SOYEBALAM
and
PIYUSH RAI

An innovative technique for improved production from depillaring panel in an underground coal mine

System analysis approach has been used in the present field based research paper for a deep and gassy coal mine of Jharia coalfields. The working panel was considered as a system that was splitted into various sub-systems. The sub-systems were statistically analyzed to compute the production capacity of various sub-systems. Reserve capacity of various sub-systems was also estimated. Additionally, a modified depillaring method has been mooted for enhanced production and productivity level from side discharge loader (SDL) in depillaring panel for given set of manpower and face preparation equipment. Tangible benefits, non-tangible benefits, requirements for its success have also been discussed.

1.0 Introduction

The supremacy of coal as prime source of energy is unlikely to be challenged in foreseeable future, particularly in the Indian context. The demand estimates of 640 million tonnnes and 980.5 million tonnes respectively during the terminal phases of XI and XII Plan respectively also substantiate the larger dependence on coal (Chaoji, 2002; Rai et al, 2005). To cater to the projected demand targets, the Indian coal industry needs accelerated growth in terms of production as well as productivity. The opencast mining, with state-of-art mechanization is likely to become uneconomical beyond a certain depth, as the opencast technology and equipment have already reached a stage of plateau beyond which further growth is mostly unforeseen (Rai, 2001). Karmakar (1996) also expressed that production of coal will depend more on underground mine as against present predominance of surface mines. Looking from this standpoint, the industry may be compelled to increase its share of coal production from underground mines in the coming years, particularly for deep seated reserves which are vastly untapped at this point of time.

At present, the underground coal mines of our country are generally stricken with problem of low production and OMS (0-0.7).This is largely because of manual/semi-mechanized operations in Indian underground mining conditions. Furthermore, the fact that production planning for the underground coal mines is a cumbersome process also casts its influence on production and productivity. As such, immediate attention and investigation into the performance and appropriate planning process for the underground mines is imperative (Bhattacherjee et al, 1996).

Md. Soyeb Alam, Assistant Professor, Department of Mining Engineering, Indian School of Mines, Dhanbad-826004, Jharkhand, India. Email: multisoyeb2004@gmail.com and Prof. Piyush Rai, Department of Mining Engineering, IIT BHU, Varanasi 221005, UP, India

The system analysis based approach appears to offer rationalized solution to solving the complex and intricate production planning from underground mines. In this approach the entire mine (or even a part of the mine) may be considered as system, the capacity of which is dependent on its related sub-systems (Ray et al, 1978). A proper understanding of any system and its related sub-systems could provide intriguing facts in order to practice this approach in any underground coal mine.

2.0 Research objectives

The underground panel as already stated, the present research paper aims at deploying the system analysis approach for a large underground coal mine of Jharia coal fields. The specific aims of the present work are enumerated as:

1. To study the panel as a system and to identify the important sub-systems of the panel.
2. To critically evaluate the sub-systems in terms of production and compute their reserve capacities.
3. To identify the weaknesses and strengths of the system on the basis of critical evaluation of its various sub-systems, in order to enhance the overall system capacity
4. Introduction of an innovative technique to enhance the production and productivity level from the depillaring panel with loading equipment (SDL).

3.0 Relevant details of the mine and mine workings

The present research work was undertaken in a large, privately owned underground mine (‘Mine-A’) of Jharia coal field. The borehole section of the Mine-A has been shown in Fig.1. The study was conducted in a panel of XI seam. The salient geo-mining details of the XIl seam are tabulated in Table 1 and details of panel workings are given in Table 2.

Fig.1: Borehole section of ‘Mine-A’

TABLE 1: SALIENT GEO-MINING DETAILS OF THE COAL SEAM

Item	XI Seam
Avge. thickness	7.31 m
Avge. depth	400 m
Avge. dip	1in 7
Dip direction	S76^o 50' W
Shape of seam	Basin like
Degree of gassiness	II degree
R.M.R. value	34.46 (not good)
Geological features	Dyke
Mode of entry	3 shafts
Type of ventilation	Central ventilation system
Coal transportation	Through incline by belt conveyor up to surface

4.0 Description of panels under study

The study was conducted in the manual as well as semi-mechanized panels in the mine. The description of the panel workings as follows:

(A) SEMI-MECHANIZED PANELS

There were three semi-mechanised panels, namely, panel ‘A1’, B1&C1 in the given mine. Bottom section of panel-A1 had been developed and depillared up to 3.0 m height by bord and pillar (B/P)working in conjunction with stowing. Some portion of bottom section had been left for the purpose of sumping. Depillaring in bottom section had been done by splitting pillar into four parts. After leaving a parting of almost 1.5 m, top section 2.8m was being developed by B/P method of working as shown in Fig.2. the arrows in the Fig.2 indicate the sequence of extraction. Panel consisted of 3 SDLs, loading coal on separate light duty chain conveyor (LDCC). Coal broken by drilling and blasting was loaded on LDCC (maximum capacity = 60 te/hr) by means of side discharge loader (SDL) with bucket capacity of 1.5 m^3. Face conveyors discharged the coal on the belt conveyor (BC) to be carried out through a system of belts up to the coal washery located on the surface. The development plan of panel ‘A1’ is illustrated in Fig.3. Operational plan of SDLs in conjunction with LDCC and BC is shown in Figs.3 and 4. The depillaring method practiced in top section panels ‘B1’ and ‘C1’ was similar to panel ‘A1’ (splitting and slicing), and is represented in Fig.4 for panel-‘B1’ (having a parting of 1.91m and average gallery heights = 3.0m). Bottom section development, depillaring and stowing for ‘B1’ and ‘C1’ was also similar to that of Panel ‘A1’. The galleries, splits and slices in these panels were supported systematically by the roof bolts (having yield load of 5 tonnes) in these panels.

(B) MANUAL PANEL

Only one manual panel in the entire mine namely panel ‘D1’was being worked. Bottom section was developed, depillared and stowed by B/P method. Depillaring plan in bottom section is shown in Fig.2. Panel consisted of 18 miners per shift. Coal broken by drilling and blasting was loaded on LDCC, manually. Mode of coal transportation from this panel was same as that in panels ‘A1’, ‘B1’, ‘C1’. The galleries, splits and slices were supported systematically by the roof bolts (having yield load of 5 tonns) in this panel also.

TABLE 2: DETAILS OF PANEL WORKING IN THE MINE-A

Panel name	Avg.depth (m)	Avg. pillar size(m)	Gallery size (m) (w□h)	Coal preparation method	Loading	Transport
Panel-A1	400	45 □ 45	4.2 □ 2.8, 4.8 □ 2.8 (slice)	Bord & Pillar (dev* & dep**, top sec.)	SDL	Conveyor
Panel-B1	370	45 □ 45	4.2 □ 3.0, 4.8 □ 3.0 (slice)	Bord &Pillar (dep., top sec.)	SDL	Conveyor
Panel-C1	325	45 □ 45	4.2 □ 2.7, 4.8 □ 2.7 (slice)	Bord & Pillar (dev. & dep., bottom & top sec. both)	SDL	Conveyor
Panel-D1	400	45 □ 45	4.2 □ 2.7,4.8 □ 2.7 (slice)	Bord & Pillar (dep., top sec.)	manual	Conveyor

*Development operation;**Depillaring operation

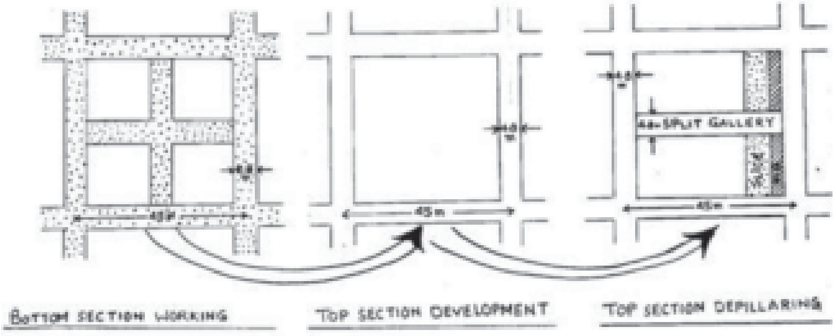

Fig.2: Sequence of coal extraction in bord and pillar mining

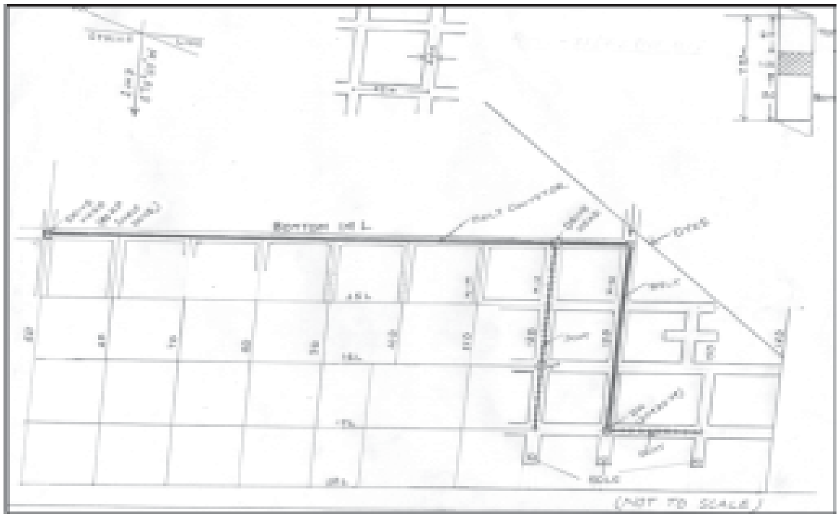

Fig.3: Development panel with dip headings in bord and pillar mining

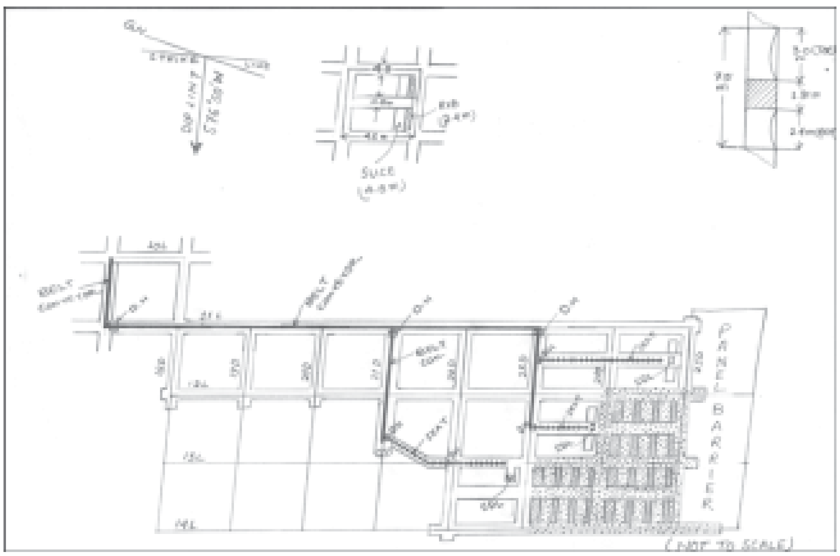

Fig.4: Development with depillaring panel with level splitting in bord and pillar mining

5.0 Research methodology

The concept of system analysis, which entails the breaking of any mining system into smaller sub-systems for finer analysis, has been adopted in the present case study, considering the working panel as a system. Accordingly, the major sub-systems, considering panel as a system, were formulated as llustrated in Fig.5.

From a perusal of Fig.5, it is obvious that panel as a part of mine-A, has been splitted into various sub-systems namely face preparation, loading, transportation, ventilation, stowing, pumping and material supply sub-sytems. Face preparation sub-system has further been splitted into various sub-sub-sytems, namely, production drilling, charging, blasting, fume clearance, water spraying and loose dressing, roof bolting, side bolting, face dressing for better understanding the facel production sub-system.

Actual capacity (in terms of production) of various sub-systems and sub-sub-systems was estimated by conducting time and motion study for various sub-systems and sub-sub-systems. Statistical mean time evaluated from time and motion study was used for actual capacity assessment of various sub-systems and sub-sub-systems as described in appendices I to VIII.

5.0 Results and discussions

5.1 Results of Capacity Estimation

As per the formulated system model (Fig.5) the time and motion studies were conducted at field scale for various sub-systems and sub-sub-systems of face production sub-system. The results used to calculate the capacity of various sub-systems are tabulated in Tables 3 and 4.

Face dimension (width and height) and pull per blast round) were also measured as tabulated in table 5. The mean face dimensions were used for capacity estimation.

5.2 Results of reserve capacity of various sub-systems

The results of reserve capacity of various sub-systems in the study panel are given in Table 6.

From the analysis of the results as given in Table 6 the following important discussion may be drawn:

(i) Production capacity of the panel has been computed on the basis of capacity of individual sub-systems of the panel.

Fig.5: Formulated model for sytsem analysis of a panel system

(ii) Planned daily production capacity is the capacity of the panel planned by the mine management keeping in view the geo-mining conditions.

(iii) Actual production is the production achieved by the mine from the said panel in real-time.

(iv) Reserve capacity of the panel is the difference of the production capacity of the panel and planned production from the said panel.

TABLE 3: MEAN TIME (MINUTES) ELEMENTS OF FACE PREPARATION AND LOADING SUB-SYSTEMS

Operations	Statistical time (mean)
Face preparation time:	
1. Roof dressing per face (minutes)	20.18
2. Roof bolting per bolt (minutes)	7.32
3. Side bolting per bolt (minutes)	3.97
4. Face dressing per face (minutes)	8.84
5. Production drilling per hole (seconds)	80
6. Charging and blasting per face (minutes)	36
7. Fume clearance per face (minutes)	4.78
8. Water spraying and Loose dressing per face (minutes)	17.98
Loading cycle time (seconds):	
1. Lead (15-20m)	99.74

TABLE 4: AVERAGE ELEMENTS OF VENTILATION, TRANSPORTATION AND STOWING SUB-SYSTEMS

Ventilation sub-system	Transportation sub-system	Stowing sub-system
Average intake quantity = 979.6 cu.m/min.	Tipper belt = 20hrs./day	Average stowing rate 93.5 te/hr (on monthly basis)
Average quantity at LVC =718.98 cu.m/min.	Trunk belt = 20 hrs./day	
Average leakages quantity =260.62 cu.m/min.	Sectional belt = 18hrs./day	
	Face chain conveyor =18 hrs./day	

Panel production capacity on the basis of ventilation sub-system is lowest. It has negative reserve capacity of 87 te/day. However, the main fan has capacity to handle a quantity 11326.73 m^3/min. and, only three panel were worked. So, there is tremendous scope to increase the air quantity requirement in the panel up to 2980 cu.m/min. (by minimizing various fan losses) from which panel production could be raised up to 792 te/day (based on face preparation sub-system), which means, optimum utilization of manpower during the shift and up to 3 hours utilization of SDL per shift just twice of existing one.

TABLE 5: MEAN ELEMENTS (METERS) OF FACE DIMENSION (WIDTH AND HEIGHT) AND PULL PER BLAST ROUND

Face dimension & pull per blast round	Mean
Width (m)	4.681
Height (m)	2.513
Pull (m)	1.345

TABLE 6: RESERVE CAPACITIES

	Sub-system	Production capacity (te/day)	Planned production (te/day)	Actual production (te/day)	Reserve capacity (te/day)
1	Face preparation	792	400	393	392
2	Loading	1980	400	393	1580
3	Transportation	2906	400	393	2506
4	Ventilation	313	400	393	-87
5	Stowing	557	400	393	157

Panel production capacity on the basis of stowing sub-system is more than capacity based on ventilation sub-system. It has positive reserve capacity of 157te/day. If the panel production capacity could be raised up to 792 te/day (based on face preparation sub-system) then stowing hours must be more than 55% of planned hours. It could be achieved by minimizing the hours loss for stowing due to various reasons. On the basis of field studies it was observed that many times achieved stowing hours was 60-80% of planned stowing hours. This implies that for matching the face production sub-system capacity (792 te), the 55% fulfilment of stowing hours is not very difficult in the existing field conditions.

Panel production capacity on the basis of transportation sub-system is the highest. This means, transportation sub-system (in term of production) is the strongest sub-system. It has positive reserve capacity of 2506 te/day which means, gross under utilization of transport sub-system. Excessive positive reserve capacity reveals improper matching of transport equipment with respect to coal availability

Panel production capacity on the basis of loading sub-system is more than capacity based on face preparation sub-system. It has positive reserve capacity of 1580 te/day which is indicative of huge under utilization of SDLs. Excessive positive reserve capacity reveals improper matching of loading equipment with respect to availability of broken coal on the faces.

Planned panel production was 400te/day with 3 SDLs but, it could be increased up to 792 te/day (based on face preparation sub-system) by providing 2 faces per SDL per shift in depillaring panel. For solving this issue a new depillaring method is proposed and described herewith.

Panel production capacity on the basis of face preparation sub-system (if two face available i.e. 792 te/day) is more than capacity based on stowing sub-system. It has positive reserve capacity of 392te/day. Panel production capacity on the basis of face preparation sub-system is 792 te/day (when two faces per SDL per shift available) which is up to the mark in terms of optimum utilization of manpower during the shift and at the same time about 3 hours utilization of each SDL per shift(which is just twice of existing one). In development panel average two faces are available for each SDL per shift per heading. This is because of large pillar size at depth of 400m (pillar size being 45m×45m).

An innovative technique of depillaring was contemplated and implemented at the field scale. The salient features of the alternative technique as recorded religiously during the field observations is shown in Fig.5. From the Fig.5 it is evident that instead of one depillaring face (conventional depillaring method Fig.4) in a pillar two faces (1 in dip and 1 in rise) as shown in Fig.5 were worked with deployment of 1SDL with provision of 4 blast round per shift (2 in dip and 2 in rise). The related scheduling of production in a pillar by 4-blast round/shift is given in Tables 7 and 8 for ‘A’ and ‘B’ shift respectively. Similarly, 4 cycles is also possible in C shift hence, the SDL productivity would be 88×3=264 te/day but on every 4th day, the SDL was compulsarily stopped due to stowing. Hence, the average production achieved from one SDLwas almost 200 te/day. The SDLproduction potential in conventional depillaring method is about 130 te/day, much lower than 200 te/day.

Diagonal line of extraction of pillar is shown in Fig.6 and faster rate of extraction in modified depillaring method with respect to conventional depillaring method is shown in Fig.9. From Fig.9 it is evident that on day 31 by conventional method we are able to depillar one pillar, whereas by modified method we can depillar 1.63 pillars, so it is about 1.63 times faster than conventional method but area of exposure is just double hence, abutment pressure will increase in modified method. So, safety is one of the important aspect in this method. Related benefits of this modified depillaring method with respect to conventional depillaring method is tabulated in Tables 8 and 9.

Legend:

1 Shows 1st day extraction

2 Shows 2nd day extraction

3 Shows 3rd day extraction

4th day is preparatory day for barricading etc (not shown in figure)

Similarly 5, 6 show 5th , 6th day of extraction

A, B, C showing different shift per day

Yellow colour showing stowed sand

Barricade

Chock support

REQUIREMENTS FOR SUCCESS OF THIS METHOD

D.G.M.S. permission, 1 shot firer per SDL per shift along

Table 7: Cycle time showing 4 cycles in A shift from 1 SDL

		description	nos	unit time	total time	cycle A1					
10 to 10:15	A	attendance				A					
10:15 to 10:30	B	safety instruction and tavelling									
10:30 to 10:45	C	roof dressing			20						
10:45 to 11	D	stone roof bolting	6	7.32	44	B	C,D				
11 to 11:15											
11:15 to 11:30	E	SDL crew reaching with explosives						cycle A2			
11:30 to 11:45	F	side bolting	4	4	16	F					
11:45 to 12							E	C,D			
12 to 12:15	G	preparation like drill bit fitting , meco tube etc			15	G					
12:15 to 12:30	H	face dressing & drilling holes	15	1.33	9+20	H					
12:30 to 12:45	I	preparation for charging like priming cartridge, sand capsule					I				
12:45 to 1	J	charging and stemming		20	20	J		F			
1 to 1:15	K	blasting and fume clearance		20	20	K		G			
1:15 to 1:30	L	face inspection, water spraying & loose dressing		18	18	L		H			
1:30 to 1:45	M	loading		37	37						
1:45 to 2					219	M		I			
2 to 2:15								J		cycle A3	
2:15 to 2:30								K		C,D	
2:30 to 2:45								L			
2:45 to 3										F	
3 to 3:15								M			
3:15 to 3:30										H	cycle A4
3:30 to 3:45										I	C,D
3:45 to 4										J	
4 to 4:15										K	F
4:15 to 4:30										L	H
4:30 to 4:45											
4:45 to 5								cycle B1		M	I
5 to5:15								A			J
5:15 to 5:30											
5:30 to 5:45											K
5:45 to 6						cycle B2		B	C,D		L
6 to 6:15											
6:15 to 6:30								F			M
6:30 to 6:45						C,D			E		
6:45 to 7								G			

TABLE 8: CYCLE TIME SHOWING 4 CYCLES POSSIBLE IN B SHIFT FROM 1 SDL

4:45 to 5								cycle B1		*M*	I	
5 to5:15								*A*			J	
5:15 to 5:30												
5:30 to 5:45											K	
5:45 to 6						cycle B2		*B*	*D,C*		L	
6 to 6:15												
6:15 to 6:30								*F*			M	
6:30 to 6:45						C,D						
6:45 to 7								*G*	*E*			
7 to 7:15						F						
7:15 to 7:30						H		*H*	*I*			
7:30 to7:45						I		*J*				
7:45 to 8								*K*				
8 to 8:15								*L*				
8:15 to 8:30						J						
8:30 to 8:45								*M*				
8:45 to 9										cycle B3		
9 to 9:15						K				*C,D*		
9:15 to 9:30						L						
9:30 to9:45										*F*		
9:45 to 10						M				*H*		
10 to 10:15										*I*	cycle B4	
10:15 to 10:30										*J*	C,D	
10:30 to 10:45										*K*		
10:45 to 11										*L*	F	
11 to 11:15											H	
11:15 to 11:30										*M*		
11:30 to 11:45											I	
11:45 to 12											J	
12 to 12:15											K	
12:15 to 12:30											L	
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1:15 to 1:30												
1:30 to 1:45												

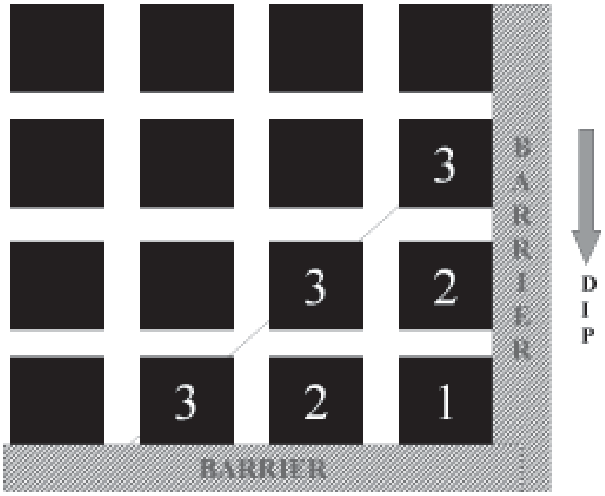

Fig.6: Diagonal line of extraction of pillar (1,2,3 showing sequence of pillar extraction) from bord and pillar method of working (not to scale)

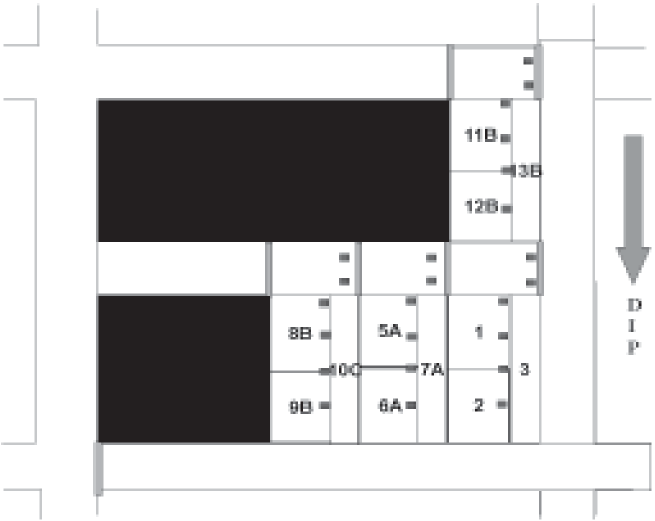

Fig.7: Conventional depillaring method of working (full extraction) in conjunction with hydraulic sand stowing, showing 1 face per SDL per shift (not to scale)

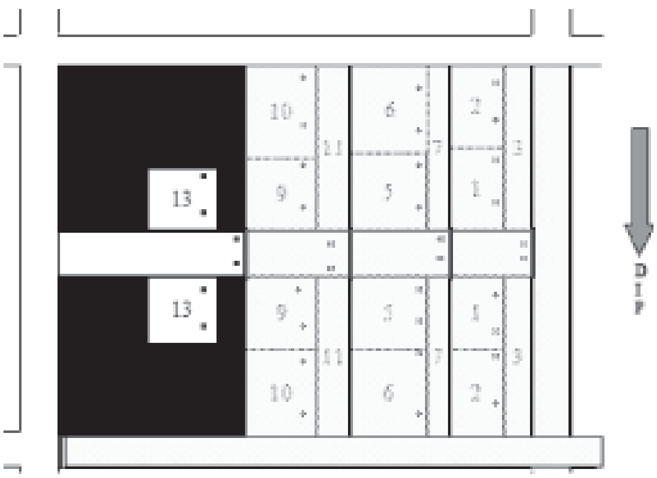

Fig.8: Modified depillaring method of working (full extraction) in conjunction with hydraulic sand stowing, showing 2 faces per SDL per shift (not to scale)

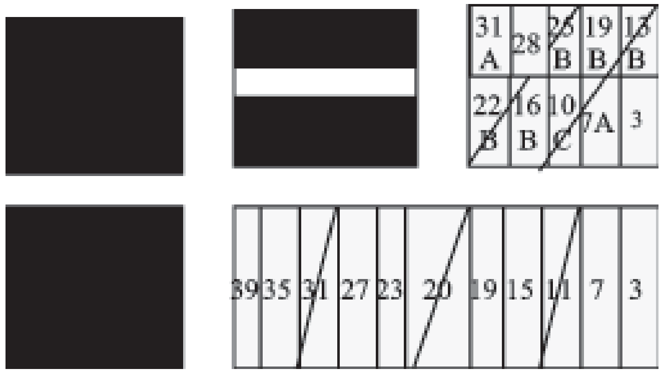

Fig.9: Faster rate of extraction in modified depillaring method w.r.t conventional depillaring method

Table 9: Tangible benefits of modified depillaring method

Description	Conventional method	Modified method
Numbers of skat with 1 SDL	2 (Rs.10 lakhs)	1 (Rs.5 lakhs)
Contract cost of skat installation, SDL marching, gate end shifting for 1 pillar extraction for 1 SDL	8 times (Rs.20 thousand in 1 month)	one time (Rs.2500 in 0.6 month)
Stowing range N-80 for 1SDL	In both main level and split ie 100m (3.5 lakhs)	In main level only ie 50 m (1.75 lakhs)
Annualized saving for 10 SDL		One time:67.5 lakhs Recurring: 21 lakhs

Table 10: Non tangible benefits of modified depillaring method

Description	Conventional method	Modified method
Stowing effectiveness	Usually empty for about 2 feet as stowing pipe is there	Totally pack as stowing from main levels only
Rework and obstruction in path	There is always rework of stowing range near slice as it has always to be removed for SDL fleeting	No such rework
Safety (under consideration)	Slower rate of extraction but area of exposure is half (in 31 days about 1 pillar)	Faster rate of extraction (in 19 days about 1pillar) but area of exposure is double, abutment pressure will increase. This calls for further instrumentation based scientific study.

with 1 mining sirdar and 1 overman in each shift, equipment availability of at least 90% (in present 92%) and stowing with a rate of more than 93.5 tph and at least 13 hours of stowing per day per panel.

6.0 Conclusions

1. System capacity analysis has revealed the discrepancies amongst the planned, actual and capacity production from given SDL panel.
2. System possesses excessive reserve loading capacity because of unavailability of face.
3. Ventilation and stowing are the weakest sub-systems which need proper planning and re-organization to enhance the system production and productivity.
4. From the overall assessment of the system, it appears that system can be easily upgraded from planned (400t/d), and actual (393t/d) to a level of 792t/d by properly organizing the stowing and ventilation sub-systems.
5. The implementation of depillaring with two faces being worked simultaneously, at field scale appears to be tangible benefits in terms of enhancing production as well as productivity by increasing the no. of blast round during depillaring for increased utilization of SDL. Although the method has been adopted on trial basis with no casualties, it needs proper and thorough justification through the instrumentation data.

7.0 References

1. Bhattcherjee, A., Maiti, J. and Mukherjee, N. (1996): “An investigation of system performance of mechanized bord-and-pillar mine working through discrete event simulation modeling”, Minetech Journal., Vol.17, No.6, Nov.-Dec., pp.33-37.
2. Chaoji, S.V. (2002): “Environmental challenges and future of Indian coal”, Journal of Mines, Metals and Fuels, Vol. 50, No.7, July, pp.257-261.
3. Karmakar, H.N. (1996): “New generation mine modeling for higher production and productivity of underground coal mines”, *Minetech Journal*, Vol.17, No.6, Nov.- Dec., pp.39-42.
4. Mathur, S.P. (1999): “Labour productivity and cost reduction in Indian coal mines”. *Minetech Journal*, Vol.11, No. 3, May-June, pp. 5-9
5. Mukhopadhyay, S.B. and Babu, A.V. (1993): “System analysis approach in mining operations-an overview”, Minetech Journal,Vol.14, No.5-6, Sep-Dec., pp.30-36.
6. Rai, P. and Singh, Arvind Kumar (2005): “System analysis approach for production planning in underground coal mine”, *Minetech*, Vol.26, No.2, Mar-Apr., pp.23-28
7. Rai, P. (2001): “Some aspects of material transportation by rear dump trucks in opencast mines”, *Minetech Journal*, Vol.22, No.3, May-Jun., pp.3-10.
8. Ray, S.C. And Sinha, D.K. (1978): “Capacity analysis of mine system-a pre-requisite for production planning in underground mines”, *Indian Mining & Engineering Journal*, Vol.17., No.5, May, pp.7-12.
9. Sachdev, B.K. (1993): “Coal industry in India development issues” Procs. 4th Asian mining international conference on geology, mining and mineral processing. Nov. 24-28, Calcutta.
10. Saha, Pinaki, (2006): “Technical Discussion for Improving Underground Production”, 1st Asian Mining Congress, Jan. 16-18, Kolkata.

APPENDIX–I

W = Average width of gallery (4.681m, as per field observation, Table 5.25)

h = Average height of gallery (2.513m, as per field observation, Table 5.25)

P = Average pull per round of blast (1.345m, as per field observation, Table 5.25)

ρ_c = Density of coal (1.4 te/cu.m)

Production per blast round is given by

$$Q = W \times h \times P \times \rho_c$$
$$= 4.681 \times 2.513 \times 1.345$$
$$= 22.15 \text{ H} \approx 22 \text{ te}$$

APPENDIX–II

PRODUCTION CAPACITY ON THE BASIS OF FACE PREPARATION SUB-SYSTEM

- Roof dressing time per face = 20.18 (minutes)
- Roof bolting time per face = 6×7.32 = 44 (minutes)
- Side bolting time per face = 3.97×4 = 16 (minutes)
- Face dressing time per face = 8.84 (minutes)
- Production drilling time per face = 1.33×15 = 20 (minutes)
- Charging and blasting time per face = 36 (minutes)
- Fume clearance time per face = 4.78 (minutes)
- Water spraying and loose dressing time per face = 17.98 (minutes)

As per Table 3

Total time taken for one face preparation =

20.18+44+16+8.84+20+36+4.78+17.98 = 167.98 minutes = 2.78 hrs.

- Shift utilization time = 6 hrs. = 360 minutes. Excluding travelling time, safety instruction time, unavailability of equipment/power and rest (there are 3 overlappingproduction shifts of 8 hrs. per day.A-shift 10am-6pm, B-shift 5pm- 1am and C-shift 12pm-8am)

If one face is available per SDL per shift:

- No. of blasting per shift per SDL = 6/2.78 2
- Production per shift per SDL = 2×22 = 44 te (production per blast round= 22 te)
- Production per day per SDL = 3×44 = 132 te
- Production per day from studied panel = 3×132 = 396 te (panel consisting 3 SDLs)

If two faces available per SDL per shift:

- No.of blasting per shift per SDL = 6/1.39 4
- Production per shift per SDL = 4×22 = 88 te (production per blast round= 22 te)
- Production per day per SDL = 3×88 = 264 te
- Production per day from studied panel = 3×264 = 792 te (panel consisting 3 SDLs)

APPENDIX–III

PRODUCTION CAPACITY ON THE BASIS OF LOADING SUB-SYSTEM

- Nominal bucket capacity (Q_n) = 1.5 cu.m.

- Actual bucket capacity (Q_c) = $Q_n \times \quad \times \rho_c$

Where,

f_f = Fill factor = 0.8
S_f = Swell factor = 1.2

- $Q_c = \dfrac{1.5 \times 1.4 \times 0.8}{1.2} = 1.4\,\text{te}$
- Av. time of cycle = 1.67 min (at lead = 15-20 m, Table 3)
- Time taken to load one blasted coal = (22/1.4) × 1.67= 27 minutes
- About10 minutes for cable handling problem, chunk in the bucket etc.
- Total time taken to load one blasted coal = 27+10 = 37 minutes
- No. of blasting loaded per shift/SDL = 360/37= 10
- Coal loaded per shift/SDL = 22×10 = 220 te
- Coal loaded per day/SDL= 3×220 = 660 te
- Coal loaded per day from 3 SDL (panel production) = 3×660 =1980te

APPENDIX–IV

PRODUCTION CAPACITY ON THE BASIS OF TRANSPORTATION SUB-SYSTEM

Carrying capacity of light duty chain conveyor (LDCC) in te/hr. with uniform feeding is

Q_{fcc} = (Rectangle cross-sectional area + Trapezoidal cross-sectional area) $\times \psi_f \times \rho_c \times v_{fcc} \times 3600$

Where,

Rectangle cross-sectional area + trapezoidal cross-sectional

area = .430×.220 + .5 (.602+.430) ×.184 = .1895 m²

ψ_f = Fill factor = .8

ρ_c = Density of coal = 1.4 te/m³

v_{fcc} = Velocity of fcc = .8 m/s

So, Q_{fcc} = 0.1895×0.8×1.4×0.8×3600 = 611 te/hr.

Operating hrs. per shift = 6

Operating hrs. per day = 18

Carrying capacity per shift = 6×611 = 3666 te

Carrying capacity per day = 3×3666 = 10998 te

Transport capacity of belt conveyor in te/hr. with uniform loading at trough angle 30º (maximum) is given by

Q = (w²/7) × $\psi_f \times \rho_c$ × V×3600

Where,

w = Width of belt in meters

ψ_f = Fill factor = .8

ρ_c = Density of coal = 1.4 te/m³

V = Speed of belt conveyor in m/s

Carrying capacity of sectional belt conveyor is given by

Q_{sbc}= (w^2_{sbc}/7)×$\psi_f \times \rho_c \times V_{sbc}$×3600
(.9144²/7)×0.8×1.4×1.62×3600 = 780 te/hr.

Operating hrs. per shift = 6

Operating hrs. per day = 18

Carrying capacity per shift = 6×780 = 4680 te

Carrying capacity per day = 3×4680 = 14040 te

Carrying capacity of trunk belt conveyor is given by

Q_{tbc}= (w^2_{tbc}/7) × $\psi_f \times \rho_c \times V_{tbc}$×3600
(.9144²/7) ×0.8×1.4×1.62×3600 = 780 te/hr.

Operating hrs. per day = 20

Carrying capacity per day = 20×780 = 15600 te

Carrying capacity of tipper belt conveyor is given by

Q_{tbc}= (w^2_{tbc}/7) × $\psi_f \times \rho_c \times V_{tbc}$×3600
= (.9144²/7)×0.8×1.4×1.71×3600 = 823.55 te/hr.

Operating hrs. per day = 20

Carrying capacity per day = 20×823.55 = 16471 te

- Total no. of SDL (Mine-'A' and Mine-'C') 9+8 = 17
- Tipper belt carry coal of both the mine

Hence,

- Production capacity per day per SDL on the basis of tipper belt conveyor = 16471/17 = 968.88 te
- Production capacity per shift per SDL on the basis of tipper belt conveyor = 323 te
- Production capacity per day with 3 SDL (panel production) = 3×968.88 = 2906.06 te

$\dfrac{f_f}{S_f}$

APPENDIX–V

PRODUCTION CAPACITY ON THE BASIS OF VENTILATION SUB-SYSTEM

$$Q_v = \left[60 * 24\{q_{\max} - (q_1)\}(d_1 - d_0)/(100 * a * k_1 * k_2)\right]$$

Where,

Q_v= Production capacity per day based on ventilation (te)

q_{max}= Maximum quantity of air entering in the panel = 979.6 cu.m/min. (as per Table 4)

q_l = Leakage losses(cu.m/min) = 260.62 cu.m/min. (as per Table 4)

d_1 = % of CH_4 in main return (0.5 to 0.75%, 0.5% is safer side)

d_0 = % of CH_4 in intake air (normally 0%)

a = Gassiness of seam expressed in cu. M of CH_4 per tonnne of daily production. (10 cu.m/te, maximum for second degree gassy mine as per Indian categorization)

k_1 = Factor which takes into account the unbalanced distribution of air in mine workings (1.1)

k_2 = Co-efficient which takes into account the variation of production level during the day (1.5)

Q_v = {60×24 ×(979.6- 260.62) × 0.5}/(100×10×1.1×1.5) = 313 te

Production capacity per day with 3 SDL (panel production)=313te

Production capacity per day per SDL = 104.57 te

Production capacity per shift per SDL = 34.85 te

APPENDIX–VI

QUANTITY OF AIR REQUIREMENT IN A DISTRICT IS CALCULATED ON THE BASIS OF FOLLOWING THREE NORMS

1. In every ventilation district, not less than 6.0 cu.m/minute per person employed in the district in the largest shift, passes along the last ventilation connection in the district.
2. In every ventilating district, not less than 2.5 cu. m/minute of air per daily tonne output, passes along the last ventilation connection in the district, whichever is larger.
3. The percentage of inflammable gas does not exceed 0.75 in general body of return air and 1.25 in any place in the mine.

Quantity of air requirement in a district on the basis of first norm

Q_{lvc} = 6×149 = 894 cu.m/min.

Quantity of air requirement in a district on the basis of second norm:

Q_{lvc} = 2.5×400 = 1000 cu.m/min

Quantity of air requirement in a district on the basis of third norm:

Q_{intake} = 413.54 cu.m/min (at 0.75% inflammable gas in general body of return air)

= 621.87 cu.m/min (at 0.5% inflammable gas in general body of return air, safer side)

APPENDIX–VII

PRODUCTION CAPACITY ON THE BASIS OF STOWING SUB-SYSTEM

- Effective hrs. of stowing per day = 24×0.39 = 9.36 (as per Table 4)
- Stowing per day achieved = 9.36×93.5 = 875.16 te (as per Table 4)
- Production capacity per day from panel = (875.16 × 1.4)/2.2 = 556.92 ≈ 557te

Where,

1.4 = density of coal (te/cu.m)

2.2 = density of sand (te/cu.m)

- Production capacity per day per SDL = 557/3 = 185.64 te
- Production capacity per shift per SDL = 185.64/3 = 61.88 te

APPENDIX–VIII

PRODUCTION CAPACITY PER SDL ON THE BASIS OF SHOT FIRER

- Exploder: multi shot exploder
- CMR 1957, Regulation ‘166’ sub clause (a) in case of gassy seam of the second or third degree or a fiery seam, forty, if a single-shot exploder is used and eighty, if a multi-shot exploder is used by per shot firer per shift.
- No. of shot firer per shift = 2
- No. of shots fired per shift = 2 × 80 = 160
- No. of faces blasted/shift = 160/15 = 10.66
- No. of faces blasted/day = 10.66 × 3 = 32
- Production capacity per day from 3 SDL (panel production) = 22 × 32 = 704te.
- Production capacity per day per SDL = 234.66 te
- Production capacity per shift per SDL = 78.22 te