1.4	3.5	40
25.	ERYTHRINA INDICA Lamk.	PHRAD	PAPILIONACEAE	0.5	1.7	40
26.	EUGENIA JAMBOLANA Lam.	JAMUN	MYRTACEAE	0.8	2.0	50
27.	EUGENIA SP.	KATH-JAMUN	MYRTACEAE	2.2	3.6	60
28.	FICUS BENGALENSIS L.	BARGAD	MORACEAE	0.4	2.0	40
29.	FICUS GLOMERATA Roxb.	GULAR	MORACEAE	0.2	1.0	24
30.	FICUS INFECTORIA Roxb.	PAKAR	MORACEAE	0.4	2.0	20
31.	FICUS RELIGEOSA L.	PEEPAL	MORACEAE	0.4	2.0	20
32.	GAMELINA ARBOREA Roxb.	GAMBHAR	VERBENACEAE	1.6	2.3	60
33.	HOLARRHENA ANTIDYSENTRICA Wall.	KORAIYA	APOCYNACEAE	9.9	10.6	80
34.	LAGERSTROEMIA PARVIFLORA Roxb.	SIDHA	APOCYNACEAE	4.0	7.6	72
35.	MADHUCA LATIFOLIA Roxb.	MAHUA	LYTHRACEAE	4.2	5.2	75
36.	MANGIFERA INDICA L.	AAM	ANACARDIACEAE	2.4	2.3	60
37.	MANGIFERA PINNATA L. f.	AMRA	ANACARDIACEAE	0.8	1.1	20
38.	MILLINGTONIA HORTENSIS L.	AKAS-NEEM	BIGNONIACEAE	0.6	0.8	20
39.	MILIUSA VELUTINA H. F. & T.	KARI	ANONACEAE	1.7	2.8	50
40.	NYCTANTHES ARBOR-TRISTIS L.	HARSINGAR	OLEACEAE	0.2	1.0	20
41.	OLAX SP.	-	OLEACEAE	0.2	0.6	20
42.	PARKINSONIA ACULEATEA L.	-	CAESALPINIACEAE	0.4	2.0	20
43.	PELTOPHORUM PTEROCARPUM	-	OLEACEAE	0.6	1.0	20
44.	PHOENIX ACAULIS Buch.	KHAJUR	PALME	2.7	4.8	46
45.	PONGAMIA GLABRA VENT.	KARANJ	PAPILIONACEAE	1.0	2.5	63
46.	PETEROSPERMUM CANESCENS Roxb.	MUCHKANDA	STERCULIACEAE	2.3	1.6	40
47.	SEMICARPUS ANACARDIUM L.	BHELWA	ANACARDIACE	6.6	8.2	60
48.	SHOREA ROBUSTA Gaertn.	SAL	DIPTEROCARPACEAE	28.6	32.8	92
49.	TAMARINDUS INDICA L.	IMLI	CAESALPINIACEAE	0.8	2.0	35
50.	TECTONA GRANDIS L.	SAGWAN	VERBENACEAE	2.6	3.1	60
51.	TERMINALIA ARJUNA Bedd.	KAHUA	COMBRETACEAE	1.6	2.6	40
52.	TERMINALIA BELERIC Roxb.	BAHERA	COMBRETACEAE	1.8	2.7	60
53.	TERMINALIA CHEBULA Retz.	HARA	COMBRETACEAE	0.8	2.0	40
54.	ZIZYPHUS JUJUBA Lank.	BER	RHAMNACEAE	2.1	3.8	60
55.	ZIZYPHUS OENOPLIA Mill.	DATHORA	RHAMNACEAE	2.6	6.8	50
56.	ZIZYPHUS XYLOPYRA Willd.	KATBER	RHAMNACEAE	2.6	7.8	40
57.	Few unidentified trees	-				

On the basis of detail observations, it was found that Tapin North open cast coal mines forest areas consists of 29 types of trees while Tapin South open cast coal mines forest areas consists of only 10 tree plants species. Parez East open cast coal mines forest areas consists of 14 types of tree plants while Kuju open cast coal mine forest areas consists of only 16 tree plants species Parez west project forest areas includes 20 tree plant species while in Parez green forest areas, 27 types of tree plants were found. Among undisturbed lands, Charhi forest areas showed the presence of 24 tree plants while Canary forest areas consists of maximum 46 tree species.

Few trees plants species were also found in Tapin North, Tapin South and Parez East open cast coal mines overburden (O.B.) areas in which mainly *Acacia catechu, Cassia siamea, Butea frondosa, Dalbergia sissoo,* ***Eucalyptus*** *citriodora,* ***Ficus bengalansis,*** *Gamelina arborea, Zizyphus* ***jujuba*** were found in overburden dump with low frequency percentage.

On the basis of mean frequency(60% to 100%), mean density and mean abundance of tree plants, the following tree plant species dominated in all selected open cast coal mines and forest areas:

Acacia milotica (Babul), *Albizia lebbeck* (Shiris), *Anogeissus latifolia* (Dhaunta), *Bauhinia variegata* (Kachnar), *Boswellia serrata* (Salai), *Butea frondosa* (Palas), *Carissa* sp. (Karunda), *Cassia fistula* (Amaltas), *Croton oblongifolius* (Putri), *Dalbergia sissoo* (Shisham), *Diospyros melanoxylon* (Kendu), *Elaeodendron galucum* (Ratangura), *Gamelina arborea* (Gambhar), *Holarrhena antidysentrica* (Koraiya), *Lagerstroemia parviflora* (Sidha), *Madhuca latifolia* (Mahua), *Mangifera indica* (Aam), *Pongamia* ***glabra*** (Karanj), ***Shorea*** *robusta* (Sal), *Tectona grandis* (Sagwan), *Terminalia belerica (Bahera), Zizyphus jujuba* (Ber), and few others.

In conclusion, the forest which occur in the investigated areas, falls under following type (A) Dry mixed deciduous forest, (B) Moist mixed deciduous forest and (C) Evergreen forest. It was also observed that above mentioned natural plants species were also found in operating open cast coal mines areas. On this basis, one can conclude that these tree plants can be considered as natural forest tree species of coal mines region. These natural growing forest plants species can be tried for reclamation of surrounding barren lands of open cast coal mines areas.

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DUST FILTERING CAPACITY OF SOME DOMINANT TREES IN OPEN CAST COAL MINES REGION OF HAZARIBAG DISTRICT

A.N Prasad*, Binay Kumar Singh* and M. K. Dangi*

चरही तथा कुजु कोयला क्षेत्र के विभिन्न खुले मुख खदानों के रास्तों के किनारे जन्म लेने वाले और बढ़ने वाले पेड़ों में 15 प्रमुख प्रजातियों पर कोल डस्ट फिल्टरिंग क्षमता तथा रंध्री इन्डेक्स का पता लगाने के लिए अनुसंधान किया गया है। इस संबंध में अच्छी कोल डस्ट फिल्टरिंग क्षमता को बनाए रखने वाले वृक्षों यथा - *B. frondosa, T. grandis, F. bengalensis, M. imdica* तथा *E. jambolana* पर विशेष विचार किया गया है। ऐसा देखा गया है कि वायु विविक्त के संचयन के लिए वृक्ष में पत्तों का क्षेत्रफल ही विशेष महत्व नहीं रखता है। यह भी अवलोकन किया गया कि नियंत्रित एवं प्रभावित दोनों ही क्षेत्रों में रंध्री इन्डेक्स करीब-करीब समान था।

INTRODUCTION

The various direct effects of air pollutants on plants have been reviewed in detail by Hill (1971); Stern (1973); Mansfield (1976); Lal & Ambastha (1980); Agarwal & Khanam (1989) and Prasad & Inamdar (1990).

Open cast coal mines are operating in Hazaribag district. Considerable amount of coal dust particles and poisonous gas (SO^2) are emitted in the environment of Charhi and Kuju open cast coal mines areas. As we know that coal mining is very important industry next to agriculture and it is also critical to the national development. The open cast coal mining will be increased to 61% by 2000. Amount of coal dust and poisonous gas will be also increased and ultimately effect the surrounding coal mines areas.

It has long been suspected that plants green areas can filter out dust, smoke and many other aerosol particles. With this view, dust filtering efficiency and stomatal index studies of different dominant trees leaves of coal mines region were investigated.

MATERIALS AND METHODS

For present investigation, 15 dominant trees species were selected on the basis of percentage frequency. These plants are grown on roadside of the different open cast coal mines areas of Kuju and Charhi coal mines region of Hazaribag district.

The leaves of different dominant forest trees species of the approximately same age and same height were carefully collected in the polythene bags. The deposition of the dust per unit of the leaf surface was measured by removing the dust particles from the surface of the leaf with the help of clear hair brushes and weighing them in a chemical balance and dust / sq.cm of the leaf surface of each plant was calculated.

Stomatal frequency were determined in the separated epidermal peelings of all the leaves of selected dominant plant species. Stomatal Index of each plant was calculated by using the following formula:

$$\text{Stomatal Index} = \frac{S}{E+S} \times 100$$

Where S = Number of stomata per microscopic field area

E = Number of epidermal cells per microscopic field area

RESULT AND DISCUSSION

Table 1 shows that leaf area is not the only character for the collection of air particulates because average leaf area of *B. frondosa* and *T. arjuna* is more or less similar, 72.2 cm^2 and 83.5 cm^2 respectively. It is observed that *B. frondosa* is best dust collector (1.149 mg/cm^2) while *T. arjuna* is observed as bad dust collector (0.371 mg/cm^2). It

*University Department of Botany, Vinoba Bhave University,Hazaribag — 825 301 (Bihar)

Table 1 : Dust filtering capacity and stomatal index studies of some dominant trees in open cast mines region

Sl.No.	Name of the plant	Area of leaf in cm^2	Dust deposited per cm^2	Stomatal index (S. I.)	S. I. of control Areas
1.	AZADIRACHTA INDICA A. JUSS.	06.34	0.015	13.20	13.25
2.	BOMBAX MALABARICUM DC.	23.30	0.135	12.65	12.65
3.	BOSWELLIA SERRATA Roxb.	66.62	0.020	11.85	12.00
4.	BUTEA FRONDOSA Roxb.	72.20	1.149	14.02	14.04
5.	DALBERGIA SISSOO Roxb.	29.80	0.022	13.07	13.07
6.	DIOSPYROS MELONOXYLON Roxb.	56.00	0.155	10.95	11.15
7.	EUGENIA JAMBOLANA Lam	52.60	0.418	18.15	18.23
8.	FICUS BENGALENSIS L.	69.20	0.592	14.72	14.60
9.	FICUS RELIGIOSA L.	39.00	0.410	15.95	16.08
10.	MADHUCA LATIFOLIA Roxb.	85.30	0.267	13.15	13.07
11.	MANGIFERA INDICA L.	62.00	0.484	14.05	14.00
12.	SHOREA ROBUSTA Gasertn.	90.20	0.206	25.02	25.03
13.	TECTONA GRANDIS L.	1140.90	0.694	20.18	20.04
14.	TERMINALIA ARJUNA Bedd.	83.50	0.371	15.75	15.70
15.	TERMINALIA BELERICA Roxb.	60.00	0.183	16.27	16.40

is also noted that smallest leaf area i.e. 29.8 cm^2 of *D. sissoo* has minimum collecting efficiency. On this observation, it is evident that leaf area plays not so important role in collecting coal dust. It is also revealed that leaf surface character is important factor for dust filtering.

In this regard, *B. frondosa, T grandis, F. bengalensis, M indica* and *E. jambolana* are considered having better dust filtering capacity.

It is interesting to note that the upper surface of the leaf collects most of the coal dust particles but the lower surface also collects an appreciable amount of the coal dust (Prasad et al., 2000)

Stomatal index of all selected leaves were also calculated in both affected and normal areas (Table No. 2). Stomatal index was observed more or less similar in both control and affected leaves.

It might be possible that coal dust has not affected the stomatal index because heavy rains in this locality washed the dust frequently.

But to come on conclusion, detail works on air pollution tolerance index should be undertaken by taking more parameter such as chlorophyll content, ascorbic acid estimation, relative water content and others on leaves of trees and shrubs growing in coal mines areas.

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