Optimum Utilisation of Continuous Miner for Improving Production in Underground Coal Mines

Vijaya Raghavan*, Dr Syed Ariff, Paul Prasanna Kumar*****

*Department of Mining Engineering, Dr.Thimmaiah Institute of Technology, Oorgaum, Kolar Gold Fields-563120
**Assistant Professor, Professor, Senior Lecturer.
***Department of Mining Engineering, Dr.T.Thimmaiah Institute of Technology, Oorgaum, Kolar Gold Fields-563120

Abstract- Conventional underground coal mining relies upon the use of continuous miners in order to extract coal reserves from underground coal seams. In combination with the continuous miners- shuttle cars are used to transport the extracted coal from the face to a transfer point (feeder breaker). From there the coal is typically tipped onto the underground conveyor system, which transports the coal to the surface in order to be distributed to customers. Effective management of the cutting, loading and tipping cycles utilised will serve as a possible area for productivity improvement.

I. Introduction

The phenomenal growth of the Indian coal mining Industry was entirely due to opencast mining, which proved to be more productive and economical to cope with growing demand of coal. With economic liberalization and consequent reduction of import duty on coal, India can no longer rely on opencast mining alone. Introduction of broad scale mechanization in underground coal sector, the production from which otherwise remained stagnant has become imperative with an aim to improve the overall quantity of coal bridge the ever increasing demand-supply gap of coal. At present, there is a need to introduce technology in Indian coal mines to boost the production from underground with due regard to the cost of production, productivity, profitability and safety.

With the increase in coal demand and growing awareness towards sustainable development, the coal industry has drawn a consensus over the need for increased production from underground coal mines. From the current share of 15 per cent, the industry aims to reach a total coal production of 30 per cent from underground mines by 2030.

The Indian coal industry is the world's third largest in terms of production and fourth largest in terms of reserves. Around 70% of the total production is used for electricity generation and the remaining by the steel, cement and other heavy industries. Coal is also used as fuel for domestic purposes. Despite having one of the largest reserves, the Indian coal industry does not hold a position in the league of global energy suppliers. This can be attributed to the soaring domestic demand.

In India the underground coal production is nearly 20% of the total coal production of the country. This is despite the fact that about 70% of the country's coal reserve is amenable to be worked by underground methods. This shows the slow pace of technology induction in underground coal mines and there is enormous scope for improvement. To meet the ongoing demand of coal in the country, immediate attention is required for improvement in the production from underground mines.

Underground mining needs to adopt mechanisation to reach the desired coal production levels. In this respect, the Continuous miner technology in board and pillar mining is the best option, which does not require virgin areas, whereas it can also be applied where development has already been done. Since board and pillar system is a well-proven technology in India. This technology helps in achieving high production and faster rate of extraction with safety.

II. LITERATURE REVIEW

2.1 Introduction

Coal seams can be mined both by underground methods and surface mining methods depending
On certain conditions like:

- Thickness of seam
- Dip of seam
- Depth of occurrence
- The ratio of overburden to coal (stripping ratio)

There are two basic methods of underground coal mining methods. They are: i) Bord and Pillar method ii) Longwall method. Although the basic principles remain the same, there could be many variants of these two methods.

In India, about 98% of underground output of coal is obtained by Bord and Pillar method and barely about 2% by longwall methods. The other countries where Bord and Pillar method predominates are Australia, USA and South Africa.

2.2. Bord and Pillar Method

Fundamentally, the bord and pillar method of mining coal seams involves the driving of a series of narrow headings in the seam parallel to each other and connected by cross headings so as to form pillars for subsequent extraction, either partial or complete, as geological conditions or necessity for supporting the surface, may permit. Ideally, the pillars should be square but they are sometimes rectangular or of rhombus shape and the galleries surrounding the pillars are invariably of square cross-section.

The bord and pillar method of mining is suited to work flat coal seams of average thickness and at shallow depths. Coal seams of 1.8 to 3 m thickness are best suited for this method, though the method has been successful in thinner seams also down to a thickness of 1.2 m and in thicker seams up to 4.8 m in thickness. In seams with gradients of more than 1 in 4 difficulties are encountered in maneuvering machines. Most of the bord and pillar method of coal mining has been done in depth range less than 300 m as at greater depths pillars experience crush. However, in India in some cases the depth of mining for bord and pillar method has been around 600 m, though mining at such depths is beset with the problems of strata control. Sometimes, low strength of coal limits the depth to which bord and pillar mining can be done.

2.3. Longwall Method of Mining

The high productivity and production associated with Longwall method of mining has made this method the most popular method of mining. The current trend is towards adopting this method of mining.

The Longwall method of mining coal involves the extraction of the panel of coal to be worked by advancing the face forward (in the case of advancing Longwall) on a wide front leaving behind the roadways serving it which are supported by packs of stone or other pickings in the area of extraction. The face is retreated on the roadways driven before opening out the face and as the face is retreated backwards, the goaf is allowed to cave in or it is filled and the gate roadway is lost in the goaf. This method can be employed almost in all geological conditions, though it is eminently suited for working thinner seams, i.e., seams less than 1.8 m thick. On the lower side, seams of up to 0.3-0.35 m thickness have been worked by this method. Coal from a Longwall especially in a thin seam is the cheapest coal a mine can produce. It is also desirable that thick seams (more than 4.8 m) be worked by this method in slices of 2 to 3 m. It can be practiced in seams at depth and also in gassy seams or seams prone to spontaneous heating.

2.4. Productivity:

Traditionally, the output per man shift measured in terms of tonnes in coal mines in India, has been low when compared to other major coal producing nations. Though the opencast mines have recorded a consistent increase in productivity over the years, the underground OMS is hovering around 0.7 t. The geo-mining and socio economic conditions in some of the developed countries are much different than ours. Targets for the productivity should be based on the prevailing conditions, optimum level of mechanization, automation, cost considerations, constraints or continued availability for imported items, etc. There is a need for benchmarking productivity targets for mining operations.

III. METHODOLOGY

3.1. Overall working of bord and pillar method of mining using Continuous Miner:

The bord and pillar method of mining is the older and traditional method of underground coal mining. This method is sometimes called room-and-pillar mining. Coal is removed from the working faces as the rooms are advanced. Cross-cuts, connecting the rooms are also mined leaving pillars of coal for support. The rooms and cross-cuts are typically about 20 feet wide and of a height consistent with that of the seam. Prior to the development of continuous mining technology, the conventional room and pillar method was composed of undercutting the coal, drilling, blasting, and loading.

3.2. Development

In case of Bord and Pillar, two sets of galleries, one normally perpendicular to the other, are driven forming pillars between them of size that currently depends on depth and size (width) of the gallery. The ultimate method of pillar extraction presently does not influence shape and size of pillar. However, the ultimate method of pillar extraction should also be taken into consideration while forming these pillars. This is one of the important factors for deciding the size and shape of the pillar. In the present scene of underground mine development by Bord and Pillar system, mostly square pillars are being formed of size dictated only by depth and width of galleries under the Coal Mine Regulations.

3.3. Support system during development

Considering that roof falls causes the largest number of mine accidents, it was decided a few years ago to support a 9 m length of a gallery immediately out bye of the working face. These supports may be temporary or permanent in nature. If temporary, they can be replaced by permanent supports, if the roof conditions so dictate or can be taken out completely, if the roof stratum is found to be self supporting. However, the current trend is to consider that practically no strata is self- supporting for the size of development galleries normally driven. Now, mine managements have to necessarily prepare support plans for the mine as a whole. The above stipulation and the past experience have encouraged installation of roof bolts in Indian Coal mines. Roof bolting as the sole system of support has been accepted by Directorate of Mines Safety for mine development galleries. Roof bolting, now, is beginning to be accepted as the sole system of support in depillaring areas and for certain geo-mining and operating conditions. Conventional supports in depillaring areas can be reduced if roof bolts are also used. Shiftable hydraulic roof bolting machines are being popular.

3.4. Pillar extraction

After pillars have been formed on the Bord and Pillar system, consideration has to be given to the extraction of coal pillars; the operation is known as pillar extraction. It is also referred to as depillaring. In a method of depillaring, known as the caving method, the coal of pillars is extracted and the roof is allowed to break and collapse into the voids or the de coaled area, known as goaf. As the roof strata about the coal seam break, the ground surface develops cracks and subsides, the extent of damage depending upon depth, thickness of the seam extracted, the nature of strata, thickness of the subsoil and effect of drag by faults. Depillaring with stowing is a method of pillar extraction in which the goaf is completely packed with incombustible material and generally plasticized where it is necessary to keep the surface and strata above the seam intact after extraction of coal. The following circumstances would require adoption of depillaring with stowing:

- Presence of water bearing strata above the coal seam being extracted. Enormous quantities of water beyond the economic pumping capacity may enter the mine through cracks in the strata.

- Railways, rivers, roads, etc. situated on the surface, which cannot be diverted.
- Presence of fire in a seam above the seam to be extracted.
- Existence of one or more seams of marketable quality extractable in the near future.
- Restrictions imposed by local or Government authorities for the protection of the surface.
- Extraction of the full thickness of a seam thicker than 6 m, as thicker seams cannot be extracted fully by caving method.
- Extraction of seams very prone to spontaneous heating, of very gassy nature or liable to pumps.
- Surface buildings which cannot be evacuated.
- Tanks, reservoirs, etc. which cannot be emptied.

3.5. Principles of Pillar extraction techniques

The principles of designing pillar extraction techniques are given below:

- Roof exposure at one time should be minimal. In the Indian coalfields, where caving is practiced, 60-90 m^2 exposure is normally allowed. But in stowing districts the exposure may be increases up to 90-100 m^2.
- The size of the panel should be such as depillaring can be completed within the incubation period. This period commonly varies between 6-9 months. But there are some seams in which fire has not occurred even thought depillaring has been going on for more than two years and yet there are some seams in which spontaneous heating has been reported within three to four months of the commencement of depillaring. In a lignite mine spontaneous heating took place within a few weeks only.
- The extraction line should be so arranged as to facilitate roof control. In practice o diagonal line, or step diagonal line of face is common. In special cases a steep diagonal line of face or even straight line of face has been selected. Diagonal or step diagonal line of face provides protection as the working places are supported by solid pillars and also when the roof caves, there is less risk of goaf flushing into the working faces. It is also claimed that diagonal line of extraction helps in the caving of the roof. In the panel worked in conjunction with hydraulic sand stowing-diagonal line of face is prepared as it facilitates water drainage without flooding the working faces in the lower level.
- The single lift extraction is limited to height of 4.8 m or less. If the thickness of the seam is more than 4.8 m, the extraction is done in multi-lifts and in that case hydraulic sand stowing is insisted upon. Seams up to 4.8 m thick can be mined by caving in one pass.
- Whatever the method of extraction, the working area is systematically supported by cogs and props.

3.6. Factors affecting production capacity of the system

The continuous miner system has got enormous capacity if deployment is done at suitable place. However following factors which influence the production capacity of the mining system are as below:

3.6.1. Seam Thickness

Seam thickness determines the availability of coal per meter of cut by continuous miner .low thickness seam yields lesser amount of coal per meter in comparison to thicker seam .hence ,the low height seams requires more frequent change of place of continuous miner and ultimately appreciable loss of productive time of continuous miner .thus ,the productive capacity of the system reduces with reduction in seam thickness .

3.6.2. Gallery Width

This is another important factors which determines the success or failure of the continuous miner technology .continuous miner and shuttle cars combination can maneuver the gallery width of 4.8m ,the maximum permissible gallery width as per CMR ,1957, but with a great difficulty .the equipments of this technology can work more efficiently and higher operational safety ,if the gallery width is more than 5m, say up to 6.0m ,or 6.6m depending up on the condition of strata .with increase in the gallery width ,the continuous miner can cut more amount of coal from a place during the same time period thereby improving the capacity.

3.6.3. Roof Condition

Prevailing condition of roof dictates the requirement of support .the weaker roof requires high support density in comparison to good roof. Not only this, in case of weaker roof the cut out distance will also be much less than that of good roof, thereby increasing in unproductive place changing time and ultimately affecting the production performance of the technology as we know the continuous miner advancement is dependent up on the cycle –time for supporting the freshly exposed cut out area by roof bolts, this is also one of the important factor for determining the production capacity of this system.

3.6.4. Pillar Size

The size of the pillars, barring the operational parameters of the equipments, predominately affects the cycle time which in turn adversely affect the production performance of the technology. The tramming distance increases with the increase of pillar size. Bigger pillar may poise more problems during the time of final extraction than the small pillars.

3.6.5. Cut out Distance

The cut out distance determines the availability of amount of coal at one place, higher the amount of coal at one place better will be the utilization of continuous miner by avoiding the unproductive shifting time to the other face. Higher the cut out distance lesser number of faces requires to be cut for an optimum production.

Besides production capacity is also affected by some mine dependent factors which can be summarized as under

- Roof support system
- Travelling time
- Awaiting shuttle car
- Machine breakdowns and insufficient maintenance
- Out-bye coal clearance
- Power unavailability

- Pick changing
- Poor roof/floor condition etc.

IV. CASE STUDY

4.1 Field Investigations

4.1.1 General Details of Venkateshkhani 7 Mine

- The mine was worked with 2 separate sections. I.e. Anandkhani section and Venkateshkhani section.
- The above sections are separated by a fault of 40 m throw on the rise side of vk7 Inc tunnels.
- Both the sections are having interconnections at 9, 10, 11 and 12 cross-cuts through index seam.
- Anandkhani section was isolated.

The vk7 mine boundary is fixed by considering two major faults of about 160 m throw. Towards south side and 80 m towards north side

Seams present in the Venkateshkhani 7 section:
Four seams are present in this mine. These seams are

- Top seam
- Index seam
- King seam
- Bottom seam

Surface features
Surface features at VK 7 mine are

- PWD road to Vijayawada.014
- Assisted railway siding.
- Coal handling plant.
- Venkateshkhani colony.
- General Manager's office, timber yard, exploration dept, petrol bunk.
- 220 kva transmission lines (proposed for rerouting).
- Tellavagunallah (proposed for diversion).

4.1.2. Location

The mine is situated between north latitude 17^o 27' 07'' to 17^o 30' 24'' and east longitude 80^o 40' 00'' to 80^o 41' 30'' as covered in survey of India topo sheet No.: 65C/10 & 11 of Khammam district in A.P. It is at a distance of 297 kms from Hyderabad. The nearest railway station, Bhadrachalam road, a branch line from Dornakal on Kazipet –Vijayawada section of the south central railway, was of about 10kms away from the mine.

4.1.3. Topography

The terrain is gently sloping towards east. The average ground level is about 138m above MSL. The maximum and minimum MSL is about 119m and 157m respectively.

Brief Description of VenkateshKhani No.7 Incline:

Mine started on = 15-8-1954
Entries = through strike tunnels and man winding shaft
Mine take area = 748 Ha
Non forest land = 155 Ha
Forest land = 593 Ha

The vk7 mine boundary is fixed by considering two major faults of about 160m throw towards south side and 80m towards north side. The mine was worked with two separate sections .i.e. ANANDKHANI section and VENKATESH KHANI section separated by a fault of 40m throw. ANAND KHANI section was isolated.

4.1.4. Geology of Present Seams

The following shows the geology of different seams in vk7 incline :

Table.4.1: Geology of the Different Seams in Vk7 Incline.

Seam	Thickness	Parting	Min.Depth	Max.Depth	RMR	Gradient
Queen	8-11 m		62 m	357 m	52	1 in 7.5
		20 m				
Index	1.2 m		80 m	220 m	-	-
		22 m				
King	6.5-10.5 m		125 m	426 m	62	1 in 7.5
		5-6 m				
Bottom	3-6 m		149 m	298 m	42	1 in 7.5

4.1.5 Technology Being Worked

- Depillaring with remote LHD's (caving).
- Continuous miner non-caving yield pillar method.
- Road headers & Longwall 1985-2004
- Blasting gallery 1998 -2006
- Conventional depillaring 2006-2009
- SDL's 2004-2009
- Continuous miner 2006 (Still Working)

4.1.6 Major Installations/Important Machinery Of The Mine

- Man winding shaft commissioned in 1974 upto bottom seam to a depth of 266 m.
- Man riding system installed in the year 1990, from 29l of king seam(shaft level) to 88l of top seam.(1.80 kms)
- Nitrogen flushing plant of 500cu m/hour.
- SDL's 3 no's.

- One no. B. G. Unit.
- Conveyors from CHP to BG panels and proposed continuous miner panel.
- 1000 t strata bunker at 1Dip/27 Level.
- Two no's of sand stowing plants.

4.2. Support System

4.2.1. Roof support of original galleries.

All the original galleries will be widened to 6.5m / 7.5m. After widening of original galleries, it shall be kept supported by 1.80m long fully resin grouted tor steel roof bolts. Distance between two bolts in the same row shall not be more than 1.50 m and between the two rows of bolts shall not be more than 1.50m. Wherever required the galleries will be widened upto 6.5/7.5m to remove the weathered rib sides up to a solid rib for bolting with the GRP bolts. These 7.5 m wide galleries will be supported with an additional bolt in a row.

4.2.2. Supporting of the sides

Supporting along the dip: Along the North and South side of a dip 1.8m long GRP bolt shall be fixed at roof level to hold Steel/Plastic wire mesh of a height of 2.0m. Two more side bolts shall be fixed at 1.0m interval to secure the side with mesh. The distance between the bolts in the same row shall not be more than 1.0m. Such bolts shall be fixed along the sides at an interval of 1.50m as shown on the SSR plan.

4.2.3. Support along level:

Support along rise &dip side of level: To support the rise & dip side of level, Steel/Plastic wire mesh shall be used. The mesh shall be used upto 2.0 m from roof. Along the rise & dip side of level, mesh shall be fixed by 1.8m long GRP bolt at roof level. Two more side bolts shall be fixed at 1.0m interval to secure the sides with mesh. The distance between the bolts in the same row shall not be more than 1.0m. Such bolts shall be fixed along the rise & dip side of level at an interval of not be more than 1.50m as shown on the SSR plan. This support system shall be followed along the rise & dip side of all levels as shown on the SSR plan.
Additional bolts shall be installed across the slips and geological disturbances as shown on the plan.

4.2.4. Supporting geologically disturbed area

All dykes, visible slips and breaks in the roof will be supported with 2.40m long fully resin grouted bolts as shown on the plan in addition to the SSR for original galleries as per para 2.0.

4.2.5. Support of ledges and overhangs:

Ledges and overhangs which cannot be dressed down and made safe shall be supported with 2.4m long fully resin grouted steel bolts

4.2.6. Quality of support:

Strata reinforcement support system components to be used, shall confirm to the standard prescribed in IS 1786 – 2008 or British Standard 7861 (Part 1)1996 (Specifications for roof bolting) or latest revision thereof.

Short Encapsulation Pull Test (SEPT) shall be conducted for every new batch of resin capsules. Resin bond strength shall be 130 KN for 300 mm encapsulation for more than 50% of bolts tested.

The standard and parameters of the roof bolts and its assembly to be used in mines should be maintained as prescribed in the DGMS circular No. DGMS/S&T/Tech. CIR.(Approval) No.3, Dhanbad, 3 rd June 2010

The diameter of the roof bolt shall not be more than designed diameter ± 4 % by mass as specified under IS 1786 – 2008 or as per the standard given in BS 7861 (Part 1) 2008. The difference of diameter between the hole and the roof bolt (annular space) shall be not less than 3.0mm and not more than 6.0mm.

Each hole shall have one set of resin capsules – one fast setting and the other slow setting with 23-24mm diameter (tolerance 5%).

The standard composition and properties required for cement capsules to be used as grouting material in roof bolting in mines shall be maintained as per DGMS circular No. DGMS/Tech.Cir.(Approval) No.4 Dhanbad, 15 th June 2010

4.2.7. Monitoring of roof bolts

To check the efficacy of the support system in the workings, anchorage testing will be done. Effective organization under the supervision of a Rock Bolt Engineer will be set up for regular anchorage testing, tightening and also for ensuring proper system of support, monitoring of roof sagging, etc. At least 10% of the installed bolts will be subjected to anchorage testing (at random), after 1 hour of installation by applying a minimum of 12T load, under the direct supervision of Overman /Under manager/Asst. Manager (First Class) and the results will be recorded in the prescribed format (DGMS (Tech) (S&T) circular No 3 of 1996)

Testing will be carried out under the properly supported roof, with a suitably designed anchorage testing machine. All directives given in the DGMS (Tech) (SAPICOM) circular No 3 of 1996 will be strictly complied. Bolts shall be pre-tensioned at a torque of 70 – 120 Nm. Results of all such tests will be recorded in a bound paged book kept for the purpose and signed by the officer-in-charge. The report will be scrutinized by the manager at least once in seven days and corrective action will be taken whenever necessary. All the provisions of Regulation 109 and 110 of the CMR 1957 regarding setting and withdrawal of supports as applicable herein will be complied with.

4.3 Working Details CMP-5A :

The following are some of the details of Continuous miner panel – 5A

Panel started on	07.03.2012
Panel completed on	23.06.12
Extraction period	109 days
No. of Pillars	20
Panel length and breadth	231 m & 170
Area of the panel	39,000 sq.mtrs
Coal extractable in the panel	2,43,100 T
Coal extracted In the panel	1,69,500 T
% of extraction	70%
Main fall occurred at	21,150 sq.mtrs

4.4 Working Details CMP-5B :

The following are some of the details of Continuous miner panel – 5B

Panel started on	08.07.2012
Panel completed on	13.10.12
Extraction period	96days
No. of Pillars	20
Panel length and breadth	233 m & 179
Area of the panel	41,707 sq.mtrs
Coal extractable in the panel	2, 18,790 T
% of extraction	73.00%
Main fall occurred at	21,550 sq.mtrs

V. RESULTS AND DISCUSSION

5.1 Production Analysis:

The detailed study of production analysis indicates the following:

- The production from LHD and Continuous miner technology is been analyzed. The continuous miner technology is giving better production comparing to LHD (3No.), in some cases its production is double.
- The Table no.5.1 indicates the production percentage of continuous miner technology and LHD, in which production of 8th months is been, analyzed at VK7 mine.

Table.5.1: Comparison of continuous miner and LHD production in percentages

Sl.No	Technology	Production (%)								
		Mar	**Apr**	**May**	**Jun**	**Jul**	**Aug**	**Sep**	**Oct**	**Total**
1	LHD	44.05	32.3	40.58	45.99	41.15	33.37	40.98	43.78	39.88
2	CM	55.95	67.7	59.42	54.01	58.85	66.63	59.02	56.22	60.12

5.2. Breakdown Analysis: The detailed study of breakdown analysis indicates the following

Continuous Miner panel - 5A:

- We observed that major percentage of breakdown in continuous miner is due to gathering problem i.e., 50.14%. Which includes gathering cylinder pin out, gathering head gear box prob, gathering spray nozzle jam.(Table.6)
- Apart from gathering problem, there are other problems like hydraulic and cutter problems which accounts 24.32% and 11.32% of breakdowns respectively.
- Here we observed electrical problem is main in shuttle car i.e,99.16%

Continuous miner panel – 5B:

- We observed that major percentage of breakdown in continuous miner is due to gathering problem i.e., 48.23%. This includes gathering cylinder pin out, gathering head gear box prob, gathering spray nozzle jam (Table.5.2.1).
- Apart from gathering problem, there are other problems like traction and hydraulic problems which account 24.49% and 22.2% of breakdowns respectively.
- Here we observe electrical problem is main problem in shuttle car (Table.5.2.4).

The table 5.2.1 and table 5.2.2 shows the total percentage of breakdown, idle hours and working hours of CMP-5A and CMP-5B panels respectively.

Table.5.2.1: Percentage of Breakdown Continuous miner CMP 5A

Sl.No	Classification of Breakdown	Percentage (%)
1	Electrical problem	7.65
2	Cutter problem	11.32
3	Conveyor Problem	4.1
4	Gathering problem	50.14
5	Traction	2.47
6	Hydraulic	24.32
7	Chassis	0
	Total	100

Table.5.2.2: Percentage of Breakdown Continuous miner CMP 5B

Sl.No	Classification of Breakdown	Percentage (%)
1	Electrical problem	0
2	Cutter problem	3.18
3	Conveyor Problem	1.9
4	Gathering problem	48.23

5	Traction	24.49
6	Hydraulic	22.2
7	Chassis	0
	Total	100

Table.5.2.3: Percentage of Breakdown Shuttle Car CMP 5A

Sl.No	**Classification of Breakdown**	**Percentage (%)**
1	Electrical problem	99.12
2	Conveyor Problem	0
3	Traction	0.88
4	Hydraulic	0
5	Chassis	0
	Total	100

Table.5.2.4: Percentage of Breakdown Shuttle Car CMP 5B

Sl.No	**Classification of Breakdown**	**Percentage (%)**
1	Electrical problem	100
2	Conveyor Problem	0
3	Traction	0
4	Hydraulic	0
5	Chassis	0
	Total	100

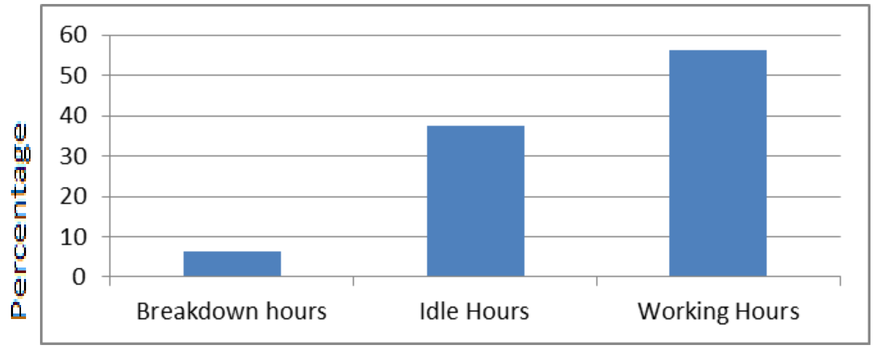
Table.5.2.5: Breakdown, Idle, Working Hours (percentage) of the CMP 5A

Sl.No	**Month**	**Percentage (%)**
1	Breakdown hours	6.32
2	Idle Hours	37.47
3	Working Hours	56.21

Table.5.2.6: Breakdown, Idle, Working Hours (percentage) of the CMP 5B

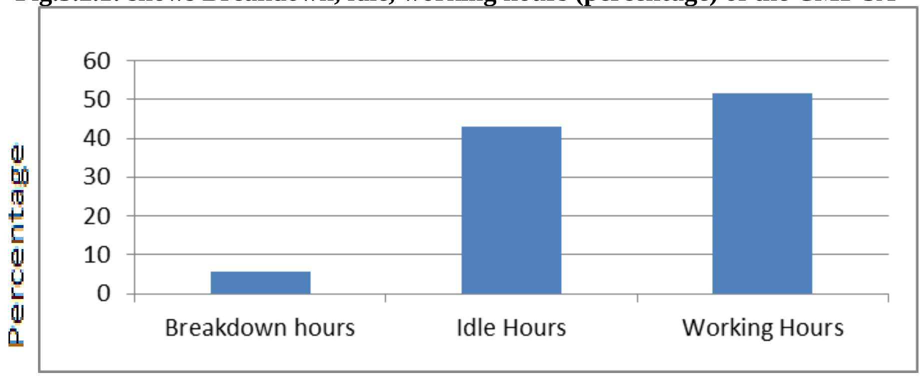
Sl.No	**Month**	**Percentage (%)**
1	Breakdown hours	5.73
2	Idle Hours	42.81
3	Working Hours	51.46

The breakdown, idle and working hours of the continuous miner panel 5A (As shown in fig.5.2.1) and the continuous miner panel 5B(As shown in fig.5.2.2) indicates that the idle hours is more compare to breakdown hours of the machine.



Time (in hours)

Fig.5.2.1: shows Breakdown, idle, working hours (percentage) of the CMP 5A



Time (in hours)

Fig5.2.2: shows Breakdown, idle, working hours (percentage) of the CMP 5B

5.3. Reliability Analysis:

The Table 5.3.1 and Table 5.3.2 are showing the breakdowns under various head in respect of the equipments of CM for CMP 5A,CMP 5B panels in VK7 mine, which are elaborated as below.

- Continuous miner has undergone the breakdowns; majority of those breakdowns is from gathering problem, which is followed by other problems like hydraulic, traction, cutter, electrical and conveyor problems.
- The reliability analysis is being done, in which the probability of failure P(F) is been calculated, the overall probability of failure for continuous miner is 0.0955. The reliability of continuous miner after calculating is 0.9044.
- Shuttle car have also got some breakdowns, the major one is due to the electrical problem, the other one is because of traction problem.
- The reliability analysis is also been done for shuttle car, the overall probability of failure P (F) of which is 0.0163. The reliability of shuttle car after calculating is 0.9836.

Table 5.3.1: Reliability analysis of Continuous Miner

Sl.No	**Classification of Breakdown**	**Hours**	**P(F)**	**%P(F)**
1	Electrical problem	10.08	0.003598214	0.36
2	Cutter problem	19.23	0.006864449	0.69
3	Conveyor Problem	7.99	0.002852156	0.29
4	Gathering problem	131.57	0.04696597	4.68
5	Traction	36.5	0.013029246	1.3
6	Hydraulic	62.18	0.022196124	2.22
7	Chassis	0	0	0
	Total		**0.095506159**	

Probability of failure P (F) = Breakdown Hours / Total Working Hours

Total P (F) = 0.0955
Reliability = 0.9044

Table 5.3.2: Reliability analysis of Shuttle Car

Sl.No	Classification of Breakdown	Hours	P(F)	%P(F)
1	Electrical problem	45.51	0.016245507	1.62455
2	Conveyor Problem	0	0	0
3	Traction	0.25	0.0000869	0.00869
4	Hydraulic	0	0	0
5	Chassis	0	0	0
	Total		**0.016332407**	

Probability of failure P (F) = Breakdown Hours / Total Working Hours
Total P (F) = 0.0163
Reliability = 0.9836

VI. CONCLUSION & RECOMMENDATIONS

6.1 Conclusion:

Continuous miner technology will drastically increase the production, productivity and safety in the underground mining. It is convenient with both caving as well as non-caving method of mining.

Using continuous miner technology high production can be achieved. It can be utilized for development as well as depillaring of developed pillars. It can give an average of 74-75% of extraction.

The machine has worked for an average of only 54.78 %.The machine can give good production rates if we can increase the working hours. This technology promises greater safety. The proper utilization of the equipments can give greater production rate

The reliability analysis shows that the machine is reliable about 95.56% where as probability of failure is only 4.45% .this indicates, it is more reliable.

6.2 Recommendations:

The following are the recommendations for effective utilization of method are:

1. Regular inspection has to be done to reduce unproductive time.
2. Immediate action to be taken on major impacting problems.
3. Belt conveyor idlers jamming have to be reduced.
4. The availability of the face has to be provided in order to reduce idle time of the machine. It is about 28%.
5. Proper layout has to be prepared for shuttle car to minimize the waiting time of machine.
6. Shifting of equipments should be done properly such that time can be reduced.
7. Properly manage the time, for all the cycle operations

REFERENCES

[1] Atul gandhe et.al.Scientific study for introduction for continuous miner at vk7, NIRM July, 2005.
[2] B.P.Verma, Rock mechanics, Khanna Publishers, Delhi 1992.
[3] Chris Oldroyd and Peter Altounyan, RMT (Rock Mechanics Technology) visit report to vk7 and DGMS in Hyderabad, May 2008.
[4] Dr.Rajendra Singh et.all, CIMFR(Central Institute Of Mining And Fuel Research) report on mechanized depillaring of king seam at V K 7 incline, Kothagudem area, SCCL
[5] Lind.G.H "key success element of coal pillar extraction in new south wale" the journal of the South African institute of mining and metallurgy, may/june2002.
[6] "Manual on pillar extraction in new south wales underground coal mines" department of mineral resource New South Wales mdgno1005august 1992.
[7] Paul Committee, report of the expert group on guidelines for drawing up support plans in bord pillar workings in coal mines. DGMS, Dhanbad, 1990.
[8] P.R. Sheorey, Design of coal pillar arrays and chain pillars by, CMRI.
[9] R.D.Singh, "Principles and practices of modern coal mining", new age international (p) limited, publishers, and copy righted 1997.
[10] Ribsidere-inforcementinstallationtrialsandperformancetestingbyjoymining,vk7,sccl.
[11] Salamon munroe [1966], a study of the strength of coal pillars. Transvaal and Orange Free State chamber of mines, research report no. 71/66.
[12] Sri Ram Chandra Nayak, "Evaluation of bord and pillar mining system in MCL coal mines", NIT Rourkela, 2010.
[13] Winton j. gale, Experience of field measurement and computer simulation methods for pillar design.
[14] http://www.novamining.com/examinations/accident-analysis/air-blast-parascole-west-colliery-on-06-07-1999
[15] www.coal info mine.com
[16] http://www.novamining.com/examinations/accident-analysis/air-blast-parascole-west-colliery-on-06-07-1999
[17] www.coal info mine.com
[18] www.teara.govt.nz
[19] www.scclmines.com

AUTHORS

First Author – Vijaya Raghavan, Assistant Professor, Department of Mining Engineering, Dr.Thimmaiah Institute of Technology, Oorgaum, Kolar Gold Fields-563120, Email ID-raghavan_pp@rediffmail.com

- Started Professional Qualification with a Diploma in Mining Engineering from School of Mines, K.G.F in 2002,
- Bachelor of Engineering in Mining Engineering from Vishvesvaraya Technological University, Belgaum in 2006,
- Master of Engineering in Environmental Engineering from Sathyabama University, Chennai in 2010 and
- Presently Pursuing PhD in Mine Environmental Engineering From National Institute of TechnologyKarnataka,Suratkal,Mangalore, Karnataka.
- Working as Assistant Professor & Head, Department of Mining,DrTTIT,K.G.F
- AREA OF INTEREST: Drilling and Blasting Mine Environmental Engineering,, Rock Mechanics.

Second Author: Dr Syed Ariff, Assistant Professor, Professor, Senior Lecturer.

Third Author: Paul Prasanna Kumar, Department of Mining Engineering, Dr.T.Thimmaiah Institute of Technology, Oorgaum, Kolar Gold Fields-563120