# PERFORMANCE APPRAISAL OF DRAGLINE MINING IN INDIA

A THESIS SUBMITTED IN PARTIAL FULFILLMENT OF THE REQUIREMENTS FOR THE DEGREE OF

**BACHELOR IN TECHNOLOGY**

**IN**

**MINING ENGINEERING**

BY

**BALAJI VEMANA**

108MN025

**DEPARTMENT OF MINING ENGINEERING**

**NATIONAL INSTITUTE OF TECHNOLOGY**

**ROURKELA-769008**

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# *CERTIFICATE*

This is to certify that the thesis entitled **"PERFORMANCE APPRAISAL OF DRAGLINE MINING IN INDIA"** submitted by **Sri Balaji Vemana** in partial fulfillment of the requirements for the award of Bachelor of Technology degree in Mining Engineering at National Institute of Technology, Rourkela (Deemed University) is an authentic work carried out by him under my supervision and guidance.

To the best of my knowledge, the matter embodied in the thesis has not been submitted to any other University/Institute for the award of any Degree or Diploma.

**Date :**

**Prof. H.K.NAIK**
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Assemblage of this nature could never have been attempted without the reference to and inspiration from the works of others whose details are mentioned in the reference section. I acknowledge my indebtedness to all of them.

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Date :                                                                 **BALAJI VEMANA**

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# ABSTRACT

Draglines have been abundantly used in coal mining for decades, either as stripper or stripper and coal extractor. As this equipment possesses certain inherent advantages, which their rivals do not, they must be operated in a round-the-clock fashion for high productivity and low costs. In India, the development of giant surface mining ventures like Bina and Jayant with setting up of higher coal production targets (upto 10 million tonnes per annum) calls for systems to remove large volume of overburden in shortest possible time. This has resulted in major changes in overburden/interburden excavation technology in surface coal mines from shovel mining to that of draglines. Coal India Limited (CIL) has now standardized the draglines in two sizes, which are 10/70 and 24/96 for their mines. Most mines depend on the dragline 24 hours a day, 7 days a week. In many coal mines, it is the only primary stripping tool and the mine's output is totally dependent on the dragline’s performance. For these reasons, dragline design requires emphasis placed on developing component’s with high levels of reliability and predictability so that repairs and replacement of components can be scheduled at times that will least affect the overall mining operation. Prior to deploying draglines in mines, various factors have to be considered for selection of suitable size. Different parameters are used to determine the production and productivity of draglines. In this thesis these points are discussed in detail.

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# Chapter 01

# INTRODUCTION

## 1.1. Dragline mining

Demand on energy is continuously increasing. Coal, which is the most homogeneously spread raw material throughout the earth's crust, is among the most demanded fossil fuels. A considerable portion of coal is produced by surface mining methods. Regarding the economics of scale extraction methods are highly mechanized and equipment with huge capacity are utilized. Draglines have been abundantly used in coal mining for decades, either as stripper or stripper and coal extractor. As this equipment possesses certain inherent advantages, which their rivals do not, they must be operated in a round-the-clock fashion for high productivity and low costs. Despite its colossal posture a dragline can be said to have a simple routine of work, which is composed of the following basic procedures: digging and walking. Among them walking is a steady process on which the mine design team has little control. Almost all walking draglines take a step of approximately 2 m within a time period of 0.75-1 min. The design of strip panels, equipping a specific unit with one operator's room on the desired side or with two on both sides and the management's strategy in coal loading operation largely affect the frequency and the length of long deadheading periods, during which the unit is unproductive (Erdem et al., 2003).

The dragline, since its inception many a decade ago, is now widely used to economically recover deposits at greater and greater depths. Extremely large draglines are now available with longer booms and higher bucket capacity parameters that go in line with an uprising productivity. Apart from a high degree of flexibility, utilization of a dragline results in entirely low cost per cubic meter and subsequent low cost per ton of the desired mineral. High initial capital outlay for dragline makes effective and efficient operation imperative to obtain low costs of overburden striping .Constant supervision, good overburden preparation, and preventive maintenance of dragline and selecting proper bench height of overburden for suitably selected machine should ensure efficient and effective operation of a dragline. Improvement in dragline productivity can have a dramatic effect on overall mining operation. Numerous field examples have led to believe that with correct application analyses, constant engineering and producing supervision, a dragline can provide an efficient solution to deep strip mining. To say the lees, its wide spread application in present mining industry is in sensible more so than ever.

## 1.2. Dragline mining in India

Indian mineral industry has contributed significantly to make the nation self-sufficient in coal. To meet the demands of thermal, cement and other users, the production trends in coal and lignite sectors have shown a remarkable increasing trend over last few years. While extracting the deep-seated coal deposit and also to increase the present production capacity, the coal mines have been compelled to modernize the mining technology, particularly in the fields of blasting.

Coal producers have already tried to open up big surface coalmines in various coalfields. This has further necessitated the importance of adopting better mining technology in the above mines by applying scientific and economic approaches while selecting the mining equipment and introducing the state-of-the-art technology. In this process it is important to adopt the blasting technology suitable for the mine as it affects the subsequent operations involved in the mining.

In India, cartridge explosives dominated the surface coal mines until the bulk explosives in the form of slurry and emulsion entered into the explosives market. Since the volume of overburden removal is increasing day by day, the majority of coal sectors have already been switched over to blasting with bulk explosives. As far as the type of explosive is concerned, the coalmines are currently using the slurry or emulsion based explosives with the gradual exit of NG based from market.

In India, the development of giant surface mining ventures like Bina and Jayant with setting up of higher coal production targets (upto 10 million tonnes per annum) calls for systems to remove large volume of overburden in shortest possible time. This has resulted in major changes in overburden/interburden excavation technology in surface coal mines from shovel mining to that of draglines.

Draglines have gained popularity in India for overburden stripping because of their flexibility and high production rate. Since the giant projects are coming up more in the coal sector in recent times, the shovel mining faces big challenges in fulfilling the production demand. Hence, the Indian surface coal mining has been switching over from shovel mining to dragline mining for removal of overburden/interburden in most of large sized coalmines to accommodate high rate of overburden removal and subsequently, high production rate with low cost of production.

The dragline mining was initially introduced in India in early 60's, and the first walking dragline was commissioned at Kurashia in 1961. Presently, there are about 43 draglines deployed to remove overburden ranging in bucket capacities from 4 cu. m to about 29-30 cu. m. Coal India Limited (CIL) has now standardized the draglines in two sizes, which are 10/70 and 24/96 for their mines. The economic life of a dragline has been assumed by CIL to be 27 years.

Northern Coalfields Limited (NCL) is the only subsidiary company of CIL, where the entire coal production is mined by opencast mining method. Another unique feature of the company is that about 40 per cent of the large volume of excavation is done with the help of larger walking draglines. Draglines are used in all the mines of NCL except in Jhingurda, Kakri and Gorbi.

## 1.3. Goal

The goal of the investigation is to measure and calculate the projected output, calculation of ownership, operating cost and cost per tonne of coal exposed by dragline by the combination of various parameters collected during field study and acquired from other sources. The secondary goal is to Develop a computer based program (C++) on dragline bench designing (keeping the cut-width constant) and determine whether or not rehandling is required and projection of annual production of overburden, calculation of ownership , operating cost operating and cost per cu.m overburden handle of the dragline

## 1.4. Objectives

1. Literature review on :
    - Draglines
    - System of working of dragline and methods of working of dragline

- Draglines used in India

2. Dragline balancing diagram and developing a computer based program (C++) on dragline bench designing (keeping the cut-width constant) and determine whether or not rehandling is required.
3. Projection of annual output, calculation of ownership, operating cost and cost per tonne of coal exposed by dragline.
4. Develop a computer based program (C++) on projection of annual production of overburden, calculation of ownership, operating cost operating and cost per cu.m overburden handle of the dragline.

## 1.5. Methodology

The specific objectives were achieved by the adoption of following methods:

1. Critical review of available literature
2. Visit to dragline mines for collecting and recording various parameters required for the study
3. Thorough calculation and computer programming to achieve the goals and objectives

# Chapter 02

## LITERATURE REVIEW

### 2.1. History and present

The dragline was invented in 1904 by John W. Page of Page Schnabel Contracting for use digging the Chicago Canal. In 1912 it became the Page Engineering Company, and a walking mechanism was developed a few years later, providing draglines with mobility. Page also invented the arched dragline bucket; a design still commonly used today by draglines from many other manufacturers, and in the 1960s pioneered an archless bucket design.

In 1910 Bucyrus International entered the dragline market with the purchase of manufacturing rights for the Heyworth-Newman dragline excavator. Their "Class 14" dragline was introduced in 1911 as the first crawler mounted dragline. In 1912 Bucyrus helped pioneer the use of electricity as a power source for large stripping shovels and draglines used in mining.

In 1914 Harnischfeger Corporation, (established as P&H Mining in 1884 by Alonzo Pawling and Henry Harnischfeger), introduced the world's first gasoline engine-powered dragline. An Italian company, Fiorentini, produced dragline excavators from 1919 licensed by Bucyrus.

In 1939 the Marion Steam Shovel Dredge Company (established in 1880) built its first walking dragline. The company changed its name to the Marion Power Shovel Company in 1946 and was acquired by Bucyrus in 1997. In 1988 Page was acquired by the Harnischfeger Corp., makers of the P&H line of shovels, draglines, and cranes.

Today, draglines are extensively used in strip mining of coal throughout the world. However, it has found wide range use in non-coal sector also, which includes surface mining of bauxite, phosphor, oil shale and tax sands. In the USSR, draglines are deployed widely for rehandling and sticking of 0/B spoil dumped by rail transport system. Occasionally, but rarely, these machines are used for loading into dumpers or bunkers as well for which special arch less buckets are available. In underwater digging such as for collecting sand and gravel, draglines are quite equipped with perforated buckets.

Presently there are five major manufacturers of draglines. They are Bucyrus Erie (US), Page (US), Marion (US), Rapier and Ransom (UK) and the Soviets. In India, Heavy Engineering Corporation is progressively manufacturing W-2000 model walking dragline indigenously in collaboration with Rapier and Ransom. Draglines used in open-cast mining typically range in size from machines equipped with 5 cubic meter drag buckets on 35 meter booms to the Bucyrus – Erie model 4250W, which is equipped with a 168 cubic meter drag-bucket on a 94.5 m boom. The longest boom length (121.9 m) dragline is offered by Bucyrus Erie, page, as well as, Marion. The largest boom from Ransom and Rapier is 105.5 m. The Soviets commissioned a long boom dragline with 120 m length during 1989. Works are now in progress to construct draglines having bucket capacity is high as 200 cubic meters. The current trend is to have machines with high bucket capacity and with short boom length. Apart from enhancing productivity and flexibility this arrangement can, most certainly, lend a degree of safety to the overall working conditions.

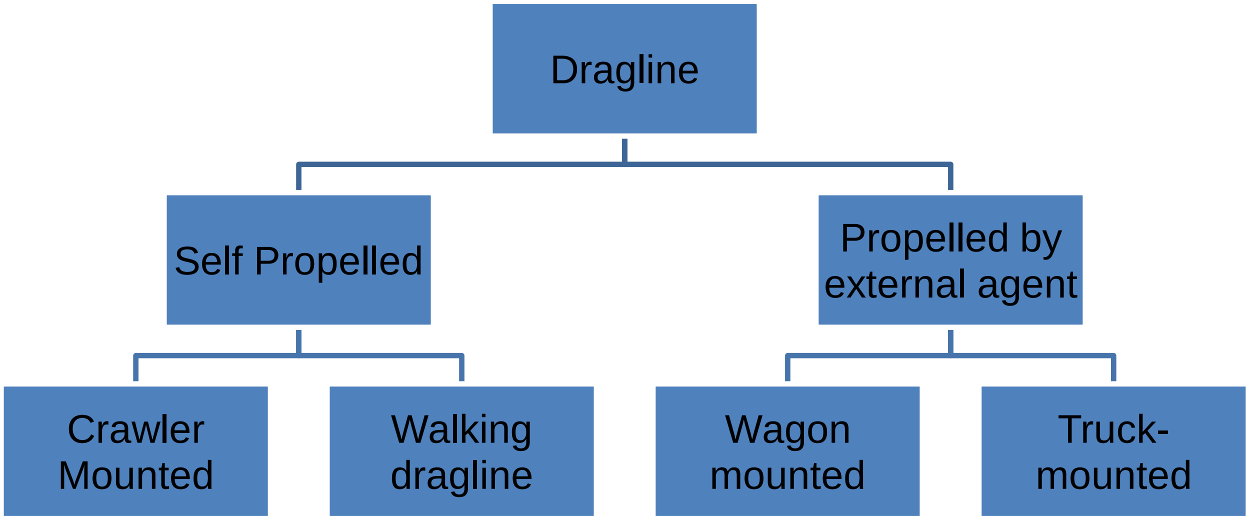
Most mines depend on the dragline 24 hours a day, 7 days a week. In many coal mines, it is the only primary stripping tool and the mine's output is totally dependent on the dragline’s performance. For these reasons, dragline design requires emphasis placed on developing component’s with high levels of reliability and predictability so that repairs and replacement of components can be scheduled at times that will least affect the overall mining operation.

Another critical designed consideration is that most repairs must be performed away from shop facilities. Although the dragline is a mobile piece of equipment, its enormous size prevents bringing to the shops for maintenance and repairs as s common with trucks and o her mine equipment. The designer must ensure that components are really accessible and that portable tools and rigging equipment are available for any eventuality.

## 2.2. Conditions for operation of dragline

1. Gradients flatter than 1 in 6
2. Seams should be free of 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

### 2.3. Classification of draglines




### 2.4. System of working

In a typical cycle of excavation, the bucket is positioned above the material to be excavated. The bucket is then lowered and the dragrope is then drawn so that the bucket is dragged along the surface of the material. The bucket is then lifted by using the hoist rope. A swing operation is then performed to move the bucket to the place where the material is to be dumped. The dragrope is then released causing the bucket to tilt and empty. This is called a dump operation.

The bucket can also be 'thrown' by winding up to the jib and then releasing a clutch on the drag cable. This would then swing the bucket like a pendulum. Once the bucket had passed the vertical, the hoist cable would be released thus throwing the bucket. On smaller draglines, a skilled operator could make the bucket land about one-half the length of the jib further away than if it had just been dropped. On larger draglines, only a few extra meters may be reached.

Draglines have different cutting sequences. The first is the side cast method using offset benches; this involves throwing the overburden sideways onto blasted material to make a bench. The second is a key pass. This pass cuts a key at the toe of the new highwall and also shifts the bench further towards the low-wall. This may also require a chop pass if the wall is blocky. A chop pass involves the bucket being dropped down onto an angled highwall to scale the surface. The next sequence is the slowest operation, the blocks pass. However, this pass moves most of the material. It involves using the key to access to bottom of the material to lift it up to spoil or to an elevated bench level. The final cut if required is a pull back, pulling material back further to the low-wall side.

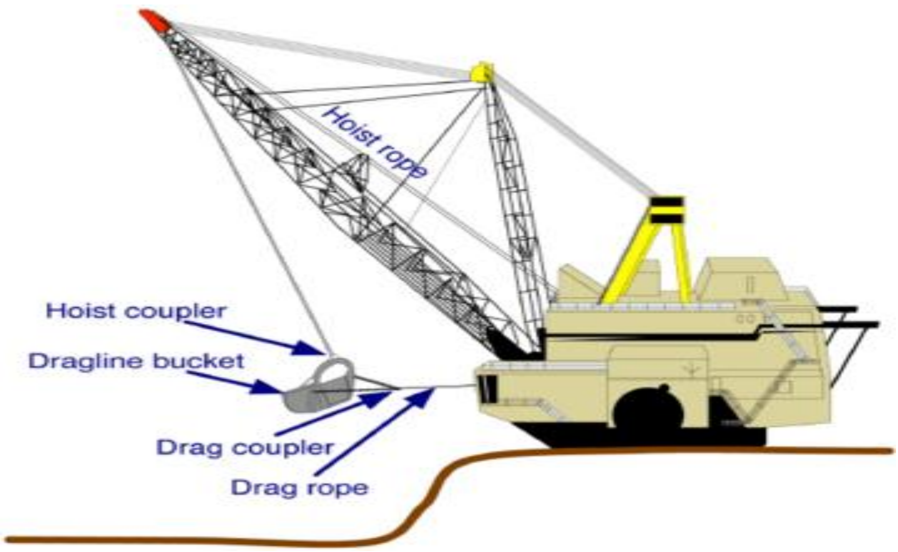



Fig. 2.1: Line diagram of dragline

The operating cycle of the dragline consists of five basic steps:

1. The empty bucket is positioned, ready to be filled.
2. The bucket is dragged toward the dragline to fill it.
3. The filled bucket is simultaneously hoisted and swung over to the spoil pile. If the swing motion must be slowed to permit hoisting, the dragline is said to be hoist critical. When hoisting to the dump position is completed before the boom is in position to dump, the dragline is said to be swing critical.
4. The material is dumped on the spoil.
5. The bucket is swung back to the cut while simultaneously being lowered and retrieved to the digging position.

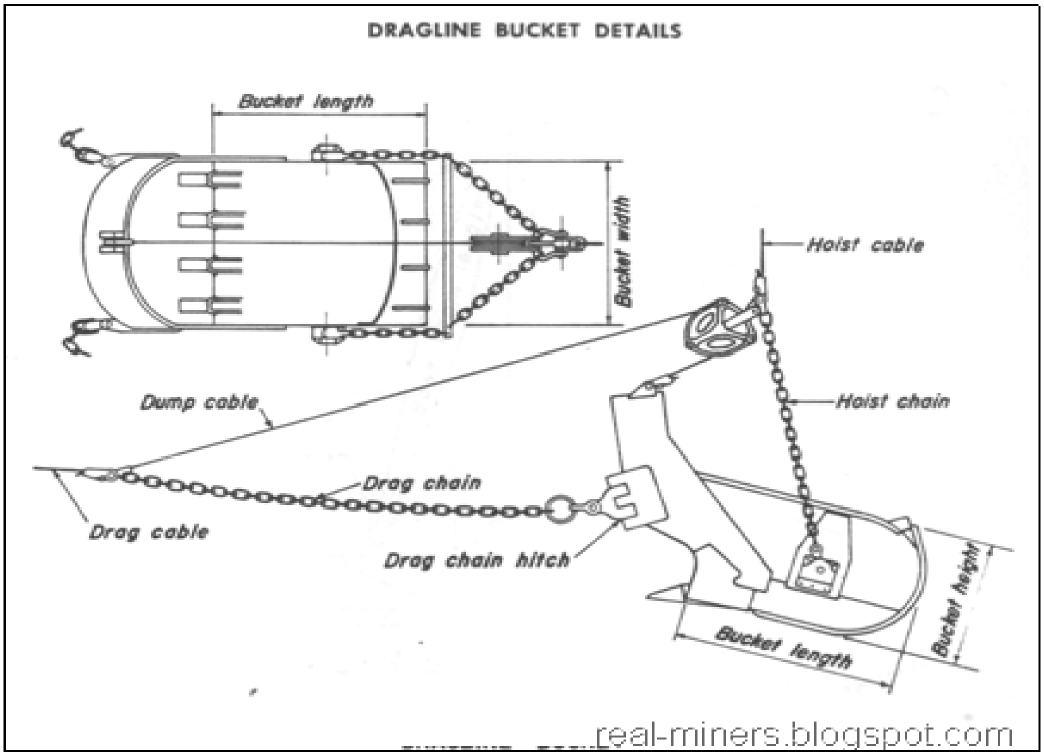



Fig 2.2: Parts of dragline bucket

## 2.5. Dragline stripping methods

The stripping cycle of dragline begins with the dragline cutting a trench, referred to as the key cut, along the newly formed highwall. The distance from the previous key cut position to the new position is referred to as the digout length. The key cut is made to maintain the panel width and uniform highwall. Without a key cut, the panel width would narrow with each subsequent digout, because the dragline could not control the bucket digging against an open face. The dragline deposits the key cut material in the bottom of the mined-out pit off the coal and against the previous spoil pile. More stable spoil from the key cut may be placed in the very bottom next to previous spoil to form the buckwall which provides a more stable spoil slope that can be steepened if deemed necessary.

When the key cut has been completed, the dragline is moved to new position to complete excavation of the digout. This is known as the production cut, and the material is cast on top of the key cut spoil. When the digout has been completed, the dragline is moved to next position, the beginning of the next stripping cycle (next digout).

Efficient dragline operation is realized by minimizing the time required to position, drag, and dump while synchronizing the swing and hoisting motions. Synchronization of hoisting and swinging is dependent on the time the boom is in motion.

**Full Key Cut vs. Layer Cut**: At the beginning of a new digout, the dragline generally is placed directly over the toe of the new highwall to be formed. From this position, the dragline can establish a uniform and safe highwall if the burden is sufficiently stable.

In this position, the dragline excavates the key cut which is more than the width of the bucket at the bottom of the cut. When the cut has been completed, the dragline moves over to make the production cut. The two positions generally are required because of the limited reach of the dragline in relation to the panel width being stripped. Large draglines, operating under ideal conditions, may be able to excavate the total digout from the one position over the highwall. Such situations are the exception, not the rule.

When operating conditions permit excavation of the dig out from one position over the highwall, the dragline generally excavates the digout in layers. The key cut is formed, one layer at a time, by excavating along the highwall before the completion of each layer. Cutting in layers can be performed from the production cut position; however, the high wall slope will require dressing by dozer while the dragline is digging. Under such circumstances, some mines also have adequately dressed the highwall by dangling a heavy section of chain from the bucket and dragging the chain along the wall. Other mines, because spoil area is critical, have progressively stepped the dragline toward the spoil while excavating in the layer cut method. This procedure has the tendency to pack spoil as tightly as possible on the spoil slope.

Layer cutting generally increases dragline productivity with a corresponding decrease in operating cost. Increased productivity is realized by progressively decreasing the average swing angle as the dragline walks in the direction of the spoil pile.

**Dragline Panel Width**: Panel width is defined as the width of the cut taken by the dragline, as it progresses from digout to digout, along the highwall from one end of the pit to the other. Panel width, one of the most important parameters affecting dragline productivity, is influenced by depth of overburden, dragline boom length, hoist and swing time, and available spoil area. Since panel width becomes the available operating area in the pit bottom, coal loading operations are also affected.

Several operational factors must be considered in the selection of panel width. A wide pit generally is favorable for coal load-out and permits greater safety for men and equipment. The minimum practical pit width is dictated by the maneuverability of coal loading and hauling equipment.

If the available space for placement of spoil is critical, such as might occur when crowding spoil to open haul roads through the spoil, narrow panels permit greater flexibility to deal with such problems. In general, the wider the panel, the less dragline walking time is required.

Productivity variations, because of panel width, are directly related to whether or not the dragline is swing critical. Small draglines can become swing critical at panel widths less than the width required for practical coal operations in the pit. Their cycle time also increases dramatically. Larger draglines may not become swing critical until the panel width exceeds 50 m (150 ft).

**Bench Height:** The height above the coal seam at which the dragline is positioned is defined as the bench height. Selection of the bench height is based on numerous operational factors and topographic restraints.

The complex relationship of bench height (which could be equal to overburden depth), panel width, dragline dumping reach and dumping height, as well as material characteristics such as swell and angle of repose, influence greatly the dragline's capability to dispose of burden off the coal. The dragline's digging depth, while related to burden depth, rarely becomes a factor in dragline performance.

The bench height must be selected primarily on the basis of fitting the dragline's specific characteristics to the required pit geometry. In general, the bench height should be as high as possible within the limit of required dragline reach.

Undulating topography may complicate a simplified selection of bench height. Two alternatives are available to alleviate the problem:

1. The dragline can be used to cut and fill to develop a common bench elevation. Cutting termed chopping or overhand digging, increases the cycle time and reduces the bucket fill factor, thereby reducing effective productivity. Fill material must be rehandled, thus reducing overall production. Chopping has very special advantages: the dragline reach required may be shortened, rehandling of burden may be avoided, fill may not be required to create a level working surface, a level return path for deadheading can be provided, and subsoil can be placed back in its relative position on top of the spoil.
2. Auxiliary equipment can be used to perform the cut and fill operation. Care must be exercised to ensure that filled areas are stable. Utilizing auxiliary equipment offers the benefit of freeing the dragline for its primary function of stripping burden from the coal.

Whether the dragline cuts and fills its working pad or auxiliary equipment is utilized to prepare a level working surface depending on several variables. Dragline chopping decreases overburden stripping productivity and may involve abnormal wear and tear on dragline equipment. Auxiliary equipment for prestripping adds to the capital and operating costs of operations. Depending on the thickness of the material to be chopped, the cost differential between chopping and using auxiliary equipment is not likely to be high when small draglines are compared. For large draglines, prestripping with auxiliary equipment will very likely be preferable.

**Digout Length:** The selection of digout length, the length between major digging cycles, is based on the relationship of the dragline's operating characteristics with respect to pit geometry. In general, the digout should be as long as possible. However, dragline size may greatly influence digout length for specific pit geometry. For example, digout length is sensitive when using a dragline with slow hoist speed working in deep overburden. Spoil critical pits may utilize a less than desirable digout length in order to pack the maximum material onto the spoil bank.

In general, a long digout with respect to dragline size reduces cycle time and increases productivity because more material is loaded under the outer end of the boom than near the fairleads of the dragline. A good dragline operator will try to fill the bucket within two and a half to three times the bucket length. Cycle components of retrieving for bucket loading, bucket dragging, payout for dumping, and swing angle all are decreased as the digout length increases. Longer digout lengths also reduce the nonproductive time required for repositioning the dragline on the succeeding digout. Obviously, digout length should not be so long as to require the dragline operator to cast the bucket beyond the limit of the boom.

**Walking Patterns**: In a dragline operation, two separate walking cycles are involved: deadheading and walking within the digout. When a panel has been completed, there are two options available for the dragline. One, the dragline can wait for the coal to be mined to the end of the pit, then turn around and begin the next panel in the opposite direction. This procedure is termed laying over at the end of a panel. Two, the dragline can turn around and travel part or all of the way down the panel to begin the next cut. This procedure is termed deadheading. If the dragline travels part way down the panel, it cuts into the next panel and strips in the opposite direction. If the dragline travels to the other end of the pit, it cuts in to the next panel and strips in the same direction.

Whether a dragline lays over or deadheads depends primarily on the production time lost. Contractual requirements, such as exposed coal inventory, may eliminate the layover option and force deadheading. Some mines opt to lay over at the end of a panel because ground conditions are not favorable for deadheading. Other mines limit layover to a maximum of two shifts. If waiting for coal production involves two or more lost dragline shifts, the dragline will be deadheaded.

When deadheading is feasible, the decision should be made on the basis of minimum lost dragline production. Deadhead time is based on 33% of the specified dragline walking speed. Such a large discount factor must be used to account for various delays in deadheading, such as maneuvering, cable handling, ground preparation, and minor breakdowns.

The greater the digout length, the less walking time will be required per panel. Repositioning in the digout can affect cycle time. Therefore, walking patterns must be considered when selecting digout length. Time spent in repositioning the dragline can be estimated by discounting the walking speed of the dragline. Generally the discount factor is much less than the factor utilized for deadhead estimates. Based on observations by the author, a discount to 15 to 20% is appropriate.

**Pit Shape**: The new dragline pit begins with the initial cut, termed the box cut, made along the outcrop, subcrop, or property boundary. To open the box cut, excavated material is spoiled to one or both sides of the cut. The material lying on the newly created highwall must be moved or spread out evenly by auxiliary equipment. The material lying on the cut wall that will become the spoil side may, or may not, have to be moved depending on reclamation requirements.

Because of rolling topography, the mine engineer may be inclined to design the box cut along a uniform contour. Generally, succeeding cuts are designed parallel to the box cut. As a result, this type of pit develops a meandering design. Obviously, outside curves provide more spoil area. Depending on depth of overburden, panel width, radius of curvature, and operating parameters, severe operating problems may occur on inside curves. Dragline cycle time will increase, spoil crowding will occur, and coal may be lost by being covered with spoil.

To remedy the problems caused by inside curves, several options can be considered. Panel width may be decreased, material may be cast short and rehandled by extending the bench, a small auxiliary dragline may be utilized on the spoil to pull back excess spoil, the spoil pile may be steepened, or the pit may be straightened by stripping a series of short panels. Generally, the most favorable solution is to straighten the pit. Spoil steepening is also an effective method for disposing of relatively small amounts of excess spoil. The dragline bucket is positioned on the spoil slope where steepening is desired and dragged down and across the top of coal. The bottom part of the spoil pile is steepened and coal is cleaned in the process. Digging efficiency during this process is reduced, cycle time increased, and rehandling reduces effective productivity. Steepened spoil slopes may present special hazards to equipment and personnel because they are more prone to failure.

Spoil Patterns: There are three basic methods of spoiling. When using short digouts and casting at a near $90^0$ angle, a uniform ridge line can be created. This configuration makes maximum use of the available spoil room. As the digout length is increased, uniformity of the ridge line is lost and individual peaks of spoil are created.

With sufficient spoiling area, the dragline operator may cast material from both the key cut and production positions at angles less than $90^0$. While the dragline stripping cycle will improve, spoil piles appear to be ragged and irregular. An aerial view of the operation will show a definitive pattern to the irregularity. In reality, spoil peak grading will be reduced by this method of spoiling.

Dragline cycle time can be reduced by dumping the loaded bucket on the fly, that is, before the dragline swings to the ultimate dumping position. This procedure, termed radial casting, gives the spoil a cross bedded appearance. Provided that there is sufficient spoil room, radial casting tends to spread the spoil more effectively, reducing spoil grading costs.

Since distance between spoil ridges is equivalent to the panel width, narrower panels will reduce spoil grading costs. However, such reductions in cost generally will be offset by increases in dragline operating cost if the dragline is not swing critical.

### 2.5.1.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 0/B from each strip is dumped into the void left by the previous strip after the coal mineral has been mined. 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 nearest high wall is affected by starting the out with the dragline in line with the crest and moving it as the out gets deeper, ending with the machine in line with the toe of the new high wall. By this means, the slope angle of the new high wall can be closely con rolled. 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.

When the key cut has been completed, the dragline is moved close to the old high wall edge from where it can excavated the reminder of the blocks. With a suitable selection of bench height and block width, as well as, proper reach, casting can be done dear off the coal bench.

However, more often than not, the spoil pile touches the crest of the coal seam for obvious advantages mentioned early. Associated demerits are also present. Rehandling is no intended as it jeopardizes the economy of operations. Advance benching with this method is also practiced due o reasons already mentioned.

The manner in which a dragline must be applied to dispose of the material is of greater significance in affecting dragline productivity. In the simple case shown in the Fig. the dragline sets on the top of the material to be excavated and swings through an arc of between 45 to 90 degrees to dump the material. A typical average cycle time for the operation is 45 seconds.

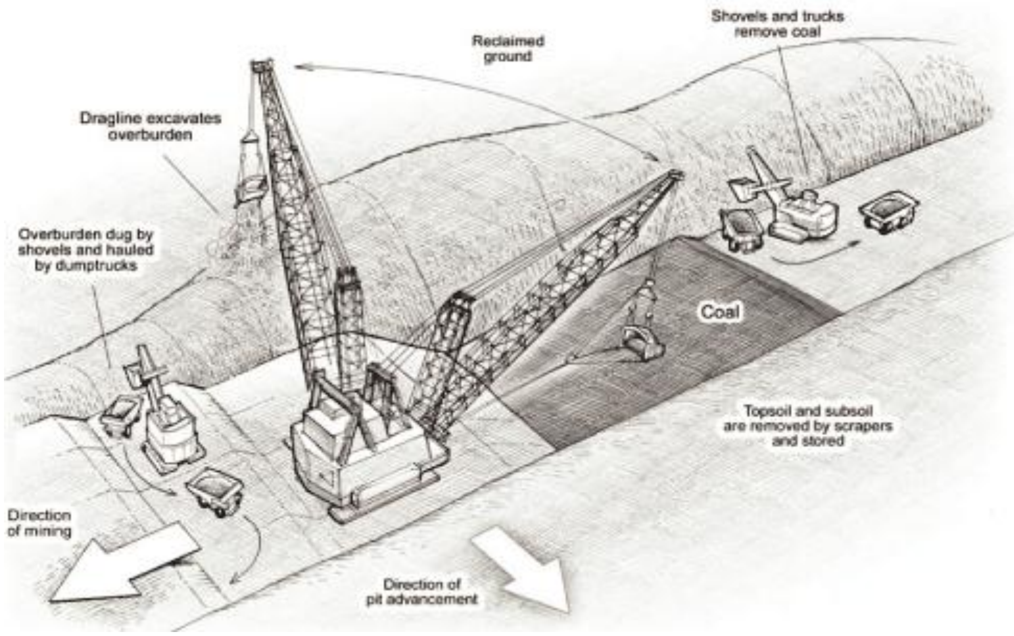



Fig 2.3: Simple sidecasting method

To obtain maximum reach, it is necessary to work the machine as close as possible to the high wall crest. In addition to the obvious risks to very expensive equipment, this practice reduces the degree of blasting which can be employed. In order to preserve a satisfactory edge from which to work, several mines 'buffer shoot' two or three strips ahead of the dragline. Buffer shooting is undoubtedly less efficient than shooting to a free face and no advantage can be taken of the material cast by the shot.

### 2.5.2. Dragline Extended bench method

Where overburden depth or the panel width exceeds the limit at which the dragline can sidecast the burden from the coal, a bridge of burden can be formed between the bank and the spoil which effectively extends the reach of the dragline. The bridge extends the bench on which the dragline is operating. The bridge is formed by material falling down the spoil bank or by direct placement with the dragline. To remove the bridge material from the top of coal, it must be rehandled.

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. In a two-dragline system, one machine must operate at the pace set by the other. Therefore, mine design must consider their respective capacities when assigning respective digging depths. The primary dragline strips overburden to the top of the first seam. Coal is removed, then a small parting dozed into the pit and the second coal seam removed. The secondary dragline strips the large interburden to the third and final seam. Extended bench systems must be designed carefully in order to maximize the dragline(s) productivity and to minimize the amount of rehandle.

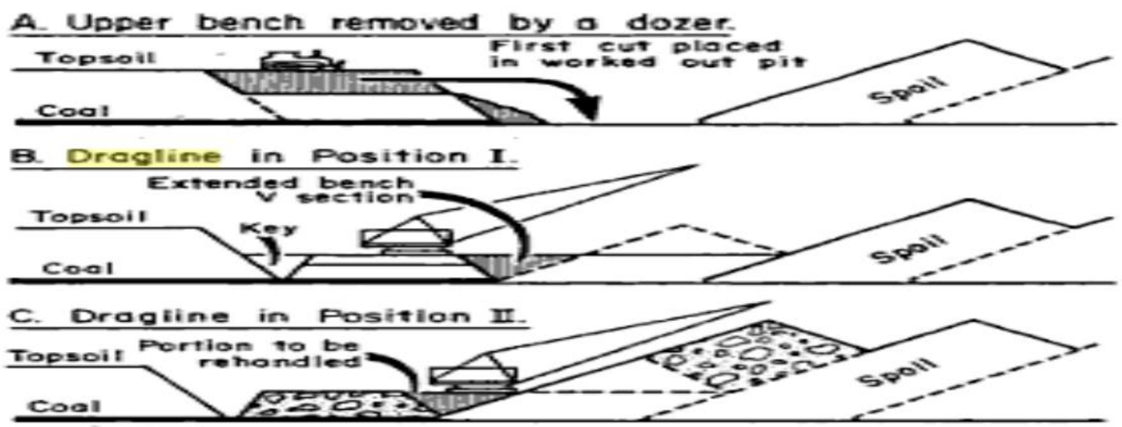



Fig 4: Positions in extended bench method

### 2.5.3. Dragline Pull-Back Method

Occasionally, overburden to be stripped will be beyond the capacity of the dragline to spoil off the coal by any of the previous methods described. 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.

Generally, rehandle volume is greater for the pull-back than an extended bench method of operation. However, it may also serve to level spoil piles in addition to providing more spoil area for the primary dragline. If the overburden/interburden is generally beyond the capability of draglines working on the highwall, the pullback method would seem to be a solution. However, great care must be given to the design of this method because of the inherent hazards of operations. Spoil slopes can be unstable, more so during periods of severe rainfall.

Draglines frequently are utilized to strip overburden from deeper coal seams than originally intended. Occasionally, spoil slopes cannot be maintained at designed angles. Various methods have evolved to stack more material into the spoil bank to alleviate these problems. The more common methods are described briefly:

1. Buck walls involve building the base of the spoil adjacent to the pit with competent material so that a steeper spoil slope near the base can be maintained.
2. Coal fenders require leaving a small wedge of coal untouched in the pit so that more spoil can be packed on the spoil slope.
3. Outside pit involves modifying the pit shape in order to develop the outside curve concept which increases the spoil area relative to the stripping area.

### 2.5.4. Tandem machine systems

Tandem machine stripping of two coal seams and/or deeper overburden can be done by using a dragline with a second system which can be another dragline, shovel, scraper dozers, etc. This system has two advantages as:

(i) One machine removing both overburden (thickness 100ft) and interburden (thickness 20ft) will probably be less efficient than two machines – one designed specifically for overburden removal and the other for interburden removal.

(ii) Production capacity, provided the operations were well planned, in tandem machine operations may be greater than that in single machine operations under similar operating conditions.

Disadvantages of the tandem system include when loading machine is down, the trailing machine will often be idled. Such situations can however be minimized by good planning.

## 2.6. Drilling and Blasting of Overburden for Draglines

Overburden drilling and blasting is more critical for dragline stripping than for shovel digging. Shovels have the ability to crowd the dipper into the bank, providing leverage to dig difficult or poorly blasted material. Draglines have leverage only by dragging the bucket over the material.

Such leverage is translated to severe strain on the bucket lip and teeth. In poorly blasted material, dragline productivity can drop more rapidly than that of shovels working in similar material.

Selective placement of explosives and blasting agents may be critical to the surface coal mine operation. Many coal seams are overlain with sedimentary beds of varying hardness and thickness. Improper placement of the charge in the blasthole can cause blast energy to travel along planes of greater weakness and through softer material. Under such conditions, harder beds of material will tend to break in large blocks or fragments. To ensure adequate placement of the blast charge, it is necessary that drill operators log differences in material or drill penetration rates and provide this information to the blasting foreman.

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. If the dragline can safely work on the spoil side of the pit, building its working pad ahead on the spoil, there may be justification for blasting material from highwall to the spoil. Increased costs of explosives, pad building costs, and highwall scaling delays must be weighed against the difference in overburden volume to be stripped (SME handbook.,).

## 2.7. Draglines in use in India

**Table 2.1- Bharat Coking Coal Ltd. (BCCL)**

| | Project | Capacity of Dragline | No. of Draglines | Geo-mining conditions |
|---|---|---|---|---|
| 1. | Block II | 24x96 | 1 | Mid seam of coking coal worked. OB dumped in coal bearing area to be removed later |
| 2. | Joyrampur | 5x45 | 1 | -------------------- |
| Total for BCCL | | | 2 | |

**Table 2.2- Eastern Coalfields Ltd. (ECL)**

| | Project | Capacity of Dragline | No. of draglines | Geo-mining conditions |
|---|---|---|---|---|
| 1. | Sonepur Bazari | 26 cu m | 1 | Multi seam deposit, bottom medium thick seam exposed by dragline |
| Total for ECL | | | 1 | |

**Table 2.3- Mahanadi Coalfields Ltd. (MCL)**

| | Project | Capacity of Dragline | No. of Draglines | Geo-mining conditions |
|---|---|---|---|---|
| 1. | Balanda | 4x45 - 1<br>10x60 – 1<br>11.5 cum– 1<br>20x90 – 1 | 4 | A thick seam (10-16 m) is split into 3 to 4 splits in part of the area. Mostly, single seam working |
| 2. | Belpahar | 10x70 | 1 | Parting between two seams taken by dragline |
| 3. | Lajkura | 10x70 | 1 | OB above a thick seam interbanded seam taken by dragline |
| 4. | Samaleshwari | 10x70 | 1 | --------------- |
| Total for MCL | | | 7 | |

**Table 2.4- Northern Coalfields Ltd. (NCL)**

| | Project | Capacity of Dragline | No. of Draglines | |
|---|---|---|---|---|
| 1. | Amlori | 24x96 | 1 | MOHER-SUB BASIN, SIngrauli Coalfield. The NCL is presently working in Moher sub-basin of Singrauli coalfield. The basin has three seams in most of its area. |
| 2. | Bina | 10x70 - 2<br>24x96 - 2 | 4 | |
| 3. | Dudichua | 24x96 | 2 | |
| 4. | Jayant | 15x90 - 1 | 4 | |

| | | | | |
|---|---|---|---|---|
| | | 24x96 - 3 | | The upper seams are 8-10 m thick with a parting of about 40 m in between. The lowermost seam is 16-22 m thick and has a parting of about 40 m between it and the second seam. The seams are flat (about 2 degree gradient). Upper seams are worked by shovel dumper combination and draglines are used only for removal of OB above the bottom most seam. When all the three seams are worked in any project of this sub-basin, the percentage of OB handled by dragline will only be 20-25 % of the total OB |
| 5. | Khadia | 20x90 | 2 | |
| 6. | Nigahi | 20x90 | 2 | |
| Total for NCL | | | 15 | |

**Table 2.5- South Eastern Coalfields Ltd. (SECL)**

| | Project | Capacity of Dragline | No. of Draglines | Geo-mining conditions |
|---|---|---|---|---|
| 1. | Bisrampur | 30 cu.m | 2 | Single thin seam at shallow depth |
| 2. | Chirimiri | 10x70 | 1 | 12-13 m thick seam developed by bord and pillar previously |
| 3. | Dhanpuri | 10x70 – 1<br>20x90 – 1 | 2 | 6-7 m thick seam |
| 4. | Dola/Rajnagar | 10x70 | 1 | Two thick seams with thin parting in between |
| 5. | Jamuna | 5x45 – 1<br>10x70 – 1 | 2 | Thin seam at shallow depth |
| 6. | Kurasia | 5x45 – 1<br>10x70 – 1<br>11.5 cu.m – 1 | 3 | Multi seam working with thin partings in between |
| Total for SECL | | | 11 | |

**Table 2.6- Western Coalfields Ltd. (WCL)**

| | Project | Capacity of Dragline | No. of Draglines | Geo-mining conditions |
|---|---|---|---|---|
| 1. | Ghughus | 24x96 | 1 | Single thick seam (16 to 22 m) developed in two sections |
| 2. | Sasti | 20x90 | 1 | Single thick seam ( 16-22 m ) |
| 3. | Umrer | 4x45 – 1<br>7 cu.m – 1<br>15x90 – 1 | 3 | Multi seam deposit, bottom seam is thickest.<br>Shovel-dumper for upper seams. |

| | | | | |
|---|---|---|---|---|
| | | | | Small dragline used for rehandling |
| Total for WCL | | | 5 | |

**Total for Coal India Ltd. = 41**

**Table 2.7- Singareni collieries Co. Ltd. (SCCL)**

| | Project | Capacity of Dragline | No. of Draglines | Geo-mining conditions |
|---|---|---|---|---|
| 1. | Ramagundam OC-I | 24x96 | 1 | Upper seams exposed by shovel-dumper. Lower seams exposed by dragline |
| 2. | Ramagundam OC-III | 30 cu.m | 1 | Parting between two seams taken by dragline |
| Total for SCCL | | | 2 | |

**Grand total for India = 43**

# Chapter 03

## PARAMETER COLLECTION, RECORDING AND ACQUIRING

The parameters such as bucket capacity, boom length, reach, dumping height, cut width, angle of repose, highwall angle, bench height, digging depth, method of working of two draglines under study were collected and the parameters such as cycle time were recorded form the field. The parameters from Samaleswari (MCL) mine of scheduled shift hours, average working hours, average idle hours, average breakdown hours, and average maintenance hours were acquired form previous recorded data and those of Singareni OC-I mines was acquired from a previous field study by Rai., 2004.

# Chapter 04

# CALCULATION AND PROGRAMMING

## 4.1. Dragline balancing diagram

Balancing diagram can be defined as the graphical representation of the scheme to be adopted for determining the suitable seating position of the dragline in order to get maximum overburden accommodation in decoaled area with least rehandling for achieving high rate of coal exposure and ensuring slope stability (Rai, 1997).

The balancing diagram assists in determining the coal exposed by a dragline, the percentage of overburden, rehandling and the volume of overburden to be accommodated in the decoaled area (Singh and Rai, 1998).

Besides these, balancing diagram shows the dragline cuts and spoil geometry (in two dimensions) cross-section, height of dragline bench and cut width taken by the dragline. The cuts sequence by dragline, key cut (box cut), first cut (next to key cut), and first-dig can be estimated through the cross-sections drawn in diagram (Pundari, 1981).

## 4.2. 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-section area taken by each dragline should be in the same ratio as their annual productivity).

(v) Calculating the percentage of rehandling.

(vi) Calculating the overburden to be accommodated in the decoaled area.

### 4.3. Preparation of Dragline balancing diagram

Let BCDE be the cross-sectional area to be removed to expose coal seam A B C O. for convenience, this area be called First-dig.

Let, A1= First-dig ……………… (1)

Now the dragline sitting on the highwall side removes the blasted overburden which lies in the cross-sectional area of first-dig. Maximum amount of overburden which can be accommodated in the dump FGKH is limited by the reach of the dragline and designed dump-slope.

Let A2= Dump Area ………….. (2)

Assuming S to be the swell factor of the overburden material, actual area of overburden required to be accommodated in dump would be A1S.

Let, A3= A1S-A2 …………….. (3)

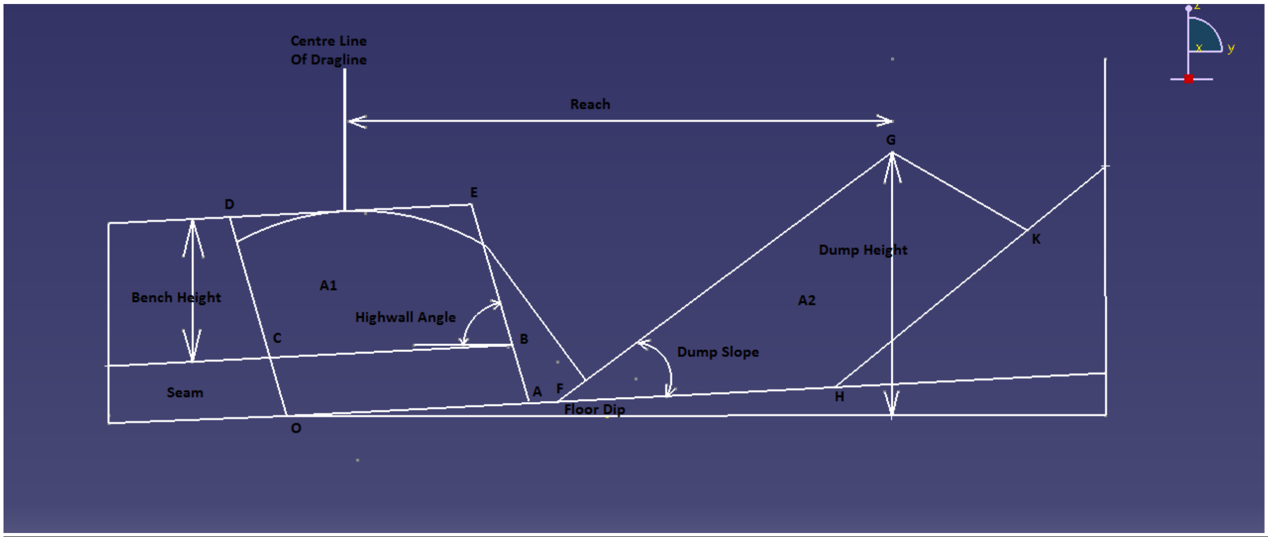



Fig 4.1: Dragline balancing diagram for total sidecasting

**Case-1. When A3<0**

In this case the dump area is incapable of accommodating overburden more than the available first-dig quantities. This implies that height of the dragline bench or cut width may be increased such that the first-dig quantity is increased. This process is repeated till the dump area is equal to the losses first dig quantity.

**Case-2. When A3=0**

This indicates an optimum solution for the simple side-casting method of dragline deployment. In simple sidecasting operation there is no rehandling of material and thereby it is the most economical operation. Any increase in the height of dragline bench or cut width would give rise to an increase in first-dig and this increase is not possible to be accommodated in the dump.

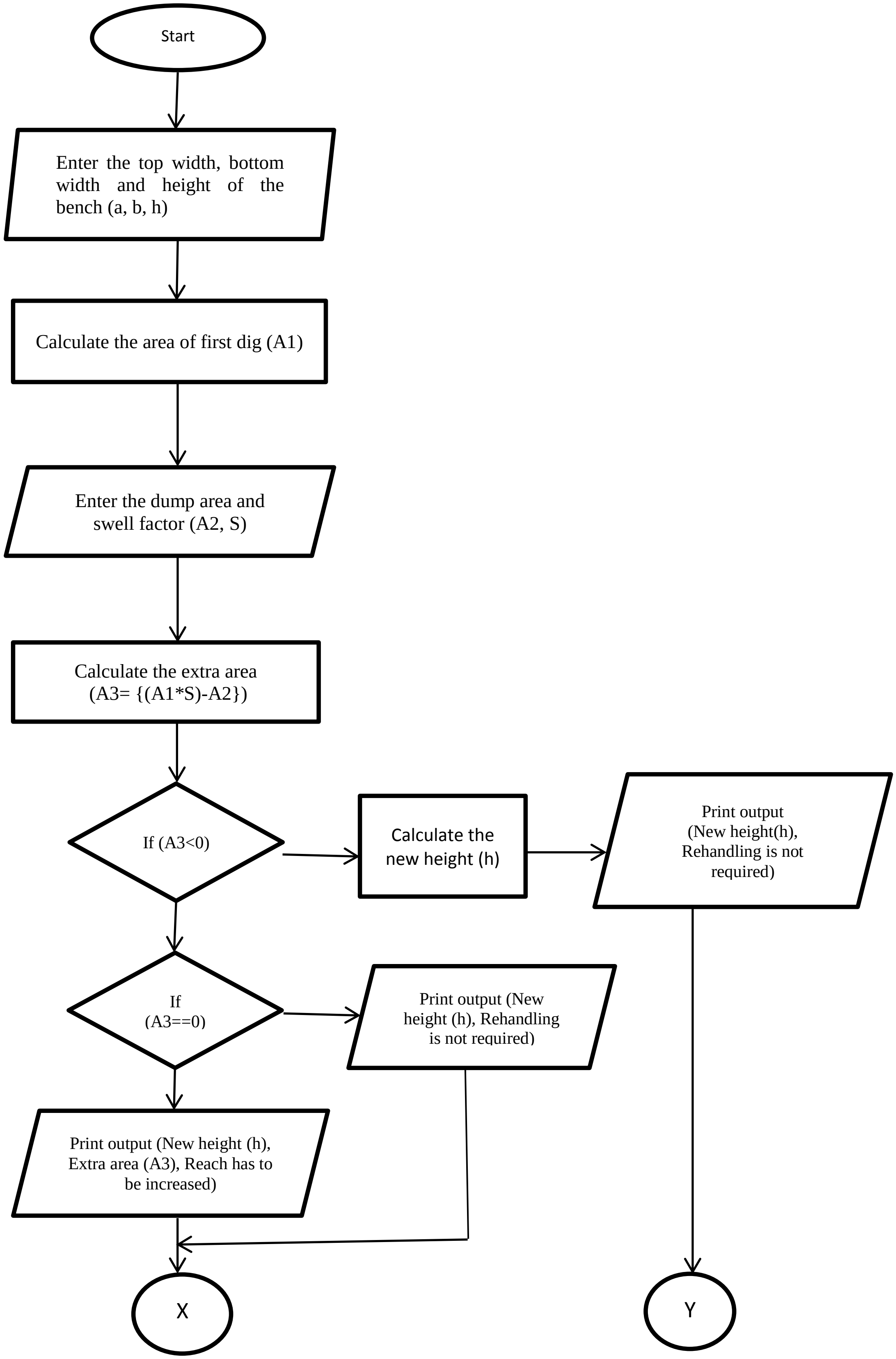
**Case-3. When A3>0**

This implies that the dump is incapable of taking the loose first-dig completely and A3 amount of overburden would be left as residual. This residual can be handled in two ways, either by transporting and dumping elsewhere or by generating extra dump capacity can be increased by increasing reach. Reach can be increased by selecting different equipment with higher reach. But the choice of availability is limited. Alternatively the reach can be increased by shifting the dragline towards the dump side. Extended bench method of dragline deployment is employed for this purpose.

### 4.4. Developing a computer based program (C++) on dragline bench designing (keeping the cut-width constant) and determine whether or not rehandling is required

```
#include<stdio.h>
#include<stdlib.h>
float a,b,h,A2,s,A3,A1;
void area1()
{
    A1=(((a+b)/2)*h);
    printf("the first dig area is now %.3f\n",A1);
}
void area3()
{
        A3=(A1*s-A2);
}
int main()
{

  printf("enter length, breadth and height\n");
  scanf("%f%f%f",&a,&b,&h);
  area1();
  printf("enter dump area and swell factor\n");
  scanf("%f%f",&A2,&s);
  area3();
  if(A3<0)
  {

            printf("the height earlier was %.3f\n",h);
            h=(2*A2)/(s*(a+b));
            printf("the height is now %.3f \n the extra area is 0 \nno rehandling
required\n",h);
```

```
    }
    else if(A3==0)
    {
        printf("the height is now %.3f \nno rehandling required\n",h);
    }
    else
    {
        printf("the height is now %.3f and the extra area is %.3f\nthe reach has to be increased\n",h,A3);

    }
    system("pause");
    return 0;
}
```


Start
Enter the top width, bottom width and height of the bench (a, b, h)
Calculate the area of first dig (A1)
Enter the dump area and swell factor (A2, S)
Calculate the extra area (A3= {(A1*S)-A2})
If (A3<0)
Calculate the new height (h)
Print output (New height(h), Rehandling is not required)
If (A3==0)
Print output (New height (h), Rehandling is not required)
Print output (New height (h), Extra area (A3), Reach has to be increased)
X
Y

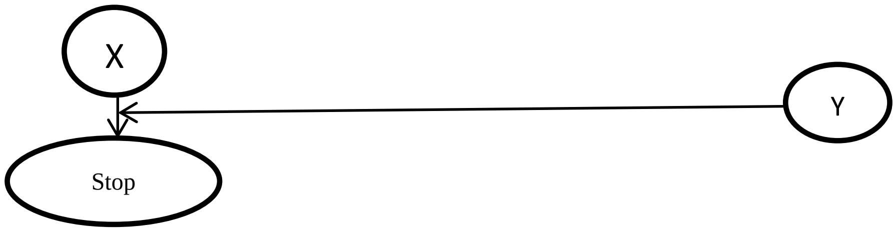



Fig 4.2: Flowchart for the on dragline bench designing (keeping the cut-width constant) and determine whether or not rehandling is required

## 4.5. Output (using user-defined data)

### 4.5.1 Case 1 (A3<0)

Input entered:

| | |
|---|---|
| Top width of cut (length) | : 80 m |
| Bottom width of cut (breadth) | : 50 m |
| Height of bench | : 28m |
| Dump area | : 2400 $m^2$ |
| Swell factor | : 1.25 |

Output:

The height earlier was 28.00 m

The new height is 29.538 m

The extra area is 0

No rehandling is required

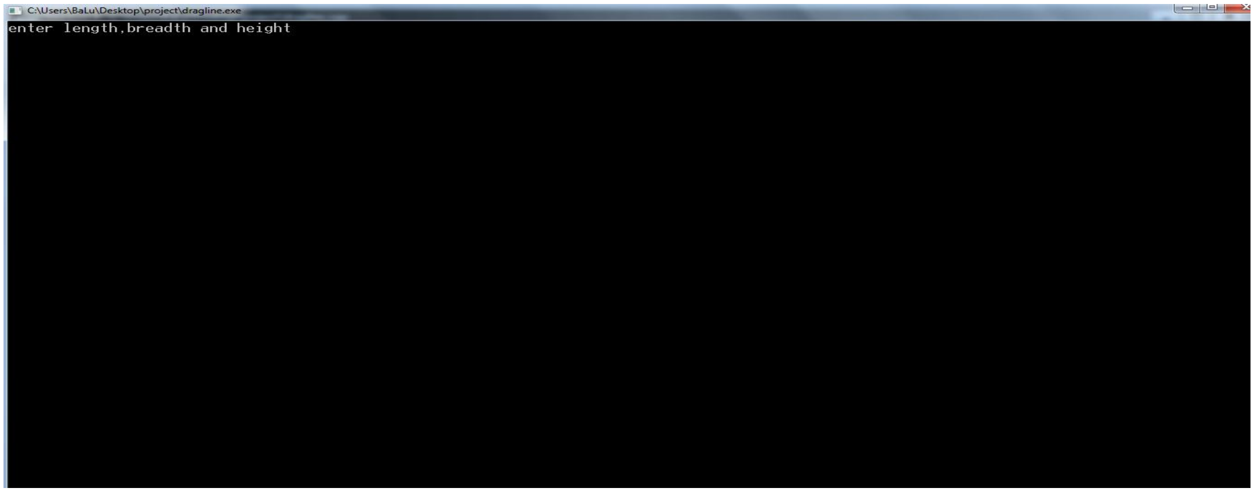



Fig 4.3: Output screenshot 1

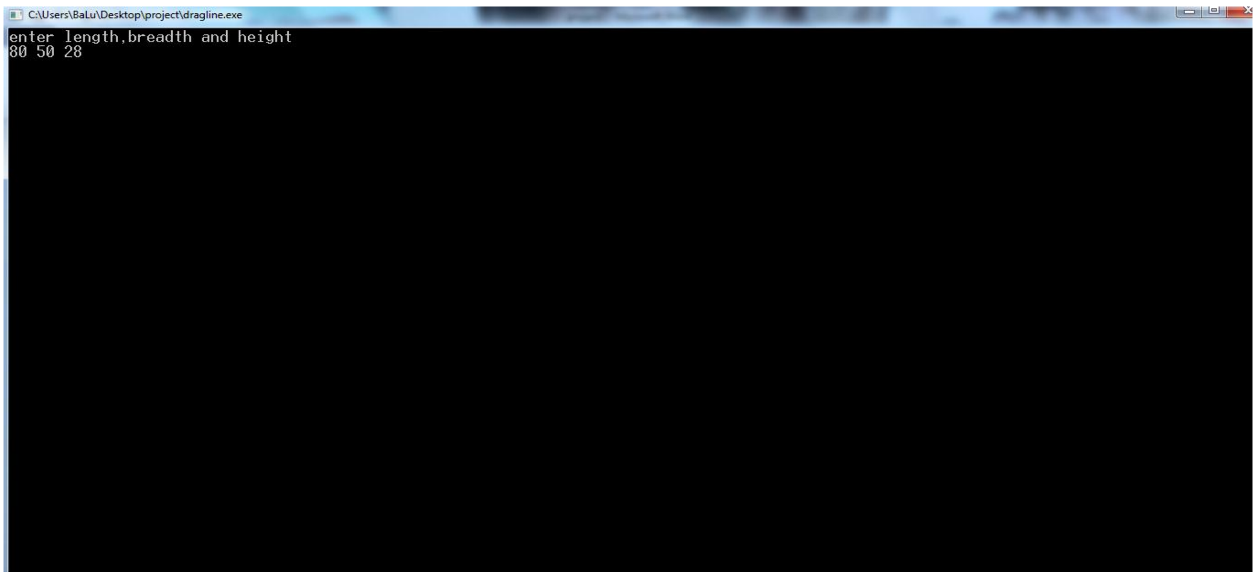



Fig 4.4: Output screenshot 2

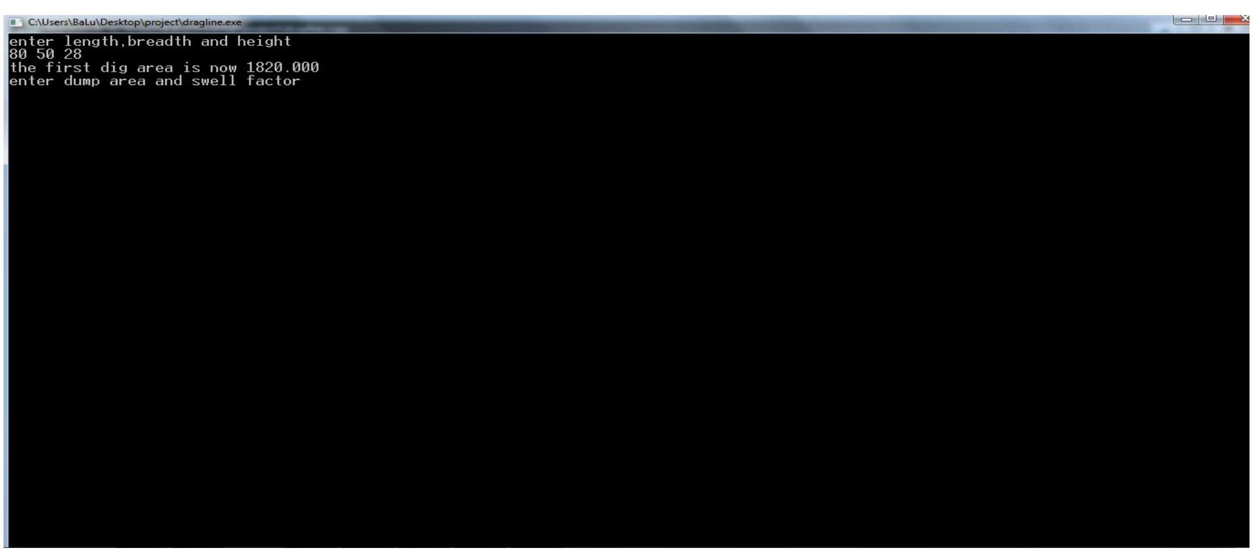



Fig 4: Output screenshot 3

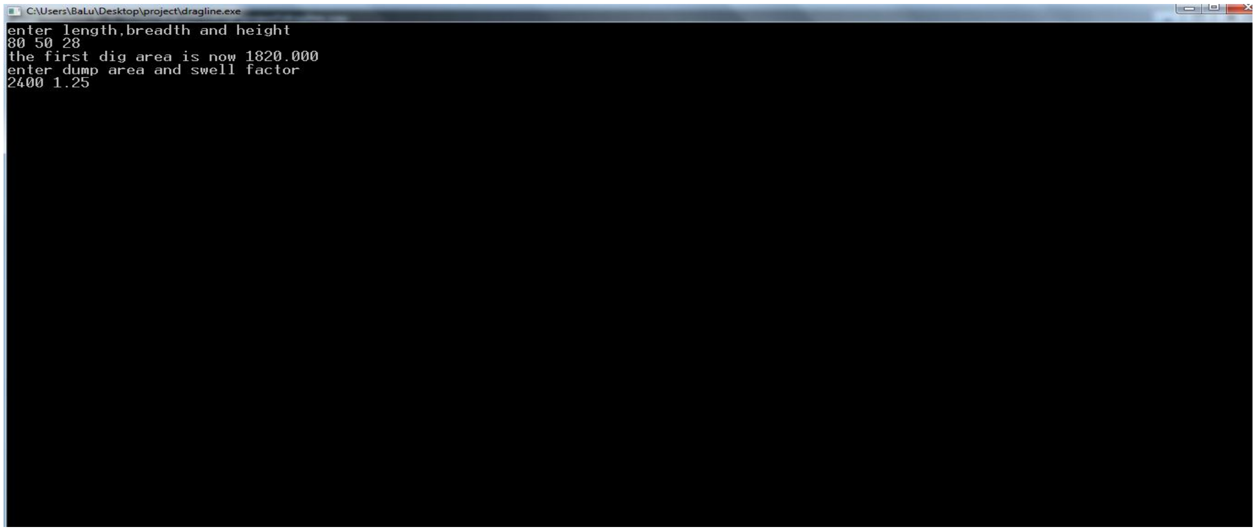



Fig 4.6: Output screenshot 4

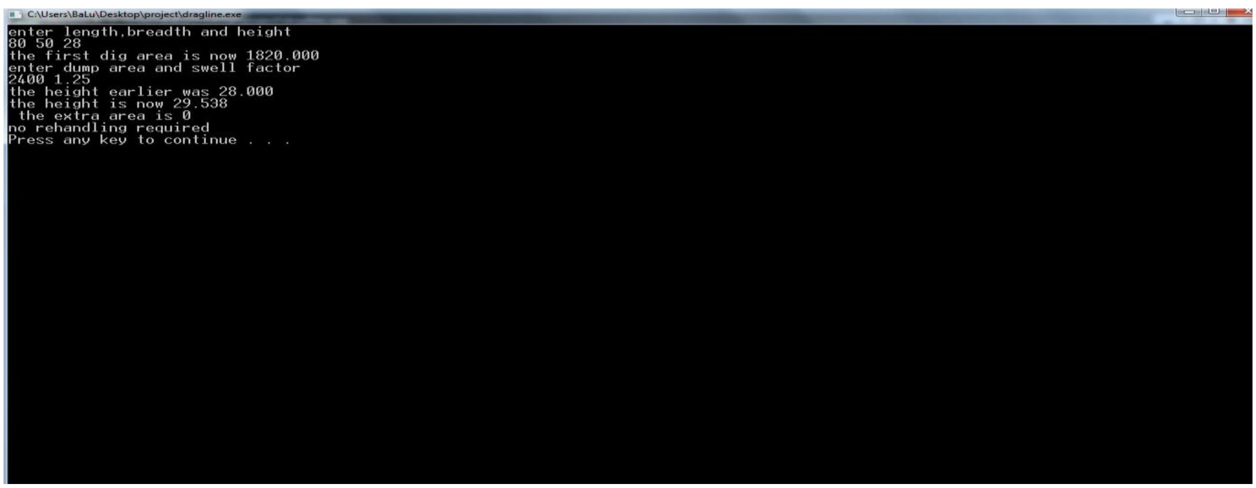



Fig 4.7: Output screenshot 5

### 4.5.2. Case 2 (A3=0)

Input entered:

| | |
|---|---|
| Top width of cut (length) | : 80 m |
| Bottom width of cut (breadth) | : 50 m |
| Height of bench | : 28m |
| Dump area | : 2275 $m^2$ |
| Swell factor | : 1.25 |

Output:

The new height is 28.00 m

No rehandling is required

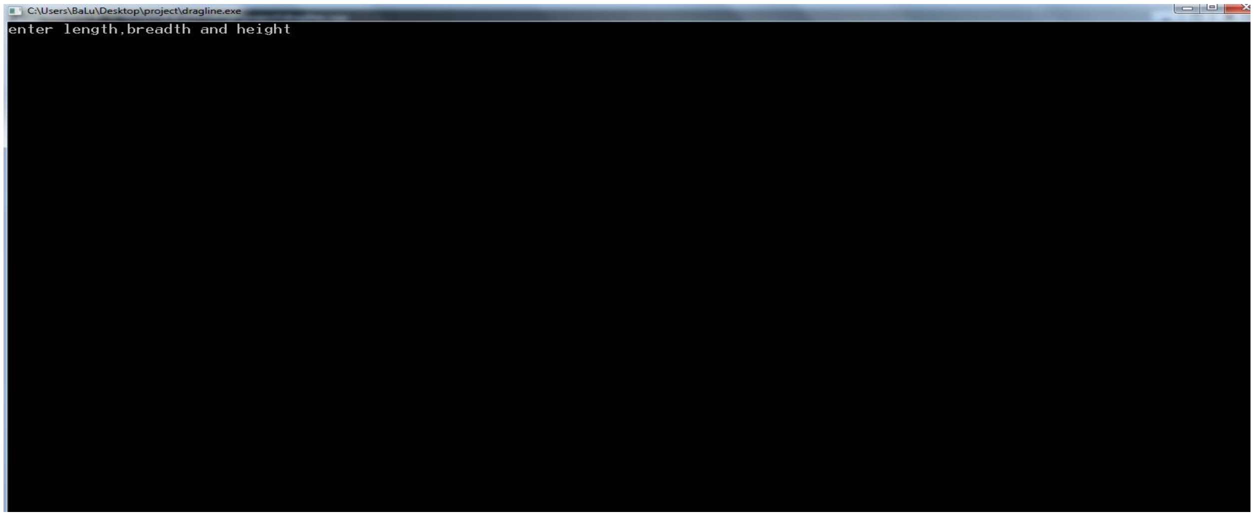



Fig 4.8: Output screenshot 6

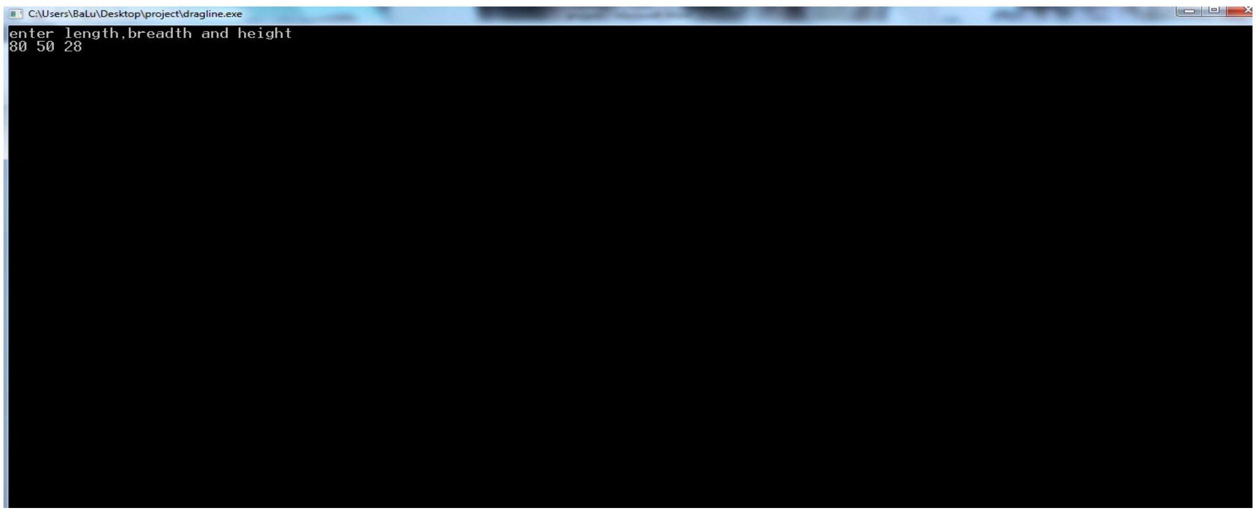



Fig 4.9: Output screenshot 7

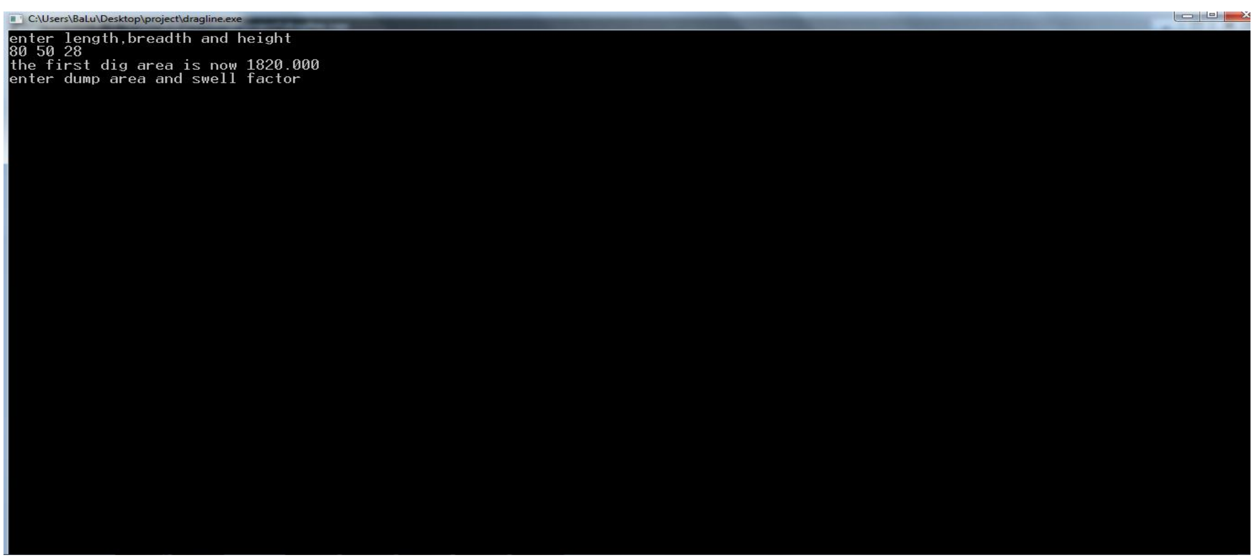



Fig 4.10: Output screenshot 8

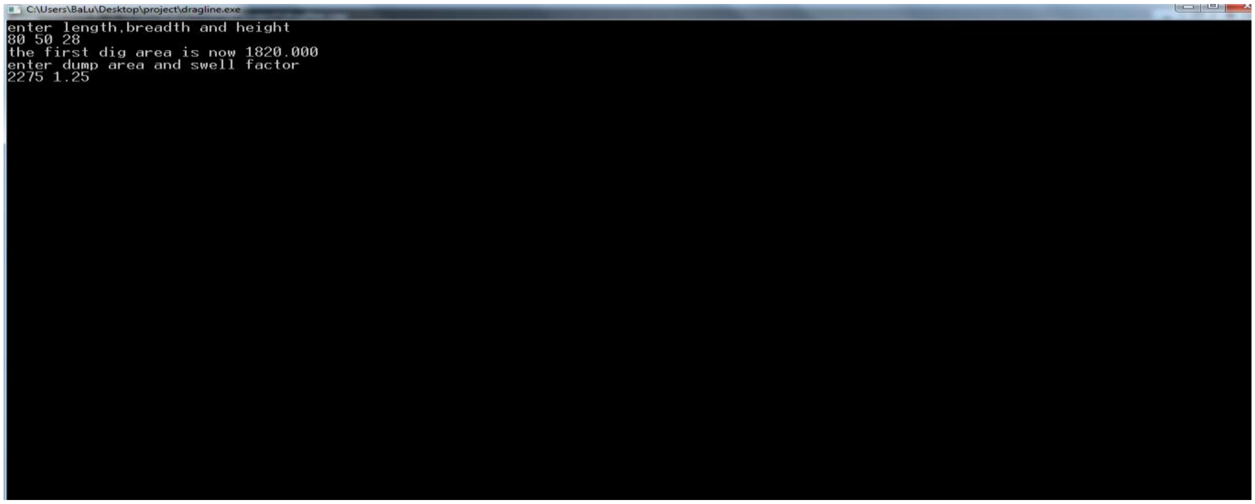



Fig 4.11: Output screenshot 9

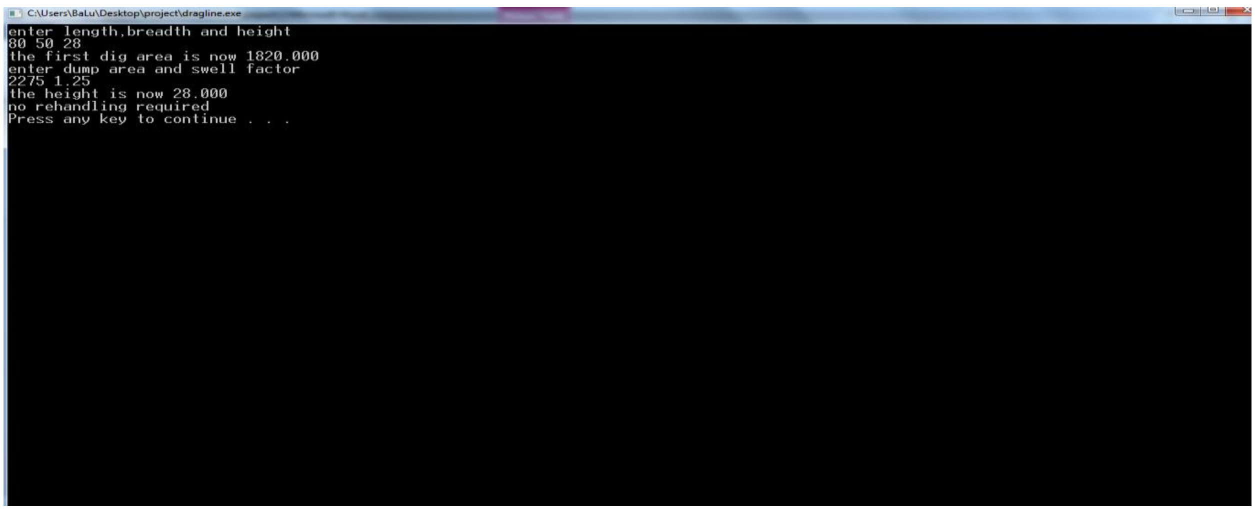



Fig 4.12: Output screenshot 10

### 4.5.3. Case3 (A3>0)

Input entered:

| | |
|---|---|
| Top width of cut (length) | : 80 m |
| Bottom width of cut (breadth) | : 50 m |
| Height of bench | : 28 m |
| Dump area | : 2000 $m^2$ |
| Swell factor | : 1.25 |

Output:
The height now is 28.00 m
The extra area is 275 $m^2$
Reach has to be increased

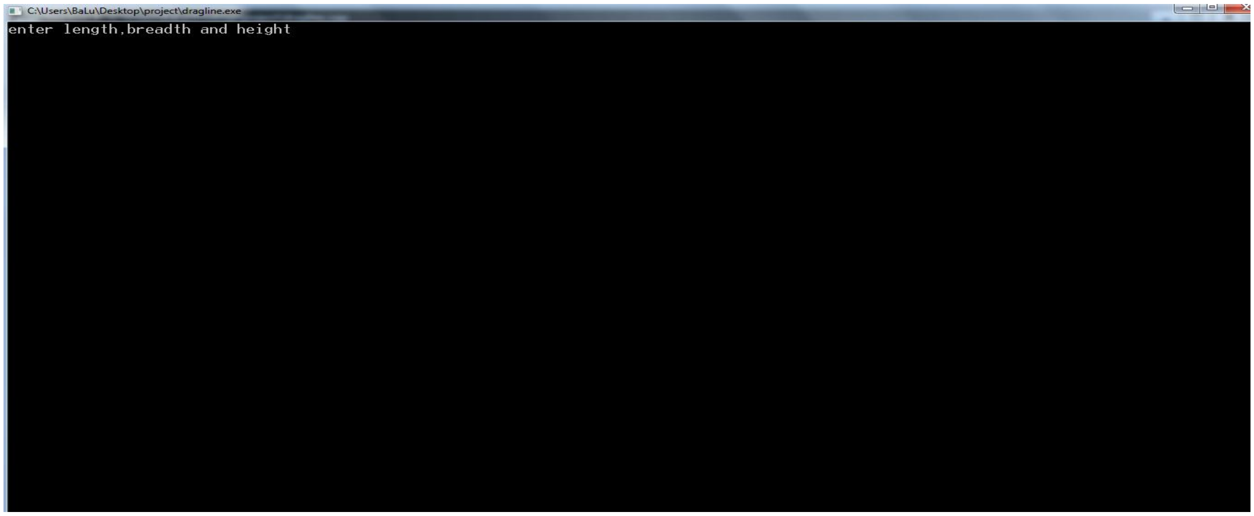



Fig 4.13: Output screenshot 11

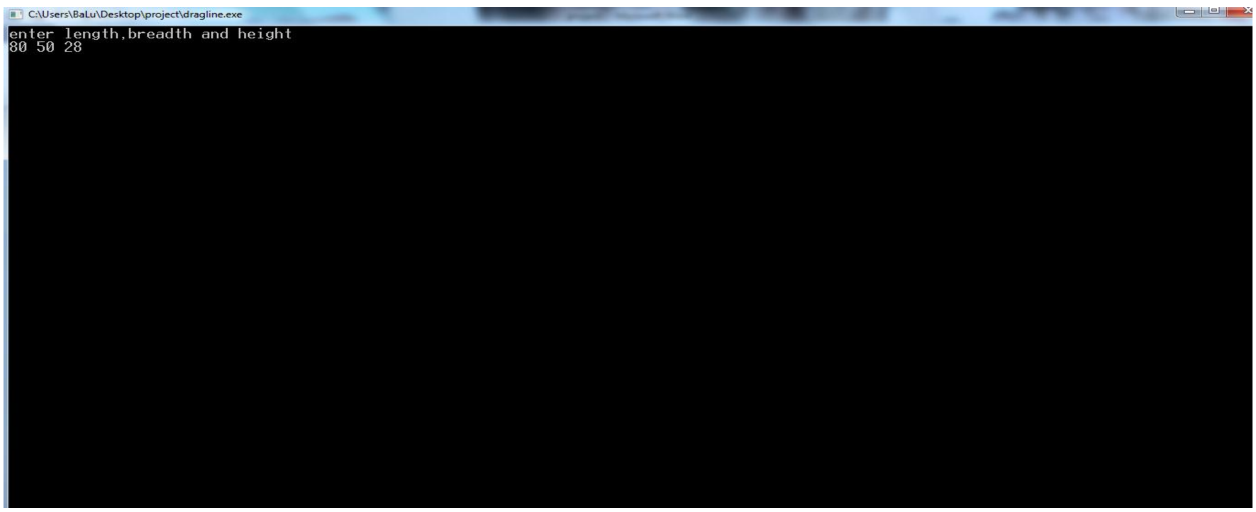



Fig 4.14: Output screenshot 12

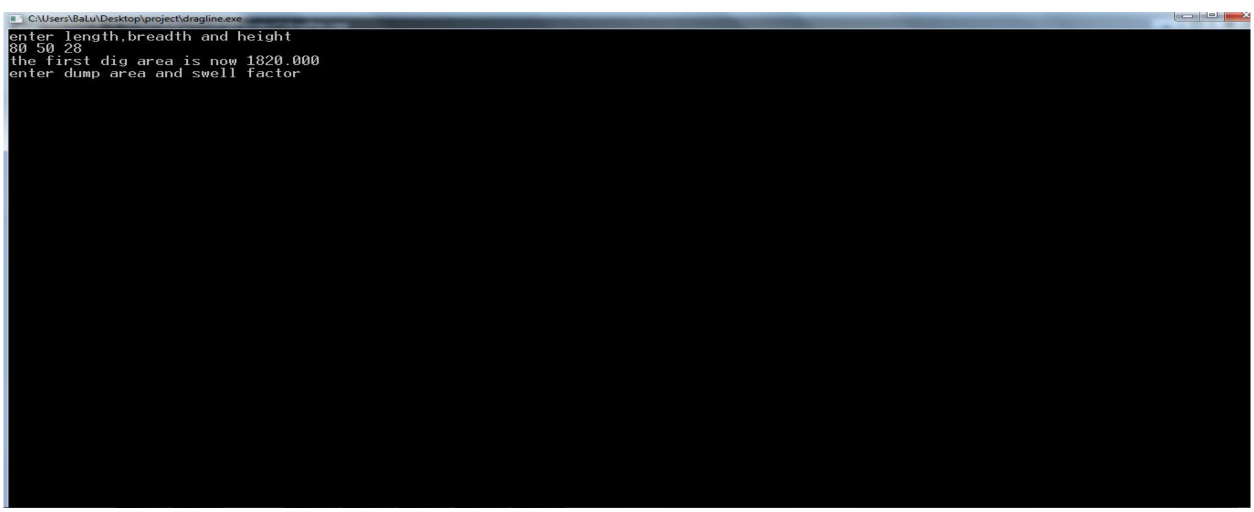



Fig 4.15: Output screenshot 13

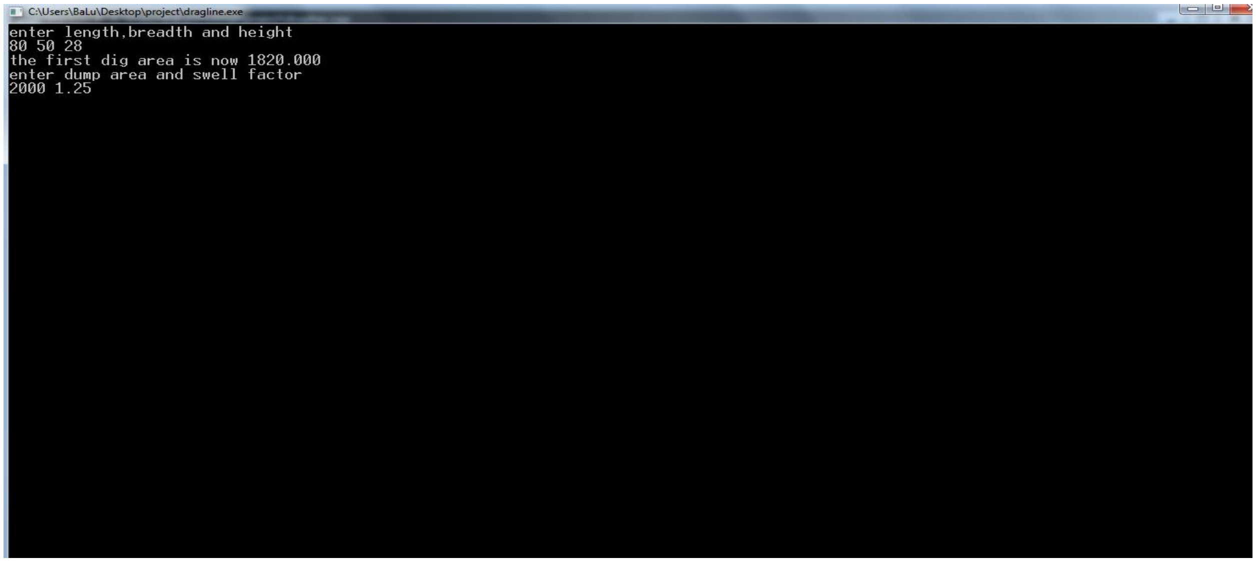



Fig 4.16: Output screenshot 14

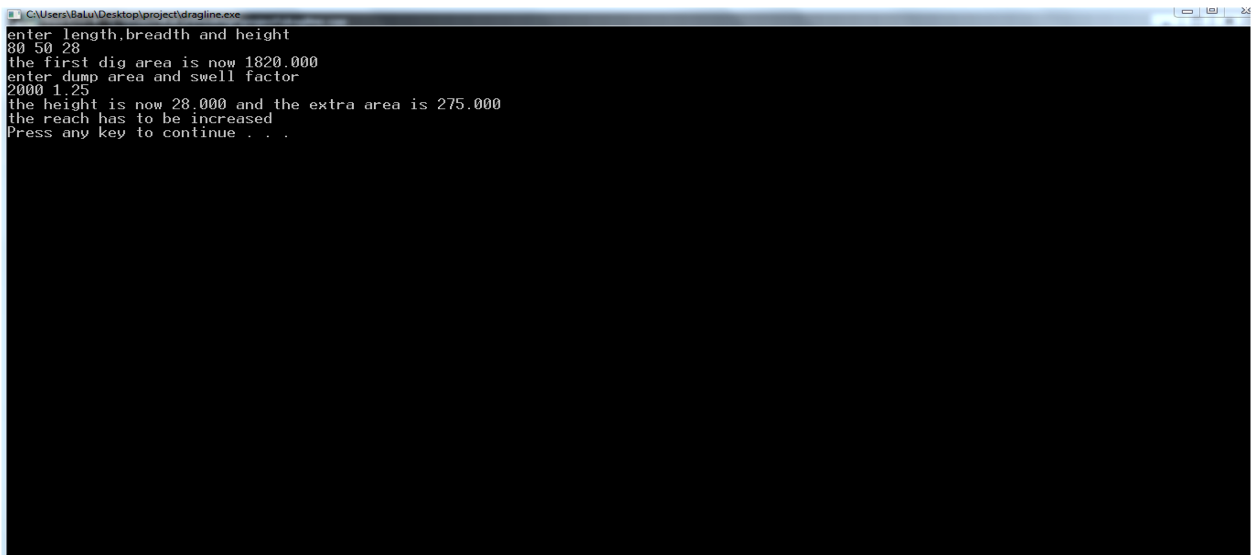



Fig 4.17: Output screenshot 15

## 4.6. Projection of annual output, calculation of ownership, operating cost and cost per tonne of coal exposed by dragline

### 4.6.1. Evaluation of Availability and Utilization

To evaluate the Availability (A) and Utilization (U) the field data acquired was substituted in the Eqs (i) and (ii).

$$A = \frac{SSH - (MH+BH)}{SSH} \qquad \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots \text{(i)}$$

$$U = \frac{SSH - (MH+BH+IH)}{SSH} \qquad \ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots\ldots \text{(ii)}$$

Where,

SSH is scheduled shift hours,

MH is maintenance hours,

BH is breakdown hours,

IH is idle hours

Based on the observed and recorded data in terms of average cycle time, A and U values the annual output (P1) of the dragline has been projected using Eq (iii) :

$$P1 = (B/C)*A*U*S*F*M*N_s*N_h*N_d*3600 \qquad \ldots\ldots\ldots\ldots\ldots \text{(iii)}$$

Where,

B is bucket capacity of the dragline in cubic meter,

C is the average total cycle time of dragline in second,

S is the swell factor,

F is the fill factor,

M is the machine travelling and positioning factor,

$N_s$ is the number of operating shifts in a day,

$N_h$ is the number of operating hours in a shift,

$N_d$ is the number of operating days in a year,

In the above equation the values of average cycle time (C), A and U were substituted as per the recorded and acquired field observations. Remaining factors in the Eq (iii) (S, F, M, $N_s$, $N_h$, $N_d$) were substituted as per the recommendations made by CMPDI in regard to the values of these factors in Indian coal mines. The suggested values for these factors are given in Table 4.1.

**Table 4.1 – Productivity factors for dragline as per CMPDI recommendations**

| Particulars | Recommended values |
|---|---|
| Swell factor (S) | 0.719 |
| Fill factor (F) | 0.733 |
| Machine travel and positioning factor (M) | 0.8 |
| No. of shifts in a day ($N_s$) | 3 |
| No. of hours in a day ($N_h$) | 8 |
| No. of days in a year ($N_d$) | 365 |

**Table 4.2 – Parameters of Singareni OCP – I Dragline**

| Sl No. | Parameters | Details |
|---|---|---|
| 1. | Dragline (bucket($m^3$ )/boom(m)) | 24/96 |
| 2. | Make | Rapier & Ransom (England) |
| 3. | Max operating radius (m) | 88 |
| 4. | Bench height (m) | 30-35 |
| 5. | Cutting width (m) | 60 |
| 6. | Highwall slope (degrees) | 70 |
| 7. | Bench slope (degrees) | 60 |
| 8. | Angle of repose (degrees) | 38 |
| 9. | Digging depth (m) | 25 |
| 10. | Reach of dragline (m) | 73 |
| 11. | Method of working | Extended Bench method |
| 12. | Thickness of coal seam (m) | 4.5 |

**Table 4.3 – Parameters of Samaleswari Dragline**

| Sl No. | Parameters | Details |
|---|---|---|
| 1. | Dragline (bucket($m^3$)/boom(m)) | 10/70 |
| 2. | Make | Russian |
| 3. | Max operating radius (m) | 58 |
| 4. | Bench height (m) | 35-40 |
| 5. | Cutting width (m) | 45 |
| 6. | Highwall slope (degrees) | 70 |
| 7. | Bench slope (degrees) | 60 |
| 8. | Angle of repose (degrees) | 38 |
| 9. | Digging depth (m) | 36 |
| 10. | Reach of dragline (m) | 58 |
| 11. | Method of working | Simple side casting |
| 12. | Thickness of coal seam (m) | 25 |

**4.6.2. The maximum depth that can be worked by a dragline is given by the formula:**

$$H = \frac{t + \tan x\,(R - W/4)}{S + \frac{\tan x}{\tan y}} \qquad \ldots\ldots\ldots\ldots\ldots\ldots\ldots \text{(iv)}$$

For OC – I Dragline (24/96) :

Maximum depth that can be worked by the dragline (H)

Thickness of coal seam (t) = 4.5 m

Angle of repose of overburden (x) = $38^0$

Reach of dragline (R) = 73 m

Swell factor (S) = 1.39

Width of cut (W) = 60 m

Slope angle of highwall to horizontal (y) = $70^0$

Using the above values in the equation (iv)

$$H = \frac{4.5 + \tan 38\,(73 - 60/4)}{1.39 + \frac{\tan 38}{\tan 70}}$$

After calculation we get H = 29.74 m

So, the maximum depth that can be worked = 29.74 m

For Samaleswari Dragline (10/70):

Maximum depth that can be worked by the dragline (H)

Thickness of coal seam (t) = 25 m

Angle of repose of overburden (x) = $38^0$

Reach of dragline (R) = 58 m

Swell factor (S) = 1.39

Width of cut (W) = 45 m

Slope angle of highwall to horizontal (y) = $70^0$

Using the above values in the equation (iv)

$$H = \frac{25 + \tan 38\,(58 - 45/4)}{1.39 + \frac{\tan 38}{\tan 70}}$$

After calculation we get H = 36.74 m

So, the maximum depth that can be worked = 36.74 m

### 4.6.3. Amount of rehandle ($P_{RM}$) (Extended bench method followed at Singareni OC-I)

$$P_{RM} = \frac{1.125\,t + 0.684\,H + 0.1\,R}{W} + \frac{0.25\,t^2 - 0.4\,Rt - 0.16\,R^2}{HW} + \frac{0.1\,t + 0.08\,R - 0.01W}{H} \quad . . (v)$$

Overburden dump height (H) = 40 m

Thickness of coal seam (t) = 4.5 m

Reach of dragline (R) = 73 m

Width of cut (W) = 60 m

$$P_{RM} = \frac{(1.125*4.5) + (0.684*40) + (0.1*73)}{60} + \frac{(0.25*4.5^2) - (0.4*73*4.5) - (0.16*73^2)}{40*60} +$$

$$\frac{(0.1*4.5) + (0.08*73) - (0.01*60)}{40}$$

$= 0.662 - 0.40 + 0.14$

$= 0.402$

So the amount of rehandle percentage = 40.2

### 4.6.4. Projection of annual Output of the dragline

**Table 4.4 - Breakup of operational hours**

| Mine | Equipment | Scheduled shift hours (SSH) | Working hours (WH) | Maintenance hours (MH) | Breakdown hours (BH) | Idle hours (IH) |
|---|---|---|---|---|---|---|
| Singareni OCP-I | 24/96 | 720 | 507 | 119 | 33 | 61 |
| Samaleswari | 10/70 | 720 | 540 | 90 | 30 | 60 |

**Table 4.5 - The average total cycle time results**

| Mine | Equipment | Standard cycle time (s) | Observed cycle time (s) |
|---|---|---|---|
| Singareni OCP-I | 24/96 | 60 | 61.7 |
| Samaleswari | 10/70 | 60 | 66.3 |

**For Singareni OC-I Dragline (24/96)**

Using the data form Table 2. In the Eqs (i) and (ii)

$$A = \frac{720 - (119+33)}{720}$$

$= 0.7888$

$$U = \frac{720 - (119+33+61)}{720}$$

$= 0.7041$

So, we get Availability = 0.7888

Utilization = 0.7041

**For Samaleswari Dragline (10/70)**

Using the data form Table 2. In the Eqs (i) and (ii)

$$A = \frac{720 - (90+30)}{720}$$

$= 0.8333$

$$U = \frac{720 - (90+30+60)}{720}$$

$= 0.75$

So, we get Availability = 0.8333

Utilization = 0.75

Availability cum utilization factor (k) = A*U = 0.625

**Table 4.6 - Availability and utilization factors**

| Mine | Equipment | Availability factor | Utilization factor |
|---|---|---|---|
| Singareni OCP-I | 24/96 | 0.7888 | 0.7041 |
| Samaleswari | 10/70 | 0.8333 | 0.75 |

By using the recorded, acquired data and recommended values in the Eq (iii):

**For Singareni OC-I Dragline**

P1= (24/61.7)*0.555*0.719*0.733*0.8*8*3*365*60*60

= 2.807 M cu.m.

So, the projected annual output of the Singareni OCP-I dragline is 2.807 M cu.m.

**For Samaleswari Dragline**

P1= (10/66.3)*0.625*0.719*0.733*0.8*8*3*365*60*60

= 1.253 M cu.m.

So, the projected annual output of the Samaleswari dragline is 1.253 M cu.m.

### 4.6.5. Calculation of Ownership and Operating cost of dragline (Singareni OC-I dragline)

A. Cost of ownership per year of the 24/96 dragline

(i) Cost of equipment

Cost of the 24/96 dragline = Rs. 1000 million

(ii) Depreciation cost for 25 year i.e. annual flat rate of 4%

Annual depreciation cost of 24/96 dragline = Rs. 40 million

(iii) Annual cost of ownership (24/96)

$$\text{Average annual investment} = \frac{N+1}{2N} * \text{cost of dragline}$$

$$\text{Where N = Life of dragline = Rs. } \frac{1000*26}{2*25} \text{ million = Rs. 520 million}$$

(iv) Annual intrest, insurance rates and taxes i.e. annual flat rate of 12.5%

= 15 % of Rs. 520 million

= Rs. 78 million

Hence the total ownership cost per year = (ii) +(iv)

= Rs. (40+78) million

= Rs. 118 million

B. Operating cost per year of the 24/96 dragline

(i) Annual manpower cost (salary and wages)

Operator cost @ Rs. 0.20 millions/operator for 2 operators in 3 shifts = Rs. 1.20 million

Helper cost @ Rs. 0.14 million for 1 operator in 3 shifts = Rs. 0.42 million

Total manpower cost = Rs. 1.62 million

(ii) Annual power and energy consumption on the basis of 13.65 MKWH for 24/96

Annual power consumption cost @ Rs. 4.89/KWH = Rs. $4.89*13.65*10^6$

= Rs. 66.75 million

(iii) Annual lubrication cost @ 30% of power consumption = Rs. 20.025 million

(iv) Annual maintenance cost @ 20% of depreciation cost = Rs. 8 million

Major breakdown cost @ 2% of cost of equipment = Rs. 20 million

Total maintenance cost = Rs. 28 million

Hence, Total Annual operating cost = Manpower cost/year + Electrical cost/year + Maintenance cost/year + Lubrication cost/year

= Rs. (1.62 + 66.75 + 20.025 +28) million

= Rs. 116.4 million

Total ownership and operating cost

= ownership cost/year + operating cost/year

= Rs. (118+116.4) million

= Rs. 234.4 million

Dragline operating cost per $m^3$ overburden handle considering annual output of 24/96 as 2.807 M cu.m = Rs. $\frac{234.4*10^6}{2.87*10^6}$ = Rs. 81.67

### 4.6.6. Calculation of cost per ton of coal exposed by Singareni OCP – I Dragline by extended bench method

Dragline deployed is 24/96 having a production capacity of 2.87 M cu.m/year

Percentage rehandling is 40.2%

Total overburden handled = overburden directly over the exposed coal + overburden rehandled

= overburden directly over the exposed coal (1+coefficient of rehandling)

Here, coefficient of rehandling = O.B rehandle/O.B removal to expose coal

Therefore, 2.87 M cu.m = overburden directly over the exposed coal *1.40

Hence the overburden directly over the exposed coal removed by the dragline = $\frac{2.87}{1.40}$ M cu.m

= 2.05 M cu.m

Amount of coal exposure = $\frac{2.05 \text{ M cu.m}}{4.2 \text{ m}^3/\text{te}}$

= 0.82 Mte

Estimated cost/tonne of coal exposed = Rs. $\frac{234.34*10^6}{0.82*10^6}$

= Rs. 285.78

= Rs. 285.78 per te of coal exposed

### 4.6.7. Calculation of Ownership and Operating cost of dragline (Samaleswari dragline)

A. Cost of ownership per year of the 10/70 dragline

(i) Cost of equipment

Cost of the 24/96 dragline = Rs. 300 million

(ii) Depreciation cost for 25 year i.e. annual flat rate of 4%

Annual depreciation cost of 10/70 dragline = Rs. 12 million

(iii)Annual cost of ownership (10/70)

Average annual investment = $\frac{N+1}{2N}$ * cost of dragline

Where N = Life of dragline = Rs. $\frac{300*26}{2*25}$ million = Rs. 156 million

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

= 15 % of Rs. 156 million

= Rs. 23.4 million

Hence the total ownership cost per year = (ii) +(iv)

= Rs. (12+23.4) million

= Rs. 35.4 million

B. Operating cost per year of the 10/70 dragline

(i) Annual manpower cost (salary and wages)

Operator cost @ Rs. 0.20 millions/operator for 2 operators in 3 shifts = Rs. 1.20 million

Helper cost @ Rs. 0.14 million for 1 operator in 3 shifts = Rs. 0.42 million

Total manpower cost = Rs. 1.62 million

(ii) Annual power and energy consumption on the basis of 9.07 MKWH for 24/96

Annual power consumption cost @ Rs. 4.89/KWH = Rs. $4.89*9.07*10^6$

= Rs. 44.35 million

(iii) Annual lubrication cost @ 30% of power consumption = Rs. 13.3 million

(iv) Annual maintenance cost @ 20% of depreciation cost = Rs. 2.4 million

Major breakdown cost @ 2% of cost of equipment = Rs. 6 million

Total maintenance cost = Rs. 8.4 million

Hence, Total Annual operating cost = Manpower cost/year + Electrical cost/year + Maintenance cost/year + Lubrication cost/year

= Rs. (1.62 + 44.35 + 13.3 +8.4) million

= Rs. 67.67 million

Total ownership and operating cost

= ownership cost/year + operating cost/year

$= \text{Rs. } (35.4+67.67) \text{ million}$

$= \text{Rs. } 103.07 \text{ million}$

Dragline operating cost per $m^3$ overburden handle considering annual output of 10/70 as 1.253 M cu.m = $\text{Rs. } \frac{103.07*10^6}{1.253*10^6} = \text{Rs. } 82.2$

### 4.6.8. Calculation of cost per ton of coal exposed by Samaleswari Dragline by simple side casting method

Dragline deployed is 10/70 having a production capacity of 1.253 M cu.m/year

Amount of overburden handled = 1.253 M cu.m

$$\text{Amount of coal exposure} = \frac{\text{Annual production of dragline}}{\text{Average stripping ratio}}$$

$$= \frac{1.253 \text{ M cu.m}}{3 \text{ cu.m/te}}$$

$$= 0.417 \text{ M te}$$

$$\text{Estimated cost/tonne of coal exposed} = \text{Rs. } \frac{103.07}{0.417} \text{ /te}$$

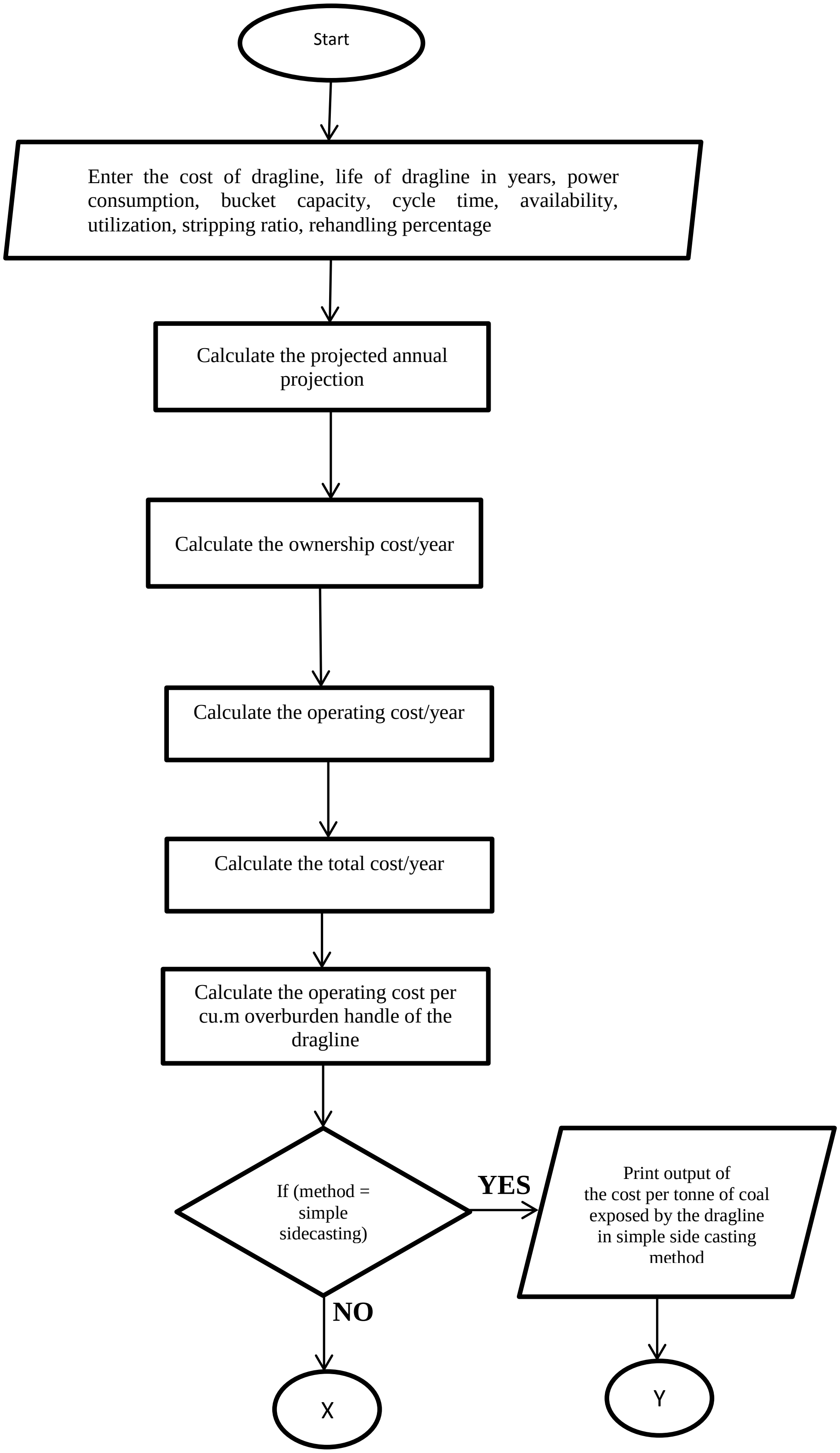
$$= \text{Rs. } 247.17 \text{ /te of coal exposed}$$

**4.6.9. Developing a computer based program (C++) for projection of annual production of overburden, calculation of ownership , operating cost operating and cost per cu.m overburden handle of the dragline**

```
#include<stdio.h>
#include<stdlib.h>
#include<conio.h>
float c,d,n,aai,ai,oc,mc,ap,al,am,p,op,ta,pa,ce,ec,tce,tec,s,r,to,u,ob,a,ut,b,ct;
char t;
void annualproduction()
{
pa=(b/ct)*a*ut*0.719*0.733*0.8*8*3*365*60*60;
printf("the annual production of the dragline is %.6f\n",pa);
}
void ownershipcost()
{
    d=0.04*c;
    aai=((n+1)/(2*n))*c;
    ai=(15.00/100)*aai;
    oc=d+ai;
    printf("the ownership cost of the dragline is %.6f\n",oc);
}
void operatingcost()
{
  mc=(0.20*6.00)+(0.14*3.00);
  ap=4.89*p*1000000;
  al=0.30*ap;
  am=(0.20*d)+(0.02*c);
  op=mc+ap+al+am;
  printf("the operating cost of the dragline is %.6f\n",op);
}
void totalcost()
```

```
{
   ta=oc+op;
   printf("the total cost of the dragline is %.6f\n",ta);

}
void overburdencost()
{
   ob=(ta/pa);
    printf("the operating cost per cu.m overburden handle of the dragline is %f\n",ob);
}
void coalexposure1()
{
   ce=pa/s;
   ec=ta/ce;
   printf("the cost per tonne of coal exposed by the dragline is %.3f\n",ec);
}
void coalexposure2()
{
 to=(pa)/(1.00+r);
 tce=(to/s);
 tec=(ta/tce);
 printf("the cost per tonne of coal exposed by the dragline is %.3f\n",tec);
}
int main()
{

    printf("enter cost of dragline,no. of years,power consumption,bucket capacity,cycle time,availability,utilization,stripping ratio,rehandling in decimals\n");
    scanf("%f%f%f%f%f%f%f%f",&c,&n,&p,&b,&ct,&a,&ut,&s,&r);

/*
```

```
    c=1000000000;
    n=25;
    p=13.65;
    b=24;
    ct=61.7;
    a=0.833;
    ut=0.75;
    s=2.5;
*/

    fflush(stdin);
    printf("Enter type of method(simple sidecasting(s) or extended benchmethod(e))::");
    scanf("%c",&t);
    annualproduction();

    //pa=(b/ct)*a*ut*0.719*0.733*0.8*8*3*365*60*60;
    //printf("the annual production of the dragline is %.6f\n",pa);

    ownershipcost();
    operatingcost();
    totalcost();
    overburdencost();
    if(t=='s')
      coalexposure1();
    if(t=='e')
      coalexposure2();
    system("pause");
    return 0;
}
```


Start
Enter the cost of dragline, life of dragline in years, power consumption, bucket capacity, cycle time, availability, utilization, stripping ratio, rehandling percentage
Calculate the projected annual projection
Calculate the ownership cost/year
Calculate the operating cost/year
Calculate the total cost/year
Calculate the operating cost per cu.m overburden handle of the dragline
If (method = simple sidecasting)
YES
Print output of the cost per tonne of coal exposed by the dragline in simple side casting method
NO
X
Y

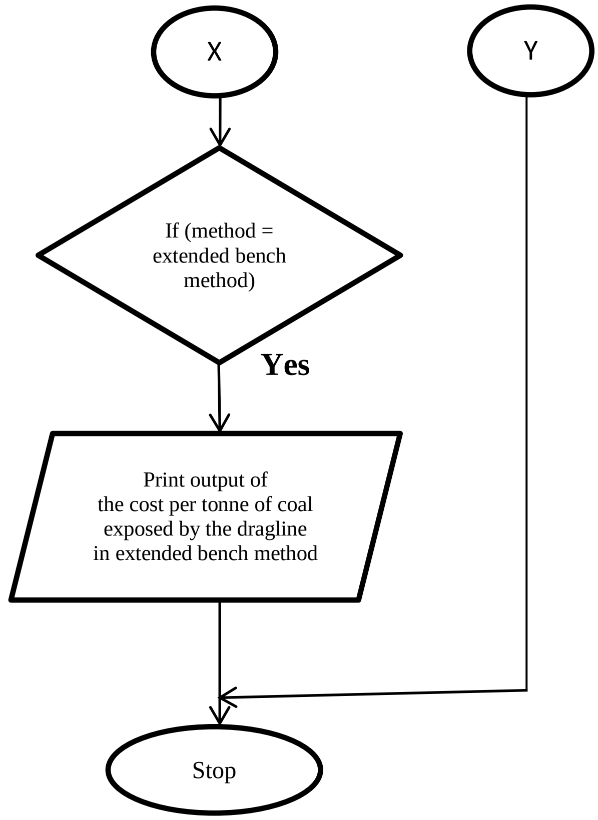



Fig 4.18: Flowchart for the on dragline bench designing (keeping the cut-width constant) and determine whether or not rehandling is required

**Output: 1 For extended bench method;**

Input entered:

| | |
|---|---|
| Cost of dragline | : Rs. 1000 million |
| No. of years | : 25 |
| Power consumption | : 13.65 KWh |
| Bucket capacity | : 24 cu.m |
| Cycle time | : 61.7 s |
| Availability | : 0.7888 |
| Utilization | : 0.7041 |
| Stripping ratio | : 2.5 |
| Percentage rehandling | : 0.40 |

To enter the type of method (simple sidecasting (s) or extended bench method (e)) : e

Output 1:
The annual production of dragline = 2.87 M cu.m
The ownership cost of dragline/year = Rs. 118 million
The operating cost of dragline/year = Rs. 114.7 million
The total costs of dragline/year = Rs. 232.7 million
The operating cost per cu.m overburden handle = Rs. 81.03
The cost per tonne of coal exposed by the dragline = Rs. 284.76

**Output screen 1:**

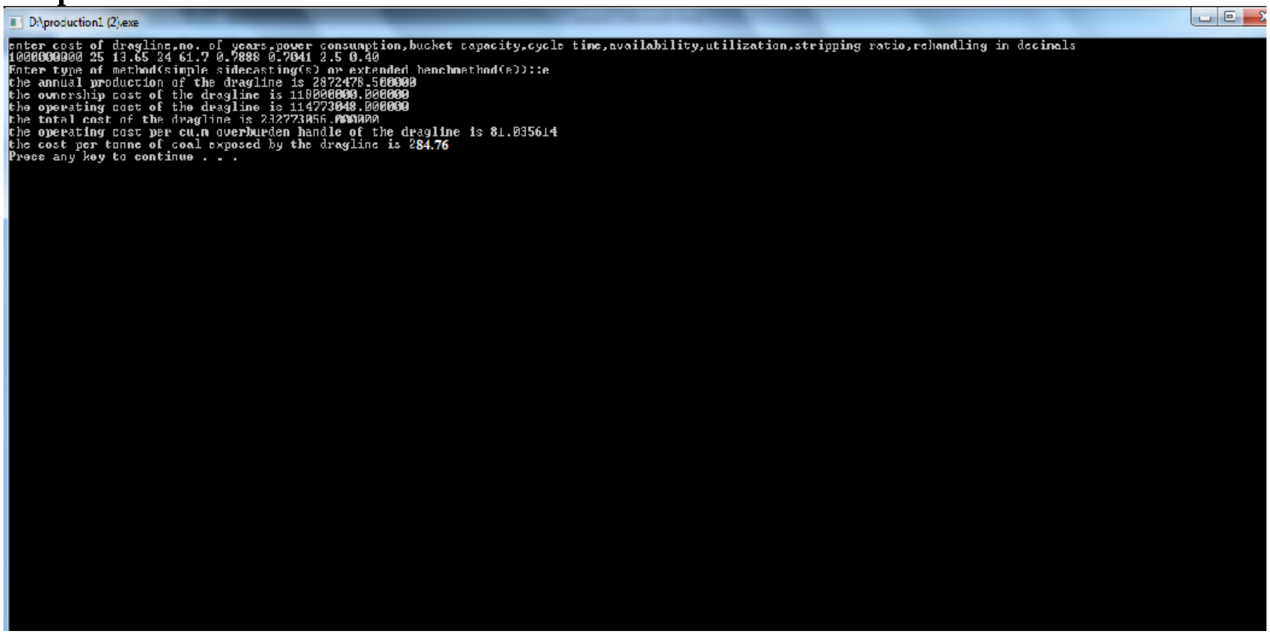



Fig 4.19: Output screen 16

**Output: 1 For simple sidecasting method;**

Input entered:

| | |
|---|---|
| Cost of dragline | : Rs. 300 million |
| No. of years | : 25 |
| Power consumption | : 9.07 KWh |
| Bucket capacity | : 10 cu.m |
| Cycle time | : 66.3 s |
| Availability | : 0.8333 |
| Utilization | : 0.75 |
| Stripping ratio | : 3 |
| Percentage rehandling | : 0 |

To enter the type of method (simple sidecasting (s) or extended bench method (e)) : s

**Output 2:**

The annual production of dragline = 1.24 M cu.m
The ownership cost of dragline/year = Rs. 35.4 million
The operating cost of dragline/year = Rs. 66.05 million
The total costs of dragline/year = Rs. 101.45 million
The operating cost per cu.m overburden handle = Rs. 81.46
The cost per tonne of coal exposed by the dragline = Rs. 244.4

**Output screen 2:**

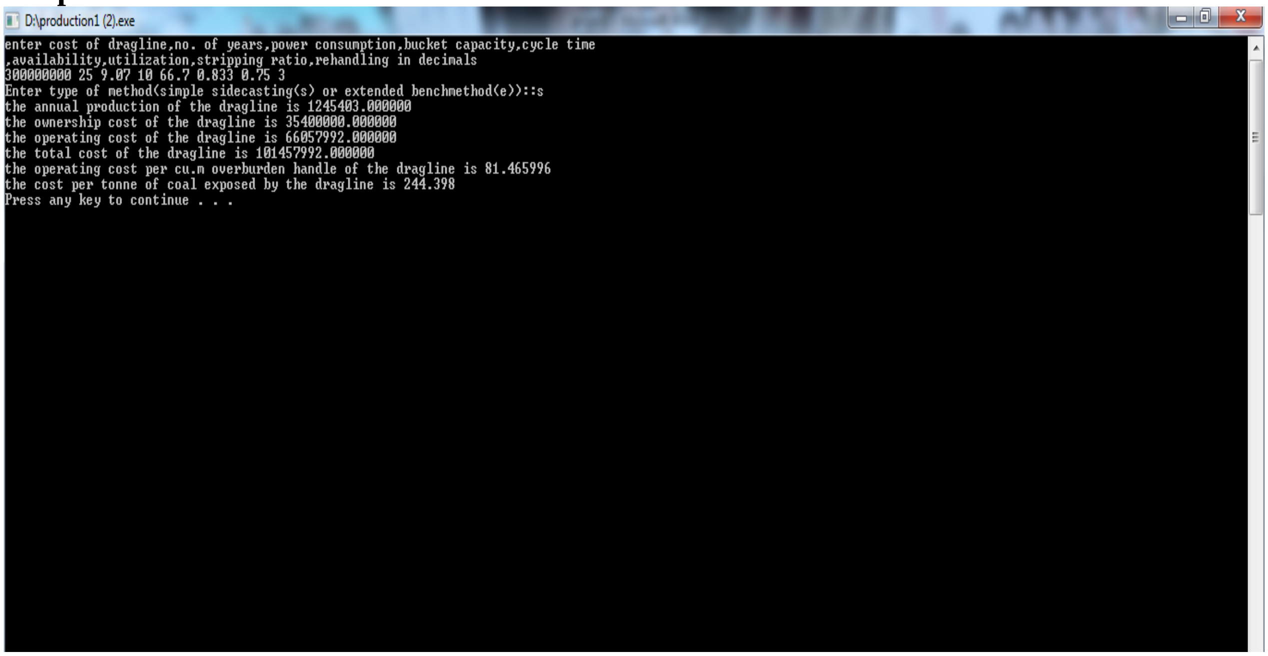



Fig 4.20: Output screen 17

# Chapter 05

## RESULTS

- The projected annual output of the Singareni OC-I dragline is 2.807 M cu.m.
- The projected annual output of the Samaleswari dragline is 1.253 M cu.m.
- Dragline operating cost per $m^3$ overburden handle considering annual output of Singareni OC-I dragline (24/96) as 2.807 M cu.m is Rs. 81.67
- Dragline operating cost per $m^3$ overburden handle considering annual output of Samaleswari dragline (10/70) as 1.253 M cu.m is Rs. 82.2
- Estimated cost/tonne of coal exposed by the Singareni OC-I dragline(24/96) is Rs. 285.78
- Estimated cost/tonne of coal exposed by the Samaleswari dragline(10/70) is Rs. 247.17

# Chapter 06

## CONCLUSION

Factors affecting the production and cost of coal exposed by dragline are:-

- Increased no. of idle hours due to non-availability of blasted muck pile, operator availability, ability and performance
- Increased breakdown time
- Increased breakdown and maintenance costs

Scope for improvement

- Increasing the Dragline Productivity through Maximizing Cast by using blasting techniques like i-kon system of blasting.
- Employing a spare dragline operator and reducing the Variability in dragline operator performance.
- Better preventive maintenance schedule to reduce the breakdown time and breakdown costs.

# Chapter 07

## REFERENCES


i. Erdem B, Paşamehmetoğlu A.G, “Dragline Cycle Time Analysis - Case Study”, lä" International Mining Congress and Exhibition of Turkey-IMCET, 2003, ISBN 975-395-605-3.

ii. Rai P, Yadav U, Kumar A, “Productivity Analysis of Draglines Operating in Horizontal and Vertical Tandem Mode of Operation in a Coal Mine—A Case Study”, Geotech Geol Eng, 2011, 29:493-504.

iii. Kishore N, Pankaj D, “Estimation of operating cost of dragline in various modes of operation”, The Indian Mining and Engineering journal, 2010.

iv. Rai P, “Performance assessment of draglines opencast mine”, Indian journal of engineering and material sciences, 11: 493-498.

v. Kishore N, Dewangan P, Dhekne P.Y, “Method of drawing balancing diagram”, The Indian Mining and Engineering journal, 2012, pp. 18-21.

vi. Satyamurthy M, “Dragline deployment in opencast mines”, Minetech Volume10 No.1, pp 13-15.

vii. Seymour C.A, “Dragline stripping extended bench method”.

viii. Mishra G.B. Surface Mining, Geominetech Publications, 2007, PP. 439-452.

ix. Reddy V R, & Dhar B B, “Dragline performance in openpit Indian coal mines”, Procs Int Conf on Mine Planning and Equipment Selection, 1988, Balkema Pub, Rotterdam, The Netherlands.

x. Rai V K, “Balancing diagram—a basic planning tool for dragline operation”, In: Proceedings of the national seminar on emerging technology in surface mines and environmental challenges, 1997, Mangalore, India, pp 85–89.

xi. Internet: Ministry of Mines, 19-Nov,2004, Dragline in opencast mines, http://pib.nic.in/newsite/erelease.aspx?relid=4960.

xii. Hartman H L, SME Mining Engineering Handbook, Colorado: Society for mining, metallurgy and exploration, Inc., 1992, 2nd Edition, Volume 1.

xiii. Das S K, Surface Mining Technology, Dhanbad: Lovely Prakashan, 1994, 1st Edition.