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STABILITY ANALYSIS OF OVERBURDEN INTERNAL DUMP MATERIAL OF AMLOHRI OPENCAST COAL MINE, INDIA

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ABSTRACT

This paper deals with geo-technical data generation and stability analysis of internal overburden (OB) dump material of dragline operated Amlohri opencast coal mine of Northern Coalfields Limited (NCL), Singrauli, India. This paper also presents the optimum and safe combination of height and slope of dragline dumps of Amlohri opencast coal mine.

Keywords: overburden dump, slope stability, opencast mining, shear strength parameters.

1. INTRODUCTION

The increased demand of coal, as a source of power, owing to rapid industrialization and population explosion, has forced the opencast coal mines to turn toward higher mechanization for extraction of coal and removal of O/B as well (Rai*et al.* 1999) [1]. Amlohri opencast coal mine of Northern Coalfields Limited (A subsidiary of Coal India Limited) is bounded by latitude $24^0$07'30" to $24^0$09'30" North and longitudes 82^0 34' 30" to 82^0 36' 30"East located in Singrauli district of Madhya Pradesh, India (EAC Minutes, 2014) [2].Itcovers an area of 2175 ha and exploits the coal depositsoccurring within the south-western part of Moher sub-basin, the North Eastern unit of Singrauli Gondwana basin i.e. the northern entity within South Rewa Master Basin.

Amlohri opencast mine has a total geological reserve of 328.55MT out of which the mineable reserve is that of 314.46MT (EAC minutes, 2014) [2]. Amlohri mine comprises of four seams i.e. Purewa top, Purewa bottom, Turra top and Turra bottom, separated by partings of sandstone, shale and carbonaceous shale (Table-1, Figures 1 and 2). These partings constitute the waste rock overburden material that has to be extracted in order to obtain the coal. The total volume of these overburden/waste rock in Amlohri opencast mine is about 1400Mm^3, with a stripping ratio of 4.18m^3/t. The OB overlying the Turra top and Turra bottom seams are being excavated using dragline and rest of OB in between the Purewa top and Purewa bottom seams and the top soil above the Purewa top are being excavated using shovel-dumper combinations (Figure-1).

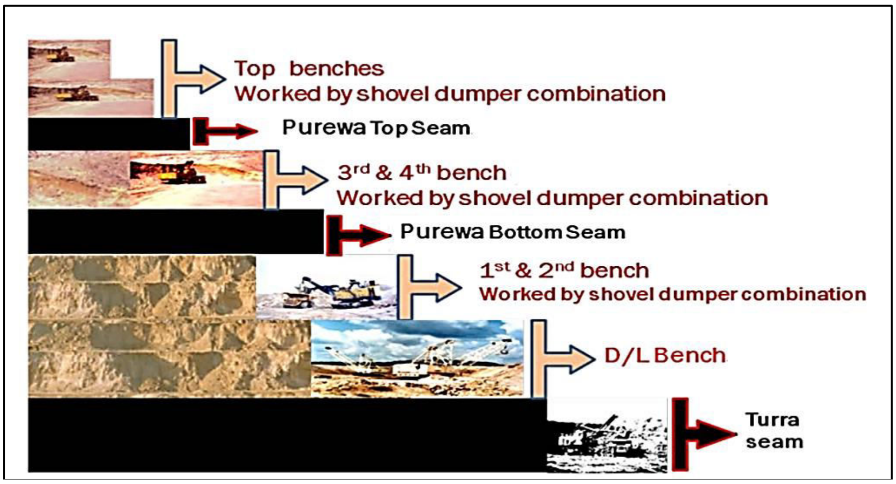
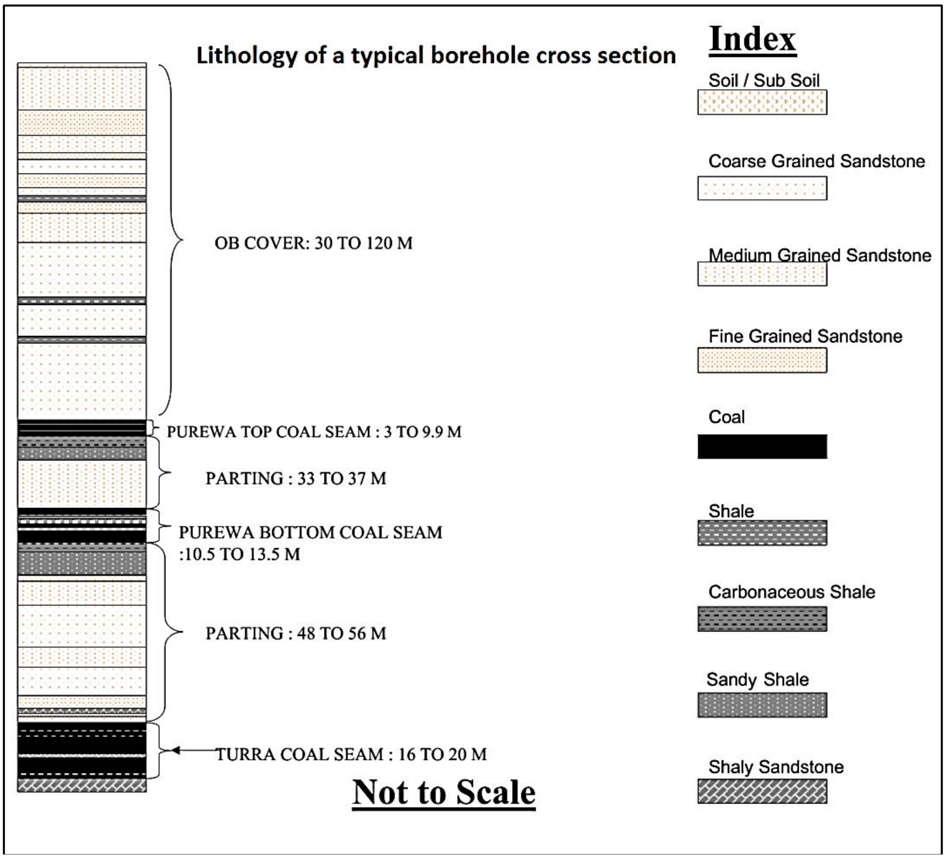


Figure-1. Disposition scheme of HEMM along the geological cross section.



(Source: NCL) [3]

Figure-2. Lithology of typical borehole cross section in Amlohri.

The huge amount of overburden material thus generated has to be accommodated in the limited quarry space in the form of internal dumps. This further result in formation of steep dragline dumps standing just at limit equilibrium condition and is animpending slope failure zone. Also a number of overburden dumps from successive strips had shown large instability in recent years in this mine. Overburden dumps in opencast coal mining operation experiences two types of failure surfaces (Hebil, 1986) [4]:

- Circular failure surface
- Circular-cum-planar failure surface (Cambell, 1986) [5].

Therefore it is important to carry out a thorough geo-technical investigation of these existing steep overburden dumps, so that case of slope instability could be minimized.This paper deals with stability analysis and determination of economic design with optimum and safe height and slope combination of such internal dragline dumps.

2. OVERBURDEN DUMP MATERIAL CHARACTERISTICS

Three discrete types of overburden dump materialexist in the above mentioned mine and details are appended below:

a) OB material, formed by dragline, is a mixture of fragmented soft whitefine to coarse grained sandstone, shale, carbonaceous shale and very softclay.

b) OB material, formed by shovel-dumper combination, is a mixture of fragmented soft sandstone and excavated soil. The shovel excavates the material and is being transported by using dumpers. The loaded dumper moves along the haul road and dumps the excavated material above the dragline dump.

c) Interface material at the base of dragline dump under submerged conditions is a mixture of fragmented white sandstone, very soft sandstone, coal and pre-dominantly excavated yellow clay lying under water. This interface material on an inclined quarry floor is

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ARPN Journal of Earth Sciences

the plane of weakness at the foundation of dump, which is one of the major causes of failure of dump.

2.1 Geo-technical parameters

The collected dump and interface material were individually compacted in large shear box apparatus (Figure 3) (40cm X 40cm) consisting of two halves. The thickness of the sample within the large box is that of 15cm. The bottom half is fixed to a base plate and top half is movable, being supported on rollers on two sides at the junction between the bottom and the top halves. A uniform normal stress was applied on the sample throughout the test using a vertical digitally controlled jack and the shearing load was digitally applied trough a horizontal jack. The applied normal loads, shear loads and the horizontal displacement i.e. displacement due to shear load were digitally recorded online in the computer system.

These materials were compacted at the level of stress actually existing within the dump mass as the dump material could only be collected from the dump surface during sampling (Sengupta*et al,* 2014) [6].The dump and the interface materials were subjected to a pre-compression pressure for total dump. (For example, for a dump height of 100 m comprising of dump material with bulk density of 20kN/m^3, the average pre-compression 50X20 pressure is of the order 1000 kN/m^2). The pre-compression pressure was maintained till the vertical compression ceased the sample was then subjected to a pre-determined normal stress and sheared. The test results are given in Table-2.

The different geo-technical parameters determined where cohesion, angle of internal friction and bulk density.

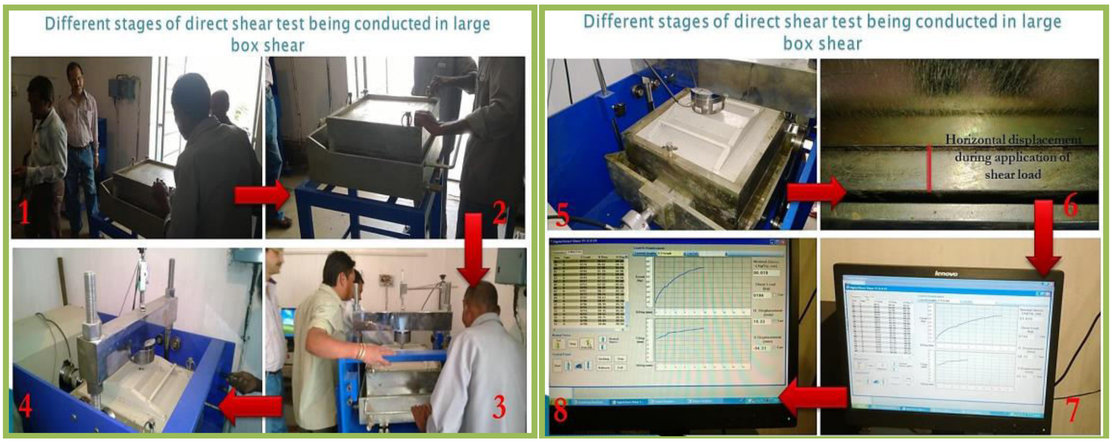

Figure-3. Determination of shear strength parameters using Large Box Direct Shear Apparatus.

2.2 Hydro-geological parameters:

Due to presence of 15m water table at the base of dump, there will be both hydrostatic pressure and seepage force of water. A schematic diagram showing the application of two forces has been shown in Figure-4b, the details of which are given below:

a) The buoyancy force of water, which is equal to the product of unit weight of water and volume of submerged overburden dump material falling within the failure mass.

b) The seepage force of water which is equal to the product of unit weight of water and volume of submerged overburden dump material falling within the failure mass and gradient of seepage line.

2.3 Blasting and Seismic effect on dump material

Ground vibration on account of earthquake and blasting, hampers the stability of dragline dumps.As per seismic map of India, Amlohri opencast mine falls under seismic zone-II with horizontal seismic coefficient of 0.02.The peak particle velocity and frequency of blast wave within the internal dragline dump due to blasting in the quarry face adjacent to the dump mass, was measured. The blasting co-efficient was found to be 0.01.(Roy, 1997)The seismicity and blast vibration force is computed by multiplying 0.03 (0.02 + 0.01) with dead load of the dump mass.

2.4 Factor of safety

The slope stability analysis in all the above mentioned dragline dump has been made for a stipulated factor of safety of 1.2 using both Fellenius as well as Bishop's Simplified method.

3. METHODOLOGY OF STABILITY ANALYSIS

The determination of safe and economic combination of slope geometry has been made in a sequence, i.e., given below (Figure-4(a)-4(d)):

a) Determination of critical slips surfaces and the strength parameters of the dump and interface material.

ARPN Journal of Earth Sciences

b) Several combination of height and slope of overburden dump were taken for analysis, a number of possible failure surfaces were shaped. Factor of safety of each failure surface was calculated and compared to find in order to find the minimum factor of safety.

c) The mode i.e. circular or circular-cum-planar that corresponds to minimum factor of safety is the most potential failure mode.

d) Stipulated factor of safety 1.2 was compared with the factor of safety of most potential failure surface. Any probable slope design i.e. height and slope combination of dragline dump was considered with a factor of safety of 1.2 and not less than that.

e) Water table profile within the dump mass behind the toe upto 60m has been calculated.

f) At 60m behind the toe of the dump, the water table was found to be 14-15m (Figure 4a and 4d).

g) A stability analysis of both, dragline dumps with coal rib from coal floor towards dragline dump and dragline dump without coal rib was also under taken. This was done in order to ascertain the increase in resisting forces acting on the dragline dump slopes in presence of coal rib.

h) Also a stability analysis with increased dragline dump height and flatter slope angle was undertaken in order to assess the feasibility of higher dragline dumps.

In order to calculate factor of safety, the forces acting on the failure surface was considered as follows:

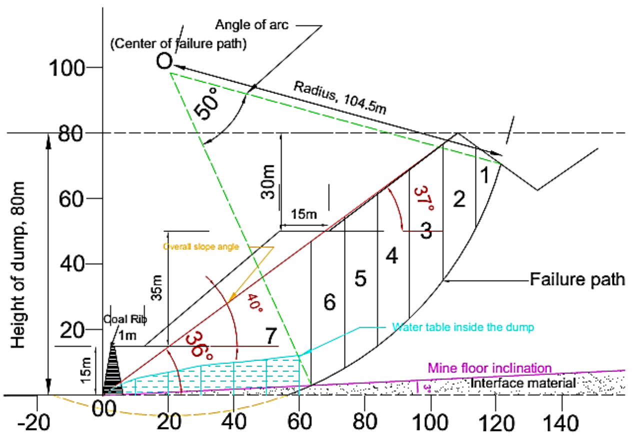

Figure-4(a). Stability analysis of existing dragline dump, Amlohri opencast mine

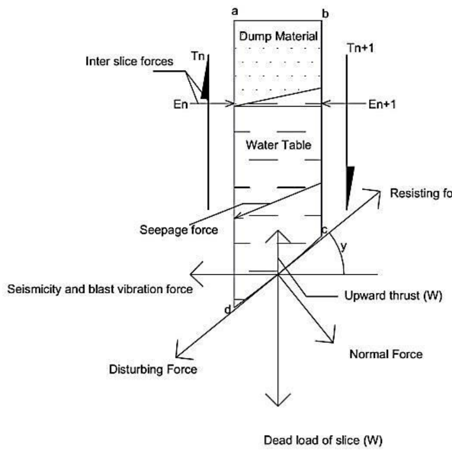

Figure-4(b). Free body diagram of an individual slice

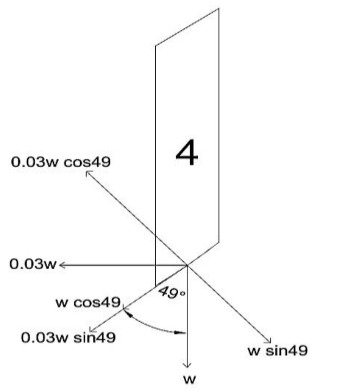

Figure-4(c). Forces acting on a slice due to dead load and seismicity.

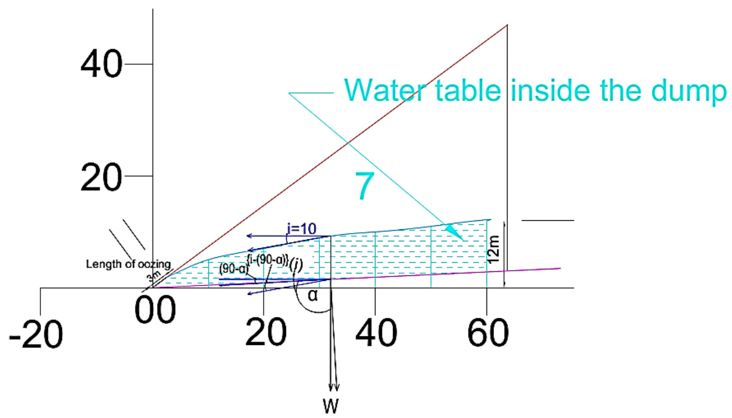

Figure-4(d). Water table formed inside the dragline dump mass.

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Disturbing force (due to dead load) = $(W-W_1) \times \sin(y)$ (1)

Where,

W and W_1 denoting the dead load of rock mass and buoyancy force respectively and y represents the angle between tangential force and normal force.

Disturbing force (due to seismic effect) = $A\,(W-W_1) \times \cos(y)$ (2)

Where

A is the seismic coefficient.

Disturbing force (due to seepage force) = $W_1 \sin(x) \times \cos(y-x)$ (3)

So, total disturbing force will be the summation of all above mentioned disturbing forces i.e., due to dead load, seismic force and seepage force.

Total disturbing force = $\{(W-W_1) \times \sin(y)\} + \{A(W-W_1) \times \cos(y)\} + \{W_1 \sin(x) \times \cos(y-x)\}$ (4)

Total frictional force = $\{(W-W_1) \times \cos(y) - A(W-W_1) \times \sin(y) - W_1 \sin(x) \times \cos(y-x)\} \times \tan$ (angle of internal friction) (5)

The cohesive and resisting forces are derived as;

Cohesive force = (area of failure surface) X (cohesion of material) (Figure-3) (6)

Resisting force = Frictional force + cohesive force + shearing resistance by coal barrier (7)

A coal rib as a barrier with minimum width of 25m at the base and 4m at the top is left at the toe of dump along the entire strike length of the quarry (Figure-2).

The factor of safety (FoS) by Fellinius Method (FS_1):

FoS (FS_1) = Resisting force/ Disturbing force (8)

But in Fellenius Method, no inter slice forces are considered in the calculation and henceforth, more accurate factor of safety for the dump slopes was also determined usingBishop's Simplified Method.

FS_2 = Resisting Force/ m(Disturbing Force) (9)

FS_2 = {cohesive force + $(W-W_1)$ tan (angle of internal friction)} ÷ M $\{W \sin(y) + A\,(W-W_1) \cos(y) + W_1 \sin(x) \cos(y-x)\}$ (10)

Where

FS_2 = Factor of safety calculated by Bishop's Simplified Method

$$m = \cos(90-y)\left[1+\frac{\tan(90-\propto)\tan\emptyset}{FoS\,\{by\ fellenius\ Method\ (FS1)\}}\right] \quad (11)$$

FS_3 = {cohesive force + $(W-W_1)$ tan (angle of internal friction)} ÷ M $\{W \sin(y) + A\,(W-W_1) \cos(y) + W_1 \sin(x) \cos(y-x)\}$ (12)

Where,

$$m = \cos(90-y)\left[1+\frac{\tan(90-\propto)\tan\emptyset}{FoS\,(FS2)}\right] \quad (13)$$

Similarly, by repetitive convergence method FSn was made equal to FSn-1, where FSn is the factor of safety by Bishop's Simplified Method.

4. RESULTS

With the above geo-engineering, hydro-geological parameters, various combinations of height and slope of dragline dumps and the corresponding factor of safety obtained by stability analysis are tabulated below in Tables 1 and 2:

Notation

- **H** = Overall height of overburden dumps with respect to horizontal plane passing through dump toe.
- **C_2** = Cohesion of dump material.
- **$Ø_2$** = Angle of internal friction of dump material.
- **C_3** = Cohesion of interface material.
- **$Ø_3$** = Angle of internal friction of interface material.
- **γ_2** = Bulk unit weight of dump material.
- **I** = Mine floor inclination.
- **D** = Depth of water within the dump.
- **C_4** = Cohesive force between coal rib and its floor.
- **$Ø_4$** = Frictional resistance between coal rib and its floor.
- **Ag** = Ground acceleration generated in dump mass in case of earthquake.
- **L** = Overall slope angle of dump with respect to horizontal plane passing through dump toe.

Table-1. Water table profile within dump mass behind the toe.

S. No.	Length within the dumpmass (m)	Height of water table (m)
1	10	5
2	20	8
3	30	10
4	40	11
5	50	13
6	60	14

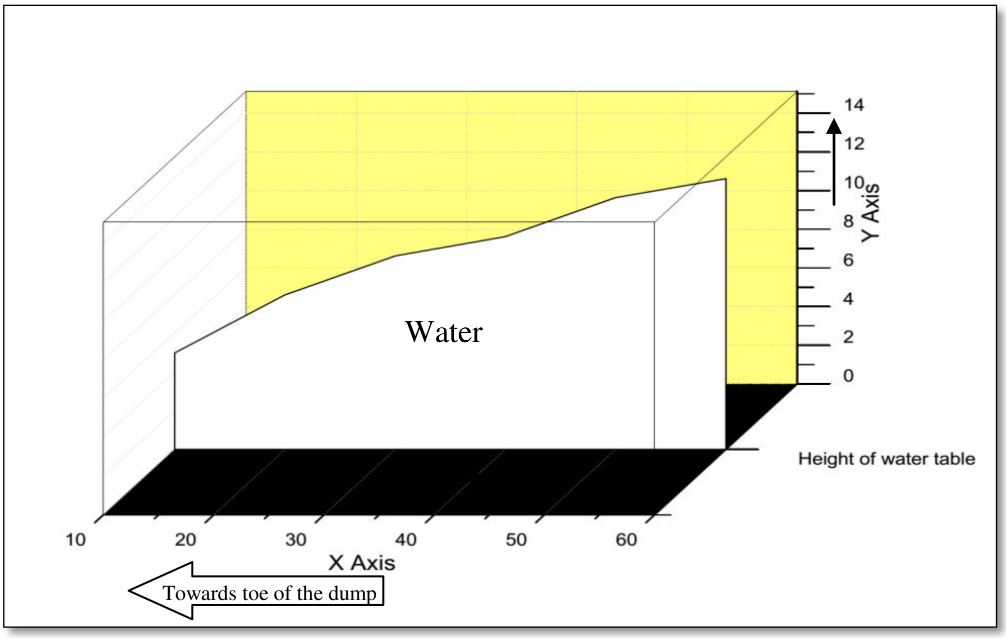

Figure-5. Graphical representation of water level within the dump mass.

Table-2. Geo-engineering parameters.

H (m)	**C_2** (KN/m^2)	**$Ø_2$** (deg)	**C_3** (KN/m^2)	**$Ø_3$** (deg)	**γ_2** (KN/m^3)	**I** (deg)	**D** (m)	**C_4** (KN)	**$Ø_4$** (deg)	**Ag** (m/sec^2)	**L+i** (deg)	**FoS Stipulated**	**FoS actual**
80	65	27	30	20	22	3	13	13000	9717	0.30	33+3	1.05-1.10	1.12
50	65	27	30	20	22	3	13	13000	9717	0.30	33+3	1.05-1.10	1.20
87	65	27	30	20	22	3	13	13000	9717	0.30	32+3	1.05-1.10	1.10

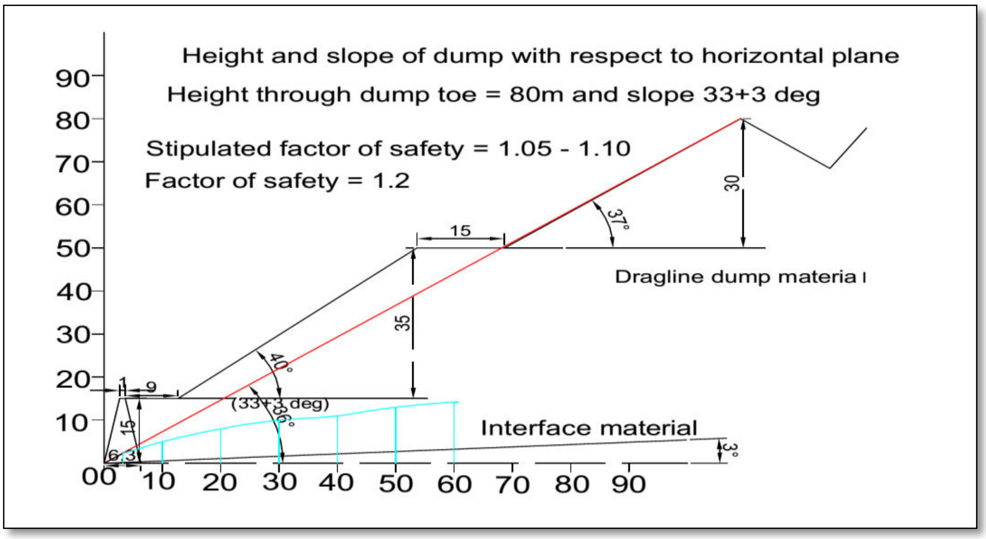

Figure-6. Optimum height and slope combination for dragline dump profiles with 80m height and overall slope angle 36^0.

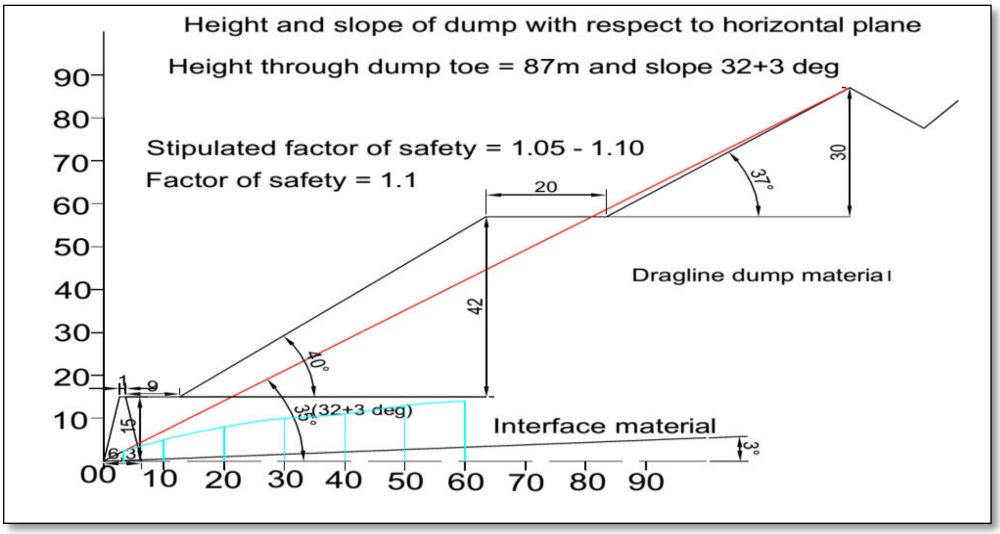

Figure-7. Optimum height and slope combination for dragline dump profiles with 87m height and overall slope angle 35^0.

5. CONCLUSIONS

Dragline dump is formed on interface material that is a de-coaled floor with slushy layer submerged under accumulated water. That slushy material is a mixture of fragmented soft white fine to coarse grained sandstone, shale and carbonaceous shale. The height of slushy material and oozing length of accumulated water at the base of dump is 15m and 4m respectively, accumulated water is upto 60m behind the toe within the dump mass.As per this study more stable and feasible profiles for internal dragline overburden dump i.e. height and slope combination in Amlohri opencast mine are as follows:

Profile 1: Height of the dump 80m, having an overall slope angle of 36^0(33+3 deg), where 3^0 is mine floor inclination.

Profile 2: Height of the dump 87 m, having an overall slope angle of 35^0 (32+3).

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