CHAPTER 3 - PLANNING OF DRAGLINE OPERATION AND DEVELOPMENT OF THREE-DIMENSIONAL BALANCING DIAGRAM

3.1 Introduction

A large opencast coal mine deploying a high-capacity dragline always requires very systematic operational planning and very strict operational controls for its strict implementation. Operational parameters such as pit geometry and sequencing of the dragline operations are usually considered in conjunction with the selected digging method. Once the suitable digging method and dragline model or size have been selected for a dragline operation, the next step is then operational planning of those factors which will influence or control the strip mine.

3.2 Planning for dragline operation

3.2.1 Dragline method selection

There are several factors to consider when selecting a dragline method for mining operations. Some of the key considerations include mining-geological and geotechnical conditions of sections (blocks) mined by direct dumping (number of coal seams, their thickness and dip angle; physical and mechanical properties of the excavated overburden, as well as the rock mass underlying the soil of the lowest seam in a suite, which will be the foundation for internal direct dump; interbed thickness and the total thickness of the coal-bearing strata. The best performance of the dragline can be achieved only when it is employed in a specific stripping method that best characterises the conditions of the mine site and its ability of selected dragline equipment.

It means that the capacity and the productivity of a dragline should be compatible with the geometry of the cut and hence the production and overburden stripping requirements. The selected dragline should meet the geometrical constraints of the stripping method, which will be applied and should be capable of removing the required volume of overburden in a given period of time. After a particular dragline method has been selected, the next steps are dragline planning in that mine. Details on dragline methods are provided in chapter 2, section 2.7 and 2.8.

3.2.2 Dragline machine size selection

The size of the dragline is often expressed by its bucket capacity and boom length. A proper dragline selection needs a simultaneous understanding of various factors, such as the geological properties of overburden material to be removed, casting method, available funding, and technology (Jameson and Thomas, 1992). Dragline selection is, therefore, a difficult and time-consuming procedure (Akhundov and Demirel,2019).

There are three basic stages to the dragline selection process (Atkinson,1992).

i. preliminary estimates of dragline operation variables (dragline availability and utilization, dragline operational periods, cycle time, ore or coal production and ore recovery, etc.).

ii. Initial mine design geometry definition

iii. Relationships between the available equipment geometry and mine design

The proper dragline is selected by reviewing the above steps.

3.2.3 Macro-level operational planning

In order to achieve a safe and productive dragline working, various macro and micro-level operational requirements have to be fulfilled. Macro-level operational requirements address all the desired conditions and environment not directly related to the direct operation of the dragline but prerequisites to the direct operation of the dragline, whereas micro-level operational requirements address all the conditions and environment for safe and productive dragline operation, that is, directly related with the operation of the dragline.

Macro-level operational planning includes the following:

3.2.3.1 Planned advance of all the benches above the dragline bench, including dragline formation level.

The advance of all the overburden and coal benches should be as per the operational planning of the mine. During the mine advance following principles must be adhered to:

A) The advance rate of different benches should be in harmony with each other.

B) the rate of advance of mine, that is, the combined advance of all the benches should be in commensuration with the rate of void/de-coaled area created by dragline.

3.2.3.2 Planned advance of dragline formation level.

This advance will ensure the independent operation of the dragline, independent drilling, charging and blasting operation at the dragline bench ahead of blasted material, and independent operation of excavators and transporting equipment deployed at the dragline formation level bench for the preparation of dragline formation level.

Less advance of dragline formation level bench will interfere with the drilling and blasting operation of dragline bench, operation of excavators and transportation equipment and adversely affect the productivity of dragline as well as safety of different heavy earth moving machinery working in the dragline bench including the safety of dragline.

3.2.3.3 Favourable de-coaled cut (previous cut) for stable dumps.

De-coaled cut or the previous dragline cut should fulfil the requirements for the creation of a stable dragline dump in addition to sufficient space for the accommodation of required overburden to side cast by dragline. The dump created by dragline is the first stage dump of the internal dump over which the dumps by dumper/tipper dumping are formed in benches/layers.

3.2.3.4 Favourable blasting of dragline bench, which ensures the required fragmentation as well as strong, stable and aligned highwall.

Well-fragmented blasted material improves the bucket fill factor and reduces the bucket filling time to a large extent. Well-fragmented blasted material also reduces the bucket line breakdown, including wear and tear of the dragline bucket to a large extent.

Blasting for the dragline operation (dragline bench blast planning aspects)

As a dragline excavates overburden material by dragging the bucket over it rather than pushing the bucket into the bank, as a shovel does; hence, most care should be taken as the blasting operation is more critical for a dragline as compared to a shovel. Dragline productivity can decrease more rapidly in poorly fragmented material as compared to shovels working in the same material (Kennedy ,1990). For dragline stripping, two common blasting methods are used, these are stand-up or standard blasting and throw or cast blasting (Harder,1988). The blasting pattern is such in the stand-up method that it loosens bank materials rather than the highwall. This method provides the dragline with a stable seating on the highwall side when excavating the material. The stand-up method is economical due to the relatively lower cost of drilling and blasting (Patir and Salgir,1996). The dozer requires less levelling work, and a safer working area is provided for the dragline in this method.

Figure 3.1: A dragline at NCL mine excavating blasted overburden material from a key cut.

The second method is called throw or cast blasting. In this method, explosive energy is utilised to push the overburden into the spoil side as much as possible, as shown in figure 3.2. Hence dragline handles less volume of the overburden material. The most significant advantage of this method is increased dragline productivity (Rai et al., 2022; Ray et al.,1999). In a thick overburden bench, a lesser dragline rehandle is required as throw blasting reduces the dragline bench working level. With the above advantages, throw blasting is costly as it may increase drilling, explosive, and dragline pad preparation costs to some extent (Kennedy ,1990). Sometimes, pre-split blasting or line drilling may be used with throw blasting to dewater permeable overburden (Raina,2021; Crosby and McDonald,1983).

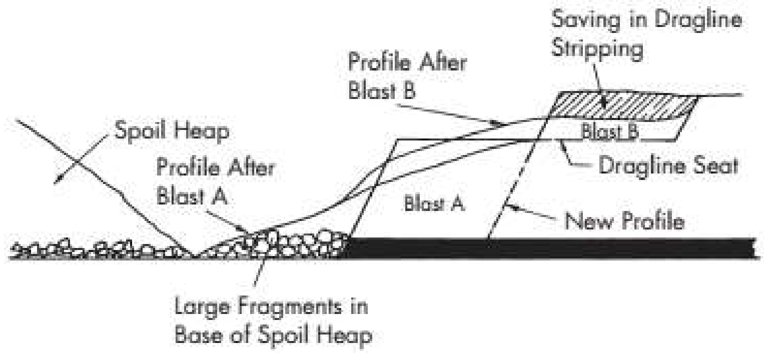

Figure 3.2: Cast blasting profile of blasted overburden bench (Darling,2011).

The height of overburden bench on which draglines are deployed remains same after blasting of intact overburden bench, because the effect of cast blasting and swell factor of overburden neutralise each other.

3.2.3.5 Dragline cut corridor of sufficient width for the deployment of a dragline in the key cut along the highwall.

Sufficient width of this corridor ensures the deployment of dragline for key cut operation by sitting along highwall. This gives the dragline the advantage to operate for key cut at a swing angle of less than 90°. Highwall dressing is also done very well from this sitting position of the dragline.

3.2.4 Micro-level operational planning

Micro-level operational planning includes the following:

3.2.4.1 Selection of most productive dragline cut parameters

A) Dragline bench height

The limiting dragline bench height is equal to the maximum permissible digging depth. As the bench height increases, there are issues with the visibility while spotting the bucket as well as difficulty in the placement of the swinging bucket at bench toe level.

The minimum bench height should not be less than one-third of the boom length of the dragline.

B) Dragline cut width or panel width

It 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. Dragline cut width is influenced by many factors like safety, the requirement of the rate of coal production, available spoil area, reach/dragline boom length, hoist and swing time, system of working (vertical tandem or horizontal tandem) and coal recovery percentage.

C) Dragline cut length strip length

Dragline cut length is the distance between the starting point of the dragline cut to the endpoint of the dragline cut. It is mainly dependent on the strike length, pit layout etc.

D) Block length/digout length

In general, the digout should be as long as possible. However, dragline size may greatly influence digout length for specific pit geometry. Block length is directly proportional to the dragline face slope angle.

E) Dragline face slope angle

The dragline slope angle should not be too small, as the dragline bucket will not be able to reach the toe of the face. It also leads to poor visibility of the bucket by the operator, which affects the cycle time and productivity.

F) Highwall slope

It is the angle that the highwall slope makes with the horizontal. The highwall slope is dependent on the angle of drilling holes. For vertical blast holes, the highwall slope angle is higher compared to the inclined holes. The highwall slope selection is decided on the basis of safety, stability as well as operational requirements.

3.2.4.2 Preparation and implementation of dig plan

One of the main objectives of developing and reviewing the operational plan is to optimise the dig methodology to achieve the safe removal of the required volume of overburden from the desired location and dumping in the desired location in the shortest possible time at the lowest unit cost. The safe and efficient means of achieving this objective requires detailed planning and scheduling and relies on good communication and coordination between the technical services department (planning), the operational engineers, supervisors and the dragline operators.

3.2.4.3 Formation of stable dragline dump

A strong and stable dragline dump foundation & complete rib less mining or coal rib extraction, as far as practicable, is ensured. The overburden cast in the void due to blasting forms the bottommost first layer over the foundation or base of the dump. The large size boulders are normally found in the bottommost first layer. Dragline dump monitoring should be done to ensure a safe dragline dump profile as per statutory requirements.

As stated earlier, the major work done by the author for this study to prepare the thesis is for tandem dragline operation. The thesis focuses on planning tandem dragline operations in Indian geo-mining conditions through software development. Tandem dragline planning may be done using a computer software-generated three-dimensional balancing diagram. The following sections describe three-dimensional balancing diagram and software development for it.

3.2.5 Balancing diagram

3.2.5.1 Introduction and definition of balancing diagram

Successful operation of large draglines is a key factor in deciding the economy of the opencast coal projects. To utilize huge capacity draglines effectively and efficiently, the balancing diagram will be very helpful. The capacity utilization of this expensive and sophisticated equipment requires proper planning with careful consideration to achieve the goal of the project. The use of large-scale draglines is a key priority in planning (Kishore and Jaiswal, 2004). Coal production, the main objective of coal mining projects, is directly impacted by the proper use of draglines. Balancing diagrams are a fundamental planning tool which can be used for the efficient utilization of draglines (Rai, 1997).

A balancing diagram is an essential planning tool that can be used for the effective deployment and operation of draglines. It is a graphical representation scheme for determining the suitable seating position of the dragline to get minimum rehandle of the dragline cuts and provides spoil geometry which determines the rate of coal exposure, rehandle volume, and cuts volume. It decides the seating position of draglines and a balanced workload for each dragline operating in tandem operation. The ultimate purpose of the balancing diagram is to achieve the targeted coal production with a minimum rehandle percentage and to decide the mode of operation (Rai, 1997). For deciding on the optimum cut width, the preparation of a balancing diagram is essential (Rai and Kishore, 1999). Dragline balancing primarily means establishing the relationship between the system’s capacity and its workload.

A dragline-balancing diagram is prepared to incorporate suitably selected machine, design, and geotechnical parameters of the quarry. Typical balancing diagrams for horizontal and vertical modes of dragline operations are shown in figure 3.4 and figure 3.5, respectively. Various important input and output parameters are also shown in table 3.1. The diagram also shows various dragline cut sections. K/C is the key cut, F/C is the first cut, F/D is the first dig, and R/H is the rehandling area. Leading and lagging dragline positions are sown by Le A1 and Le A2, respectively.

3.2.5.2 Feature and purpose of balancing diagram

The primary purpose of drawing a balancing diagram is to develop a plan that will be followed during tandem dragline operations in order to meet the targeted coal production set by the organisation/project. The balancing diagram will include the following features and purposes.

(i) It demonstrates the dragline cut sections, for example, the key cut, the first cut (next to the key cut), the first dig (next to the first cut), and the rehandled section (according to the mode of operation).

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

(iii) Rate of coal exposure (daily/monthly or annually) can be determined through it.

(iv) Workload distribution calculation 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) Deciding the scheme to be adopted for dragline operation i.,e., horizontal or vertical tandem.

(vii) Deciding the seating position of draglines so that workload should be balanced in respect of their annual productivity.

(viii) Determining the volume of overburden to be accommodated in the de-coaled area.

(ix) Balancing diagram also calculates the advancement of each dragline and hence the rate of advance of the mining operation.

3.2.5.3 Method of Drawing Balancing Diagram

The following procedure through the below steps may be used when drawing a balancing diagram.

STEP 1: Calculate the annual production of each dragline

P_H =B/C x K x S x F x M x 3600 m^3 ……………………………. (1)

Where, Ph = Hourly production (m^3)

B = Bucket capacity (m^3)

C = Average cycle time (sec.)

K = Availability-cum-utilization factor (operational factor)

S = Swell factor (Geological factor)

F = Fill factor (Combined geological and operational factors)

M = Machine travel and positioning factor (Operational factors)

Hence, Annual Production Considering 8760 hours per year

P_A = P_H x 8760 (m^3) (CMPDIL Norms) ……………………………. (2)

STEP 2: Decide Work Load on Each Dragline

There should be a proportionate amount of overburden removal by the leading (primary/independent) dragline and the lagging (secondary/dependent) dragline in terms of the annual production capacity of these draglines. Hence, the cut sections taken by each dragline should be in the same proportion as their annual capacity. Erdem (1999) described it as synchronization of a tandem dragline system as it has uniformity in the rate of linear advance of the units.

STEP 3: Assume Different Design Angles for Drawing Balancing Diagram

Depending on past experience and records, estimate appropriate angles for drawing the balancing diagram. For example,

Angle of dragline cut in highwall side - 70^0

Angle of coal extraction - 80^0

Angle of dragline cut in blasted overburden - 60^0

Permitted cutting angle of dragline dump at the height up to 35m - 40^0

(Bench slope angle for loose overburden)

Angle of repose for loose overburden - 38^0

STEP-4: Decide the Width of Cut

The width of cut should be as large as possible in order to maximize the coal exposure and minimize the percentage of rehandling. Large cut width reduce relative time losses of standby and unproductive moves of excavators. On the other hand, it should be optimized in order to obtain a greater height of the stripping bench (Rzhevsky, 1987). It is also restricted by the operating parameters of dragline i.e. digging radius and dumping radius and the system of working as vertical tandem or horizontal tandem to be adopted

(Rzhevsky, 1987). The suggested cut width is a contentious value and is a matter of subjective judgement. First, the maximum cut width is determined on the basis of the required rate of coal exposure and operating radius of the dragline. The minimum cut width is dictated by the manoeuvrability of coal loading and hauling equipment.

The maximum width of a cut with a dragline on the basis given above is calculated using the following equation:

$W = CE \times A / (P_A \times T \times D \times R)$ (CMPDIL calculation)………………. (3)

Where,

W = Cut width (m)

CE = Coal exposure (Mte/year)

P_A = Annual production of the draglines (Mm^3)

A — Cross-section area taken by both draglines (m^2)

T — Thickness of coal seam (m)

D = Density of coal (te/m^3)

R = Recovery factor

and, CE = Rate of advance of dragline x W x T x D x R (Mte)

(P_A/A= Rate of advance of dragline)

Taking into account dragline reach,

$W_{max} = R + 0.5\,B + b$ (Rzhevsky, 1987) …………………………. (4)

Where,

W_{max} = Maximum cut width (m)

R = Reach of dragline (Operating radius) (m)

B = Width of dragline travel (m) (Overall width over shoes)

b = Width of safety berm (m) (2-5m)

For deploying two shovels, minimum width of cut should be kept 60m (Rai,1997).

STEP-5 : Decide the stripping bench height of the dragline

The height above the coal seam at which the dragline is positioned is defined as the bench height. The selection of the bench height is based on numerous operational factors. The complex relationship of bench height, cut width, dragline dumping reach and dumping height, as well as material characteristics such as swell and angle of repose, greatly influence the dragline's capability to dispose of the burden of the coal. 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 the required dragline reach (Kennedy, 1990).

The height of the stripping bench is restricted by the height of the waste heap and the height of the waste heap is always restricted by the dragline dumping radius, and not by dumping height (Rzhevsky, 1985). The height of the dragline bench, considering dragline operational characteristics, is given by the following norms. Height of the bench ≤ 0.8 times of the dumping height of the dragline used (Rzhevsky, 1985). For the tandem operation of the draglines, rate of advance (linear) of both the dragline is the same so that the entire operation may continue smoothly without hindrance in the system (Erdem, 1999).

Hence, if

P_A = Annual production of either dragline (Mm^3)

T = Seam thickness (m)

CE = Required rate of coal exposure (annual) (Mte).

H = Height of dragline bench (to be determined) (m)

D = Density of coal (te/m^3)

Then, $H = (P_A \times T \times D) / CE$ ……………………………. (5)

To be more specific and precise, loss of coal and rehandling by dragline should also be considered.

Rate of the advance of either dragline per annum = P_A/ (Corresponding cross-sectional area taken by dragline)

The corresponding cross-sectional area taken by dragline is the function of bench height, which is calculated on graph paper taking into account the workload distribution principle (Step-2) where bench height is inversely proportional to coal exposure. It means as cut width increases, the rate of coal exposure also increases and vice-versa, and as the bench height increases, the rate of coal exposure decreases. Therefore, optimum cut width and bench height should be decided simultaneously on graph paper by iterative procedures till we arrive at required rate of coal exposure. It should always be borne in mind that cut width and bench height are restricted by dragline reach and dumping height, respectively. The optimization of cut width and bench height can be accomplished by applying computer graphics methods more easily and accurately.

STEP 6 : Decide seating position for key cut and key cut width

Key cut width is such that it must conform to a stable bench slope angle in a pit. The minimum width of the key cut bottom is equal to 1.5 times width of the dragline bucket (Rzhevsky, 1987).

Since the key cut is in the form of an inverted truncated cone (wedge-shaped) hence, with the increase in overburden thickness, the width of the key cut top also increases. Highwall angle (bench slope angle) should be flatter with the increase in the overburden height which means that at the height of 50m, the bench slope angle may be around 500 (Shevyakov, 1965). This would terribly increase the width of key cut top and would subsequently decrease the first cut width (next to the key cut). At this situation, the load distribution on the draglines operating in tandem would be severely disrupted (Rai, Kishore, and Nath, 2000).

The seating position of the dragline is decided by its reach, bench height and cut width taken by draglines. The seating position of the dragline may be end-cut or side-cut position for key cut development. In the end-cut position, the dragline is positioned at the end of the cut along the axis of the key cut and strips all overburden from this position. The swing angles are generally less than 900. This method is applicable where the overburden depth is relatively shallow compared with the dragline dimensions. In the side-cut position, the dragline is positioned on the spoil (decoaled area) side of the key cut and is applicable where the overburden is so deep that all of it can not be stripped without rehandling from the end-cut position. The spoil can be dumped farther away from this position. The swing angles are, however, increased up to 1200 (Chugh, 1980).

STEP-7: Deduce seating position of draglines for first cut (next to key cut), first dig (production cut) and rehandling

Depending upon technological system of face operation (whether horizontal or vertical tandem operation) and the reach of the dragline i.e., the distance between the dumping point and the axis of travel of the dragline for each cut, decide the seating position of the draglines. It should be noted here that a wider cut width is possible in horizontal tandem operation of draglines and a larger height is possible in the vertical tandem operation (Rzhevsky, 1987).

STEP-8: Calculate cross-sectional areas to be excavated by leading and lagging draglines

The cross-section area of cuts and spoil geometry is determined by trapezoidal rule, which is based on measurements and calculations. The abilities of the computer should be used to eliminate the tedious and time-consuming tasks involved in drawing the balancing diagram, such as repetitive calculations and measurements.

STEP-9: Calculate coal exposure per year

$CE = P_A/A \times W \times T \times D \times R$ (Mte/year) ……..……………………. (6)

Where,

CE = Coal exposure (Mte/year)

P_A = Annual production of the dragline (Mm^3)

A = Cross-sectional area taken by the dragline (m^2)

W = Cut width (m)

T = Thickness of coal seam (m)

D = Density of coal (te/m^3)

R = Recovery factor

Here, P_A/A = Rate of advance of dragline.

STEP-10: Calculate Rehandling Percentage

Rzhevsky (1987) defined the ratio of the volume of repeatedly excavated overburden to the total volume of initially excavated overburden as the re-excavation ratio.

Huddart and Runge (1979), define it as

Rehandle (%) = (Prime volume of material rehandled from a certain block)/(Total prime volume of block) x 100

But the definition used here is

Rehandle (%) = (Rehandled cross-sectional area)/(Excavated cross-sectional area) x 100

…….……………………. (7)

3.3 Development of computer program for the present study

The manual drawing of cut and spoil diagrams on a regular basis could become time-consuming and very expensive. This can be accomplished by applying computer methods at a relatively low cost and in less time (Pundari, 1981). The majority of the commercial packages that are presently available to facilitate dragline mine planning have been developed for specific mine properties or are limited by the coal geology and mining methods that they simulate. Many variations of dragline methods have been reported by mine planners. Since the deployment and operation of a dragline in modes different from conventional schemes will affect productivity, it will be necessary to study the effects on the production of changing the dragline operational schemes. Therefore, mining techniques (operational parameters) used in software packages have to be modified. For instance, tandem dragline operation in horizontal and vertical mode requires a change in operational schemes (deployment schemes of dragline) and therefore, mining techniques (operational parameters) used in the program have to be modified. Although programs can be specifically designed and packaged for mine planning purposes, there seems to be no universally accepted definition of what mine planning actually entails (Carter, 1989). Every program is based on some specific model or assumptions, which are limitations in general. At this point, it is appropriate to mention that not a single software package has been developed to simulate single coal seam mining with tandem dragline operation (i.e., in horizontal or vertical tandem).

Keeping in view the complex tandem dragline operations, a computer program has been developed in C++ language using OpenGL graphics API for both horizontal and vertical tandem operations (Seervi, Kishore and Verma, 2022). The computer program for generation of this graphical balancing diagram was written, taking into account the various formulae and calculations incorporating all the field parameters as revealed in the flow chart. For the development of the program, the critical parameters that influence the production of a dragline and hence coal exposure rate in tandem dragline operation were identified. The inter-relationship between these factors has been incorporated into the program (Seervi et al.,2022).

In tandem operation of draglines, several decision-making parameters, such as geo-mining, machine and operational parameters, need to be considered. It is worth noting that the tandem operation of draglines calls for a very intricate and comprehensive study and analysis. With this in mind, the balancing diagram (Cross-sections proportionate to draglines productivity) is a very important tool for planning tandem dragline operations, while coal exposure and rehandle percentage are important indices for appraising the performance of draglines (Rai et al., 2001).

In order to alleviate the manual drawing of the balancing diagram and carry out volumetric calculations quickly and precisely to select the best alternative, the computer graphics program has been developed for the purpose. This program can handle any variable parameter influencing tandem dragline operation like geo-mining, machine (equipment specifications) and operational. This has been achieved by developing a program that follows the same basic steps as those taken when planning is carried out for the manual preparation of the balancing diagram. The planner (engineer) will control the planning operation at every stage. All actual decision-making, such as the determination of optimum bench height (stripping bench height), cut width, workload distribution on each dragline (To maintain uniform advancement of both draglines), specification of the cut dimensions (cross-sections), coal exposure rate, rehandle percentage etc. is carried out interactively at the computer terminal.

As stated earlier, every software is based on some assumptions or models and has some limitations in general. Some assumptions are as follows:

(i) Average productivity of each dragline is considered taking into account the average cycle time for dragline deployment in tandem operation.

(ii) Overburden thickness overlying the coal seam is constant throughout, i.e. the coal seam and the overburden layers are having the same gradient (As pre-stripping with shovel-dumper combination and overburden height left for dragline stripping is actual practice) in the mines where case studies have been done.

(iii) The swell factor, fill factor, availability cum utilization factor and machine travel cum positioning factor are considered in the average productivity of the draglines. These factors may vary according to geology, machine and method of operation and are almost constant for the same mines for the same method of operation (Appendix - 1).

(iv) Reach consideration is taken into account in the end-cut position.

(v) Angle of repose (spoil dump) and various cut angles are considered constant (according to past experience and various rules of thumb based on operating experience).

(vi) Coal rib is left for full seam thickness.

(vii) Single coal seam is considered in the package.

(viii) Draglines operate in tandem with the extended bench method.

The inputs of the program, as given in table 3.1, are factors associated with geo-mining (seam thickness, seam gradient, overburden height, the density of coal), dragline specification (annual productivity and reach of both the draglines) and operational parameters (cut width, stripping bench height, utilization, coal recovery etc.). The output is in the form of balancing diagram (as shown in figures 3.4 and 3.5) generated on the screen with all information and calculations like coal exposure, rehandle percentage, cross-section area of key cut, first cut, first dig and rehandle.

Table 3.1: Input design parameters in the program.

Sl.No.	Main Parameters	Sub-Parameters	Particulars
1.	Geology	Seam Thickness, m	Variable
		Seam Gradient, degree	Variable
		Density of coal, t/m^3	Variable
		Annual productivity of leading dragline Mm^3	Variable
2.	Dragline specification	Reach of leading dragline, m	Variable
		Annual productivity of lagging dragline, Mm^3	Variable
		Reach of lagging dragline, m	Variable
		Bench height, m	Variable
		Cut width, m	Variable
		Bench slope angle for intact ground, degree	Fixed (60)
		High wall angle, degree	Fixed (70)
		Bench slope angle for loose ground, degree	Fixed (40)

3.	Pit Design (operational)	Angle of repose, degree	Fixed (38)
		Key cut bottom width, m	Fixed (5)
		Berm (rehandling area), m	Fixed (10.5)
		Berm (lower bench in vertical tandem), m	Fixed (20.0)
		Coal rib, m	Left for full height
		Dragline availability cum utilization, %	Variable
		Coal recovery, %	Variable
		Angle of coal extraction, degree	Fixed (80)

Apart from the above input parameters, some more input parameters like swell factor, fill factor and machine travel cum positioning factor are utilized in calculating annual production keeping in view each dragline operating in tandem operation. The methodology adopted in the program is the average annual production rate approach of draglines. Such approach has been considered by (Hrebar and Cook, 1987) to estimate dragline productivity in the interactive computer program. The input and output parameters of the program in the form of a block diagram are shown in figure 3.3. The computer program was written considering various formulae and cross-section area calculation procedures incorporating the required field parameters, as seen in the block diagram.

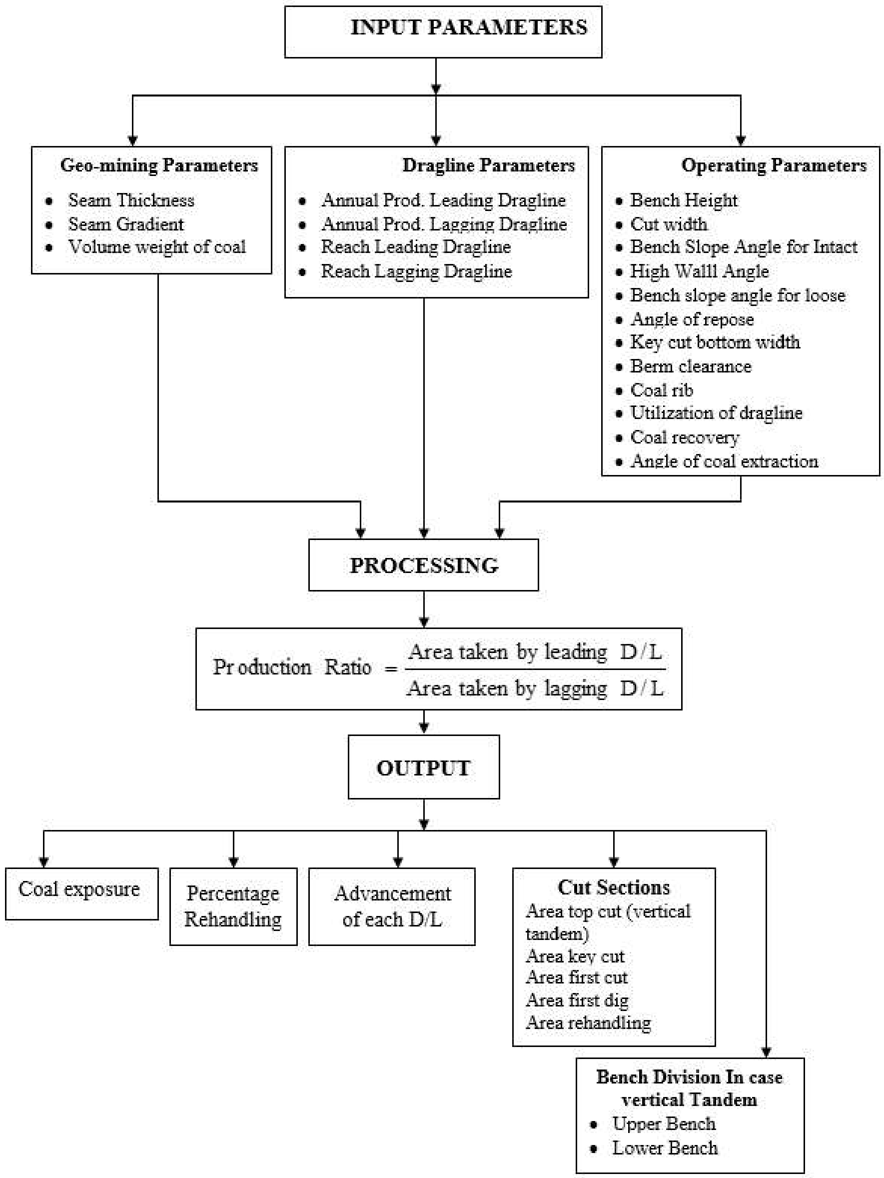

Figure 3.3: Block diagram of input and output parameters of the program.

According to the requirements of two different modes of tandem operation, i.e., horizontal tandem and vertical tandem, two programs have been developed since a single program cannot cater to both. Slight differences exist between them due to stripping bench division. In vertical tandem operation, stripping bench division requires that the cut sections taken by each dragline should be in the same proportion as their annual production. This has been done by an iterative procedure. The main contribution of both programs is to optimize the stripping bench height and cut width by varying these parameters till we obtain the required rate of coal exposure. The output is displayed as shown in figure 3.4 and figure 3.5 in the form of a typical three -dimensional balancing diagram. Research methodology of this study is shown through figure 3.6.

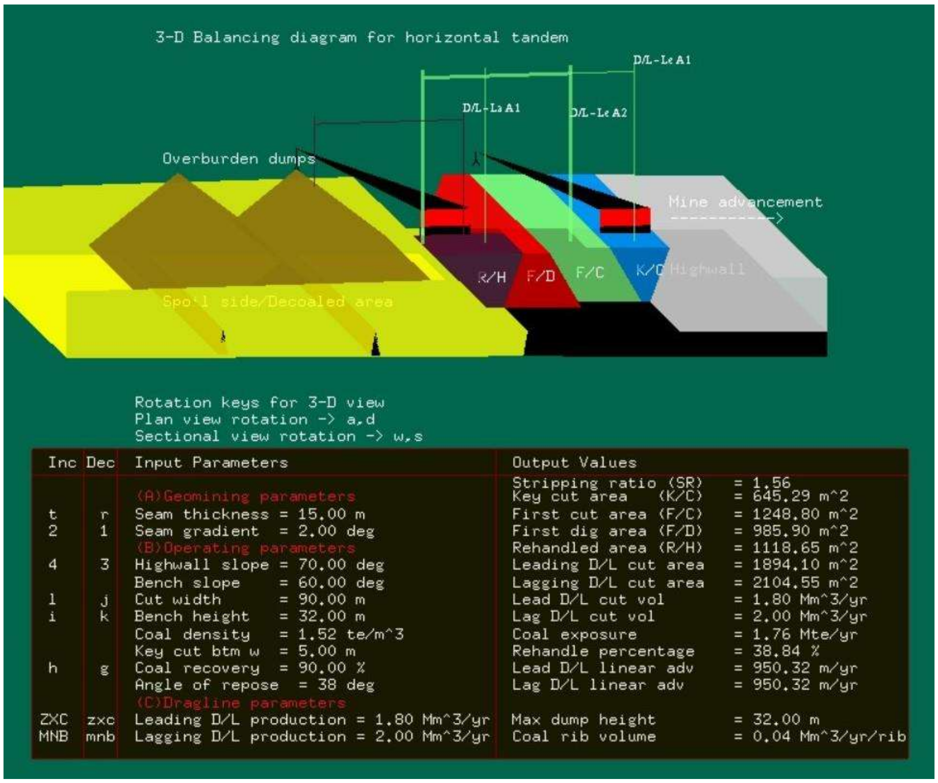

Figure 3.4: A typical 3-D balancing diagram for horizontal tandem.

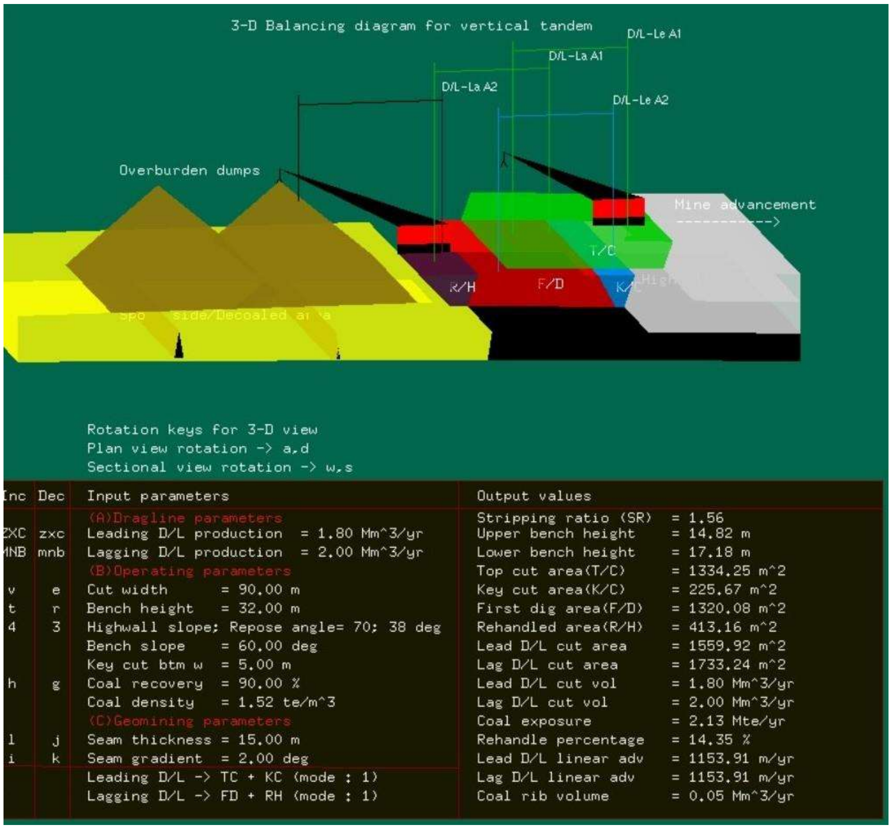

Figure 3.5: A typical 3-D balancing diagram for vertical tandem.

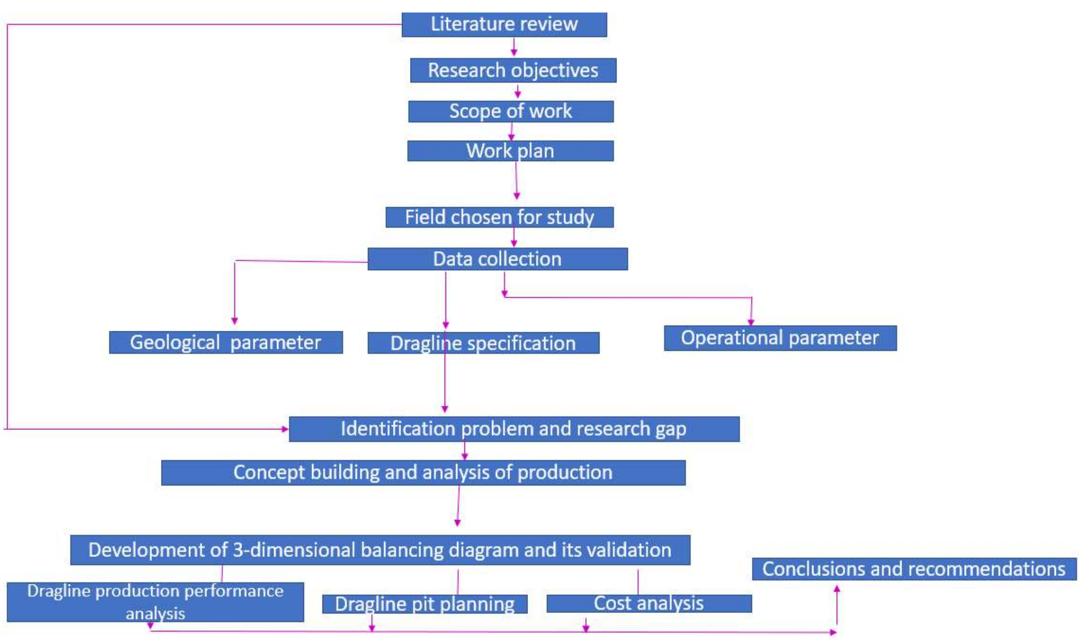

Figure 3.6: Research methodology.