Economical Operation & Estimation of Operating Cost of Draglines in Open Cast Mines

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Abstract

Draglines are the most expensive piece of excavating equipment at the mine site and it is important that they are operated safely, efficiently and economically. Draglines are versatile and provide a low cost mining method. In order to achieve high production and productivity of heavy earth moving machineries in opencast mines, it is necessary to have high percentage of availability and utilization of the said equipment. The present study focuses on the importance of economic appraisal of dragline productivity parameters, such as, cycle time, swing angle, seating position, availability, utilization, etc., in the field scale application. In this contest, calculation of operating cost of dragline operation in various methods has been done and compared critically on case to case study basis. Principally, it will serve as a guideline to the method employed in determining the economical operating cost in various mining methods adopted in opencast mines.

Key words: *Cycle time, idle time, availability, utilization, stripping ratio, production index*

Introduction

A **dragline excavator** is a piece of heavy earth moving equipment used in civil engineering and surface mining operations. Draglines used in civil engineering are almost always of this smaller, crane type. These are used for road, port construction, pond, and canal dredging, and as pile driving rigs.

Selection of Mining Equipment is based on:

- Economic Stripping Ratio
- Life of the mine
- Infrastructure available
- Proposed annual targeted output
- Technology available

SUITABILITY FOR DRAGLINE APPLICATION

1. Gradients flatter than 1 in 6
2. Seams should be free from faults & other geological disturbances
3. Deposits with Major Strike length
4. Thick seams with more than 25m thick are not suitable
5. A hilly property is not suitable
6. Property should have at least 15 years of life
7. The '*King pin*' of the mine
8. 40% of the OB should be handled by the Draglines
9. High Cost, for 24/96 Dragline is Rs. 160 Crores
10. Loss per hour Rs. 2,00,000/- approx., if not worked

DRAGLINE – Methods of Working

1. Simple side-casting method with or without rehandling.
2. Extended bench method.
3. Horse shoe method/Across Pit method.
4. Continuous bench two pass method.
5. Tandem dragline working i.e., two draglines working together, one dealing with part of the material dumped by the other.

Advantages

- Direct waste disposal
- Can also be used to dig above working level at reduced production rate
- Less frequent movement
- Not significantly affected by the adverse weather
- Safe from unstable slopes, pit flooding, etc
- Flexible, can be applied in different operating techniques
- Lowest cost per unit moves

Disadvantages

- Requires bench preparation; Max slope: 2% Max ramp : 10 %
- Does not dig poor blast well
- High investment Cost
- Rehandling reduces the productivity
- Additional leveling Cost of spoil peaks

Fig: 1 Dragline Abhimanyu at NCL Amlohri

WALKING DRAGLINE
MODEL: WD 24/96
MAKE: HEC – RANSOM & RAPIER

WORKING DATA	
BUCKET CAPACITY	24 CUB. METRE
BOOM LENGTH	95.6 METRE
BOOM ANGLE	30^0
OPERATING RADIUS	88 METRE
MAX. DUMP HEIGHT	39 METRE
MAX. DIGGING DEPTH	53 METRE
AVERAGE GROUND PRESSURE	1.03KG/SQ.CM
CLEARANCE RADIUS	22 METRE
WIDTH OVER BOTH SHOES	23 METRE
BOOM HEADS HEIGHT	54 METRE
WEIGHT OF THE MACHINE	2000 TONNE
WALKING SPEED	240 MPH (35 SECONDS A STEP)
LEGNGTH OF A STEP	2.3METRE
PRODUCTION CAPACITY	3.6 MILLION CUB. METRE PER ANNUM
DATE OF COMMISSIONING	1.10.1983
PROGRESSIVE WORKING HRS	1,77,632 HOURS
PROG. MATERIAL HANDLED	111 MILLION CUB. METRE
AVG. WORKING HOUR/DAY	18.5 HOURS
BUCKET	
WEIGHT	32 TONNE
MAX. SUSPENDED LOAD	77 TONNE
AIR SYSTEM	
MOTOR	15 KW
MAX. WORKING PRESSURE	14 BAR
FREE AIR DELIVERY	29.37 LITRE/SECOND
ELECTRICALS	
POWER SUPPLY	6.6 KV, 50 HZ, 3 PHASE
HOIST MOTORS	1300 HP , 475 V
DRAG MOTORS	1300 HP, 475 V
SWING MOTORS	640 HP, 475 V
WALK MOTORS	640 HP, 475 V
MOTOR GENERATOR SET	2 x 1750 HP
EXCITERS	65 KW, 33 KW
AUXILLIARY TRANSFORMER	440V, 50 HZ, 3 PHASE

YEAR-WISE % OF TOTAL VOLUME HANDLED BY DRAGLINES IN NCL DURING '96-'06

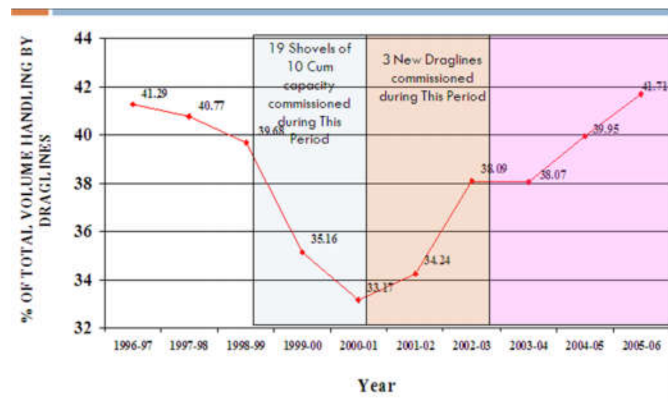


Fig: 2 Year-wise % of total volume handled by dragline in NCL during "96-06"

ANNUAL PRODUCTION CAPACITY

Production index, which can be defined as the bank measure of O/B volume moved per period per rated bucket volume. This measure is quite helpful in comparing productivity of different draglines of various make and capacities. The index can also be used in the machine selection procedure when used to calculate required bucket capacity per period given the O/B volume per period. "Production Rate" can be defined as the bank measure of O/B per period. This bank measure is an index helpful in production forecasting and scheduling.

Production Index: 2, 50,000 m^3 / m^3 bucket /year

Factors Affecting the Dragline Productivity

- Digging hour
- Cycle time
- Digging pattern
- Method of digging
- Method of excavation
- Blasting

- Dig position plans
- Operational delays(idle hours)
- Maintenance delays(down time hours)

Idle hours

- Shift change
- Field switch/cable shifting
- Face dozing
- Blasting
- Platform preparation
- Power failure

IDLE HOUR ANALYSIS

Total Idle hours	516	% of idle hours	% of shift hours
Field switch shifting/cable shifting	33	6.4	0.4
Face Dozing	218	42.2	2.5
Blasting	67	12.9	0.8
Power problem	182	35.3	2.1
Others	16	3.1	0.2

Cycle time

One cycle time consists of

1. Drag to fill
2. Hoist & swing to dump
3. Dump
4. Lower & return swing
5. Position bucket

Cycle time 24/96 dragline at rated parameters at 120^0 swing angle, 50 metre depth, 38 metre dump height

= 67.21 seconds

If the swing cycle time is reduced by *one second*, dragline productivity increases by 90,000 m^3 /year i.e. *2.5% increase in productivity.*

Factors Affecting Cycle Time

- Material characteristics
- Bank preparation
- Digging depth
- Hoisting height
- Swing angle
- Rope & swing speeds
- Operator's proficiency
- Equipment placement & scheduling
- Slope of digging face

To Reduce Cycle Time

- Sharp bucket teeth and shroud increases bucket fill factor, reduces bucket fill time
- A well balanced light weight bucket is used
- Good pick up and carry angle – reduces spillage
- Digging in systematic pattern
- A good carrying angle and hoisting under boom prevents tight lining, reduces bucket fill time by increased hoist speed

Maximizing payload is dependent upon the following factors:

- Geology
- Blasting
- Monitor weighing accuracy
- Engage location
- Disengage Location
- Rigging
- Suspended load
- Bucket characteristics
- Operator skills
- Minimizing rehandle

Estimating Dragline Production

A basic approach to estimating dragline production involve use of a standard cycle excavator equation such as the one given below that has been tailored to monthly dragline output.

$$O = \frac{B \times BF \times HS \times A \times J \times 3600}{(1+S) \times C \times (1+R)}$$

Where,

O= output (bcy/month)
B= bucket size (yd)
BF= bucket fill factor
HS= hours scheduled /month
A= maintenance availability (% of schedule time machine as available for stripping / 100)
J= job factor (% of time that machine is available for stripping)
S= swell factor / 100
C= average cycle time
R= rehandling percentage /100

Evaluation of Availability (A) and Utilization (U)

To evaluate A and U field data was required and maintained on day to day on the basis of dragline under study.

$$A = \frac{(\mathrm{SSH} - (\mathrm{BH} + \mathrm{MH}))}{\mathrm{SSH}}$$

$$U = \frac{(\mathrm{SSH} - (\mathrm{BH} + \mathrm{MH} + \mathrm{ID}))}{\mathrm{SSH}}$$

Where,

SSH = scheduled shift hour
MH = maintenance hour
ID = idle hour
BH = breakdown hour

Computation of Efficiency of Draglines

The computation of earthmoving efficiency for each dragline was done by using

$$\text{Efficiency of dragline}(\eta) = \frac{\text{Computed annual output}(P_1) \times 100}{\text{Annual output as per the balancing diagram } (P_2)}$$

The determination of annual output as per the balancing diagram (P_2) was done from the prepared balancing diagram as per the balancing for HT and VT modes of operation, respectively.

Computation of Coal Exposure

The coal exposed by the draglines working in tandem operation was estimated by using the generalized equation as per the balancing diagram concept.

$$\mathrm{CE} = (P_{\mathrm{FD}}/A) \times W \times T \times D \times R(\mathrm{M\ te})$$

Where CE is the coal exposure (M te),
PFD the annual output of the lagging dragline from the first dig (M m^3),
A the area of first dig (m^2),
W the cut width (m), T the thickness of coal seam (m), D the specific gravity of coal, and R is the recovery factor.
The term (PFD/A) represents the annual linear advance of the draglines

Estimation of Operating Cost

The Cost calculation has been made on the basis of information supplied by **NCL Singrauli Amlori Project.**

Approach to the problem

No. of annual working day- 300
No. of daily shift- 3
Duration of each shift- 8 (hour)
No. of operators for dragline in each shift- 2
No. of helper- 1

Oilers, Electrician and Mechanics are included in maintenance cost.

Calculation of Ownership and Operating Cost of Abhimanyu dragline in Amlori project NCL Singrauli:

1. **Cost of Ownership per year of 24/96 dragline**

(a) Equipment Cost
Cost of 24/96 dragline = Rs.160 crore

(b) Depreciation Cost for 25 year i.e. annual flat rate of 4%
Annual depreciation cost of 24/96 dragline = Rs. 6.4 crore

(c) **Annual Cost of Ownership**
Average annual investment

$$= \frac{N+1}{2N} \times \text{cost of dragline}$$

$$= \frac{25+1}{2\times 25} \times 160 \text{ crore}$$

= Rs. 83.2 crore

Where N = Life of dragline

(d) **Annual interest, insurance and taxes i.e. annual flat rate of 12.5%**

= 12.5 % of Rs. 83.2 crore

= Rs. 10.4 crore

Hence, Total Ownership Cost per year = [depreciation cost per year + annual interest, insurance and taxes]

=Rs. [6.4 +10.4] crore

= Rs. 16.8 crore

2. Operating Cost per year of 24/96 dragline

(a) Annual Manpower Cost (Salary & Wages)

Operator Cost @ Rs. 0.02 crore/operator for 2 operators in 3 shift = Rs. 0.12 crore

Helper cost @ Rs. 0.01 crore for 1' 3 = Rs. 0.3 crore

Hence, Total Manpower Cost = Rs. (0.12 + 0.03)

=Rs. 0.15 crore

(b) **Annual power and energy consumption on the basis of 13.65 MKWH**

Annual power consumption cost @ Rs. 4.89/KWH

Rs. 4.89×13.65 MKWH

Rs. 6.675 crore

(c) **Annual lubrication Cost**

Annual lubrication cost @ 30% of power consumption cost

= Rs. 2.0025 crore

(d) Annual Maintenance Cost

Annual maintenance cost @ 20% of depreciation cost = Rs. 1.28 crore

Major breakdown maintenance cost @ 2% of cost of equipment = Rs. 3.2 crore

Total maintenance cost = Rs. 4.48 crore

Hence, Total annual Operating cost = Rs. [0.15+ 6.675+ 2.0025+ 4.48]

= Rs. 13.3075 crore

Total annual ownership cost and operating cost

= Rs. [16.8+ 13.3075] crore

= Rs. 30.1075 crore

Considering annual output of 24/96 dragline as 3.4Mm³

$$= \frac{30.1075\text{crore}}{3.4\ \text{Mm}^3}$$

= Rs. 88.55 per/ m³

Calculation of cost per tonne of coal exposed

1. Single dragline Operation without rehandling

Amount of effective overburden handled = 3.4 Mm³.

Amount of coal exposure =

$$= \frac{\text{Annual production of dragline}}{\text{Avearge stipping ratio}}$$

$$= \frac{3.4\ \text{Mm}^3}{2.5\ \text{m}^3/\text{te}}$$

= 1.36 Mte

Estimated Cost per tonne of coal exposed =

$$= \text{Rs.}\ \frac{30.1075\ \text{crore}}{1.36\ \text{M te}}$$

= Rs. 221.37/te of coal exposed

2. Single dragline extended bench operation

Percentage of rehandling is 61.23%

Total O/B handle = O/B directly over the exposed coal + O/B rehandled

= O/B directly over the exposed coal (1+ coefficient of rehandled)

Here,
Coefficient of rehandling

$$= \frac{\text{O. B. rhandle}}{\text{O. B. removal to expose coal}}$$

Therefore, 3.4 Mm³ = O/B directly over the exposed coal, 1.61

Hence O/B directly over the exposed coal removed by the dragline = 3.4/1.61
= 2.11 Mm^3
Amount of effective O/B handled = 2.11 Mm^3

Amount of coal exposure =
$$= \frac{2.11\ \mathrm{Mm}^3}{2.5\ \mathrm{m}^3/\mathrm{te}}$$
= 0.844 Mte

Estimated cost per tone of coal exposed =

$$= \frac{30.1075\ \text{crore}}{0.844\ \text{M te}}$$

= Rs. 356.72 per tonne of coal exposed

3. **Horizontal Tandem Operation**

When leading and lagging dragline is deployed 24/96 having annual production of 3.4 Mm^3 each

Total Ownership and Operating cost per year of two draglines (24/96 + 24/96) =

= Rs. 60.215 crore

Estimated cost per tonne of coal exposed in tandem dragline operation on the basis of average stripping ratio 2.5 m^3/te

Percentage rehandling is 42.48%
Amount of effective O/B removed = 6.8Mm^3/ 1.425

= 4.77Mm^3

Amount of coal exposed = 4.77Mm^3/2.5 m^3/te
= 1.91 Mte

Estimated cost per tonne of coal exposed

$$= \frac{60.215\text{crore}}{1.91\ \text{Mte}}$$

= Rs. 315.26 per tonne of coal exposed

4. **Vertical Tandem Operation**

When the leading and lagging dragline having annual production 3.4 Mm^3

Percentage rehandling is 25.95%

$$Amount\ of effective O.B.\ handled = \frac{6.8\ \mathrm{Mm}^3}{1.2595}$$

= 5.40Mm^3

$$Amount\ of\ coal\ exposed = \frac{5.40\mathrm{Mm}^3}{2.5\ \mathrm{m}^3/\mathrm{te}}$$

= 2.16 Mte

Estimated cost per tonne of coal exposed

$$= \frac{60.215\ \text{crore}}{2.16\ \text{M tel}}$$

= Rs. 278.77 per tonne of coal exposed

Hence, different mode of operation of dragline and estimated cost calculate.

Digging hours

If the dragline is operated one extra minute per shift, this equates 18.2 hrs/year =15,000m^3 extra production

Dig rate

- Swing angle more than 120^o decrease by 15 %
- When cleaning the terrain to be excavated -decrease by 10%
- When damp to be excavated – decrease by 10%
- With oversize content – decreases by 10-15%
- With re-excavation –increase by 10%
- With top cutting – rocks decrease by 10-15%

Digging pattern

- Walk up on digging face
- 45^o digging face
- Systematic bucket spotting
- Reduce bucket fill length
- Pushing in the roll of materials
- Control bucket depth
- Follow the previous operator’s pattern

Method of Digging

- Box cut
- Key cut
- Strip cut
- Chop down cut
- Ramp cut

Chop down cut

- produces only 75 % of conventional cut
- low bucket fill factor
- increased drag time to fill
- increased dozer work
- increased down time
- increased repair cost

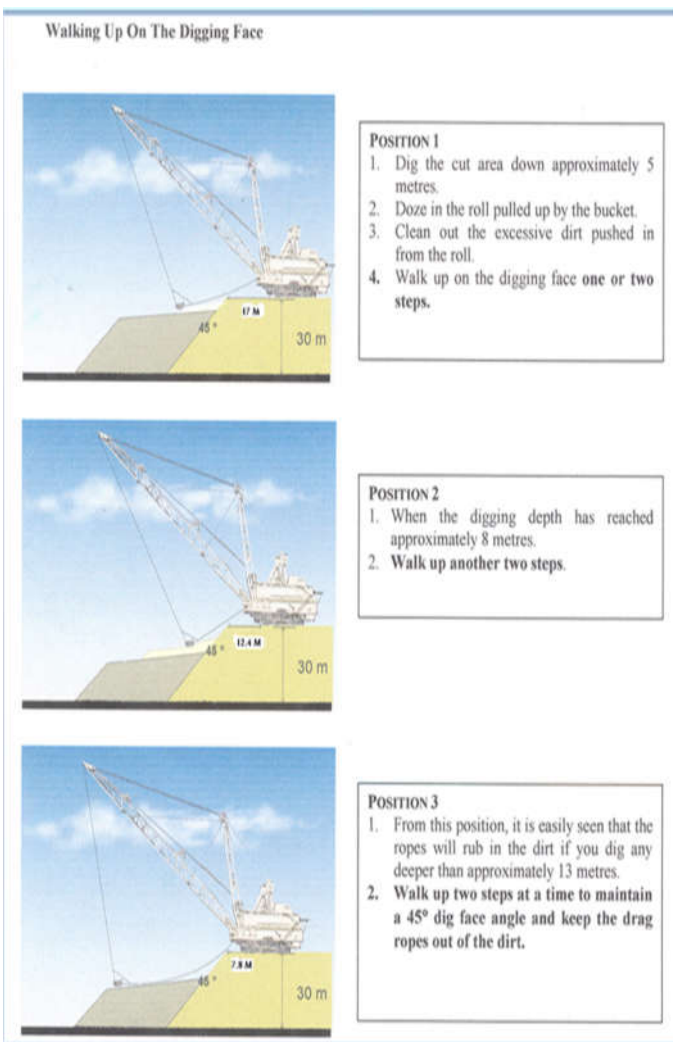


Fig: 3 different postion of digging face

Condition Monitoring

To determine its current state of health in order to prevent & prevent impending failures systematically, arise from human observation, checks & laboratory tests, from instruments etc.

CM Levels

- Visual
- Assistance Monitoring - Using instruments
- Lubrication Monitoring, physio-chemical testing, Wear debris analysis
- Monitoring with alarm

CM Methods

- Visual
- Potential Failure effects Monitoring
- Performance Monitoring
- Fore casting

Rubbing effects:
Particle effects: Lubricant Analysis
Wear debris analysis
Chemical effects: Element Monitoring
Dynamic effects:
Shock pulse monitoring
Vibration Analysis
Noise Monitoring
Physical effects:
NDT
Temperature effects: Thermography
Electrical effects:

- Condition Monitoring is done every 6 months by a Separate Cell at Jayant
- Detailed report sytsem wise is submitted to the all levels

The methods were developed at Peak Downs Mine and are in use throughout the BHP Coal group of mines.

Analysis Methods

Analysis methods can be categorized into 4 areas, as outlined

Motion analysis

- Electrical motor characteristics- Voltage/Current (Speed/Torque) curves.
- High speed / short term measurements:- Motion rates (hoist, drag and swing individually) with a step change in operator reference from full stop position

Cycle analysis

- Cycle times, such as swing to dump time, return time, and fill time.
- Hoist dependent swings.
- Payload weight.
- Other Key Performance Indicators (KPI's)

Production demographics analysis

- Production rate per dig hour
- Total production vs. target

Qualitative analysis

- Survey / interview of operators

Fig; 4 *demographics analysis*

DRAGLINE IN TANDEM-VERTICAL OR HORIZONTAL

Dragline operation in vertical tandem may be considered in the following conditions:

1. Where the OB thickness to be handled by dragline is more than the effective and efficient digging depth of dragline.

2. Bench height is limited by drilling depth of the drilling rigs.

3. Dumping height poses no problem for dragline deployed at lower bench due to accommodation of material.

4. Increased rate of coal exposure is required and thickness of dragline bench is to be kept less.

5. Shovel capacity is not matching with dragline capacity and unable to provide sufficient working space for the dragline.

Sl. No.	Parameters	Details
1	Draglines	
	D/L-1(LeDL-VT)	15/90 with max. horizontal reach of 82 m (top portion)
	D/L-2(LaDL-VT)	24/96 with max. horizontal reach of 88 m (bottom portion)
2	Bench height	Approx. 42 m (14 m as top portion and 28 m as bottom portion)
3	Cut width	85 m
4	Strip length	2,000 m
5	Highwall slope	70°
6	Bench slope	60°
7	Key cut width at top	38 m
8	Key cut width at bottom	5 m
9	Coal rib	Left for full height (triangular section)
10	Angle of repose	38°

Table 1 Key field parameters and machine parameters for construction of balancing diagram in vertical tandems (VT) mode of operation in mine 2

Horizontal tandem operation of dragline is deploying draglines at the same level have been added advantages in the under mentioned conditions:

1. Increased width of the dragline cut is possible which will reduce idle time for marching and power shut down.

2. Area of operation of both draglines is concentrated and thus better supervision is possible.

3. Due to increased OB bench thickness, dumping height for the lagging dragline, which is exposing coal, poses no problem.

4. Marching of the dragline at the end of the cut is easier than in vertical tandem because dragline will be sitting at a higher level.

5. Productivity is increased due to less swing angle and less problem due to dumping height.

In case, deployment of draglines in horizontal tandem and two shovels for extraction of coal is a necessity, width of the dragline cut has to be kept more for smooth and safe operation.

Sl. No.	Parameters	Details
1	Draglines	
	D/L-1(LeDL-HT)	24/96 with max. horizontal reach of 88 m
	D/L-2(LaDL-HT)	24/96 with max. horizontal reach of 88 m
2	Bench height	Approx. 35 m
3	Cut width	90 m
4	Strip length	1,700 m
5	Highwall slope	70°
6	Bench slope	60°
7	Key cut width (at top)	38 m
8	Key cut width (at bottom)	5 m
9	Coal rib	Left for full seam height (triangular section)
10	Angle of repose	38°

Table 2. Key field parameters and machine parameters for construction of balancing diagram in horizontal tandem (HT) mode of operation in mine

Sl. No.	Mode of operation	Swing angle (°)	Observed cycle time for 24/96 (LaHT-D/L) for blasted muck (s)	Observed cycle time for 24/96 (LaHTD/L) for re-handled muck (s)	Weighted cycle time for blasted muck (s)	Weighted cycle time for re-handled muck (s)	Overall cycle time (COA) for 24/96 (LaHT-D/L) (s)
1	HT	Up to 90	74.96	65.11	81.16	71.25	**77.20**
2	HT	>90-120	81.68	73.93			
3	HT	>120-150	89.45	78.57			
4	HT	>150-180	98.40	84.29			

Table 3 Weighted and overall cycle time for 24/96 lagging dragline (LaHT-D/L) working in HT mode in the blasted muck and rehandled muck (loose O/B) for mine

Method of Excavation

- simple side casting
- advance bench excavation
- extension bench excavation
- pull back excavation
- box cut excavation
- tandem excavation

The database also allows for the monitoring and analysis of such items as: alarms, overloading, bucket performance, rigging schemes, digging rates at various depths, and comparing productivity of different digging modes of draglines. This information can be used to change operating or maintenance practices, for operator training, and to alter mining plans resulting in an overall improvement in productivity and reduction of costs.

Simple sidecasting method

This is the simplest form of strip mining, which involves excavation of the overburden in a series of parallel strips. The strips are worked in a series of blocks. The O/B from each strip is dumped into the void left by the previous strip after the coal/ mineral has been mined out. It is customary to start the excavation of each block by digging a wedge shaped key cut with the dragline standing in line with the new high wall. From this position, the machine can most easily dig a neat and competent high wall. The width of each strip is usually designed so that the material from the key cut can be thrown into the previous cut without the need for rehandle.

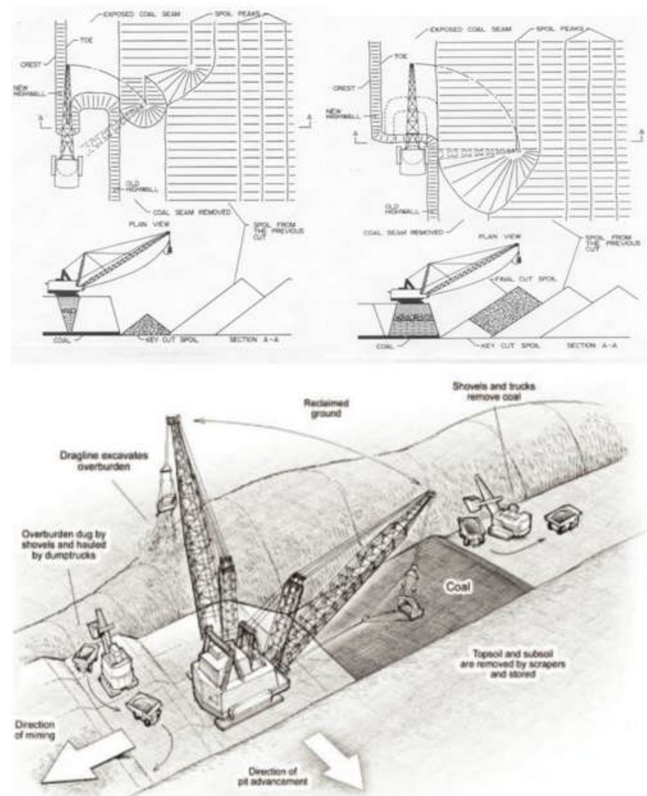


Fig: 5 Simple sidecasting methods

Dragline Extended bench method

Extended bench systems are adaptable to many configurations of pit geometry. In this method the dragline forms its working bench by chopping material from above the bench and forming the bridge, then moving onto the bridge to remove it from top of coal. The primary dragline strips overburden and spoils it into the previously excavated panel. This material is leveled, either by tractor-dozers or the secondary dragline, to form the bench for the secondary dragline. The secondary dragline first strips material near the highwall, then moves on to the bridge to move the rehandle material. Extended bench systems must be designed carefully in order to maximize the dragline(s) productivity and to minimize the amount of rehandle.

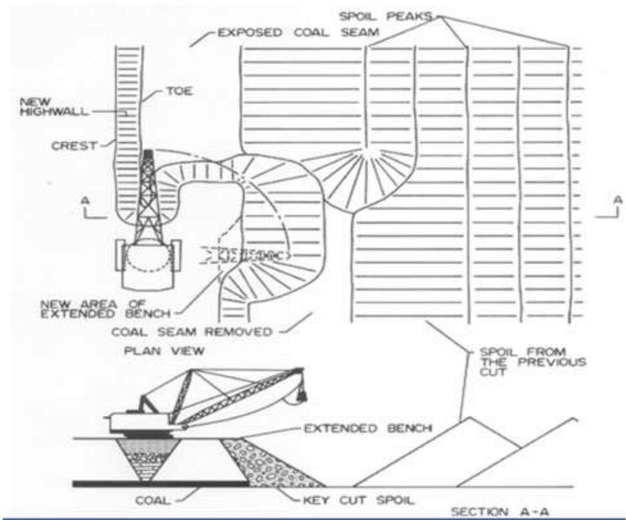
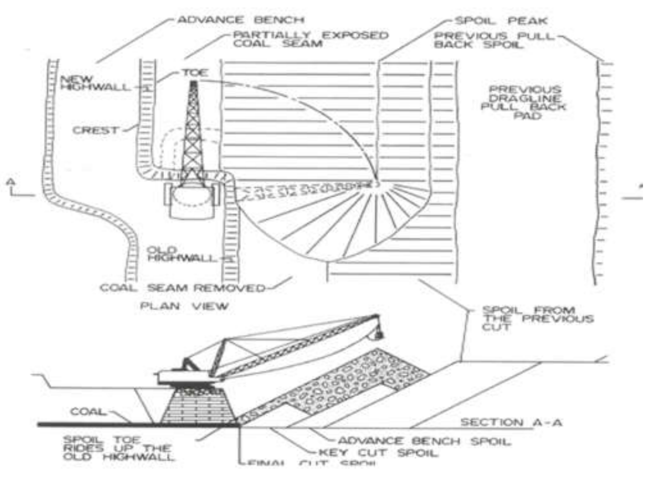


Fig: *6 Extended bench method of extravation*

Dragline Pull-Back Method

If the overburden/interburden is generally beyond the capability of draglines working on the highwall, the pullback method would seem to be a solution. In this case, a secondary dragline can be placed on the spoil bank to pull back sufficient spoil to make room for complete removal of overburden.



Fig; 7 Dragline Pull-Back Method

KEY CUT

The key cut is taken along the high wall.

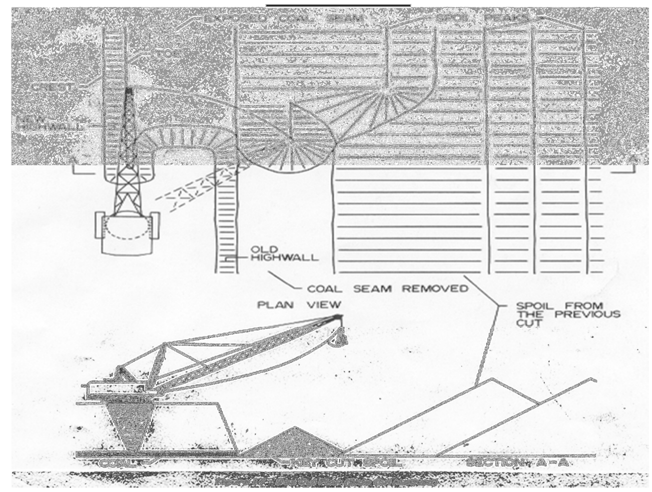


Fig: 8 Key Cut

Advantages of key cut:

1. The high wall is cleaned and dressed, so that there is no danger of looming loose and Boulder (this is also a recommendation of the inquiry committee of the accident of PH3 at **Dudhichua project NCL Singrauli**)

2. The swing angle of working of dragline while key cutting operation is 90° or <90°.

3. If the situation demands for extending bench method, the material of key cut is used for extending the bench.

Disadvantages of key-cut

1. The reach of the dragline is limited as the dragline is sitting along the high wall.

2. If the void available is sufficient and the cut width is less and permit the accommodation of over burden in void, without extending the bench. If key cut material cannot be accommodated without extending the bench, then re-handling percentage increases unnecessarily.

3. Re-handling percentage is more with key cut method, generally for single dragline.

Preparation of Balancing Diagram

The following points must be considered while preparing balancing diagrams:

1. The cut shell be divided in different sections as per the uniformity of the cut width, available dumping space (void) of previous cut and cut dumping, availability of void near central entry and mid entry. It is better to prepare different balancing diagrams for these sections. This will give the realistic picture for the actual deployment of the dragline.

2. The possibility of preparation/availability of formation/face, approach roads/ramps at proper time as per the requirement of balancing diagram shall also be checked, taking into consideration the actual geometry of the shovel benches.

3. Special attention shall be given for mid entry, central entry for extra overburden accommodation and safe dumping geometry.

4. The cut dumping, silt deposition, band dumping, dump- Bell, etc shall be taken into account for availability of void space in previous cut.

5. The possibility of simple side casting shall be tested, if sufficient void is available. This will avoid the necessity of extended bench method and will result in reducing the re-handling percentage.

6. The quantity of cut dumping shall be judiciously worked out. The detailed study is needed for proper analysis of optimizing the permissible quantity of cut dumping.

7. The detailed study is needed for gainful use of blast casting, determining profile of muck and volume of overburden casting by blasting, which may reduce the workload of dragline and also the effect on working level of dragline. If proper designed blasting reduces the bench height, the re-handling percentage can considerably be reduced. The blast profile shall be such that the level of total bench uniformly reduced, and heaving or uneven muck profile is minimum.

8. The necessity of key cut shall be properly analyzed.

9. The advance of both the dragline is balanced as per the requirement and the distance between the two draglines is constantly maintained. This can be achieved by proper distribution of workload (volume of overburden, solid and re-handling) between both the dragline.

10. If the operating level of dragline exposing coal is reduced then the percentage of re-handling will be considerably reduced.

11. The power layout and cable layout for both the dragline shall be given due consideration when designing balancing diagram.

12. The approach road for dragline working on lower bench for exposing coal shall be maintained for operation and maintenance crew as well as for dozers.

13. It is always better to blast both benches of dragline in vertical tandem in one go.
This will reduce the idle time for unproductive activities like shifting, marching, re-shifting, drilling twice, etc.

14. The swing angle for key cut is taken as 90°.

15. The swing angle for box cut is taken as 120 degrees.

16. The swing angle for re-handling is taken as 90°.

17. Therefore the productivity of dragline will differ for key cut, box cut and rehandling.

18. Coal exposure (million tons)

$$= L * W * H * g * MF * BF$$

Where,

L = solid advance of the dragline engaged in exposing coal (m)

= (Solid productivity of dragline engaged in exposing coal (million Cum)/(cross sectional area of volume which is handled by the dragline engaged in exposing coal (square meter))

W = cut width (m)
H = height of the coal seam (m)
g = specific gravity of coal
MF = mining factor (for rib and other operational losses) = 0.9
BF = band factor

Purpose of drawing balancing diagram

(i) It shows the dragline cut sections i.e. key cut, first cut (next to key cut), first dig (next to first cut) and rehandled section (as per mode of operation).

(ii) It shows the dragline bench height, cut width taken by draglines, thickness of coal seam and gradient and various slope angles.

(iii) Determination of rate of coal exposure (daily/monthly or annually).

(iv) Calculation of workload distribution on each dragline in respect of their annual productivity (i.e. cross-sectional area taken by each dragline should be in the same ratio as their annual productivity).

(v) Calculating the percentage of rehandling

Drilling and Blasting of Overburden for Draglines

For dragline stripping, there are two general methods of blasting overburden in common use. One method utilizes a blast-hole pattern with a buffer zone to contain the blasted material against the highwall. The advantage of this pattern is to contain the blasted material within the dragline working area and avoid large broken material that must be handled with difficulty. The widths of the buffer zone, combined with the powder factor, are critical elements in efficient utilization of this method. This method is useful, especially if the dragline is performing a chop cut prior to the key and production cuts.

The other method of blasting overburden is similar to the standard open pit blasting procedure. Its purpose is to blast as much material into the spoil area as possible, thereby reducing the amount that must be stripped by the dragline. The resultant advantage is debatable in the author's opinion, since considerable grading is necessary before the dragline can begin casting. Frequently the dragline is called upon to rebuild its working pad by retrieving material from the pit. Time lost in pad preparation may completely offset the original reduction in stripping volume

Present status of Draglines in NCL

SNo	Project	Model	Name	DOC	Prog Whr
1	Amlohri	24/96	Akshay	01.07.93	118232
2	Bina	24/96	Gaurav	10.03.86	162914
3		24/96	Saurav	06.04.87	160960
4	Dudhichua	24/96	Pawan	10.10.01	67431
5		24/96	Jyoti	01.09.91	108822
6		24/96	Vindhya	15.08.95	99856
7		24/96	Jwala	26.12.99	70857
8	Jayant	24/96	Saraswati	08.03.89	160564
9		24/96	Ganga	01.10.83	195824
10		24/96	Yamuna	02.03.85	183936
11	Nigahi	24/96	Krishna	13.04.02	68902
12		2/96	Balram	15.04.03	61424
13	Nigahi	20/90	Bairana	01.04.92	123682
14		20/90	Matang	25.03.94	120829
15	Khadia	10/90	Baijnath	02.10.92	123837
16		20/90	Vishwanath	26.06.95	113101
17	Jayant	15/90	Vijay	01.04.79	193896
18	Bina	10/70	Ram	15.05.78	201560
19		10/70	Shyam	08.05.79	206171

Dragline operation of India

DRAGLINES OPERATING IN INDIA

	Mine	30/70	24/96	20/90	15/90	10/70	10/60	11.5/45	6/45	5/45	4/45	6.9/70	Total
NCL	Amlohri		1										1
	Bina		2			2							4
	Dudhichua		4										4
	Jayant		3		1								4
	Khadia			2									2
	Nigahi		2	2									4
			12	4	1	2							19
SECL	Bisrampur	2											2
	Chirmiri					1							1
	Dhanpuri			1		1							2
	Jamuna					1				1			2
	Kurasia					1							1
	Rajnagar					1							1
	SUB TOTAL	2		1		5				1			9
MCL	Balanda							1					1
	Belpahar					1							1
	Bharatpur			1									1
	Hingula										1		1
	Lajkura					1	1						2
	Samleshwari					1							1
				1		3	1	1			1		7

	Mine	30/70	24/96	20/90	15/90	10/70	10/60	11.5/45	6/45	5/45	4/45	6.9/70	Total
WCL	Ghugus		1										1
	Sasti			1									1
	Umrer				1							1	2
	SUB TOTAL		1	1	1							1	4
BCCL	Block-II		1										1
	S.Tisra								1				1
			1						1				2
ECL	Sonpur Bazari		1										1
SCCL	Ramagundam-1		1										1
	Ramagundam-3	1											1
		1	1										2
INDIA		3	16	7	2	10	1	1	1	1	1	1	44

Conclusion

From the above calculation, it can be concluded that single dragline operation without re-handling is the preferable method if the production requirement can be satisfied. However, if large production is desired and extended bench method with single dragline adopted then tandem operation of dragline may be seriously considered. However this will entail capital expenditure on the purchase of a second dragline but the substantial savings which will result in terms of cost per tones of coal exposed will ultimately result in better economy for the mine. The discrepancies between the balancing diagram results and the results obtained from actual field performance study must be thoroughly examined in order to improve the productivity. In the present case, through the rigorous field monitoring the discrepancies were examined in swing angle, overall cycle time, availability, utilization and differential overall cycle time in blasted and re-handled mucks.

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