EVALUATION OF DRAGLINE MINING IN INDIAN COAL MINES

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

BACHELOR IN TECHNOLOGY

IN

MINING ENGINEERING

BY

SOUMYAKANT NAG

109MN0500

DEPARTMENT OF MINING ENGINEERING

NATIONAL INSTITUTE OF TECHNOLGY

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CERTIFICATE

This is to certify that the thesis entitled **"EVALUATION OF DRAGLINE MINING IN INDIAN COAL MINES"** submitted by **Soumyakant Nag** 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: 13.05.13

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Date : 13.05.13 **SOUMYAKANT NAG**

CONTENTS

ABSTRACT

Draglines have been explicitly used in coal mining for a no. of decades, either as stripper or stripper and coal extractor. As this equipment retains certain intrinsic benefits, which their rivals don’t have, they should essentially be operated in round-the-clock fashion for high productivity and low expenditures. In India, the advancement of giant surface mining projects like Bina and Jayant with setting up of high coal production targets (upto 10 MT/year) demands for systems to remove huge volume of overburden in minimum possible time. This has led to major changes in overburden/interburden excavation technology in surface coal mining from shovel mining to dragline mining. Coal India Limited (CIL), now has standardized the draglines in two sizes, namely 10/70 and 24/96 for their mines. Most mines depend on the dragline 24 hours/day, 7 days /week. In many of the coal mines, it’s the only primary stripping tool and the mine's yield is totally dependent on the dragline’s performance. For these reasons only dragline design requires a greater emphasis placed on developing component’s with higher levels of reliability and predictability so that repairs and replacement of components can be programmed at times that will minimum affect the overall mining operation. Prior to installing draglines in mines, various aspects have to be taken into consideration for selecting suitable size. To determine the production and productivity of draglines different parameters are used. These points have been discussed in detail in this thesis.

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Chapter 01

INTRODUCTION

1.1. Dragline mining

Demand on energy is unremittingly increasing. Coal, the most homogeneously spread raw material all over the earth's crust, is amongst the most necessitated fossil fuels. A substantial portion of coal is extracted by surface mining methods. Concerning the economics of scale extraction, methods are mechanized highly and equipment with gigantic capacity are utilized. Draglines have been profusely used in coal mining for a no. decades, either as stripper or stripper and coal extractor. As this equipment holds certain inherent advantages, which their competitors do not, they must be operated in round-the-clock fashion for high productivity and low expenditures. In spite of its colossal posture, dragline can be said to have a simple routine of work composed of the following basic events: digging and walking. Among them walking is a firm process on which the mine design team has slight control. Nearly all walking draglines take a step of roughly 1.8 m within a time period of 0.75-1 min. The design of strip panels, equipping an explicit unit with one operator's room on the anticipated side or with two on both sides and the management's stratagem in coal loading operation mainly affect the frequency and the length of long deadheading periods, during which the unit is unproductive (Erdem et al., 2003).

The dragline, since its initiation many decades ago, is now extensively used to economically extract deposits at greater and deeper depths. Enormously large draglines are available now with longer booms and huge bucket capacity, these are the 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/cubic meter and succeeding low cost/ton of the desired mineral. High initial investment outlay for dragline makes effective and competent operation imperative to get low costs of overburden striping .Constant supervision, decent overburden preparation, and preventive care of dragline and selecting appropriate bench height of overburden for befittingly selected machine should safeguard efficient and effective operation of a dragline. Up gradation in dragline productivity can have an intense effect on overall mining operation. Myriad field examples have led to trust that with correct application analyses, constant engineering and production supervision, a dragline can foster an efficient explanation to deep strip mining. To say the lees, its wide spread application in current mining industry is in sensible more so than ever.

1.2. Draglines in Indian coal mines

Indian mineral industry has contributed expressively to make the nation self-reliant in coal. To meet the demands of cement, thermal and other users, the production drifts in coal and lignite sectors have revealed a remarkable growing trend over last few years. While extracting the deep-seated coal deposit and also to raise the present production capacity, the coal mines have been duty-bound to modernize the mining technology, chiefly in the fields of blasting.

In India, cartridge explosives subjugated the surface coal mines till the bulk explosives in the form of slurry and emulsion arrived into the explosives market. Subsequently the volume of overburden removal is growing day by day, the majority of coal sectors have now been switched over to blasting with bulk explosives. As far as the category of explosive is concerned, the coalmines are presently using the slurry or emulsion based explosives with the gradual exit of NG based from market.

Coal producers have already tried to excavate big surface coalmines in numerous coalfields. This has further demanded the importance of implementing better mining technology in the above mines by applying economic and scientific approaches whilst selecting the mining equipment and presenting the state-of-the-art technology. In this process it is imperative to adopt the blasting technology appropriate for the mine as it affects the successive operations involved in the mining.

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

Since the gigantic projects are coming up supplementary in the coal sector in recent times, the shovel mining faces big contests in fulfilling the production demand. Therefore, the Indian surface coal mining has been converting from shovel mining to dragline mining in most of large sized coalmines for removal of overburden/interburden to accommodate high rate of overburden removal and afterwards, high production rate with low expenditure of production.

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

The only subsidiary company of CIL, where the entire coal production is mined by opencast mining method is Northern Coalfields Limited (NCL). Another exclusive feature of the company is that about 40% of the huge volume of excavation is carried out with the help of larger walking draglines. Draglines are used in all the mines of NCL excluding Jhingurda, Kakri and Gorbi.

1.3. Aim

The aim of the project is to evaluate dragline mining in India; to perfectly identify sort comings, wrong procedures and suggest solutions for them. And then to measure and calculate the projected outputs, calculation of ownership, operating cost and cost/tons of coal exposed by dragline by the combination of several parameters collected during field study and assimilated from other sources. The secondary intent is to develop a computer based program (C++ language) bench designing of dragline(keeping the cut-width constant) and to determine whether or not rehandling is required; and to estimate yearly production of overburden, calculation of ownership , operating cost and cost per cu.m overburden handle of the dragline.

1.4. Objectives of study

1. Literature review on :
   - Draglines and mining with them
   - The systems and method of working of dragline
   - Draglines that are used in India

2. Dragline balancing diagram and developing a computer based program (C++ language) on bench design for dragline (keeping the cut-width constant) and determine whether or not to rehandle.
3. Estimation of yearly output, calculation of ownership, operating cost and cost/ton of coal exposed by dragline.
4. Develop a computer based program (C++ language) on projection of yearly production of overburden, calculation of ownership, operating cost and cost per cu.m overburden handle of the dragline.

1.5. Methodology used

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

1. Critical review of existing literature.
2. Visits to dragline mines for recording and collecting various parameters required for the study.
3. Systematic calculation and computer programming to attain the goals and objectives.

Chapter 02

LITERATURE REVIEW

2.1. History and present

In 1904, the dragline was invented by John W. Page of Page Schnabel Contracting for using it to digging the Chicago Canal. It became the Page Engineering Company in 1912, and a walking mechanism was developed few years later, providing the draglines with mobility. Page also invented arched dragline bucket; a design still usually used today by draglines from several other manufacturers, and in the 1960s established an arch less bucket design.

In 1910, Bucyrus International came in the dragline market with the procurement of manufacturing rights for the Heyworth-Newman dragline excavator. In 1911, their "Class 14" dragline got introduced as the first crawler mounted dragline. In 1912, Bucyrus helped pioneer the usage of electricity as a power source for huge stripping shovels and draglines used in mining.

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

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

Nowadays, draglines are widely used in strip mining of coal throughout the world. Nevertheless, it has found wide range use in non-coal sector also, which take account of surface mining of bauxite, phosphor, tax sands and oil shale. In the USSR, draglines are installed widely for rehandling and sticking of O/B spoil dumped by rail transport system. Occasionally, these machines are used for loading into bunkers or dumpers as well for which exceptional arch less buckets are available. In sunken digging such as for collecting sand and gravel, the draglines are quite equipped with perforated buckets.

Five major manufacturers of draglines do exist at present. They are Bucyrus Erie (US), Page (US), Rapier and Ransom (UK), Marion (US), and the Soviets. In India, the Heavy Engineering Corporation manufactures progressively W-2000 model walking dragline indigenously in collaboration with Rapier and Ransom. Draglines used in O/C mines typically range in size from machines equipped with 5 cu. m drag buckets on 35 m booms to the Bucyrus – Erie model 4250W, which is fortified with a 168 cu. m drag-bucket on a 94.5 m boom. Bucyrus Erie, Page, and Marion offer the longest boom length (121.9 m) dragline. The largest boom offered by Ransom and Rapier is 105.5 m. The Soviets commissioned a long boom dragline of 120 m in length during 1989. Works are now in headway to construct draglines having bucket capacity as high as 200 cubic meters. The current scenario is to have machines with high bucket capacity and short boom length. Apart from improving productivity and flexibility this design can most certainly, lend a step of safety to the overall working conditions.

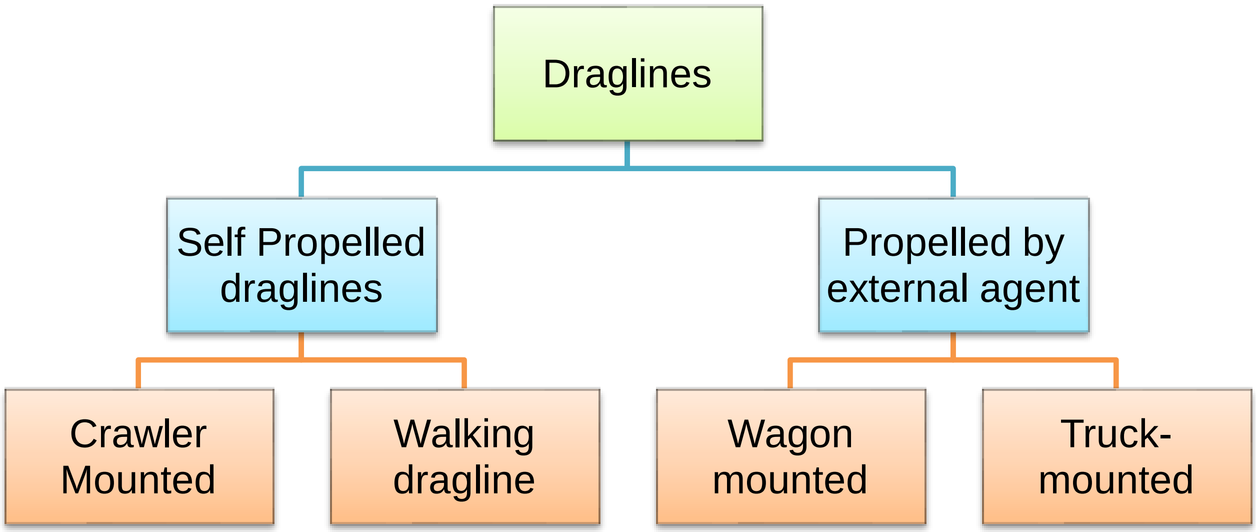
Most mines depend on the dragline 24 hours per day, 7 days per week. In many coal mines, it's the only primary stripping tool and the mine's output is totally depends on the dragline's performance. For these reasons, dragline design needs emphasis placed on developing components with high levels of consistency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

Another serious designed consideration is that most repairs essential be performed away from shop facilities. Even though the dragline is a mobile piece of equipment, its enormous size averts bringing to the shops for maintenance and repairs as common with trucks and other mine equipment. The designers must safeguard that components are really accessible and that portable tools and rigging equipment are accessible for any eventuality.

2.2. Conditions for operation of dragline

1. Gradients should be flatter than 1 in 6
2. Seams must be free of faults & any other geological disturbances
3. Deposits has to be with Major Strike length
4. Thick seams with more than 25m thickness are unsuitable
5. A hilly property is unsuitable

2.3. Classification of draglines



2.4. System of working

In a distinctive cycle of excavation, the bucket is placed above the material to be excavated. The bucket is then depressed and the dragrope is drawn so that the bucket gets dragged along the surface of the material. The bucket is then raised by using the hoist rope. A swing operation is then accomplished to move the bucket to the place where the material has to be dumped. The dragrope is then freed causing the bucket to tilt and empty. This is termed as dump operation.

The bucket can also be first 'thrown' by winding up to the jib a and then releasing a clutch on the drag cable. This would then after swing the bucket like a pendulum. Once the bucket passes the vertical, the hoist cable would be freed thus throwing the bucket. On small size draglines, a skilled operator could make the bucket down about one-half the length of the jib additionally away than if it had been just dropped. On bigger draglines, only a few extra meters may be grasped.

Draglines have dissimilar cutting sequences. The first one is the side cast method using offset benches; this includes throwing the O/B sideways onto blasted material to make a bench. The second one is a key pass. This pass initially cuts a key at the toe of the new highwall and also it shifts the bench further towards the low-wall. This may also necessitate a chop pass if the wall is blocky. A chop pass in tern may involve the bucket being dropped down onto an slanting highwall to scale the surface. The next sequence is the leisureliest operation, the blocks pass. Yet, this pass moves most of the material. It includes using the key to access to bottommost of the material to lift it up to spoil or to an raised bench level. The final cut if required is pull back, pulling material back further to the low-wall side.

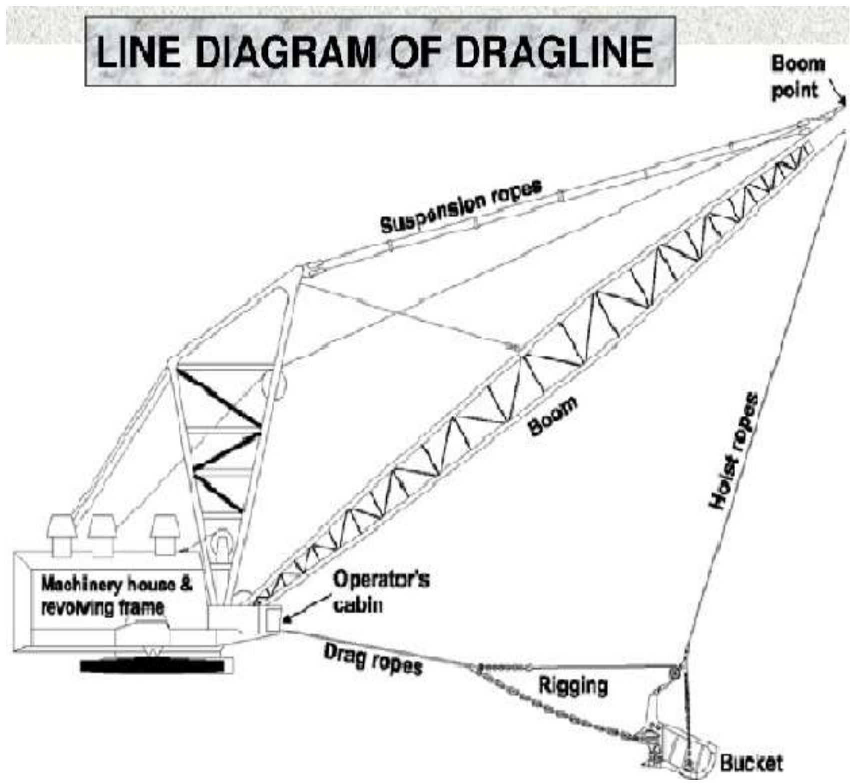


Fig. 2.1: Line diagram of dragline

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

1) The empty bucket is positioned, and made ready to be filled.
2) The bucket is dragged towards the dragline to fill it.
3) The filled bucket is concurrently hoisted and swung over to the spoil pile. The swing motion must be decelerated to permit hoisting; the dragline is called to be hoist critical. When hoisting to the dump position gets completed before the boom is in position to dump, the dragline is called to be swing critical.
4) The material is then dumped on the spoil.

5) The bucket is swung back to the cut while simultaneously being lowered and retrieved to the digging position.

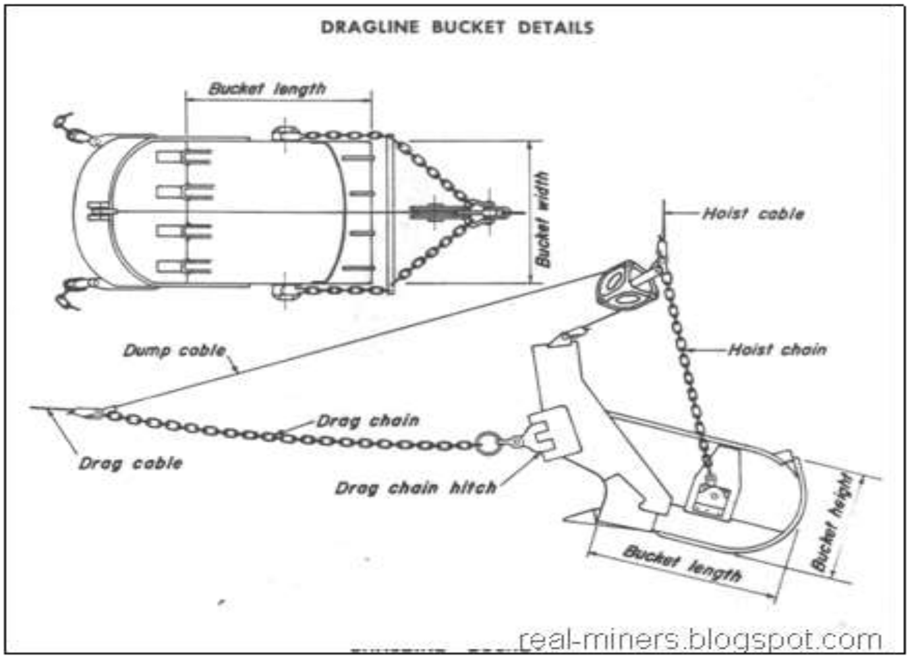


Fig 2.2: Different Parts of dragline bucket

2.5. Dragline stripping methods

The dragline stripping cycle begins with the dragline cutting a trench, commonly referred to as the key cut, along the newly formed highwall. The distance i.e. from the previous key cut position to the new position is generally referred to as the digout length. The key cut is made to uphold the panel width and uniform highwall. Deprived of a key cut, the panel width would narrow with every subsequent digout, because the dragline could not be able to control the bucket digging against an open face. The dragline deposits the key cut materials in the bottom of the pit mined-out off the coal and against the previous spoil pile. More even spoil from the key cut may be positioned in the very bottom next to preceding spoil to form the buckwall which delivers a more stable spoil slope that can be steepened if believed necessary.

When the key cut gets completed, the dragline is moved to new location to complete excavation of the digout. This is termed as the production cut, and the materials are cast on top of the key cut spoil. When the digout has been finished, the dragline is moved to subsequent position, the beginning of the next stripping cycle (i.e. next digout).

Well-organized dragline operation is realized by curtailing the time required to position, drag, and dump while harmonizing the swing and hoisting motions. Synchronization of hoisting and swinging is reliant on the time the boom is in motion.

Layer Cut vs Full Key Cut: At the beginning of a new digout, the dragline in general is placed directly over the toe of new highwall to be formed. From this position only, the dragline can establish an even and safe highwall if the burden is adequately stable.

In this position, the dragline digs the key cut which is more than the girth of the bucket at the bottom of the cut. When the cut has been finished, the dragline moves over so as to make the production cut. The two positions usually are required because of the restricted reach of the dragline in relation to the panel thickness being stripped. Large draglines, functioning under ideal conditions, may be able to dig the total digout from the one site over the highwall. Such circumstances are the exception, not the rule.

When operating conditions license excavation of the dig out from one position over the highwall, the dragline mostly excavates the digout in layers. The key cut is designed, one layer at a time, by excavating along the highwall formerly the completion of each layer. Cutting in layers can be accomplished from the production cut position; nevertheless, the high wall slope will need dressing by dozer while the dragline is excavating. Under such circumstances, some mines also have sufficiently dressed the highwall by dangling a weighty section of chain from the bucket and then dragging the chain along the wall. Other mines, because of critical spoil area, have progressively stepped the dragline towards the spoil while excavating by layer cut method. This procedure has the propensity to pack spoil as tightly as likely on the spoil slope.

Layer cutting generally raises dragline productivity with a corresponding reduction in operating cost. Increased productivity is realized by gradually decreasing the average swing angle as the dragline walks in the course of the spoil pile.

Dragline Panel Width: Panel width could be defined as the width of the cut taken by the dragline, as it grows from digout to digout, along the highwall from one end of the pit to another. Panel width, one of the most significant parameters affecting dragline productivity, is prejudiced by depth of overburden, dragline boom length, swing and hoist time, and available spoil area. Since panel width grow into the available operating area in the pit bottom, coal loading maneuvers are also affected.

Several operational factors must be well-thought-out in the selection of panel width. A wide pit usually is favorable for coal load-out and certificates greater safety for men and equipment. The minimum practical pit breadth is dictated by the maneuverability of coal loading and transportation equipment.

If the accessible space for placement of spoil is serious, such as might occur when herding spoil to open haul roads through the spoil, narrow panels license greater flexibility to deal with such problems. In overall, the wider the panel, the less would be dragline walking time is required.

Productivity dissimilarities, because of panel width, are directly linked to whether or not the dragline is swing critical. Small draglines can turn into swing critical at panel widths less than the width essential for practical coal operations in the pit. Their cycle time also rises dramatically. Bigger draglines may not become swing critical till the panel width exceeds 50 m (150 ft).

Bench Height: It could be defined as the height above the coal seam at which the dragline is situated. Selection of the bench height is completely based on numerous operational factors and topographic manacles.

The intricate relationship of bench height (which could be equal to O/B depth), panel width, dragline dumping grasp and dumping height, the material characteristics such as swell and angle of repose, effect greatly the dragline's ability to dispose of burden off the coal. The dragline's digging depth, while linked to burden depth, hardly becomes a factor in dragline performance.

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

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

1. The dragline can be in use to cut and fill to develop a common bench raise. Cutting labeled chopping or overhand excavating, increases the cycle time and lessens the bucket fill factor, thereby reducing operative productivity. Fill material should be rehandled, hence reducing overall production. Chopping has very special benefits: the dragline reach mandatory may be shortened, rehandling of burden may be circumvented, fill may not be obligatory to create a level working surface, a level coming back path for deadheading can be provided, and subsoil can be located back in its relative position on topmost of the spoil.
2. Supplementary equipment can be used to perform the cut and fill process. Care must be taken to ensure that filled areas are steady. Utilizing auxiliary equipment bids the benefit of freeing the dragline for its chief function of stripping burden from the coal.

Digout Length: The selection process of digout length, i.e. the length between major digging cycles, is completely based on the relationship of the dragline's operating features with respect to pit geometry. In overall, the digout should be as long as it's possible. Nevertheless, dragline size may greatly influence digout span for specific pit geometry. For example, digout length is delicate when using a dragline with slow hoist speed employed in deep overburden. Spoil serious pits may utilize a less than required digout length in order to pack the extreme material onto the spoil bank.

A long digout with respect to dragline size reduces cycle time and upsurges productivity because more material is loaded under the external end of the boom than near the e fairleads of the dragline. A good dragline operator tries to fill the bucket within two and half to three times the bucket length. Cycle components of recovering for bucket dragging, loading, payout for dumping, and swing angle all are reduced as the digout length increases. Longer digout lengths also decrease the nonproductive time required for relocating the dragline on the succeeding digout. Clearly, digout length should not be so long as to necessitate the dragline operator to cast the bucket beyond the bound of the boom.

Walking Patterns: In a dragline operation, two distinct walking cycles are involved: dead-heading and walking within the digout. When a panel has been finished, there are two options available for the dragline. One, the dragline can wait for the n 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 called layering over at the end of a panel. Two, the dragline can turn about and travel part or all of the way down the panel to begin the next cut. This procedure is named 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 voyages to the other end of the pit, it cuts in to the next panel and strips in the identical direction.

Whether a dragline lies over or dead-heads depends primarily on the production time lost. Contractual requirements, such as unprotected coal inventory, may eliminate the layover choice and force deadheading. Some mines opt to untrained over at the end of a panel because ground conditions are not favorable for deadheading. Other mines boundary layover to a maximum of two shifts. If waiting for coal production includes two or more lost dragline shifts, the dragline will be deadheaded.

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

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

Pit Shape: The new dragline pit instigates with the initial cut, termed as the box cut, made along the outcrop, sub crop, 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 supplementary equipment. The material lying on the cut wall which will become the spoil side may, or may not, have to be enthused depending on reclamation requirements.

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

To remedy the problems caused by inside curves, numerous options can be considered. Panel width may be diminished, material may be cast short and rehandled by spreading the bench, a small auxiliary dragline may be employed 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 promising solution is to straighten the pits. Spoil steepening is also could be an real method for disposing of relatively small quantities of excess spoil. The dragline bucket is located on the spoil slope where steepening is looked-for and dragged down and across the top of coal. The bottom part of the spoil pile steepened and coal is cleaned in the process. Digging efficiency during this process is reduced, cycle time amplified, and rehandling reduces actual productivity. Steepened spoil slopes may offer special hazards to equipment and personnel because they are more prone to failure.

Spoil Patterns: There are basically three methods of spoiling. When using short digouts n casting at a near 90^0 angle, an unchanging ridge line can be created. This configuration makes extreme use of the present spoil room. As the digout length is raised, uniformity of the ridge line is gone and individual peaks of spoil are created.

With sufficient spoiling area taken, 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 progress, spoil piles appear to be ragged and uneven. An aerial view of the operation will show a conclusive pattern to the irregularity. In reality, spoil peak grading will be abridged by this method of spoiling.

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

Since distance between spoil ridges is alike to the panel width, narrower panels will diminish spoil grading costs. However, such reductions in cost mostly will be offset by increase in dragline operating cost if the dragline is not swing critical one.

2.5.1. Simple sidecasting method

It's the simplest form of strip mining, involving 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 preceding strip after the coal mineral has been mined. It is customary to start the dig 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 m/c can most easily dig a neat and competent high wall. The nearest high wall is affected by starting the out with the dragline in stripe with the crest and moving it as the out gets deeper, ending with the machine n line with the toe of the new high wall. By this means, the slope angle of the newer high wall can be closely con rolled. The width of each strip is typically designed so that the material from the key cut can be thrown into the preceding cut without the need for rehandle.

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

However, more often the spoil pile touches the crest of the coal seam for obvious merits 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 functional 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 l to be excavated and swings about an arc of between 45 to 90 degrees to dump the material. A typical average cycle time f for the operation is 45 seconds.

Fig 2.3: Simple sidecasting method

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

2.5.2. Dragline Extended bench method

Where O/B depth or the panel width exceeds the boundary at which the dragline can sidecast the burden from the coal, a bridge of burden can be designed between the bank and the spoil which effectively extends the reach of the dragline. The bridge encompasses the bench on which the dragline is operating. The bridge is formed by material falling down t 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 outlines of pit geometry. In this method the dragline forms its working bench by chopping material from above the bench and forming the bridge and 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 r the secondary dragline, to system the bench for the secondary dragline. The secondary dragline first most strips material near the highwall, then moves on to the bridge to as move the rehandle material. In a two-dragline system, one m/c must in operate at the pace set by the other. Therefore, mine design must consider to their respective capacities when assigning respective digging depths. The primary dragline strips O/B to the top of the first seam. Coal is uninvolved, then a small parting dozed in to the pit and the second coal seam removed. The secondary dragline strip s 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.

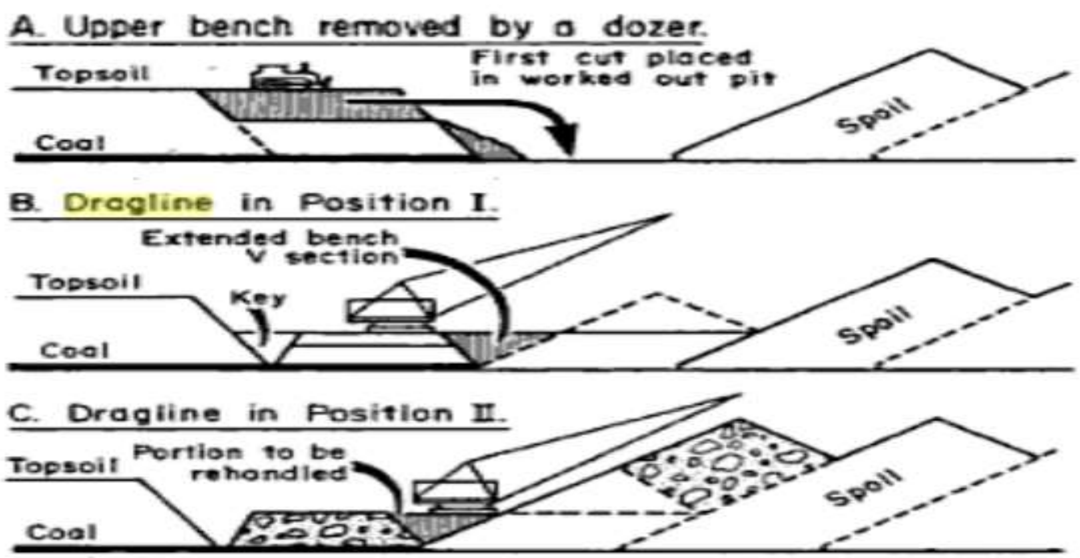


Fig 4: Positions in extended bench method

2.5.3. Pull-Back Method

Infrequently, overburden to be stripped will be beyond the capacity of the dragline to spoil off the coal by any of the preceding methods described. In this case, a secondary dragline can be placed on the spoil bank to pull back adequate spoil to make room for complete removal of overburden.

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

Draglines often are utilized to strip overburden from deeper coal seams than initially intended. Irregularly, spoil slopes cannot be maintained at designed to angles. Various methods have evolved to stack extra material into the spoil bank to alleviate these problems. The very common methods are described precisely:

1. Buck walls include building the base of the spoil adjacent to the pit with capable material so that a steeper spoil slope near the base can be maintained.
2. Coal fenders need leaving a small wedge of coal untouched in the pit so that more amount of 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 no. of coal seams and/or deeper overburden can be done by using a dragline with a second system which then can be another dragline, shovel, scraper dozers, etc.

This system has two benefits as:

(i) One machine removing both O/B (100ft thickness) and interburden (20ft thickness) will perhaps be less proficient than two machines – one designed for overburden removal and the other for interburden removal.

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

Drawbacks of the tandem system include when loading machine is depressed, the trailing machine will often be idled. Such conditions can however be minimized by good planning.

2.6. Drilling and Blasting of the Overburden for Draglines

O/B drilling and blasting is more serious for dragline stripping than for shovel digging. Shovels have the ability to crowd the dipper in to the bank, providing influence to dig difficult or poorly blasted material. Draglines have influence only by dragging the bucket over the material. Such leverage is interpreted to severe strain on the bucket lip and teeth. In poorly blasted material, dragline output can drop more rapidly than that of shovels working in alike material.

Discerning placement of explosives and blasting agents may be critical to the surface coal mine operation. Many coal seams are superimposed with sedimentary beds of varying hardness and thickness. Improper placement of the charge in the blast hole can source blast energy to travel along planes of greater weakness and through softer material. Under such situations, harder beds of material will tend to break in large blocks or fragments. To safeguard adequate placement of the blast charge, it is necessary that drill operators log alterations 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. 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 used in India

Table 2.1- Bharat Coking Coal Ltd. (BCCL)

	Project	Dragline Capacity	Dragline Capacity	Geo-mining conditions
1.	Block II	24x96	1	Mid seam of coking coal worked. O/B 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	Dragline Capacity	Dragline Capacity	Geo-mining conditions
1.	Sonepur Bazari	26 cu m	1	Multi seam deposit, bottom and medium thick seam exposed by dragline
Total for ECL			1	

Table 2.3- Mahanadi Coalfields Ltd. (MCL)

	Project	Dragline Capacity	Dragline Capacity	Geo-mining conditions
1.	Balanda	4x45 - 1 10x60 – 1 11.5 cum– 1 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	Dragline Capacity	
1.	Amlori	24x96	1	MOHER-SUB BASIN, Singrauli Coalfield. The NCL presently works in Moher sub-basin of Singrauli coalfield. The basin has three seams. The upper seams are 8-10 m thick with parting 40 m in between. The lowermost seam is 16-22 m thick and has a parting of about 40 m among 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
2.	Bina	10x70 - 2 24x96 - 2	4	
3.	Dudichua	24x96	2	
4.	Jayant	15x90 - 1 24x96 - 3	4	
5.	Khadia	20x90	2	
6.	Nigahi	20x90	2	
Total for NCL			15	

Table 2.5- South Eastern Colafields Ltd. (SECL)

	Project	Capacity of Dragline	Dragline Capacity	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 board and pillar previously
3.	Dhanpuri	10x70 – 1 20x90 – 1	2	6-7 m thick seam
4.	Dola/Rajnagar	10x70	1	Two thick seams with thin parting
5.	Jamuna	5x45 – 1 10x70 – 1	2	Thin seam at shallow depth
6.	Kurasia	5x45 – 1 10x70 – 1 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	Dragline Capacity	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 7 cu.m – 1 15x90 – 1	3	Multi seam deposit, bottom seam thickest. 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	Dragline Capacity	Geo-mining conditions
1.	Ramagundam OC-I	24x96	1	Upper seams exposed by shovel-dumper. Lower seams by dragline
2.	Ramagundam OC-III	30 cu.m	1	Parting between two seams taken by dragline
Total for SCCL			2	

Total for India = 43

Chapter 03

DATA COLLECTION, RECORDING AND ACQUIRING

The very common parameters such as bucket capacity, boom length, dumping height, reach, 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 accumulated from Samaleswari (MCL) mine of scheduled shift hours, average working hours, average idle hours, average breakdown hours, and average maintenance hours were acquired form earlier 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

The dragline 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 yearlyly).

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

(v) Calculating the percentage of rehandling.

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

4.3. Designing dragline balancing diagram

Let say BCDE be the cross-sectional area to be removed to expose coal seam A B C O. for convenience, call it as First-dig.

Let, A1= First-dig ……………… (1)

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

Let, A2= Dumping Area ………….. (2)

Assuming that S to be the swell factor of the O/B material, actual area of overburden needed to be accommodated in dump would be A1S.

Let, A3= A1S-A2 …………….. (3)

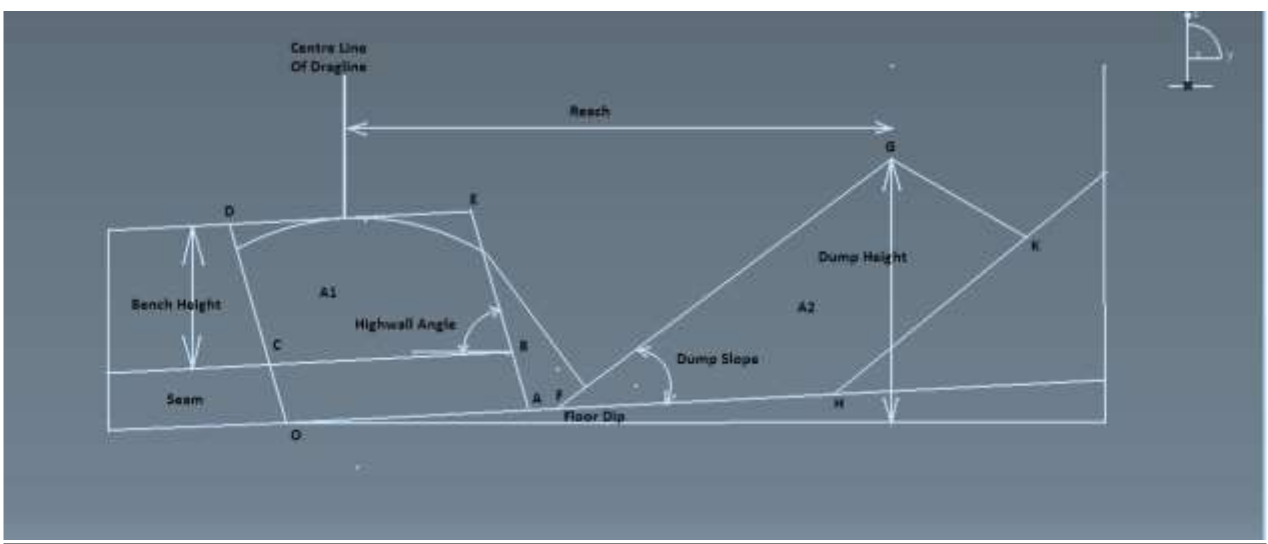


Fig 4.1: Dragline balancing diagram for total sidecasting

Case-1. When A3<0

In this case the dump area is capable of accommodating O/B more than the available first-dig quantities. This infers that height of the dragline bench or cut width may be raised 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 implies 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 rise in the height of dragline bench or cut width would give upsurge to an increase in first-dig and this increase is impossible to be accommodated in the dump.

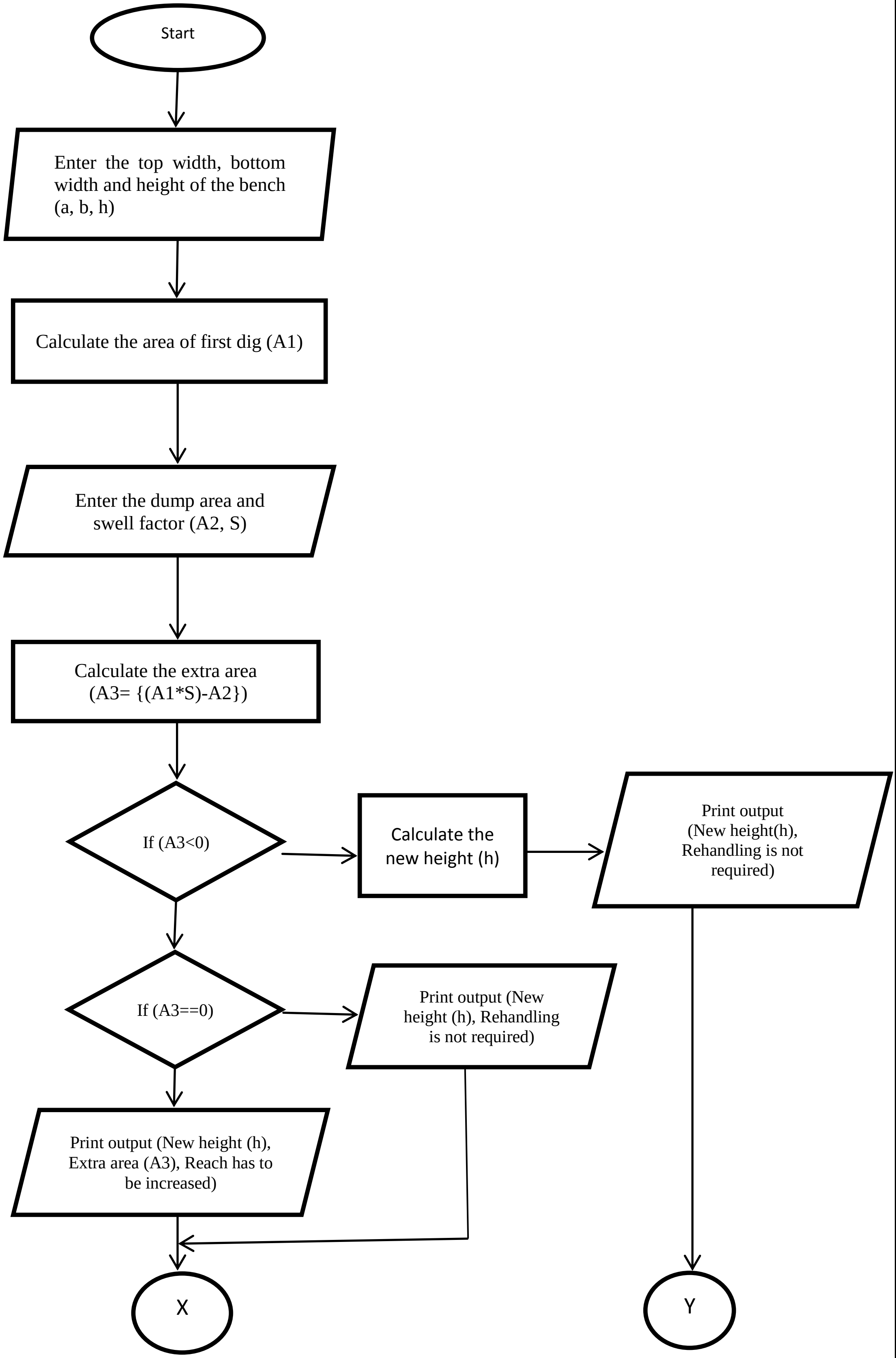
Case-3. When A3>0

This infers that the dump is incapable of taking the loose first-dig entirely and A3 amount of overburden would be left as residual. This left out matter can be handled in two ways, either by transporting and dumping somewhere else 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. Otherwise 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. A computer based program (C++) developed on dragline bench designing (restricting the cut-width to be constant) and determine whether or not to rehandle

```
#include<stdio.h>

#include<stdlib.h>

float a,b,h,A2,s,A3,A1;

void area1()

{

    A1=(((a+b)/2)*h);

    printf(" 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

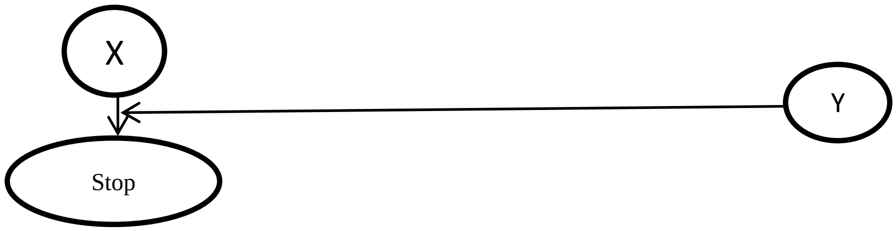


Fig 4.2: Flowchart for the dragline bench designing (keeping the cut-width constant) and determine whether or not rehandling is required

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 No.1

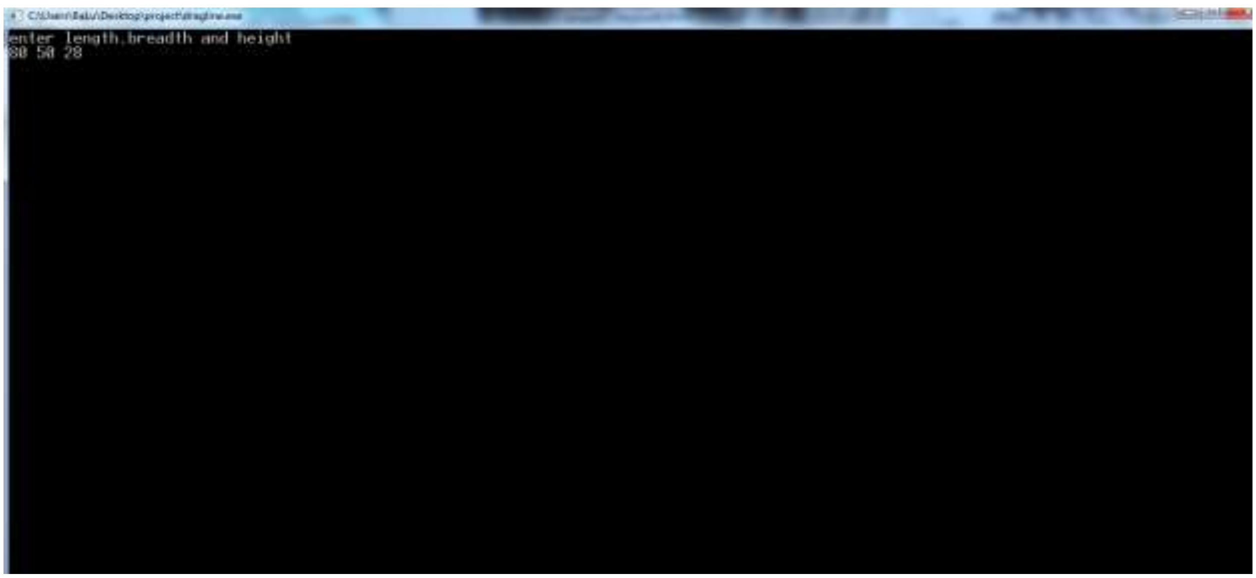


Fig 4.4: Output screenshot No.2

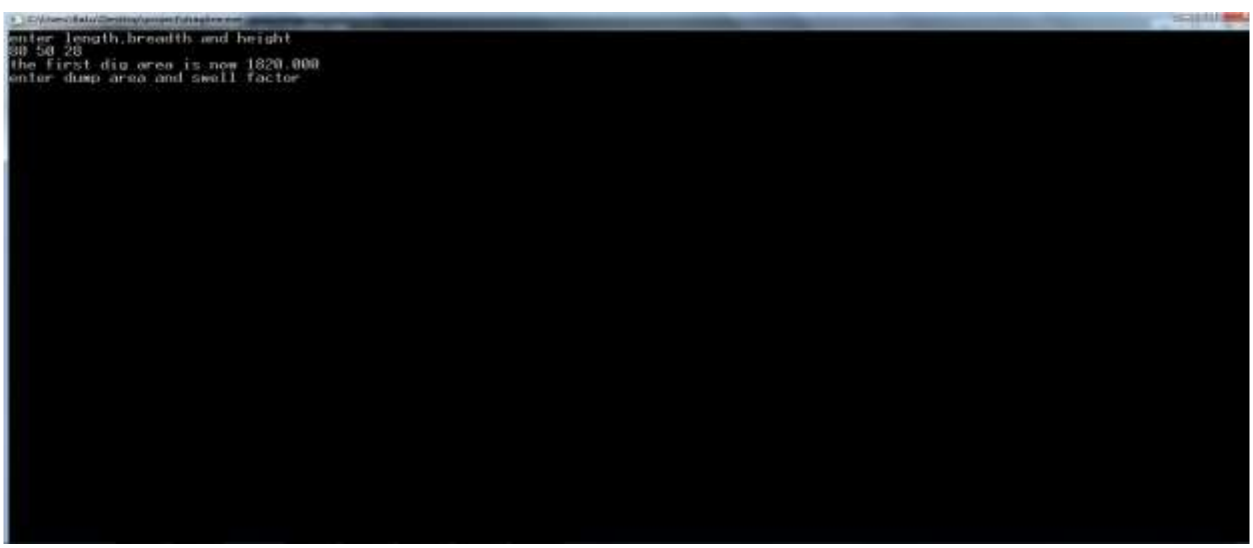


Fig 4: Output screenshot No.3

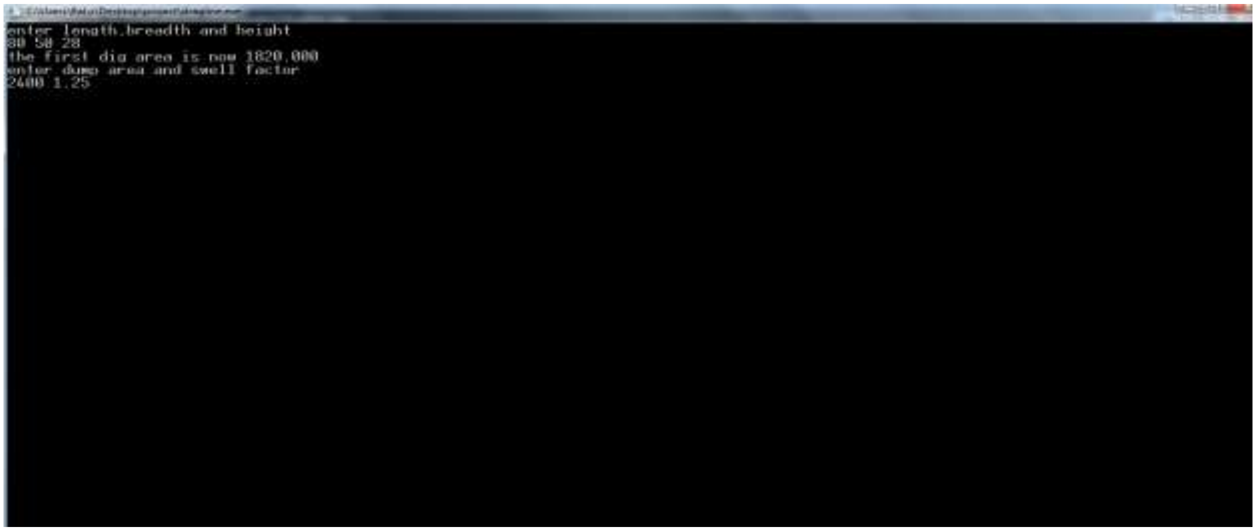


Fig 4.6: Output screenshot No.4

Fig 4.7: Output screenshot No.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 No.6

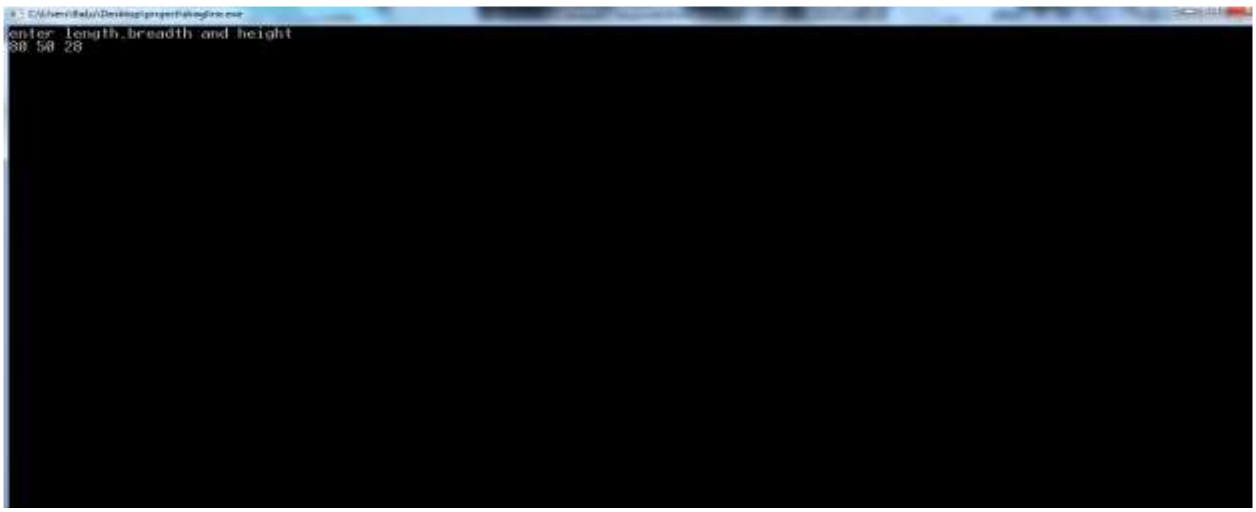


Fig 4.9: Output screenshot No.7

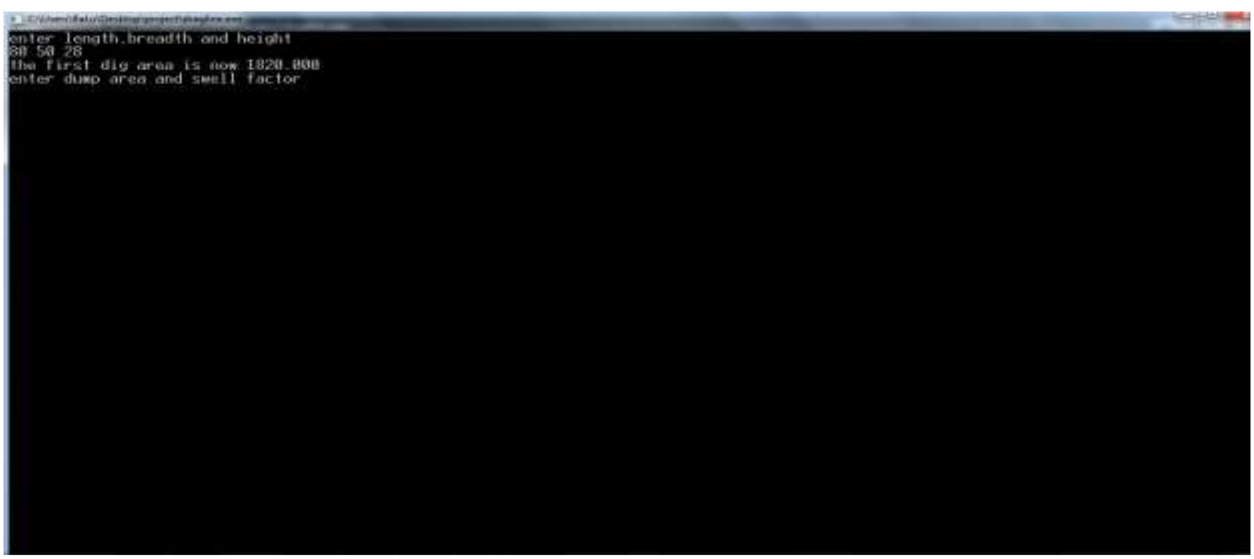


Fig 4.10: Output screenshot No.8

Fig 4.11: Output screenshot No.9

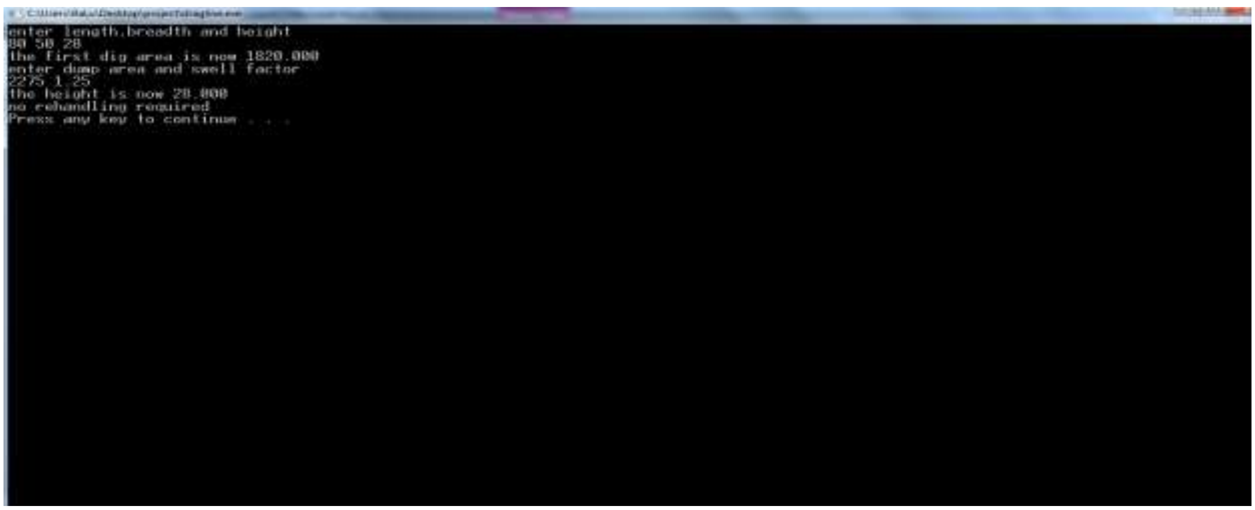


Fig 4.12: Output screenshot No.10

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 No.11

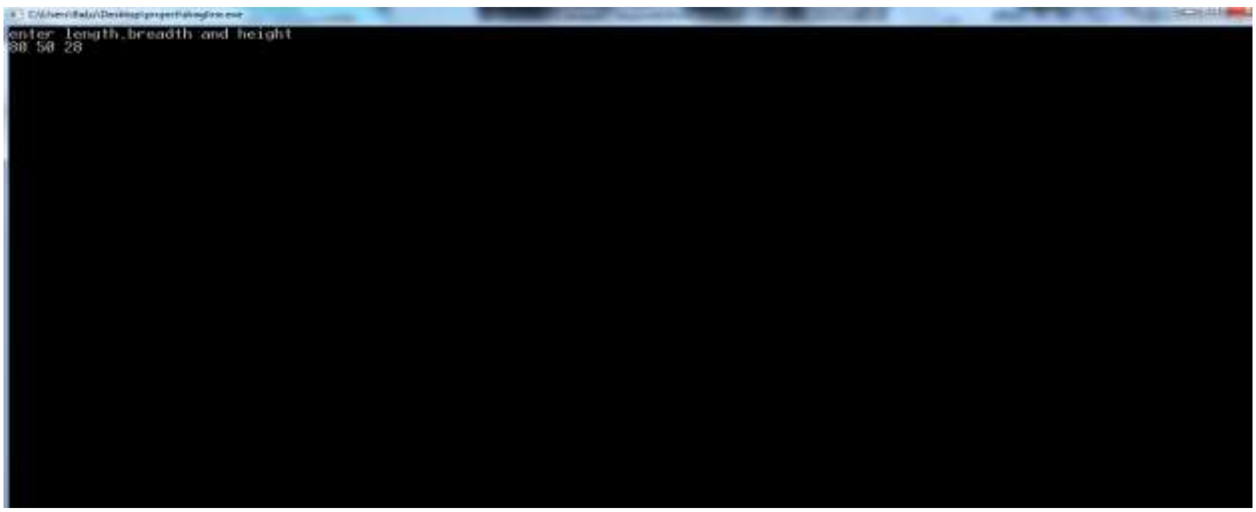


Fig 4.14: Output screenshot No.12

Fig 4.15: Output screenshot No.13

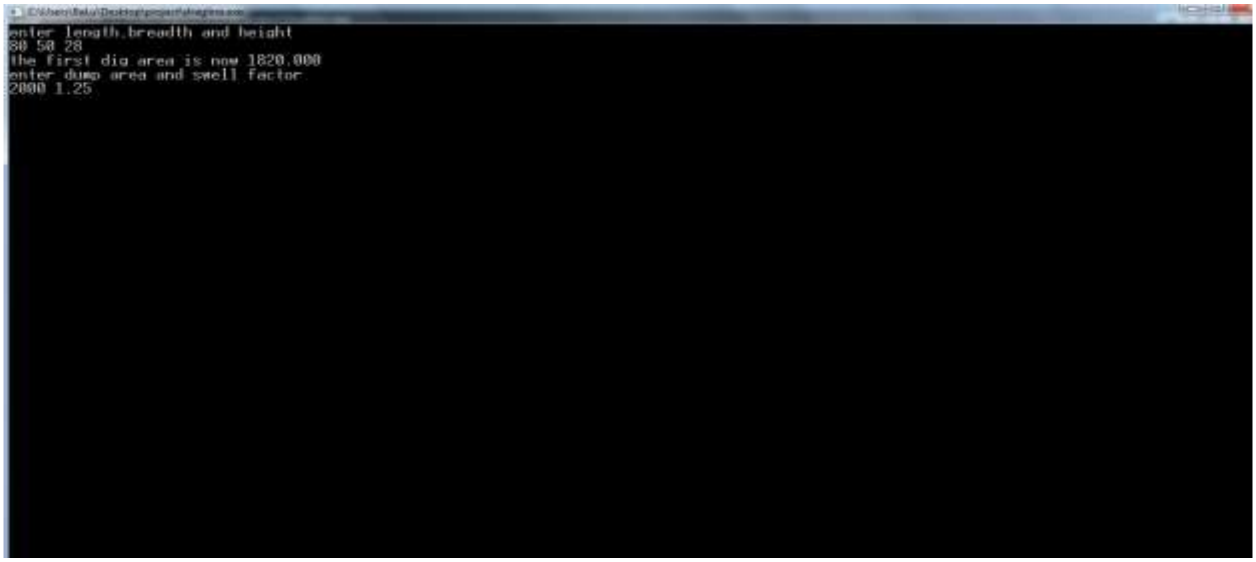


Fig 4.16: Output screenshot No.14

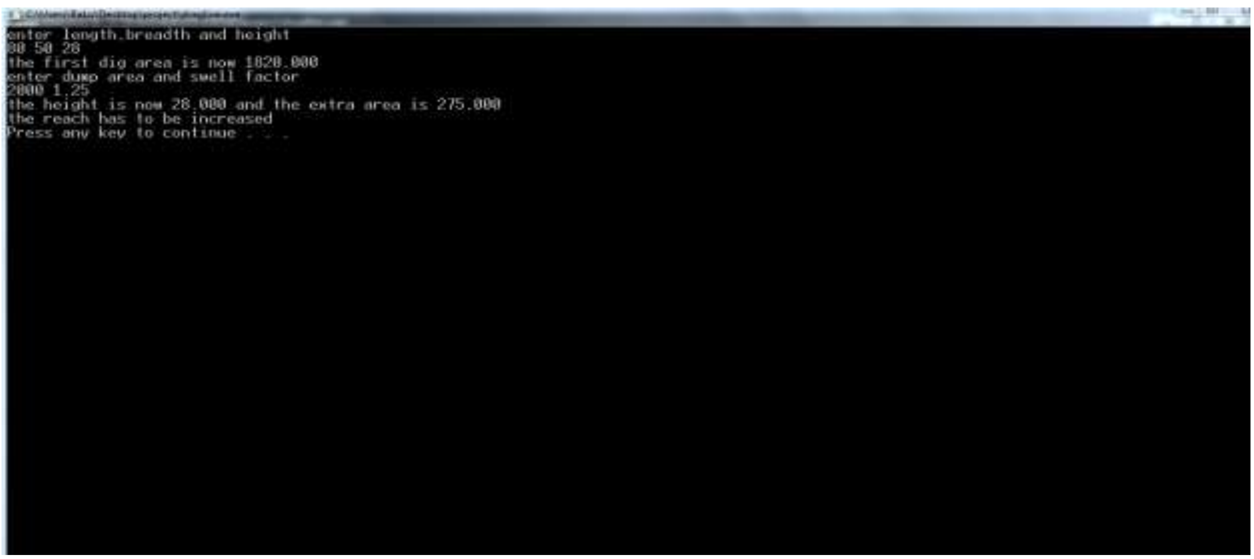


Fig 4.17: Output screenshot No.15

4.6. Evaluation of yearly 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 learned was substituted in the Eqs (i) and (ii).

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

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

Where,

SSH or scheduled shift hours,

MH or maintenance hours,

BH or breakdown hours,

IH or idle hours

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

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

Where,

B, bucket capacity of the dragline in cubic meter,

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

S, the swell factor,

F, the fill factor,

M, the machine travelling and positioning factor,

N_s, the number of operating shifts in a day,

N_h, the number of operating hours in a shift,

N_d, 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)	35-41
5.	Cutting width (m)	50
6.	Highwall slope (degrees)	70
7.	Bench slope (degrees)	60
8.	Angle of repose (degrees)	38.5
9.	Digging depth (m)	37
10.	Reach of dragline (m)	58.77
11.	Method of working	Simple side casting
12.	Thickness of coal seam (m)	24

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) = 4.55 m

Angle of repose of overburden (x) = 38.5^0

Reach of dragline (R) = 73 m

Swell factor (S) = 1.38

Width of cut (W) = 60 m

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

Using the above values in the equation (iv)

$$H = \frac{4.55 + \tan 38.5\,(73 - 60/4)}{1.39 + \frac{\tan 38.5}{\tan 70}}$$

After calculation we get H = 29.78 m

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

For Samaleswari Dragline (10/70):

Max. depth that can be worked by the dragline (H)

Thickness of coal seam (t) = 25 m

Angle of repose of overburden (x) = 38.5^0

Reach of dragline (R) = 58.77 m

Swell factor (S) = 1.39

Width of cut (W) = 50 m

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

Using the above values in the equation (iv)

$$H = \frac{25 + \tan 38.5\,(58.77 - 45/4)}{1.39 + \frac{\tan 38.5}{\tan 70}}$$

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) = 42 m

Thickness of coal seam (t) = 4.51 m

Reach of dragline (R) = 72 m

Width of cut (W) = 60 m

$$P_{RM} = \frac{(1.125*4.51) + (0.684*42) + (0.1*73)}{60} + \frac{(0.25*4.51^2) - (0.4*73*4.51) - (0.16*73^2)}{40*60} +$$

$$\frac{(0.1*4.51) + (0.08*73) - (0.01*60)}{40}$$

= 0.662 – 0.40 +0.14

= 0.402

So the amount of rehandle percentage = 40.2

4.6.4. Projection of yearly 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	508	119	33	61
Samaleswari	10/70	720	541	91	31	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	61.71
Samaleswari	10/70	60	66.35

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 = 0.7899

Utilization = 0.7042

For Samaleswari Dragline (10/70)

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

$$A = \frac{720 - (91+31)}{720}$$

$= 0.8332$

$$U = \frac{720 - (91+31+60)}{720}$$

$= 0.75$

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	0.7899	0.7041
Samaleswari	10/70	0.8332	0.751

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

= 2.817 M cu.m.

So, the projected yearly 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 yearly 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. 1001 million

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

Yearly depreciation cost of 24/96 dragline = Rs. 40.4 million

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

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

Where N = Life of dragline = Rs. $\frac{1000*26}{2*25}$ million = Rs. 521 million

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

= 15 % of Rs. 520 million

= Rs. 78 million

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

= Rs. (41+78) million

= Rs. 119 million

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

(i) Yearly manpower cost (salary and wages)

Operator cost @ Rs. 0.20 millions/operator for 2 operators in 3 shifts = Rs. 1.21 million

Helper cost @ Rs. 0.14 million for 1 operator in 3 shifts = Rs. 0.425 million

Total manpower cost = Rs. 1.622 million

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

Yearly power consumption cost @ Rs. 4.89/KWH = Rs. $4.89*13.615*10^6$

= Rs. 66.75 million

(iii) Yearly lubrication cost @ 30% of power consumption = Rs. 20.035 million

(iv) Yearly 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 Yearly operating cost = Manpower cost/year + Electrical cost/year + Maintenance cost/year + Lubrication cost/year

= Rs. (1.621 + 66.715 + 20.015 +28.5) million

= Rs. 117.4 million

Total ownership and operating cost

= ownership cost/year + operating cost/year

= Rs. (118+117.4) million

= Rs. 235.4 million

Dragline operating cost per m^3 overburden handle considering yearly output of 24/96 as 2.817 M cu.m = Rs. $\frac{234.4*10^6}{2.817*10^6}$ = 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 2.817 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 = $\frac{2.817}{1.40}$ 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. 286.78

= Rs. 286.78 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. 301 million

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

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

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

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

Where N = Life of dragline = Rs. $\frac{300*26}{2*25}$ million = Rs. 157 million

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

= 15 % of Rs. 157 million

= Rs. 23.41 million

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

= Rs. (12+23.4) million

= Rs. 35.5 million

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

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

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

= Rs. 44.35 million

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

(iv) Yearly 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 Yearly 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

= Rs. (35.4+67.67) million

= Rs. 103.07 million

Dragline operating cost per m^3 overburden handle considering yearly output of 10/70 as 1.253 M cu.m = Rs. $\frac{103.07*10^6}{1.253*10^6}$ = Rs. 81.2

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

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

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

= 0.417 M te

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

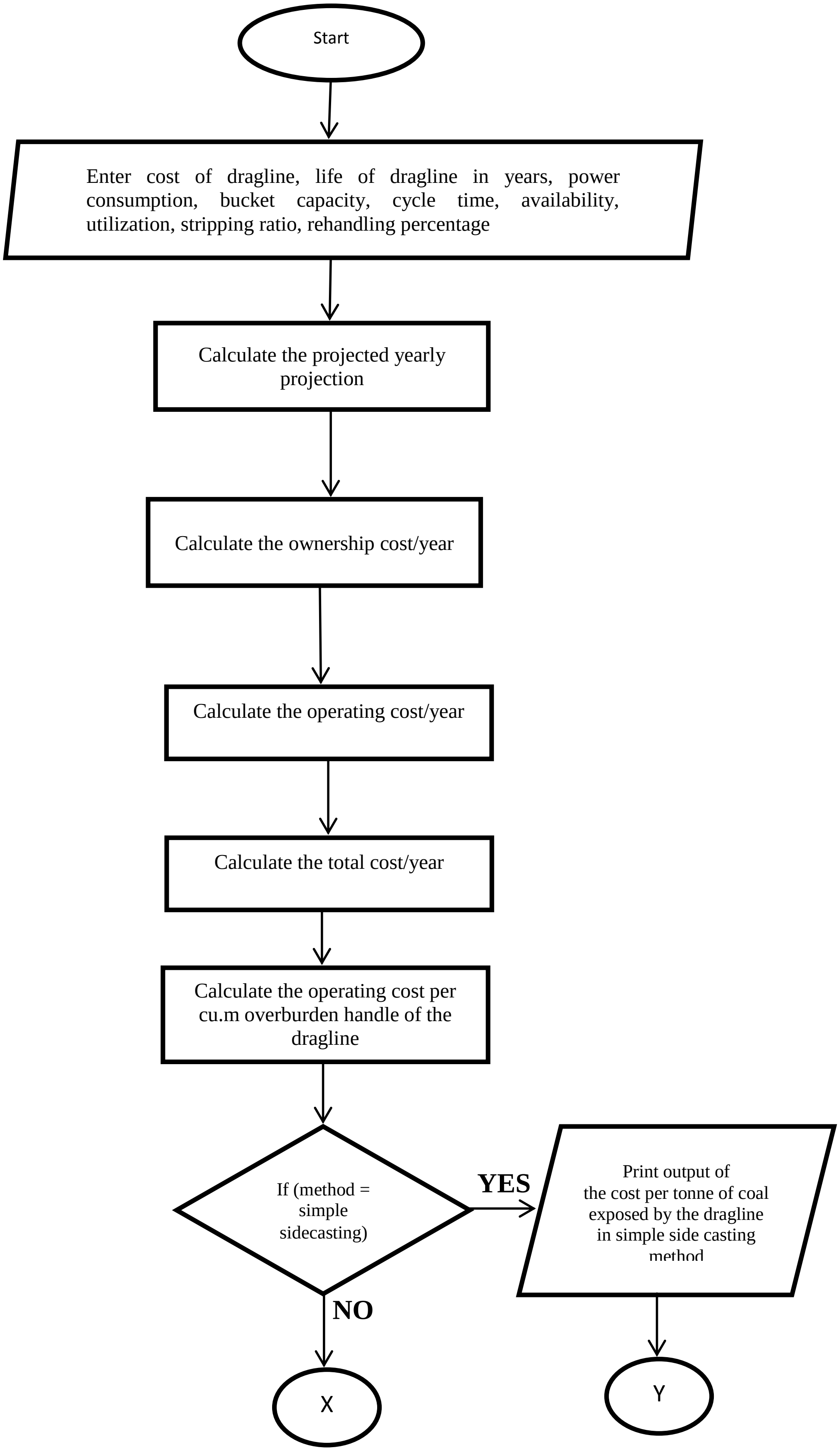
= Rs. 248.17 /te of coal exposed

4.6.9. A computer based program (C++) developed for projection of yearly 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 yearly production()
{
pa=(b/ct)*a*ut*0.719*0.733*0.8*8*3*365*60*60;
printf("the yearly production of the dragline is %.6f\n",pa);
}
void ownership cost()
{
    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 operating cost()
{
  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 total cost()
```

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

}
void overburden cost()
{
   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 coal exposure2()
{
 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%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 bench method(e))::");
    scanf("%c",&t);
    yearly production();

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

    Ownership cost();
    Operating cost();
    Total cost();
    Overburden cost();
    if(t=='s')
      coal exposure1();
    if(t=='e')
      coal exposure2();
    system("pause");
    return 0;
}
```


Start
Enter cost of dragline, life of dragline in years, power consumption, bucket capacity, cycle time, availability, utilization, stripping ratio, rehandling percentage
Calculate the projected yearly 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

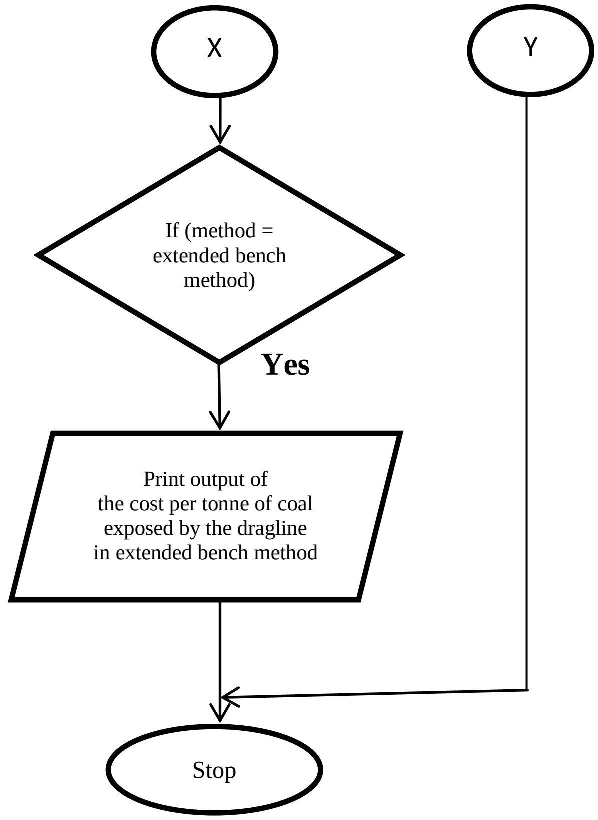


Fig 4.18: Flowchart for the dragline bench designing (restricting the cut-width to be constant) and determine whether or not rehandling is required

Output: 1 For extended bench method;

Input entered:	
Cost of dragline	: Rs. 1002 million
No. of years	: 25
Power consumption	: 13.85 KWh
Bucket capacity	: 24 cu.m
Cycle time	: 60.7 s
Availability	: 0.7898
Utilization	: 0.7053
Stripping ratio	: 2.5
Percentage rehandling	: 0.41

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

Output 1:
The yearly production of dragline = 2.87 M cu.m
The ownership cost of dragline/year = Rs. 118 million
The operating cost of dragline/year = Rs. 114.71 million
The total costs of dragline/year = Rs. 242.7 million
The operating cost per cu.m overburden handle = Rs. 81.03
The cost per tonne of coal exposed by the dragline = Rs. 285.76

Output screen no.1:

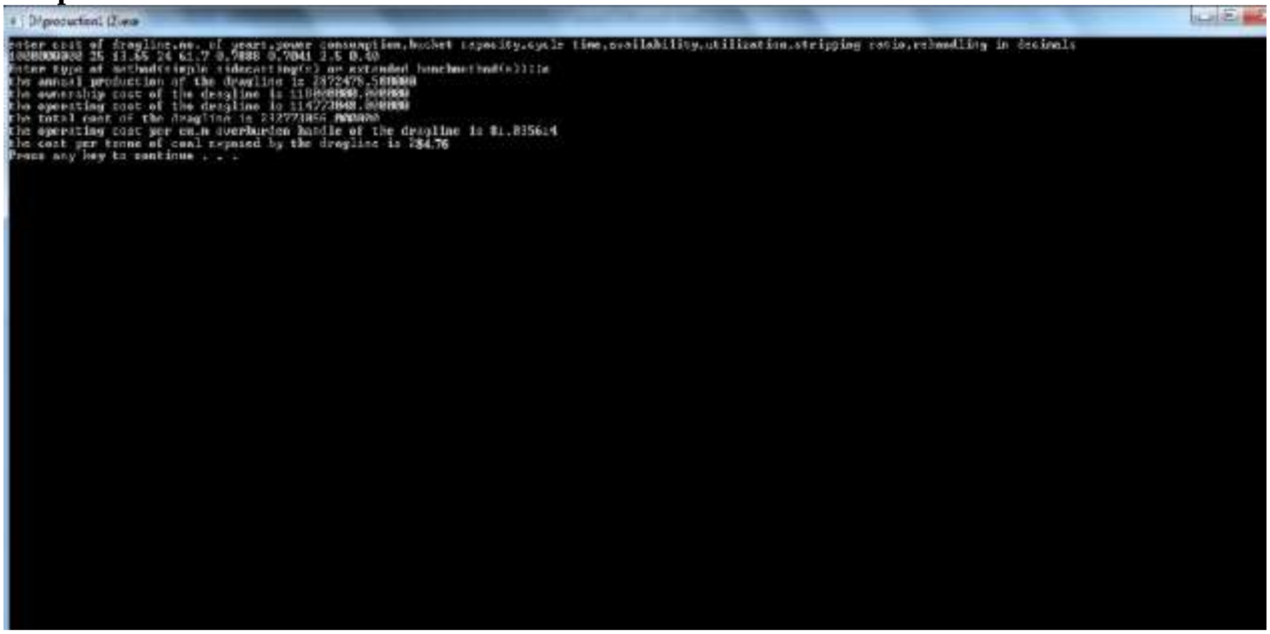


Fig 4.19: Output screen No.16

Output: 1 For simple sidecasting method;

Input entered:

Cost of dragline	: Rs. 300 million
No. of years	: 25
Power consumption	: 9.17 KWh
Bucket capacity	: 10 cu.m
Cycle time	: 66.35 s
Availability	: 0.8533
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 yearly production of dragline = 1.24 M cubic.m
The ownership cost of dragline/year = Rs. 35.45 million
The operating cost of dragline/year = Rs. 66.055 million
The total costs of dragline/year = Rs. 101.451 million
The operating cost per cu.m overburden handle = Rs. 81.46
The cost per tonne of coal exposed by the dragline = Rs. 244.40

Output screen no.2:

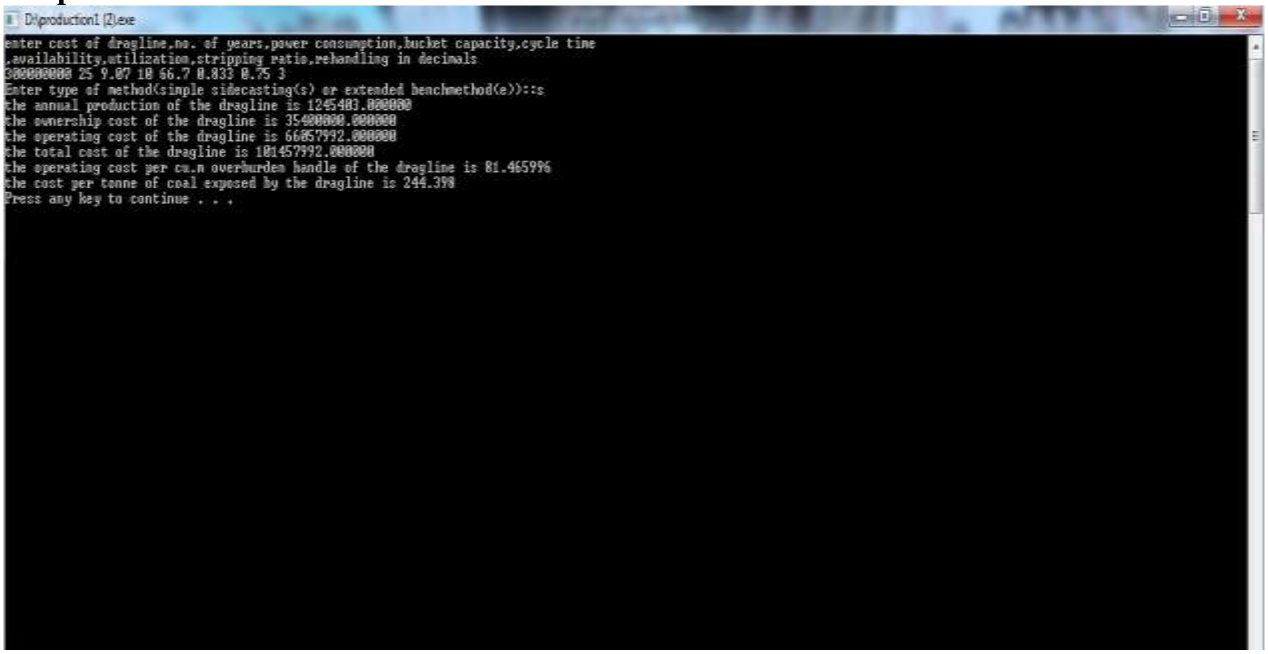


Fig 4.20: Output screen No.17

Chapter 05

RESULTS

- The projected yearly output is 2.817 M cu.m for the dragline of Singareni OC-I.
- The projected yearly output is 1.253 M cu.m for the dragline of the Samaleswari mine.
- Dragline operating cost per m^3 overburden handle considering yearly output of Singareni OC-I dragline (24/96) as 2.817 M cu.m is Rs. 81.670
- Dragline operating cost per m^3 overburden handle considering yearly output of Samaleswari dragline (10/70) as 1.253 M cu.m is Rs. 82.20
- Estimated cost/tonne of coal exposed by the Singareni OC-I dragline(24/96) is Rs. 285.781
- Estimated cost/tonne of coal exposed by the Samaleswari dragline(10/70) is Rs. 247.172

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

- Enhancing the Dragline Productivity through Maximizing Cast by using blasting techniques like i-kon system of blasting.
- A must focus on the balancing diagram.
- Employing skilled workforce.
- Maintenance of the machine should be of prior importance.
- Employing a spare dragline operator and reducing the Variability in dragline operator performance.
- Taking proper decision for a keycut (its location, width, depth etc.) and then the boxcut.
- Better preventive maintenance schedule to reduce the breakdown time and breakdown costs.
- Communication system has to be of modernized.
- Water sprinkling on the dump will suppress the dust.
- The dragline cycle time has to be reduced to ensure efficient production.

Chapter 07

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