Analytical Model To Determine Of The Optimal Block Size In The Block Caving Mining Method

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ABSTRACT

Nowadays along with population growth, industry development, consumption of mineral resources and the fact that the reserves on hand are running out, the depth of surface and underground mines for further exploitation are increasing. The block caving method for low-grade and large-scale deposits has shown a growing rate of application. The dimensions of blocks are one of the most important parameters which should be taken into account since it has been proved to have a great deal of effect on technical issues such as commencement of caving and mine design. In this study, firstly for the purpose of facilitation, some assumptions were considered and having used these assumptions for estimation of optimized length and width of block, a relationship based on rock mechanics and physics was explored and finally, it was transformed into an inequality. Solving this inequality provides us with the optimized length and width of the block. The explored relationship was analyzed and the graphs thereof were drawn. The analysis showed that mineral thickness and density had the least effect. The impact of hydraulic radius on caving showed that square blocks are best suited for caving. While using square blocks, the relationship becomes a quadratic inequality with one unknown which is easy to be solved. Proved with a slight decrease in the safety factor of about 5%, we can increase the block length about 2 times while safety remains unchanged. Considering the capacity of transportation systems and the production of the interfaces, presented that the block length is measured in terms of production. Regarding to provide relationship for the production and manufacture of the blocks and parameters. Gangue Density and Distance have the most negative impact on the block length and fallowed by the Safety Factor and Width/Length (Y/X) and Depth respectively. Angle of Internal Friction and Ore Density and Work Hours have the positive impact on the block length. Ore Hight and Ore Density and Job Efficiency have the least impact on the block length. In 90% of cases, the block length is between 32.9 m and 91.3 m. It is clear from the graph that approximately 60 m for block length has the most Probability Density which is more than 0.025. In terms of transportation, in 90% of cases, the block length is between 85.76 m and 109.24 m and approximately 97 m for block length has the most Probability Density which is more than 0.057. In the simulation by Phase2 software, the results compared with the different modes of the block. In blocks of 55, 60 and 65 m, Total Displacement, Yielded Elements (percent) and Yielded joints reached to a satisfactory condition and perfect caving will occur. In block 70m, these values reached to maximum. As the purpose of this paper is obtaining the optimal block length Thus, blocks which are between 55 and 65 meter, can represent the optimal size.

KEYWORDS

Block-caving, Cavability, Block Size, Length and Width of the Block, the Optimal Block Size in Block Caving Method

INTRODUCTION

In classification of underground mining methods, caving methods are regarded as high production methods. These techniques are usually very costly and need great preparation. However, the high production rate of these methods, make them cost-effective ones. The Block Caving method is an example of these methods, which is similar to surface mining methods in terms of production, in that the cost of production is low. Block caving is described by Laubscher (1994) as the lowest cost underground mining method provided that the extraction layout is designed to suit the caved material and the draw horizon can be maintained for the life of the draw [5]. Wide range of minerals including Gold, Copper, Diamonds, Iron and Nickel are produced using this technique in countries such Canada, Australia, USA, South Africa, Sweden, Zambia and China.[8]. The production scheduler aims to maximise the Net Present Value (NPV) of the mining operation whilst the mine planner has control over the development rate, vertical mining rate, lateral mining rate, mining capacity, maximum number of active drawpoints and advancement direction [9]. Block caving is based on material flow as a result of gravity. Northparkes was the first mine in Australia to use a variation of the cost-effective block cave mining technique in its underground operations. Northparkes is currently mining its third block cave mine. Block caving has allowed them to achieve very low mining costs and a high productivity by industry standards, mainly through the application of efficient automated material handling and comminution systems that minimises ore re-handle, including high speed electric load haul dump units, jaw-gyratory crushers, high-speed conveyors and shaft hoisting systems [1]. In general, three exploiting systems are assumed for this method

which are: Block Caving system, Panel Caving system and Mass Caving system. Mineral mass in the block system is divided into a number of square or rectangular blocks. Mineral mass in the panel system is divided into several large panels and in the mass system, all of the minerals are considered to be a large block and operation of extracting and undercut begins from one side. One of the most important factors to consider in block caving systems is the size of the blocks (length & width). The reason for this is the fact that the length and width of the blocks have a great impact on technical issues such as the design of the mine and the commencement of caving. By calculating the proper length and width for a block, we might achieve an appropriate caving and draw in this method. The optimum block length and width are the length and width for which it has always done and will not stop caving.

DESCRIPTION

This article aims to obtain the optimal width and length needed for the caving to be carried out in a way that it does not get interrupted and stopped.. In order to achieve this, forces that are applied when performing caving, should be higher than the opposite forces. The influence of dimensions of the blocks at the start of the caving makes the importance of this study two-fold. Because the weight of the block − which is an intensifier of caving − is a function of the volume of the block and the surface area of undercut and the fact that the surface area is a function of the length and width of the block. As expected, any unsupported rock mass will cave if it is undercut over a sufficient area. Caving occurs for two reasons – gravity and the stresses induced in the crown or back of the undercut or cave. The mechanisms by which caving occurs will depend on the relationship between the induced stresses, the strength of the rock mass, the geometry and strengths of the discontinuities in the rock mass [2]. Therefore, it is important to obtain the right dimensions of the block. On the other hand, if the dimensions are too big, it causes the blocks to degrade rapidly and therefore make it unstable. This is due to the natural shape of the rock mass and fractures scads and joints and cracks in the system. Furthermore, a large surface area will cause the pressure on the roof of the tunnel to be increased and therefore make it unstable. The block size must be chosen in a way that provides safety and also timely caving of blocks without stopping.

ASSUMPTIONS

The first step to achieve the above aim is to make some assumptions. Based on these assumptions, in order to perform the caving method, the original relationship has been proposed. This relationship simply states the caving condition. Then, known influencing factors and boundary conditions were added to the equation. Sensitivity analysis was then carried out using the MATLAB software in order to analyze the obtained relationship.

The assumptions are as follows:

1. There is only one rectangular block length (x), width (y) and height (z) in the mine (Figure 1).
2. The whole block is assumed to be integrated. The rules of physics have been used to solve the equations. Figure 2 shows the way in which these rules have been applied in order to balance the forces involved.
3. All the results are related to the final shape of mines after fracture and caving at the end of the mine's life.
4. It is assumed that the subsidence has taken place at the end of the mine's life. Subsidence spreads in the long term with a slope of 45 degrees from the extracted blocks to the ground surface. Figure (1) illustrates this.
5. Fraction of the rectangular block is mineral with γ_1 density and the remaining is composed of $\gamma_2$density.

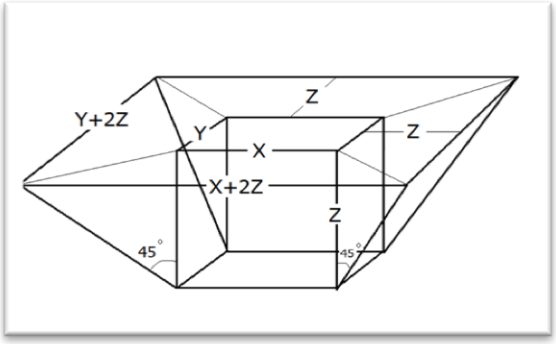


Figure 1- Three-dimensional scheme of a Block

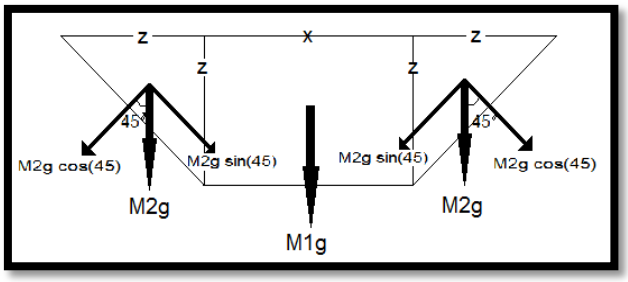


Figure 2- Vertical cross section of the block and the balance of forces

CALCULATION

In order to perform the caving method, the weight of the frustum (figure 1) has to be more than the cohesion forces that exist on the four sides of the block which prevent it from falling. This is because apart from the weight of the block, the tailings above and around the block, there is no force to contribute to the collapse and caving is determined only by gravity. Therefore, in order to cave the block we have:

$$W_o + W_c = W > \sum_{i=1}^{4} F_i \quad (1a)$$

$$F = C + \sigma_n . tan(\varphi) \quad (1b)$$

W_o: Weight of the block or mineral
W_c: Weight of the waste that is on the block
C : Cohesion
σ_n : Horizontal Normal
φ : Angle of internal friction
F : Shear Strength

In the next step, the variables are added to equation (1). Block volume was calculated using geometric relationships; its density was multiplied to obtain the total weight of the block and the destructive forces. In order to make the dimensions equal, the cohesion factor was multiplied to the surface which is applied to. The weight of the block is then used instead of the horizontal normal stress in order to convert all the relationships so that they are written according to Newton's laws. The above equation could now be modified as follows:

$$W > \sum_{i=1}^{4} F_i$$

$$\boldsymbol{x.y.h.\gamma_1 + x.y.\gamma_2(z-h) + 2\left(\frac{z^3}{3}+\frac{x.z^2}{2}\right).\gamma_2.\frac{\sqrt{2}}{2} + 2\left(\frac{z^3}{3}+\frac{y.z^2}{2}\right).\gamma_2.\frac{\sqrt{2}}{2} > 2\left[\sqrt{2}C.z.(z+x) + \left(\frac{z^3}{3}+\frac{x.z^2}{2}\right).\gamma_2.\frac{\sqrt{2}}{2}.tan(\varphi)\right] + 2\left[\sqrt{2}C.z.(z+y) + \left(\frac{z^3}{3}+\frac{y.z^2}{2}\right).\gamma_2.\frac{\sqrt{2}}{2}.tan(\varphi)\right]} \quad (2)$$

After simplification, the above equation can be written as follows:

$$\boldsymbol{x.y\big(h\gamma_1 + \gamma_2(z-h)\big) - \left\{\begin{matrix} 2\sqrt{2}C.z(2z+x+y) + \\ \sqrt{2}.\gamma_2\left[\frac{2z^3}{3}+\frac{z^2}{2}(x+y)\right](\tan(\varphi)-1)\end{matrix}\right\} > 0} \quad (3)$$

x : Block length
y :Block width
h : Thickness of the ore
z :Depth
γ_1: Mineral density
γ_2: Gangue density
φ : Angle of internal friction
C : Cohesion

This relationship is an inequality with two unknowns. Therefore, we need to define a condition in order to solve it. In the block caving method, it is always tried to design blocks are square shaped. However, rectangular blocks tend to be better designs for other applications, including transportation and draw points. Sometimes, the store also provides a way to design a rectangular block that is economical. Therefore, in order to solve the inequality, the ratio of width to length was assumed to be a constant parameter that depends on the designer's choice. Obviously, whenever this ratio becomes zero, the block is squared shape and the hydraulic radius is increased thus the caveability is increased as a result. The hydraulic radius is a term used in hydraulics and is a number derived by dividing the area by the perimeter. The hydraulic radius required to ensure propagation of the cave refers to the unsupported area of the cave back, that is, space into which caved material can move. No pillars can be left and caved material must be removed [4]. When the hydraulic radius (K) equals to 1, then caveability is at maximum.

$$k = \frac{y}{x} \rightarrow y = k.x$$

$$x.y\big(h\gamma_1 + \gamma_2(z-h)\big) - \left\{\begin{matrix} 4\sqrt{2}C.z^2 + 2\sqrt{2}C.z(x+y) + \\ \frac{2\sqrt{2}}{3}.\gamma_2.z^3.(\tan(\varphi)-1) + \frac{\sqrt{2}}{2}.\gamma_2.z^2(x+y)(\tan(\varphi)-1)\end{matrix}\right\} > 0 \quad (4)$$

$$k.x^2\big(h\gamma_1 + \gamma_2(z-h)\big) - \left\{x.\left(\frac{1+k}{2}\right)\left[4\sqrt{2}C.z + \sqrt{2}.\gamma_2.z^2(tan(\varphi)-1)\right]\right\} - \left[4\sqrt{2}C.z^2 + \frac{2\sqrt{2}}{3}.\gamma_2.z^3.(tan(\varphi)-1)\right] > 0 \quad (5)$$

And in each square are equal in length and width and we have:

K=1

$$\boldsymbol{x^2\big(h\gamma_1 + \gamma_2(z-h)\big) - x\left[4\sqrt{2}C.z + \sqrt{2}.\gamma_2.z^2(tan(\varphi)-1)\right] - \left[4\sqrt{2}C.z^2 + \frac{2\sqrt{2}.\gamma_2.z^3}{3}(tan(\varphi)-1)\right] > 0} \quad (6)$$

SAFETY FACTOR

Safety factor is defined as the ratio of the stabilization to instability. In this paper, the weight is considered as instability or destructive force and the shear force is considered as stable force. Because the goal is to cave the blocks, so unlike other surge instability is desirable.

Considering that when undercutting, no any supporting equipment is used and also the safety of undercuted roof zone depend on rocks and undercuting dimension. So in the relation (4) we can add the safety factor and also change the length and width of the block depends on the desired safety.

The safety factor is to consider the relative safety undercut below. It is due to that when ceilings undercut no any supporting is performed. But in some cases it is done locally. The value of this factor could be changed between (0.95) and (1); it is observed that the small amount of change has a great impact on the length of the block. Meanly by changing the safety factor (0.05), the length of work place could be largely increased without losing the safety.

$$\text{Safety Factor} = \frac{\text{Stability Force}}{\text{The instability}}$$

According to equation (2) we have:

$$SF = \frac{\sum_{i=1}^{4} F_i}{W} \rightarrow \frac{2\sqrt{2}C.z(2z+x+y) + \sqrt{2}.\gamma_2.\tan(\varphi)\left[\frac{2z^3}{3} + \frac{z^2}{2}(x+y)\right]}{x.y\big(h\gamma_1 + \gamma_2(z-h)\big) + \sqrt{2}.\gamma_2.\left[\frac{2z^3}{3} + \frac{z^2}{2}(x+y)\right]} \tag{7}$$

$$SF\big[x.y\big(h\gamma_1 + \gamma_2(z-h)\big)\big] - \left\{\begin{matrix} 2\sqrt{2}C.z(2z+x+y) + \\ \sqrt{2}.\gamma_2\left[\frac{2z^3}{3} + \frac{z^2}{2}(x+y)\right](\tan(\varphi) - \mathrm{SF}) \end{matrix}\right\} > 0 \tag{8}$$

THE ULTIMATE RELATIONSHIP

Relationship number (7) shows the overall calculation of the optimal block size with respect to the conditions above:

$$k.x^2\big(h\gamma_1 + \gamma_2(z-h)\big) - \left\{x.\left(\frac{1+k}{2}\right)\left[4\sqrt{2}C.z + \sqrt{2}.\gamma_2.z^2(\tan(\varphi) - 1)\right]\right\} - \left[4\sqrt{2}C.z^2 + \frac{2\sqrt{2}}{3}.\gamma_2.z^3.(\tan(\varphi) - 1)\right] > 0 \tag{9}$$

The above equation is a quadratic inequality unit age length where the ratio of width to length has also been considered. For any one mine, conditions such as depth, thickness, mineral and gangue mineral density, cohesion and internal friction angle could be added into the above equation and as a result, the optimal block length for the mine can be achieved.

BOUNDARY CONDITIONS

It is initially explained that it is assumed that there is only one block. But in fact there is more than one block in the mines. It also explains that the cost of preparation is economical and time-consuming. In general, the position of a block in a mine might be different. Below is a plan view of an overall Block caving mine-block system is depicted. As maybe seen a block might be located in the middle or border or at the corners of the waste material as follows:

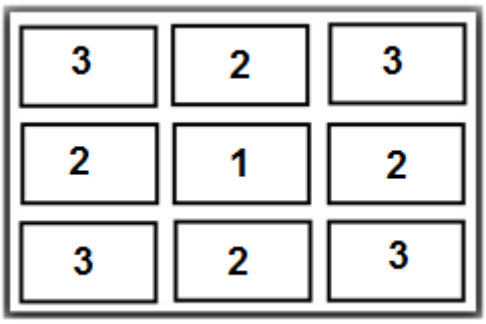


Figure 3- Plan of a mineral mass and dividing it into different blocks

Block (1) is a block that its cube and environs wedge are placed in the mineral and the only the top part of them is in the gangue mineral. The relationship of Block (1) while assuming a cube is following:

$$x^2\big(h\gamma_1 + \gamma_2(z-h)\big) - x\{4\sqrt{2}C.z + \sqrt{2}.(\tan(\varphi)-1)[h^2\gamma_1 + \gamma_2(z^2-h^2)]\} - \left\{4\sqrt{2}C.z^2 + \frac{2\sqrt{2}}{3}(\tan(\varphi)-1)[h^3\gamma_1 + \gamma_2(z^3-h^3)]\right\} > \tag{10}$$

Blocks (2) are those which two wedges of them are located in gangue and two others in mineral. The relationship of Block (2) if assuming a cube is following:

$$x^2\big(h\gamma_1 + \gamma_2(z-h)\big) - x\left\{4\sqrt{2}C.z + \frac{\sqrt{2}}{2}.(\tan(\varphi)-1)[h^2\gamma_1 + \gamma_2(2z^2-h^2)]\right\} - \left\{4\sqrt{2}C.z^2 + \frac{\sqrt{2}}{3}(\tan(\varphi)-1)[h^3\gamma_1 + \gamma_2(2z^3-h^3)]\right\} > 0 \tag{11}$$

Blocks (3) are those which one wedge of them is located in the gangue and three others in mineral. The relationship of Block (3) while assuming a cube is following:

$$x^2\big(h\gamma_1 + \gamma_2(z-h)\big) - x\left\{4\sqrt{2}C.z + \frac{\sqrt{2}}{4}.(\tan(\varphi)-1)[3h^2\gamma_1 + \gamma_2(4z^2-3h^2)]\right\} - \left\{4\sqrt{2}C.z^2 + \frac{\sqrt{2}}{2}(\tan(\varphi)-1)\left[h^3\gamma_1 + \gamma_2\left(\frac{4z^3}{3} - h^3\right)\right]\right\} > 0 \tag{12}$$

INPUT PARAMETERS

One of the most important parameters is the depth of mine that is achieved in exploration stages, also thickness, density and specific gravity of mineral and gangue material upside and also around the mineral will achieve by the stage mineral exploration and test of Cores. Cohesion and angle of internal friction will be obtained by RMR[1] and GSI[2] or test of Core. Safety factor and the ratio of width to length (k) values, which are decided by the designer.

Table 1- Designing parameters and engineering properties of the rock mass [6]

Rock mass rating (grade ore)					Parameters, rock mass properties	No
(V) <20	(IV) 40-21	(III)60-41	(II)80-61	(I) 100-81		
Very Weak	Weak	Fair	Good	Very Good	Rock mass classification	1
0.1>	0.1-0.2	0.2-0.3	0.3-0.4	0.4<	Cohesion (MPa)	2
15	15-25	25-35	35-45	45<	Angle of internal friction	3

Hook and Brown did a lot of assessments for linear approximation of Mohr Coulomb strength for a criterion rock mass and found that the parameters of cohesion and internal friction angle depends on σ_3. That's why they have proposed a chart to determine the mean values of cohesion and angle of internal friction to rapid evaluation of these parameters. Note that the cohesion and internal friction angle are reduced by GSI non-linearly.

Cohesion in rocks is15 to 20 MPa, and the internal friction angle is 44to 45degrees.Thea mount of them in Fractures and discontinues is1 to 2MPa and internal friction angle is equal to 27 degrees. [7]

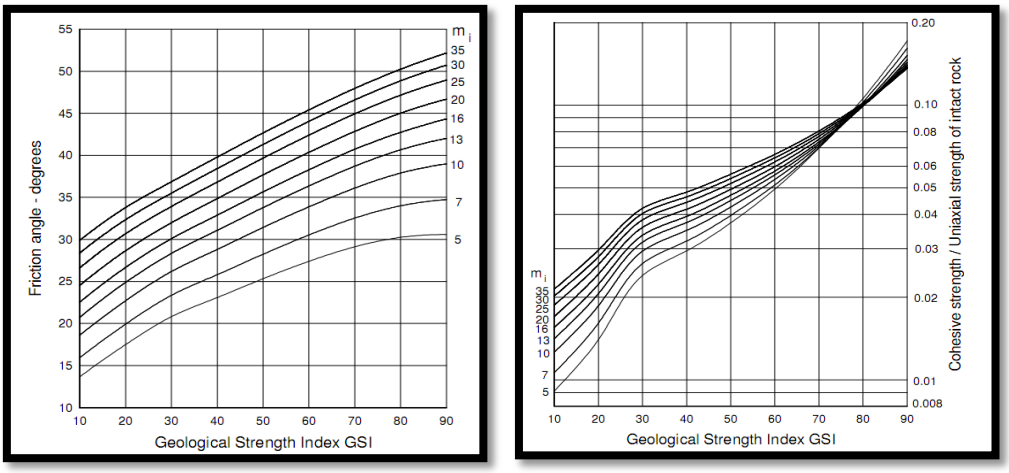


Figure 4- Relation between cohesion strength to the uniaxial compressive strength of Virgin rock $\left(\frac{c}{q_c}\right)$ and GSI for different values of m_r (right) and friction angle for different values of GSI and m_r (left) [3]

m_r= The rock constant that the three axis of the rock on rock cores is achieved.

[1] - Rock Mass Rating
[2] - Geological Strength Index

ANALYSES OF THE DIAGRAMS

Changes in length vs. depth according to cohesion variation

In the diagram of figure 5, it is assumed that all parameters are fixed. However, the depth and the cohesion can be changed. It should also be noted that the ratio of length to width is considered to be 1 (k=1).

In this diagram (figure 5), the horizontal axis shows the depth and the vertical axis shows the block length. Each line presents a value of the cohesion that is drawn from 1MPa to1.5MPa. As it can be seen, proposed length to a depth of 600m with a 1.3 MPa cohesion is about 55m and proposed length to the 1.4MPa cohesion is more than 100m.

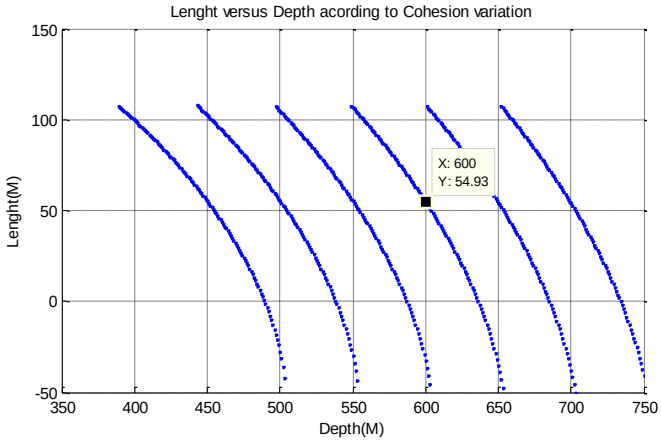


Figure 5- Changes in length vs. depth according to cohesion variation

Changes in length vs. depth according to y/x variation (K)

In the diagram below (figure 6), each line illustrates a ratio of width to length − which varies in the range of 0 to 1− with distance of 0.2 (which is changeable). It is observed that when the ratio is closer to zero, the block length is larger because the block has changed its shape to a rectangular shape and therefore it is less destructible. Therefore, the length should be greater so that as a result the weight of the blocks becomes heavier and caving is carried out.

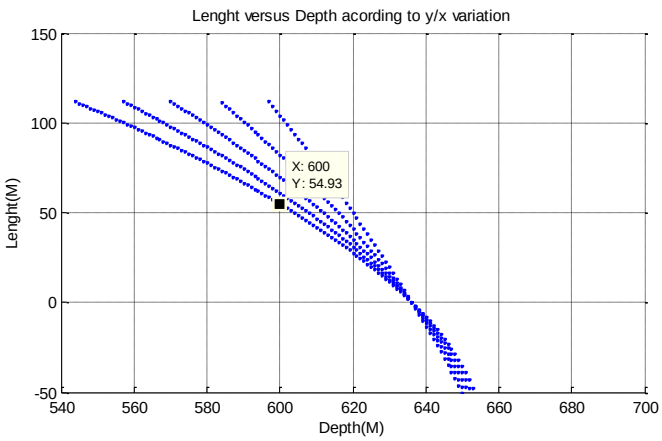


Figure 6- Changes in length vs. depth according to y/x variation (K)

Changes in length vs. depth according to the angle of internal friction variation

In the diagram of figure 7, the internal friction angle varies between 20 and 30 degrees which shows that it is directly related to the length of the block. That means that the proposed length is increased with an

increase in the internal friction angle, and vice versa. The reason for this is that when the resistance forces are also increased as a result of an increase in the internal angle of friction. Therefore, in order to overcome these forces, the block weight becomes heavier.

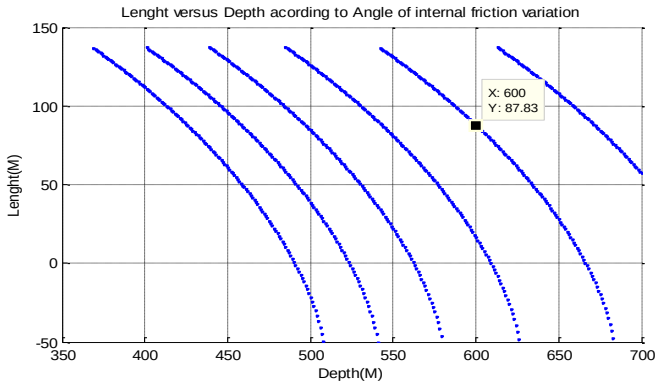


Figure 7- Changes in length vs. depth according to angle of internal friction variation

Changes in length vs. depth according to mineral density variation

The diagram of figure 8 shows that the mineral density does not influence the length of the block. Furthermore, there were no significant changes in the proposed length. This can be due to the small proportion of mineral blocks as weights. Because in this example, it is assumed that the mineral thickness is 100m and the depth is 600m. Therefore, the impact of mineral density is far less. It should be noted that in this diagram, mineral density changes between 25KN to 50KN per cubic meter.

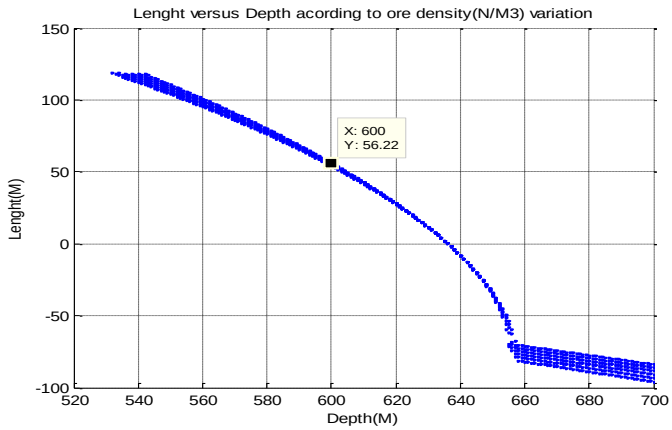


Figure 8- Changes in length vs. depth according to mineral density variation

Changes in length vs. depth according to gangue density variation

Unlike the previous diagram, in diagram 9 it is evident that the variation of gangue density plays an important role in the proposed block length. This is because of the fact that a large proportion of the blocks weight is composed of gangue. Whenever the gangue density becomes higher, the length of the block becomes smaller. These changes in the following diagram are between 25KN and 35KN per cubic meter.

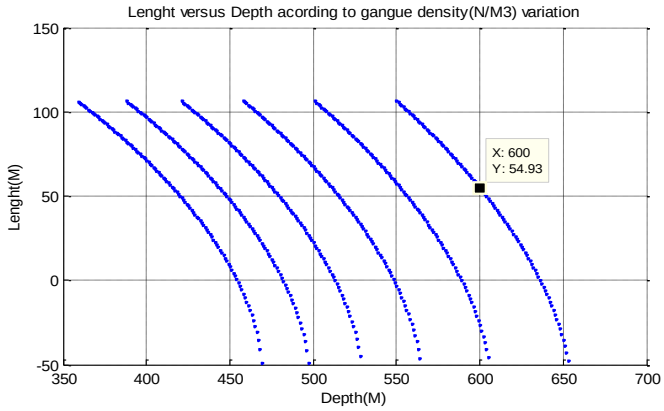


Figure 9- Changes in length vs. depth according to gangue density variation

Changes in length versus depth according to thickness of the mineral variation

The thickness of the block does not have a big influence on the length of the block. In this diagram, the thickness of the mineral varies between 100m to 200m.

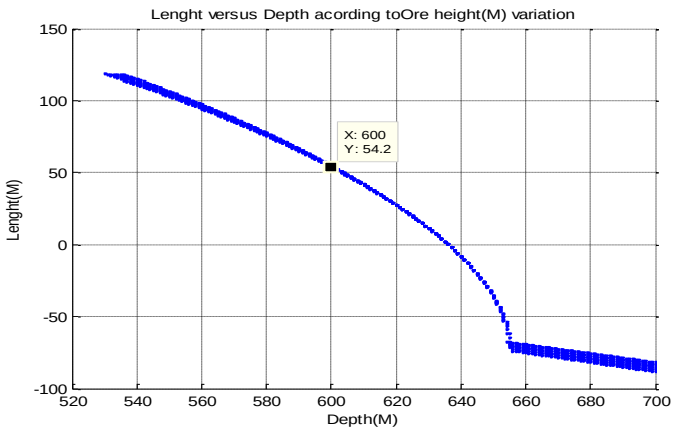


Figure 10- Changes in length vs. depth according to thickness of the mineral variation

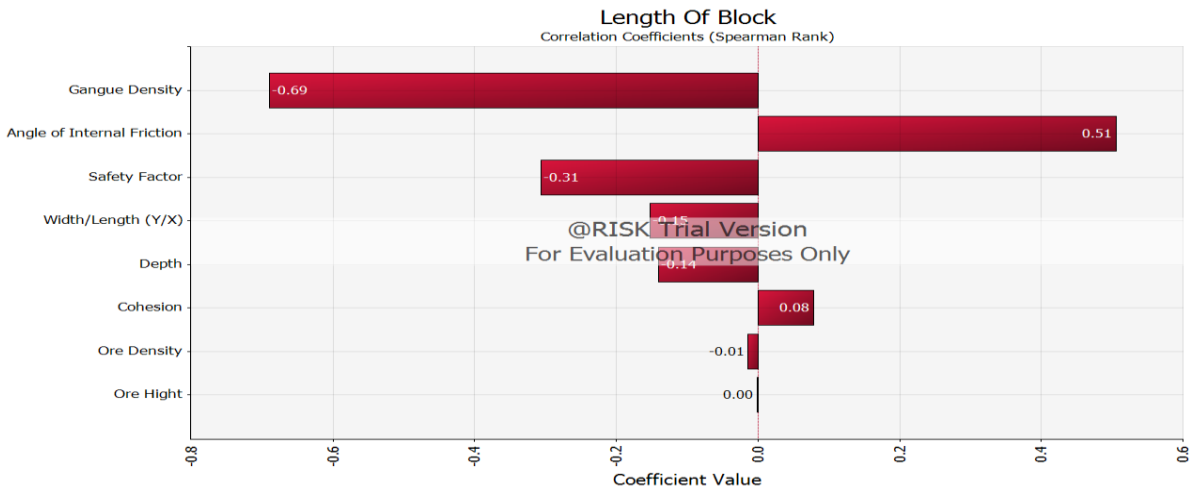


Figure 11- Tornado Graph, Correlation Coefficients, the effect of each factor on the length of the Block

Figure 11 compares the effect of each factor by Correlation Coefficients respectively. Horizontal axis shows the Coefficient Values and vertical axis shows different factors. This analyze is performed by @risk software and shows that Gangue Density has the most negative impact on the block length and fallowed by the Safety Factor and Width/Length (Y/X) and Depth respectively. Angle of Internal Friction and Ore Density have the positive impact on the block length. Positive correlation coefficients have direct impact and negative correlation has the opposite effect on the block length. Ore Hight and Ore Density have the least impact on the block length.

BOUNDARY CONDITIONS

As you can see there is not much difference in the boundary conditions. All of them are like the hypothetical situation that there is only one block. There are up to 3 meters difference by default. Of course this 3m difference in length, also this value will become 9 meters considering for space which is so wonderful. However boundary conditions are also considered in this study and depending on the inclination of the designer can be considered.

First Condition

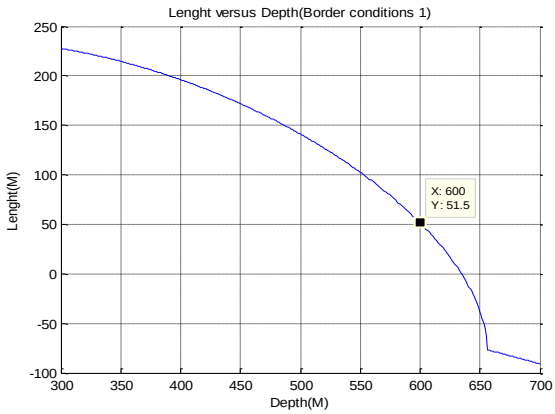


Figure 12- Change in length versus depth (Border condition 1)

Second Condition

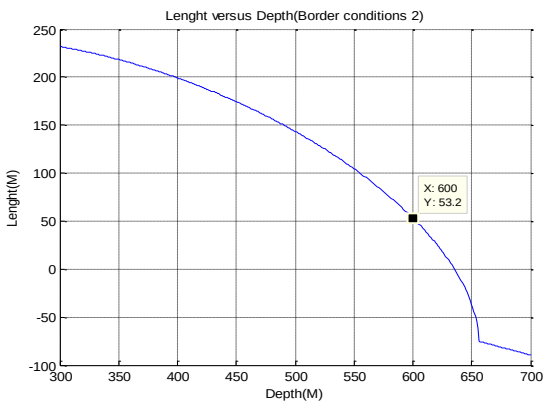


Figure 13- Change in length versus depth (Border condition 2)

Third Condition

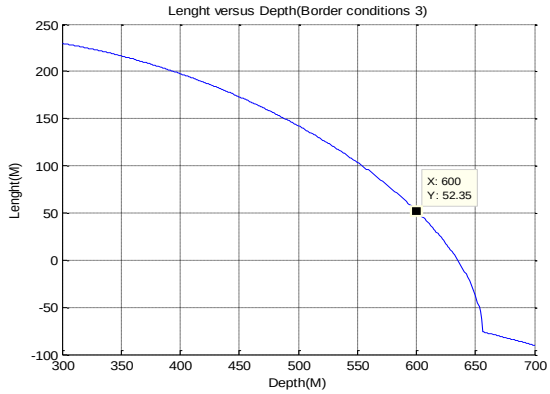


Figure 14- Change in length versus depth (Border condition 3)

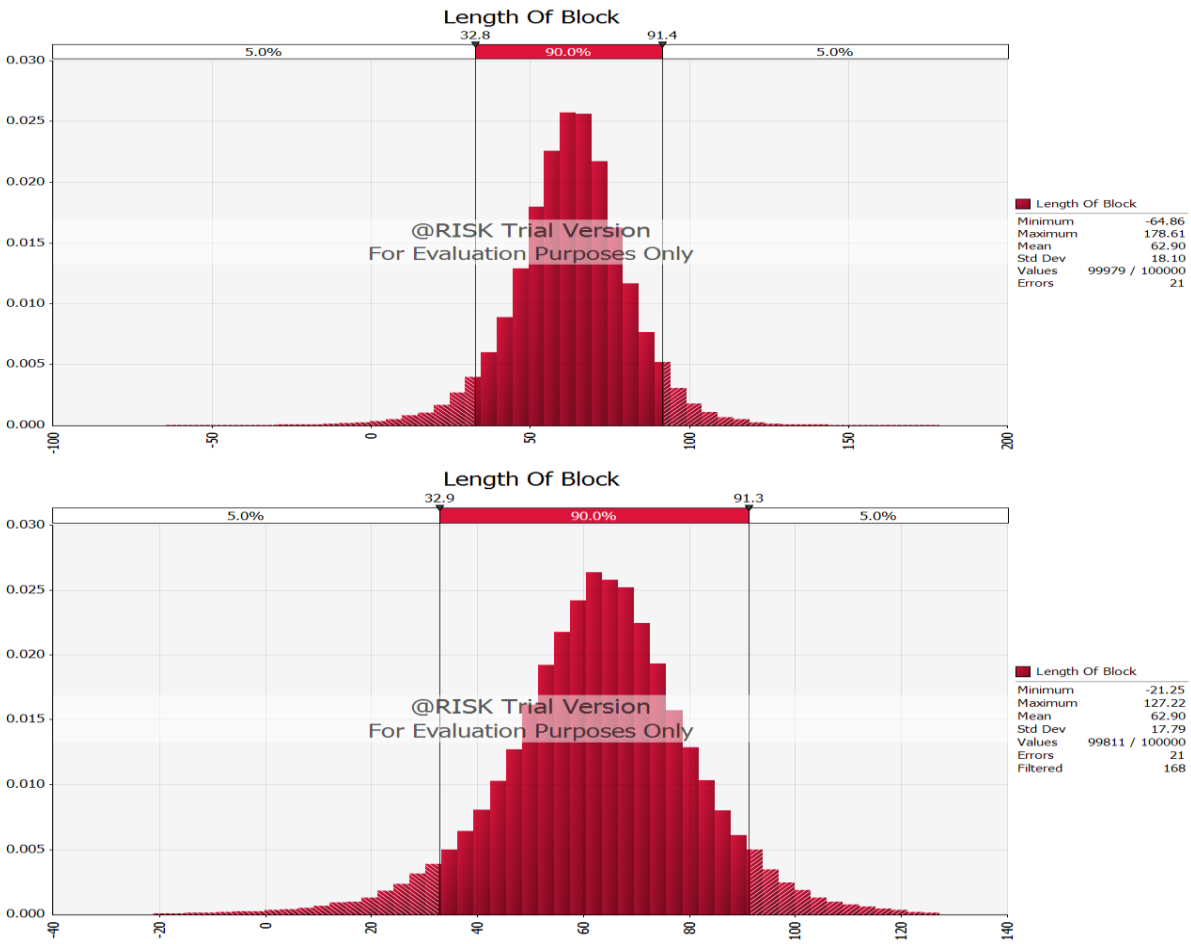


Figure 15- Probability Density Graph for Block Length

Figure 15 shows the Probability Density Distribution of the Block length. Horizontal axis shows Block Length and vertical axis shows the Probability Density of the Block length.In 90% of cases, the block length is between 32.9 m and 91.3 m. It is clear from the graph that approximately 60 m for block length has the most Probability Density which is more than 0.025.

CALCULATION METOD OF SQUARE BLOCKS DUE TO PRODUCTION RATE AND TRANSPORTATION

Using equation (9), block length can be calculated for optimal conditions according to the specific mine. But the obtained number is just a starting point, For example, greater than 60 meters in length may be the answer to inequality. But here the question arises as to how much this length can be changed? Can it also be used in larger numbers or not?

In this method, production is draw; volume of daily draw must be equal with daily volume of caving rocks which occurs due to coming down the page of caving in the block. Therefore, this factor can be used to limit the length of the interval. So according to the available transportation facilities and equipment, must define the size of the block so we can drive out 100% of caved materials-the same as our production.

Size of a block

$$V_1 = X.X.Z \tag{13}$$

Volume of slices and surrounding

$$V_2 = 4 \times \left(\frac{Z^3}{3} + \frac{X.Z^2}{2} \right) \tag{14}$$

Caving page, depending on how long the block's life span, comes down 20 to120 cm per day. We consider the maximum height which is 120 cm. [10] z = h = 1.2 m

So the maximum size of a fractured block in terms of cubic meters per day for a specified length of block (x) is obtained from the following equation:

$$1.2X^2 + 4\left(\frac{1.2^3}{3} + \frac{1.2X^2}{2} \right) = 1.2X^2 + 2.88X + 2.304 \tag{15}$$

If we consider:

nw = the number of wagons
Δv =average speed of wagons (kilometers per hour)
wh =Working hours in mines (h)
xΔ =distance (km)
v =size of wagons (cubic meters)
F_f = Fill Factor (percent)
α =Job Efficiency (je) (percent)

$$\frac{\alpha.wh.nw.(v.F_f)}{\Delta x / \Delta v} = \frac{\alpha.wh.nw.(v.F_F).\Delta v}{\Delta x} \tag{16}$$

Assume that the mine has a rail transportation system. It is usually two lines of railway track that by one of them, filled wagons are conducting out of mine and by the other one empty wagons return to the mine. If we divide the Working hours mines (wh) to the time taking wagons back the distance, considering average speed (T), The number of times wagons carrying minerals in working hours can be obtained. The Volume that could carry the rail systems per day is gained by the obtained number multiplies to the number of wagons.

In equation (16) four, the parameters Fill Factor and Job Efficiency are also considered. Fill Factor illustrate the ratio between the actual volume of rock or soil in, to its nominal volume of wagons.

$$F_f = \frac{V_a}{V_N} \tag{17}$$

Fill Factor depends on the quality of the stone, excavating conditions (easy to difficult) and other factors. Table (2) shows the estimated value of the Fill Factor.

Table 2- Estimated Value of the Fill Factor [11]

Very hard	Hard	Semirigid	Easy Terms
40%-60%	70%-80%	80%-90%	85%-100%

Dilatory factors

Some factors cause to delays that going to reduce in production or transportation. These delays are divided into two groups: fixed delays and variable delays. Fixed delays are those that the time and duration are specified, such as, lunch and dinner and changing shifts… the time and the duration of each one is half-hour. Although, the variable delay time is predictable, like filling up wagons, their maintenance and the like, but quantity and latency are not specified. The less variable delay, the higher efficiency will be. Sometimes the ratio of the real time operation to an hour(60minutes) and the time that was held to be the performance of the operation is called(Job Efficiency (je)).table(3)

Table 3- Job Efficiency [12]

Percent or ratio between the real time work and available time	Time (in minutes) to one hour (60minutes)	Job Efficiency
92	55	Very good
83	50	Good
67	40	Weak

The final equation

$$1.2X^2 + 2.88X + 2.304 = \frac{\alpha.wh.nw.(v.F_f).\Delta v}{\Delta x} \tag{18}$$

Here is an example that Working hours in mines is 24 hours, the number of wagons is 30, distance 5 km and nominal size wagons is considered 6 cubic meters which are in the normal range. Only 3 variable factors are left that include Fill Factor, Job Efficiency and the average speed of wagons in which they are discussed below:

Changes in Job Efficiency

The following table shows the Job Efficiency in very good, good and weak conditions, has been reviewed. And the mean value of Fill Factor and average velocity, respectively, 0.8 and 25 km per hour is considered.

Table 4- Change in Job Efficiency and proposed maximum length

Job Efficiency	Average Fill Factor	The average speed	Max Suggested length
67%	0.8	25	97 m
83%	0.8	25	108 m
92%	0.8	25	114 m

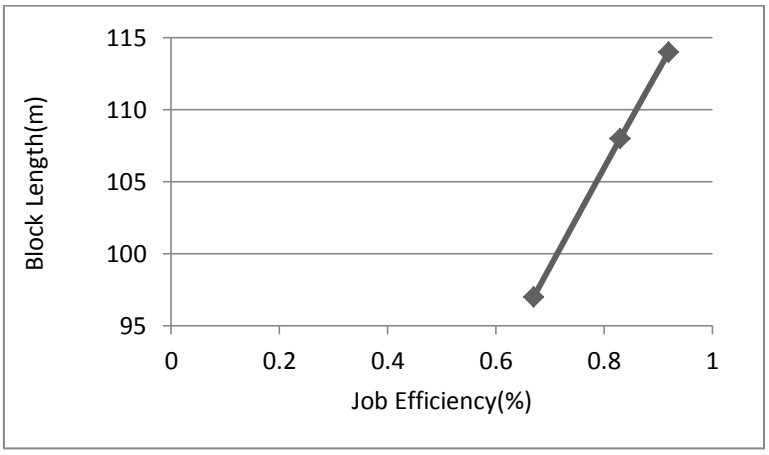


Figure 16- Graph of Job Efficiency -block length

Changes in Fill Factor

Fill Factor in the following table changes between50%to95%. And the mean value of the Job Efficiency (83%) and average speed of 25 kilometers per hour is considered.

Table 5- Change in Fill Factor and proposed maximum length

Fill Factor	The average Job Efficiency	The average speed	Max Suggested length
50%	83%	25	85 m
75%	83%	25	104.5 m
85%	83%	25	111.5 m
95%	83%	25	118 m

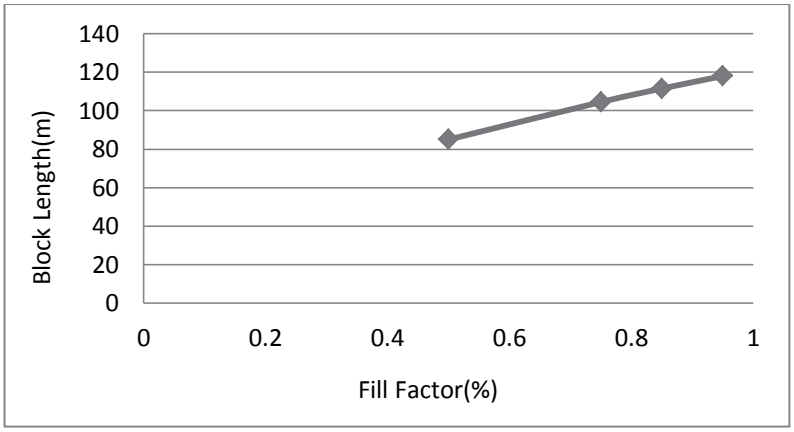


Figure 17- Graph of Fill Factor- block length

Changes in average speed

The following table shows the average speed of wagons traveling between 18 and 30 kilometers per hour is investigated. The percentage of the Job Efficiency and Fill Factor, the mean value was considered.

Table 6- changes in speed average and proposed maximum length

The average speed (Km/h)	Average Fill Factor	The average Job Efficiency	Max Suggested length
18	0.8	83%	91.5 m
25	0.8	83%	108 m
30	0.8	83%	118.5 m

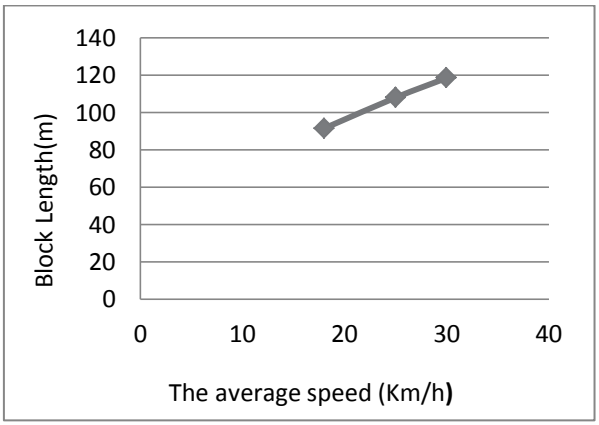


Figure 18- average speed Diagram of wagon-the block length

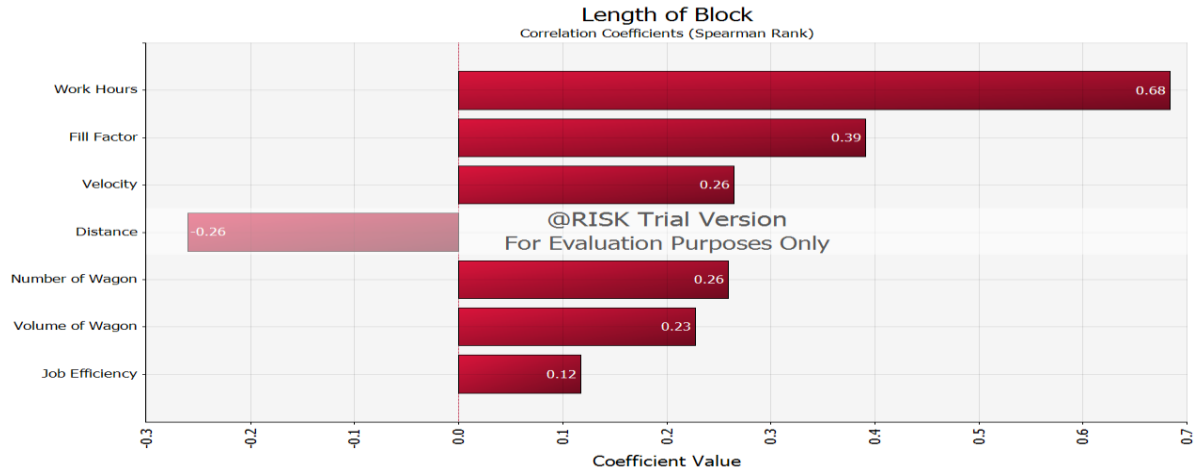
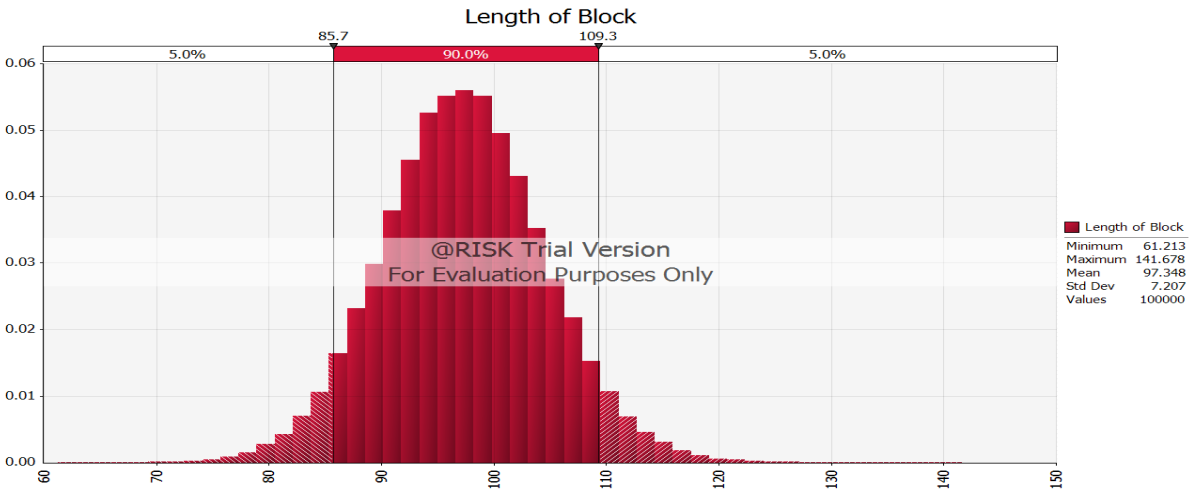


Figure 19- Tornado Graph, Correlation Coefficients, the effect of each factor on the length of the Block

Figure 19 compares the effect of each factor by Correlation Coefficients respectively. Horizontal axis shows the Coefficient Values and vertical axis shows different factors. This graph shows that Work Hours has the most positive impact on the block length and fallowed by the Fill Factor and Velocity and Number of Wagon respectively. Distance has negative impact on the block length. Positive correlation coefficients have direct impact and negative correlation has the opposite effect on the block length. Job Efficiency has the least impact on the block length.



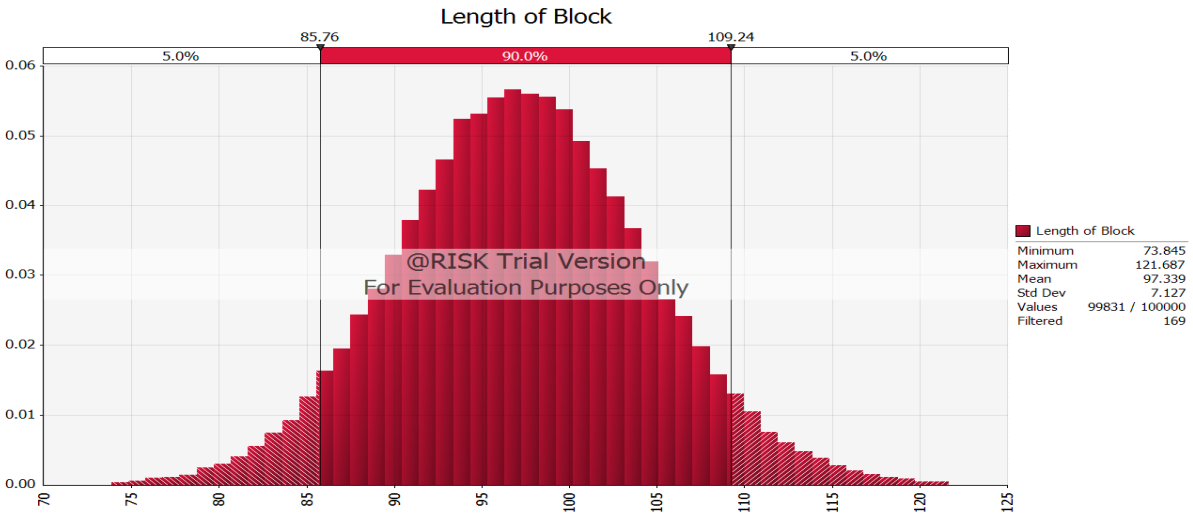


Figure 20- Probability Density Graph for Block Length

Figure 20 shows the Probability Density Distribution of the Block length. Horizontal axis shows Block Length and vertical axis shows the Probability Density of the Block length. In 90% of cases, the block length is between 85.76 m and 109.24 m. It is clear from the graph that approximately 97 m for block length has the most Probability Density which is more than 0.057.

SIMULATION BY *PHASE2*

At this stage, the above analysis has been simulated by *Phase2* software. To do this, we can model an example data by Phase2 software and compare the results with the different modes of the block. In these models at first, the length of the undercut was considered 30 meters and then this length reached to 70 meters in length take the steps to 5 meters. Each side of undercut is three times of the amount of undercut length, in order to eliminate the impact of the Borders. It is assumed that the undercutting height is 2 m for all cases. Mineral height is 100 meters and gangue height is 500 meters.

For better modeling, just 100 meters of the gangue is modeled and the rest of it the force with 10 Mega Newton is the top model. For each part of the model (minerals, gangue and undercut) random joint sets with the slope of -35 and 55 degrees and perpendicular random joint sets of angle -55 and 35 degrees is applied. *Barton-Bandis* model is used for both Waste and minerals joint.

Table 7- Input Data for Ore and gangue in Phase2

Gangue		Ore	
GSI	**60**	GSI	**40**
Mi	**10**	Mi	**25**
D	**0**	D	**0.1**
ICS	**25 Mpa**	ICS	**45 Mpa**
JCS	**20 Mpa**	JCS	**40 Mpa**
JRC	**10**	JRC	**10**
Phir	**28°**	Phir	**28°**
SG	**2500 Kg/m3**	SG	**4500 Kg/m3**
Kn	**100000 Mpa/m**	Kn	**100000 Mpa/m**
Ks	**10000 Mpa/m**	Ks	**10000 Mpa/m**
γ	**0.3**	γ	**0.3**

Hook - Brown model for mineral and gangue is used and k=1 and material type is *plastic* elastic kind is *isotropic*. The remaining data was considered to be the default application.

For each model, two stages with 500 computational steps is considered. The first stage is the initial equilibrium point and then Undercut is done. Each model shows the Total Displacement, Yielded Elements (percent) and Yielded joints respectively. Yielded joints are shown with red color in the third figures. In these models the Caving of the blocks can be achieved by these three displayed factors. I this way that each block that the amount of yielded joint and yielded elements is more, and the Total Displacement has been gone up to the surface, can be represent of a perfect caving.

As shown in Figures 21 to 29, these values in the block that their length is greater than 55m is the maximum and the caving is more than the rest. But since the goal of this paper is to obtain the optimal block length, very large values, has not been modeled.

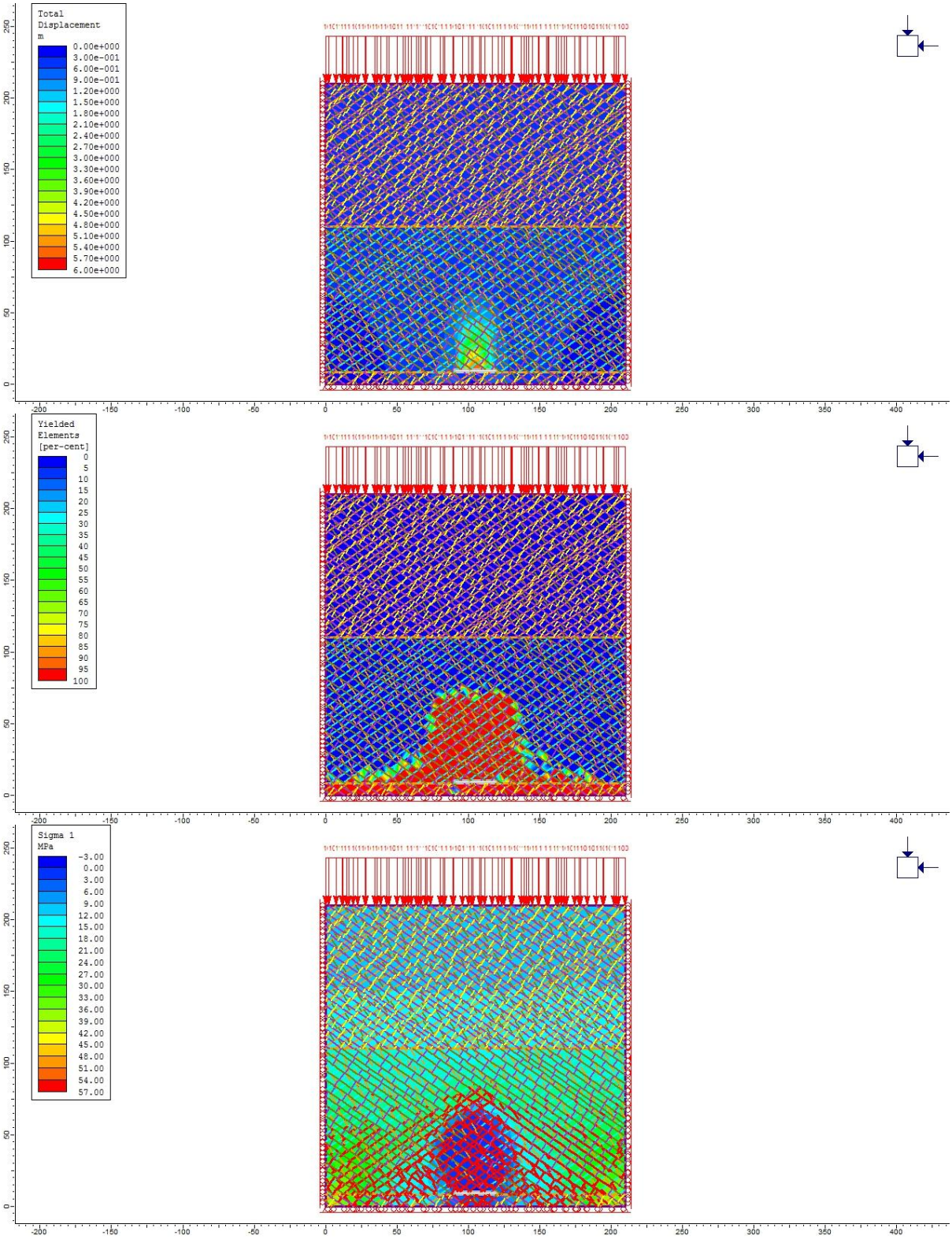


Figure 21- Block with 30m length (Total Displacement, Yielded Elements (percent) and Yielded joints respectively)

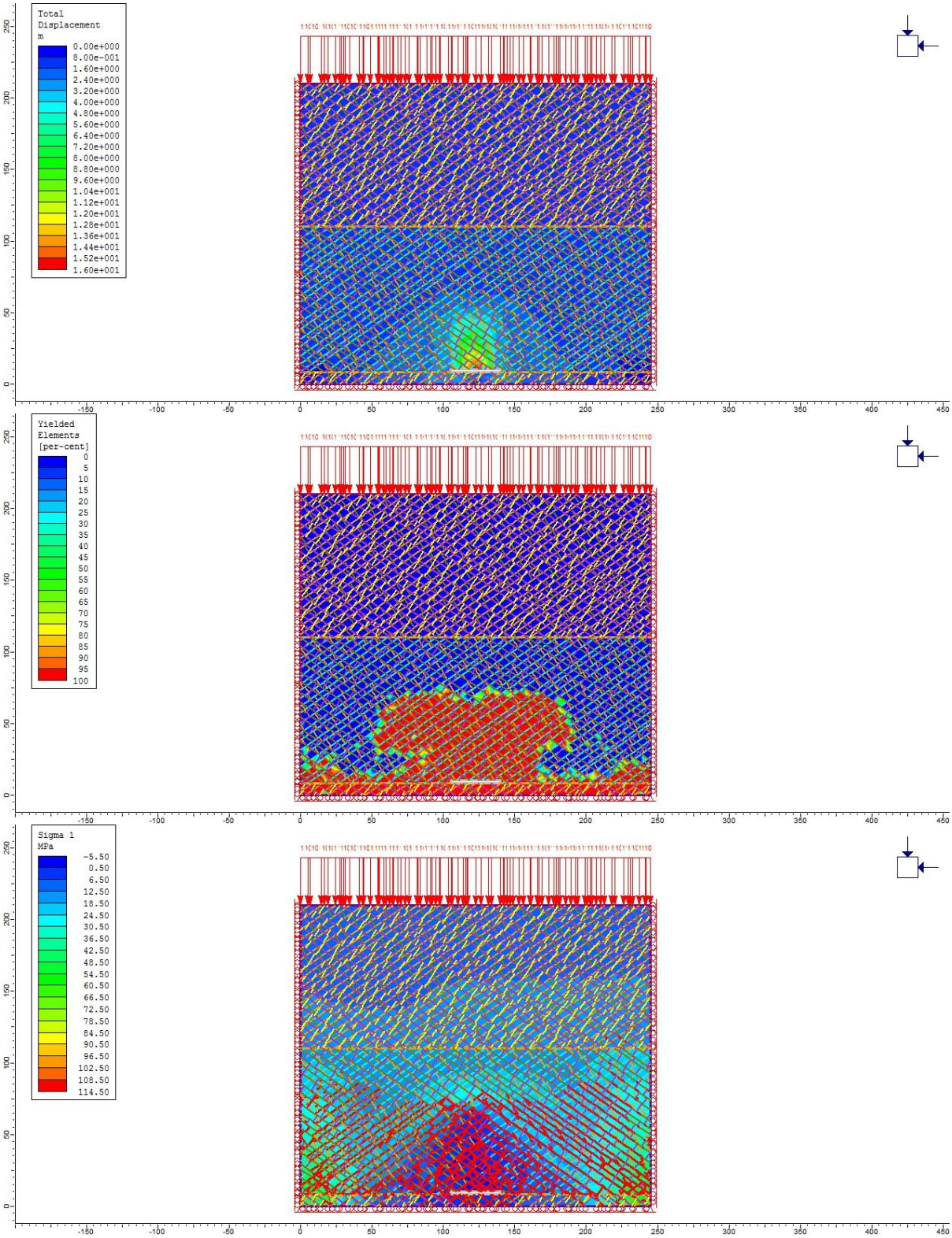


Figure 22- Block with 35m length (Total Displacement, Yielded Elements (percent) and Yielded joints respectively)

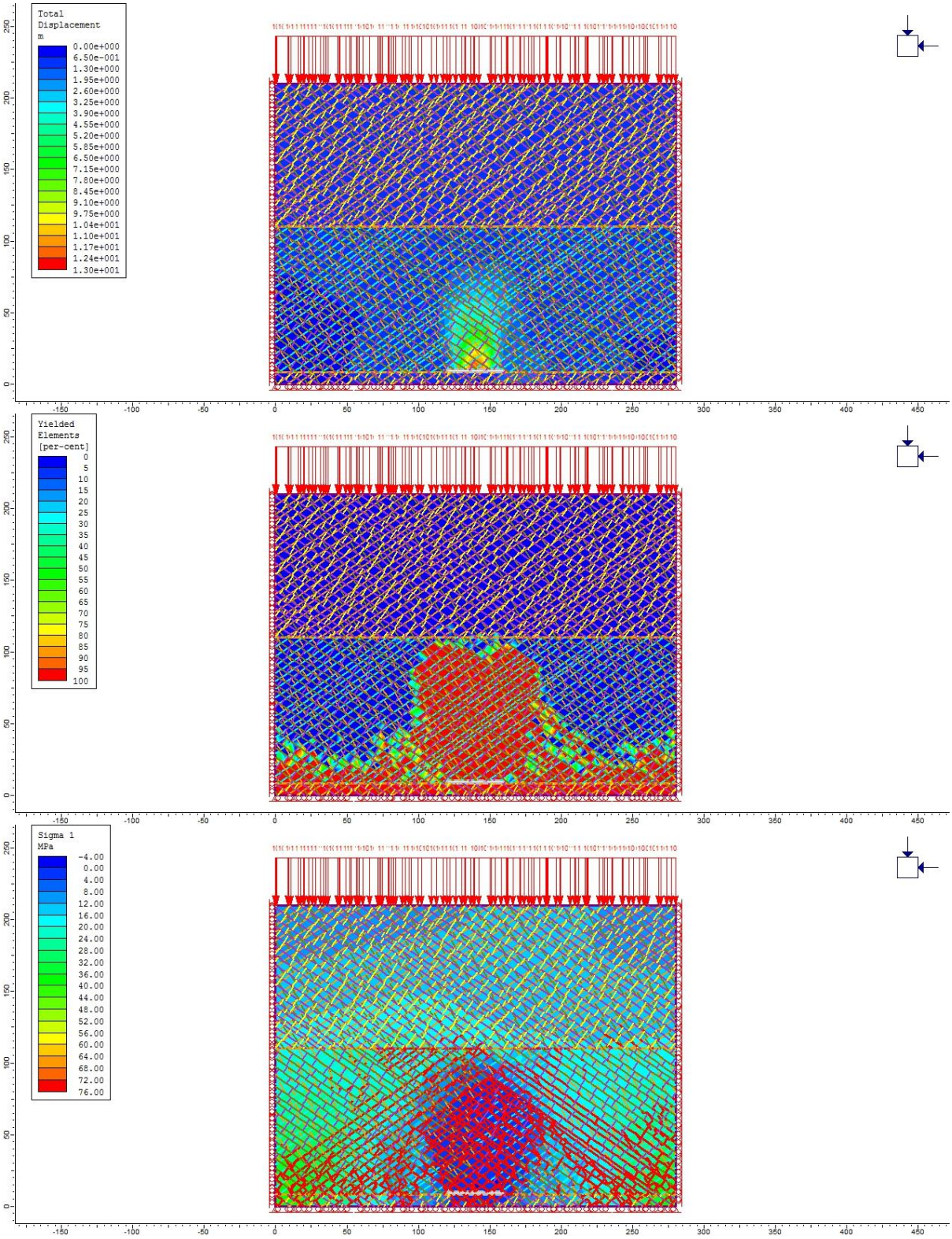


Figure 23- Block with 40m length (Total Displacement, Yielded Elements (percent) and Yielded joints respectively)

As can be observed in blocks 30, 35 and 40 m, the Total Displacement, Yielded Elements (percent) and Yielded joints goes to the border of material and gangue which. This amount is not sufficient for caving.

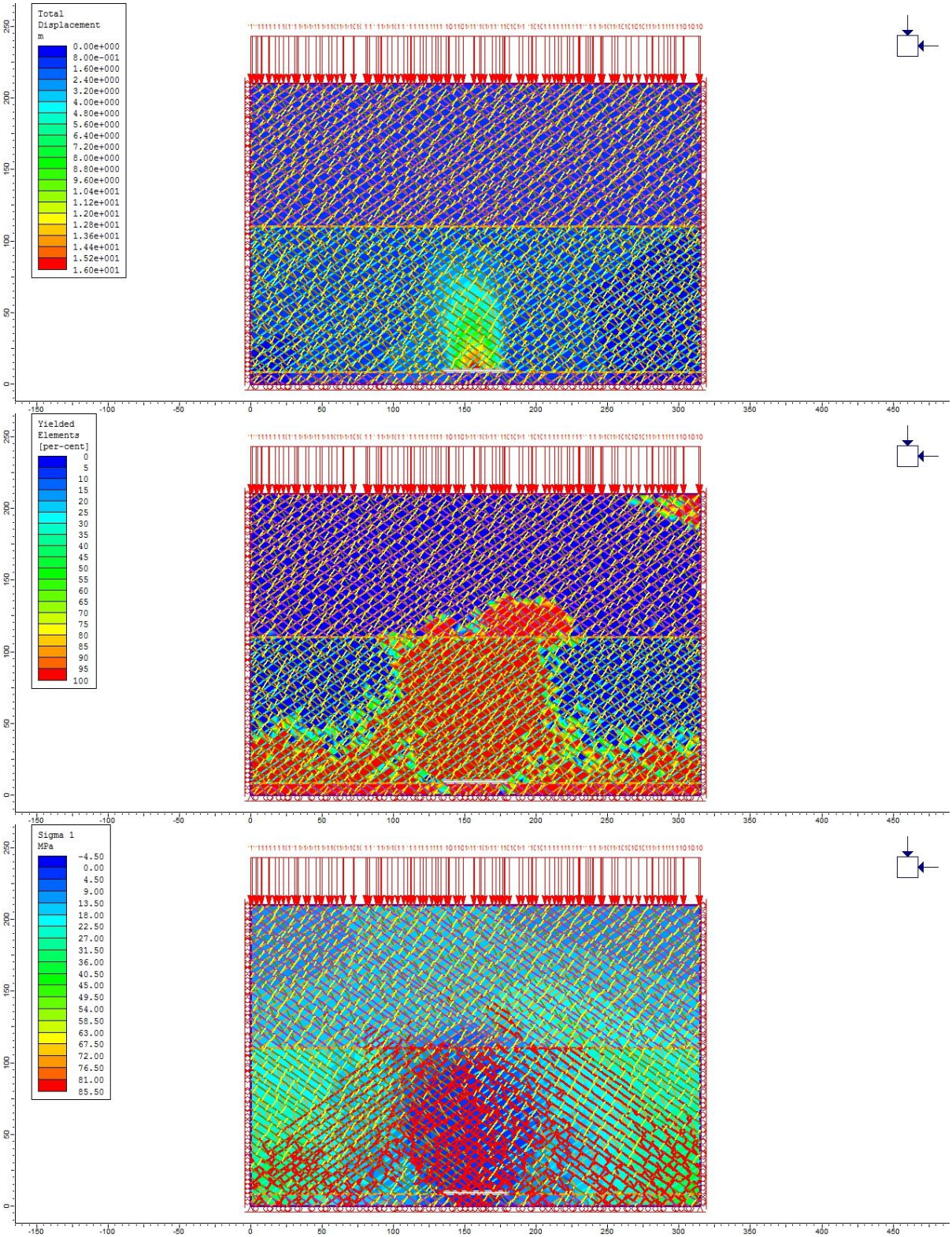


figure 24- Block with 45m length (Total Displacement, Yielded Elements (percent) and Yielded joints respectively)

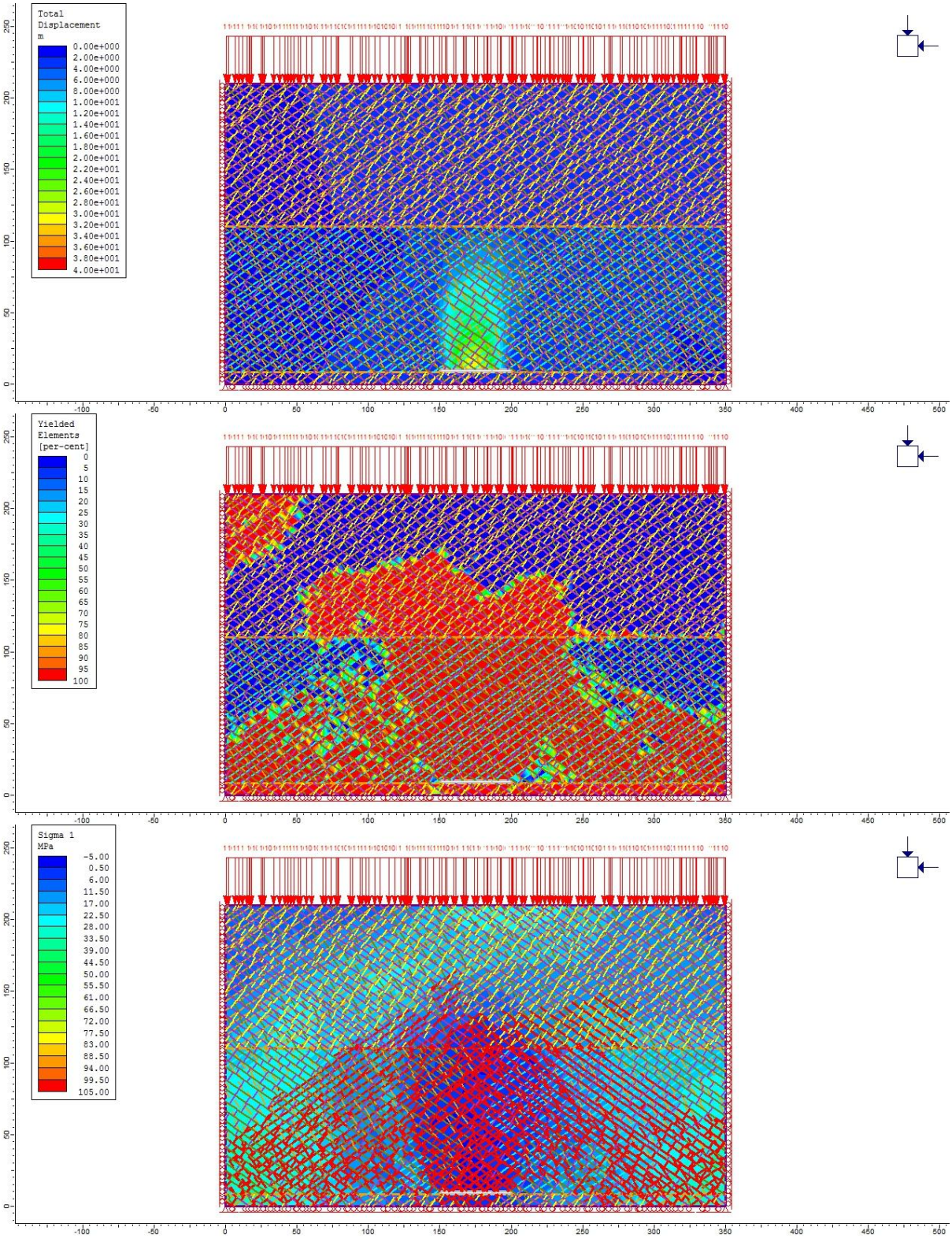


Figure 25- Block with 50m length (Total Displacement, Yielded Elements (percent) and Yielded joints respectively)

In blocks 45 and 50m, these values pass from the boundary of the minerals and gangue.

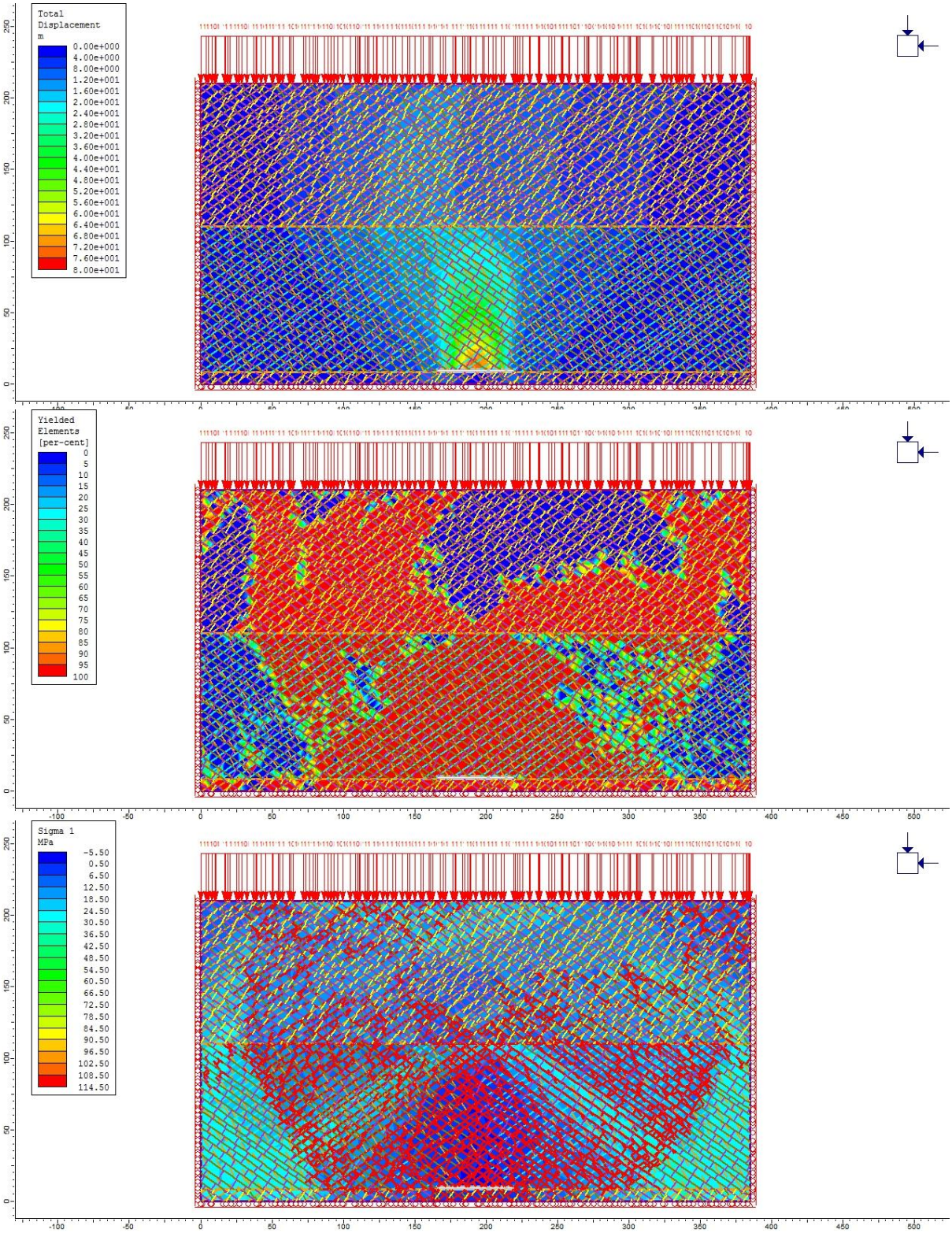


Figure 26- Block with 55m length (Total Displacement, Yielded Elements (percent) and Yielded joints respectively)

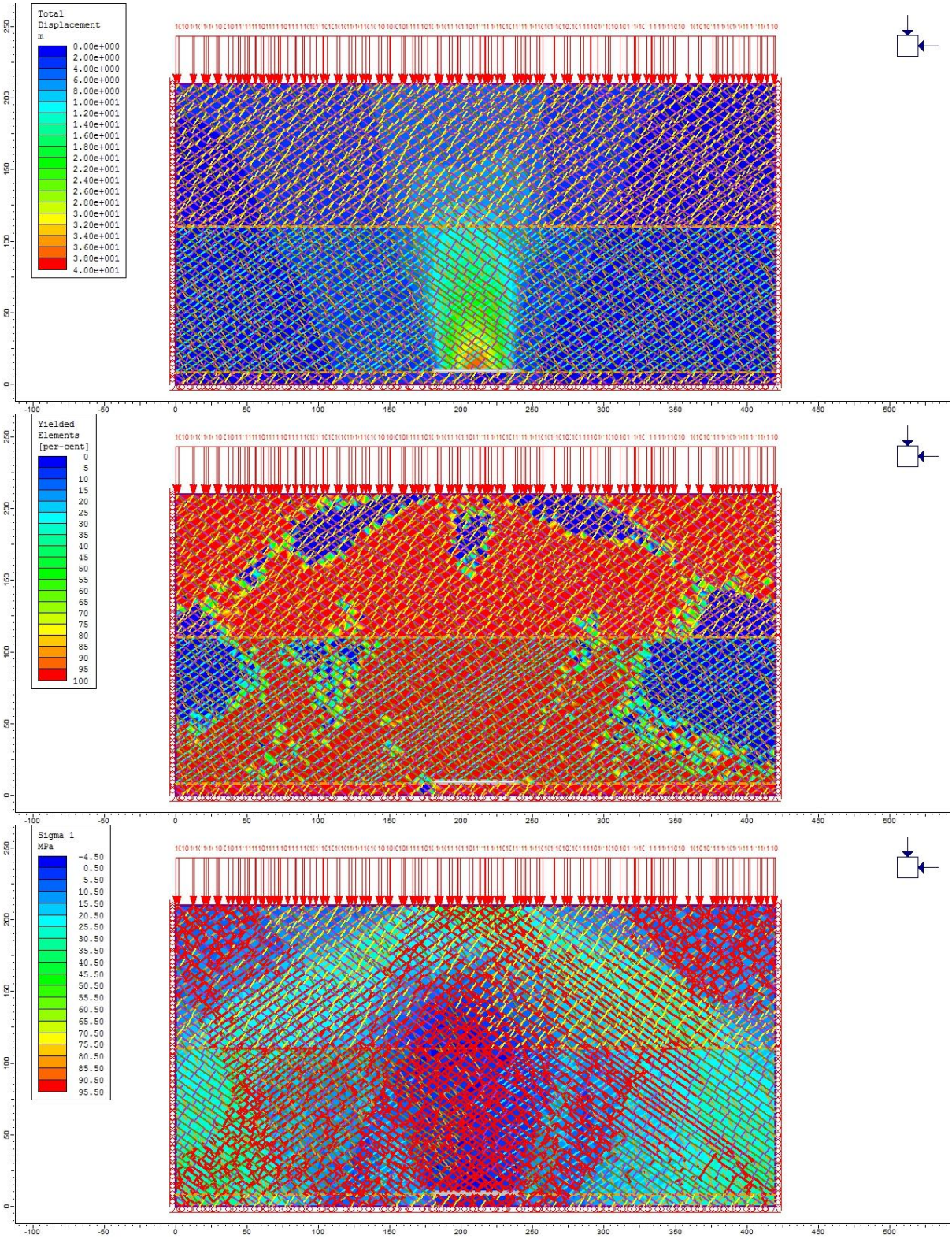


Figure 27 -Block with 60m length (Total Displacement, Yielded Elements (percent) and Yielded joints respectively)

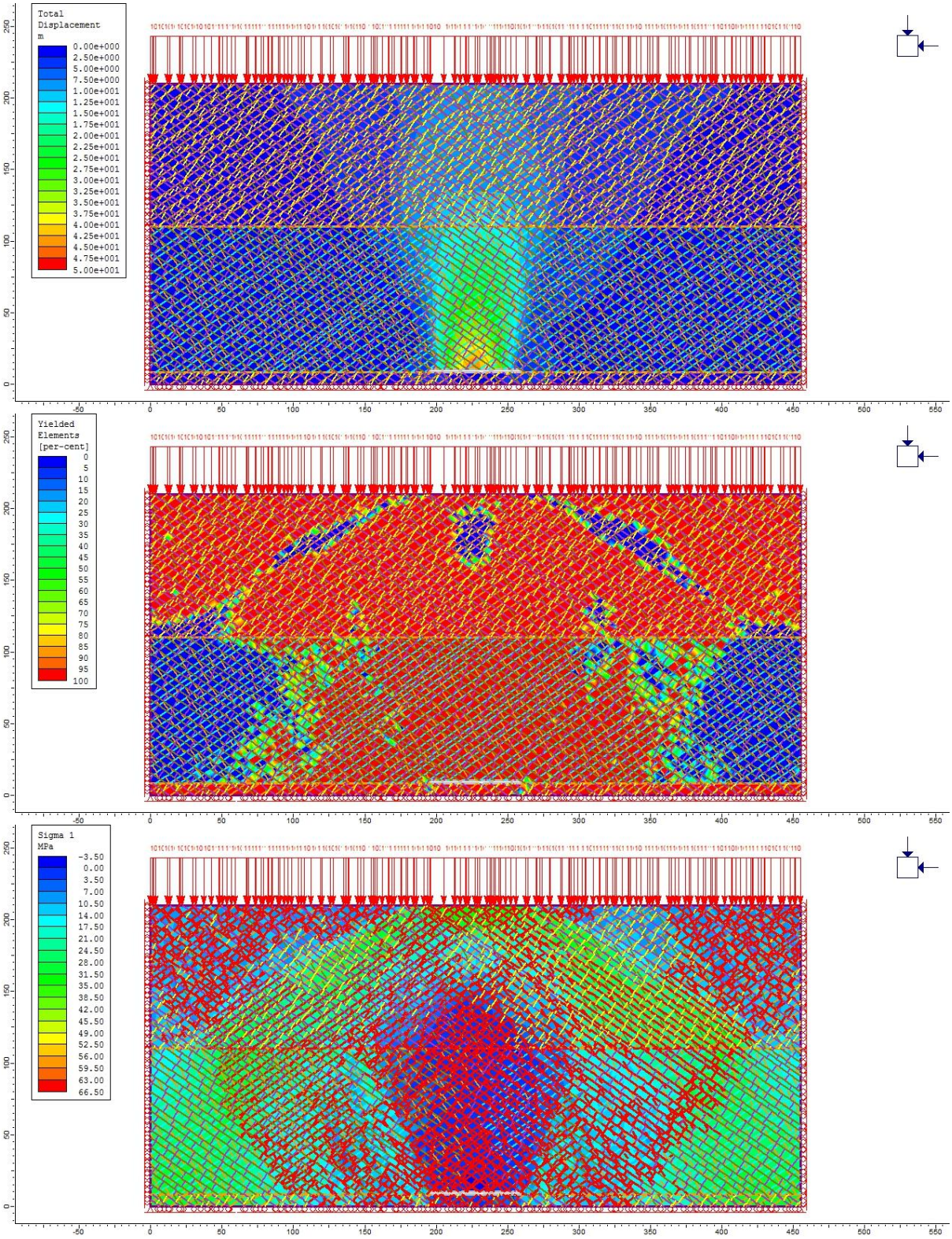


Figure 28- Block with 65m length (Total Displacement, Yielded Elements (percent) and Yielded joints respectively)

In blocks of 55, 60 and 65 m, Total Displacement, Yielded Elements (percent) and Yielded joints reached to a satisfactory condition and perfect caving will occur.

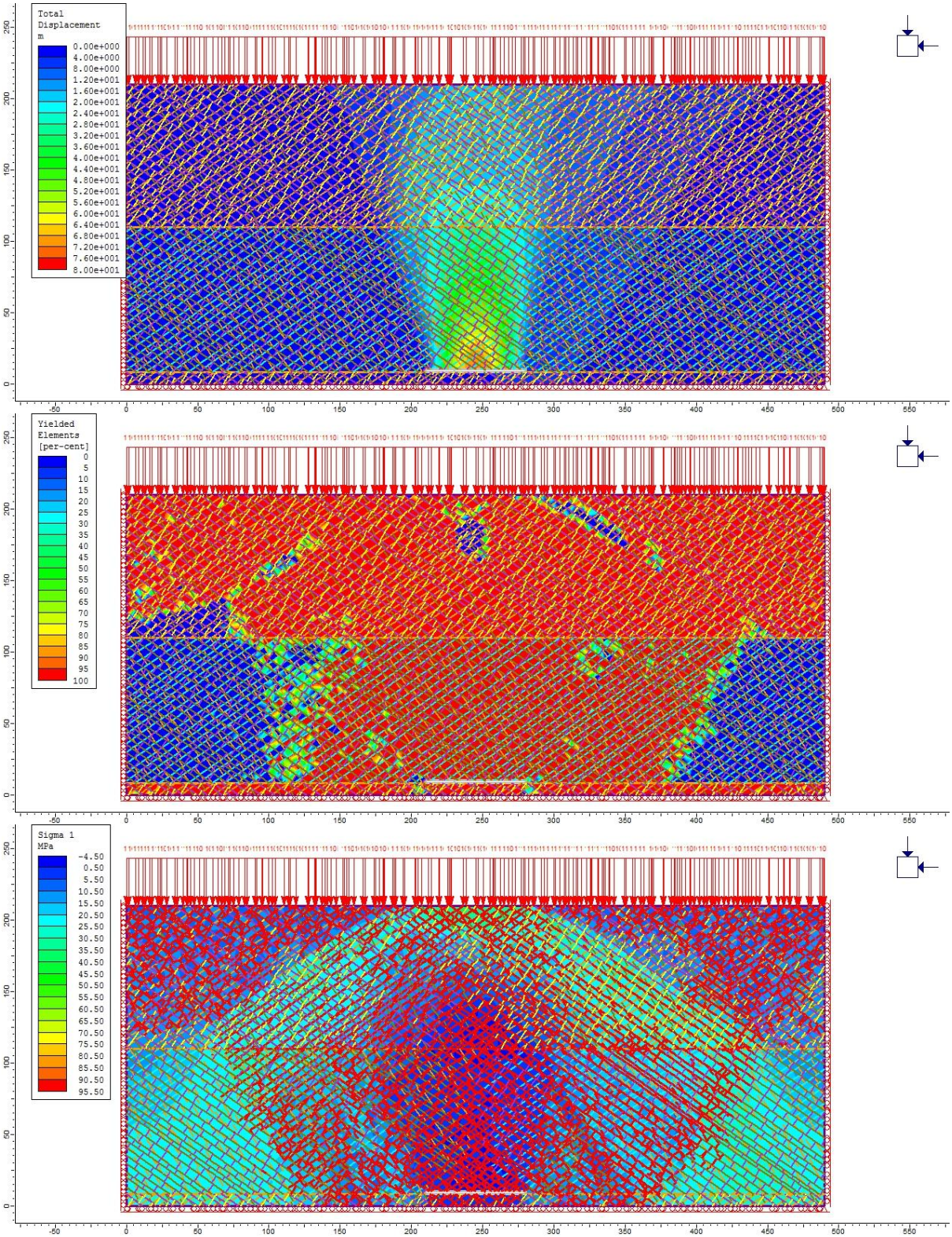


Figure 29- Block with 70m length (Total Displacement, Yielded Elements (percent) and Yielded joints respectively)

In block 70m, these values reached to a maximum and bigger blocks will cave very well. The purpose of this paper is to obtain the optimal block length. Thus, blocks which are between 55 and 65 meter, can represent the optimal size.

CONCLUSIONS

In order to perform caving, instability forces should always be greater than stability force. Thus, based on this, the length and width of the block can be achieved by solving the inequality that has been presented in this paper. Regarding the presented relationships, sensitivity analysis was carried out on various parameters such as cohesion, internal friction angle, mineral thickness, density and mineral gangue. The analysis showed that mineral thickness and density had the least effect. The impact of hydraulic radius on caving showed that square blocks are best suited for caving. While using square blocks, the relationship becomes a quadratic inequality with one unknown which is easy to be solved. However, in the case of rectangular blocks, the ratio of length to width (parameter K) is calculated graphically which has been presented in this paper in different lengths according to the depth of the block which comes in various widths. There is not much difference in the different boundary conditions and all of them are like the hypothetical situation that there is only one block. There are maximum (3) meters difference by default. The difference in length of 3m, when the area is considered is 9 m that is to be a wonder. Proved with a slight decrease in the safety factor of about 5%, we can increase the block length about 2 times while safety remains unchanged. Considering the capacity of transportation systems and the production of the interfaces, presented that the block length is measured in terms of production. Regarding to provide relationship for the production and manufacture of the blocks and parameters. The effects of factors such as average speed, and Job Efficiency of operations on the block Fill Factor was analyzed.

Gangue Density has the most negative impact on the block length and fallowed by the Safety Factor and Width/Length (Y/X) and Depth respectively. Angle of Internal Friction and Ore Density have the positive impact on the block length. Ore Hight and Ore Density have the least impact on the block length. In terms of transportation, Work Hours has the most positive impact on the block length and fallowed by the Fill Factor and Velocity and Number of Wagon respectively. Distance has negative impact on the block length. Job Efficiency has the least impact on the block length. In 90% of cases, the block length is between 32.9 m and 91.3 m. It is clear from the graph that approximately 60 m for block length has the most Probability Density which is more than 0.025. In terms of transportation, in 90% of cases, the block length is between 85.76 m and 109.24 m. It is clear from the graph that approximately 97 m for block length has the most Probability Density which is more than 0.057.

In the next stage, the analysis has been simulated by *Phase2* software. To do this, we modeled an example data by Phase2 software and compared the results with the different modes of the block. In these models at first, the length of the undercut was considered 30 meters and then this length reached to 70 meters in length take the steps to 5 meters. Finally, in blocks of 55, 60 and 65 m, Total Displacement, Yielded Elements (percent) and Yielded joints reached to a satisfactory condition and perfect caving will occur. In block 70m, these values reached to a maximum and bigger blocks will cave very well. The purpose of this paper is to obtain the optimal block length. Thus, blocks which are between 55 and 65 meter, can represent the optimal size.

REFERENCES

1. A Butcher, R Cunningham, K Edwards, A Lye, J Simmons, C Stegman and A Wyllie, **NORTHPARKES MINES**, Publication AMMOP
2. Brady, B. H. G., Brown E. T., (2005), **Rock Mechanics For Underground Mining**, Springer Science + Business Media, Inc., USA
3. Hoek, E., Brown, E.T (1997), **Practical estimates of rock mass strength**, Published in the International Journal of Rock Mechanics
4. Laubscher, Dennis H., **Cave Mining Handbook**, Mining Sciences, Vol 34, No 8
5. R.J. Butcher (1999), **Design rules for avoiding draw horizon damage in deep level block caves**, The Journal of The South African Institute of Mining and Metallurgy
6. Singh, Bhawani, R.K. Goel (1999), **Rock mass classification**, University of Rookee, India, Central Mining Research Institute Rookee, India
7. V.S. Vutukuri, K.Katsuyama (1994), **Introduction to ROCK MECHANICS**, Industrial Publishing & Consulting
8. Woo, K., Eberhardt, E., (2009), **Characterization and Empirical Analysis of Block Caving Induced Surface Subsidence and Macro Deformations**, Proceedings of the 3rd CANUS Rock Mechanics Symposium, May (2009), Toronto, Canada
9. Yashar Pourrahimian; Hooman Askari-Nasab (2012), **Mixed-Integer Linear Programming formulation for block-cave sequence optimisation**, Dwayne Tannant International Journal of Mining and Mineral Engineering (IJMME), Vol. 4, No. 1
10. Oraee, Kazem, (2008), **UNDERGROUND METALLIFEROUS MINING**, Tehran, Publication Amirkabir Industrial University
11. Basir, Hassan, (2008), **TRANSPORTATION IN MINES**, Esfahan, Publication Esfahan Industrial University
12. Osanloo, Morteza, (2010), **SUFACE MINING METHODS,** Tehran, Publication Amirkabir Industrial University