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Primary Blasting Optimization under Sub Level Caving: Application to the Kipushi Underground Mine, DR Congo

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Abstract: ***Mining in which any extraction occurs below the surface of the Earth is called underground mining. It consists of mining the ore from an excavation created below the surface of the ground, underground, without having to remove the entirety of the sterile materials which surmount it. The choice of an underground mining method is often closely related to the geology of the deposit and the degree of soil support required to make the methods productive and safe. The Kipushi underground mine is exploited by the Sublevel caving method which is a vertical extraction method in which a large open construction site is created in the vein. It turns out that its implementation requires a considerable amount of preparatory work and generally in waste rock. Now in these days, most studies converge towards the noticeable diminution of work with the rocks. To remedy this, we proceed for our study by an approach of empirical formulas. We started with an evaluation of mining ratios in Sublevelcaving by making a calculation of the explosives and fireworks by a theoretical determination of the load and by a presentation and analysis of diagrams in tracing and unstacking. Then followed by an analytical study of the techniques of caving by dimensioning the works of the mines, the determination of the exploitation height, in order to seek a rational solution for the primary mining in sublevel caving which is among the preoccupations of the underground mine from Kipushi. The analyzes and technical solutions envisaged to rationalize the operations of the primary mining in sublevel caving, prove that one will invest less for a sub-level of 17,50 m instead of two sub-levels of low height of 12.50 m. The advantage of increasing the height of the sub-levels results in a gain in dry ton ore per unit of time and a decrease in the cost of implantation of the sub level. After optimum study of existing fire patterns, and ultimately, the result confirmed that that of 11 holes for the sub-level of 12.5m and 16 holes for the sub-level of 17.5 m are rational schemes. This will not prevent the operator from choosing one or the other in the choice of the operating sub-level (12.5 m or 17.5 m).***

Keywords: fragmentation; drilling ring schemas, sub level caving, specific explosives charge

1. Introduction

Mining is the extraction of rocks or ores with economic value. Several mining techniques exist but can be divided into three main families: the open pit; the underground mine; dissolving and in situ leaching [2]. Mining in which any extraction occurs below the surface of the Earth is called underground mining [1]. The operation of an underground mine consists in mining the ore from an excavation created below the surface of the ground, underground, without having to remove all the sterile materials which surmount it. For an underground operation, a minimum amount of overburden is removed to access the deposit and corresponds to structural work (eg ramps, declines, drifts, wells) [2]. The choice of an underground mining method is often closely related to the geology of the deposit and the degree of soil support required to make the methods productive and safe [1]. Generally, three classes of methods are recognized: unsupported, supported, and caving; depending on the extent of support used [1]. According to the classification, are considered unsupported methods: room-and-pillars, stop-and pillars mining, shrinkage stoping, sublevel stoping; Supported methods: Cut-and-fill stoping, Stull stoping, Square-set stoping and Caving methods: Longwall mining, sublevel caving, block caving. The Kipushi underground mine is exploited by the Sublevel caving method which is a vertical extraction method in which a large open construction site is created in the vein. It turns out that its implementation requires a considerable amount of preparatory work and generally in waste rock. Now in these days, most studies converge towards the noticeable diminution of work with the rocks. This hypothesis led to the abandonment of sublevel caving and a possible application of sublevel stopping, to reduce the cost of extracting a dry ton. Based on the evaluation of the operating costs, it appears that the cost of fragmentation in the production stages is the highest, it represents about 35% of the whole, followed by the cost of loading and transport. This means that the cost of producing sublevel caving is higher than that of sublevel stopping. Compared to the yield, the method of Sublevel caving is around 80 to 90% that of Sublevel soping which is 75% [1], we seek to maintain the application of sublevel caving and create a reduction in operating costs . Several hypotheses are considered, including (1) adapting rational fragmentation schemes to reduce drilling and mining costs; (2) manipulation of sub-grade heights ranging from 12.50 m to 17.50 m to avoid much preparatory work in tracing in waste rock and (3) acting on drilling using lower cost machines . For our study, we will mainly rely on the optimization of fragmentation schemes. This optimization in question consists, on the one hand, in an evaluation and rational determination of the parameters of boreholes and on the other hand in the evaluation of the explosive charge; the specific rational load, the respect of the line of least resistance, the determination of the meter drilled for these schemas, with the aim of confirming the possibility of application of the sublevel caving for the continuation of the exploitation of the main vein of the Mine from Kipushi below level 1150.

2. Methodology

We will proceed for our study by an approach of empirical formulas. We will start with an evaluation of the mining ratios in Sublevelcaving by making a calculation of the explosives and fireworks by a theoretical determination of the load and by a presentation and analysis of diagrams in tracing and unstacking. Then will follow an analytical study of the techniques of caving by dimensioning the structures of the mine, the determination of the height of exploitation as well as the loading and connection of the holes of mines.

2.1. Evaluation of mining ratios in sub level caving

2.1.1. Calculation of explosives and fireworks

2.1.1.1. Theoretical determination the load

It can be done by empirical formulas.

a) According to Henri LASER: $Q = f_1.S.V.\frac{2}{D}.d$ [Kg / m3]; With Q = specific consumption (in Kg / m3); e = index of explosive breaking force; V = compaction index of the ore mined depending on the free surfaces (clearance); S: index of the structure of the rock; D: density of loading explosives into the hole; d: index of the quality of the stuffing; f1: index giving the resistance of the rock against the action of the explosives. f1 = 1/20 f with f: compactness index of the rock according to M. PROTODIAKONOV (en Kg/cm^2) ; f1 = $\frac{1}{100}$ RC. RC: compressive strength.
b) According to B.J. BOKY: $Q=\frac{NS}{V}$; With: N: number of holes: N = 2,7$\frac{S}{T^2}$; where T: depth of the hole (in m); S: surface of the clearance section; V: Volume of the rock to be felled [in m^3]; S: average size of a load [in Kg / hole]; Q: Explosive specific consumption [in Kg / m^3]
c) Q = q. V[in Kg]
d) Q= q. Sh

With: Q: specific explosive consumption [in Kg / m^3] . q $=\frac{K_0\,\sigma_1 N_1}{S.\eta_n}$; Ko: coefficient of filling of the borehole with the explosive, it varies from 0.30 to 0.35, ie an average of Ko = 0.55; η.S = Nt = Total number of mines in a drilling pattern; σ: linear loading density which is a function of the type of explosive and the mode of loading; ηn: practical use coefficient or mining yield. it varies between 0.75 and 0.85; S: section of the gallery. Considering: $n_1 = 2{,}7\sqrt{\frac{f}{7}}$; n_2 = 1,67 + 0,17 f – s (0,003 f + 0,027).. We then deduce η (mean) $= \frac{n_1+n_2}{2}$; Thus, q = $\frac{K_0.\sigma_1.N_{moy}}{\eta_n}$. l: the length of the hole: [m]. Using bulk explosives (the case of the Anfo), the average weight of the charge Q_m for a single borehole can be calculated by the formula: $Q = \frac{\pi D^2.L_u.d}{4}$; With: D: hole diameter; π: 3.14; Lu: useful length for the loading: it is determined in the following way: $L_u = L_t - (L_b + L_c)$ where L_t: total length drilled of the hole; Lb: length of stuffing; Lc: cartridge length; L_u = 3,20 – (0,45 + 0,2) : [m] = 2.55 m; d: density of the explosive: when loaded with compressed air, it varies from 0.85 to 1.00kg / dm^3.

The quantity of explosives per firing pattern can still be determined as follows:

a) For short mines: Q = q. Vr; With: Q: quantity of explosives per firing pattern; q: specific consumption or specific charge of explosives (kg / m^3); Vr: Volume of rocks to be felled (m^3) with Vr = S.L where S = the section of the gallery; L = drilled length of a mine; S = L.H With L = drilled length of a mine; H = Height. The specific explosive consumption is given by: $q=\frac{k_\circ.p.\eta m}{S.\eta\mu}$ With $k_\circ$: coefficient of filling mines with explosive. It varies from 0.3 to 0.8; P: amount of explosive by putting a hole. P is a function of the hole diameter, the type of explosive and the loading method. For the gallery, P varies between 1.3 and 1.4 kg / m; ηm: number of mine holes in the firing pattern; ημ: effective coefficient of utilization of a short mine, it varies between 0.75 and 0.85. The quantity of explosives will be: Q = q.Vr. After determining the consumption per shot scheme, we will determine the average consumption per hole by referring to the formula: $Lc=\frac{Qtr.L_\circ}{Q_\circ}$: Weight of a cartridge, $L_\circ$: length of a cartridge. The number of cartridges for each hole will be determined by the following formula: $n=\frac{Lc}{L_\circ}$
b) For long mines: It is generally necessary to consider that the dimensions of the deposit are considerably known. There are two main drilling patterns that can be used to attack the deposit by long mines: Single row or multiple rows with holes parallel to one another; Single or multiple rows with holes drilled. In both cases, the main element to be determined remains the line of least resistance. Considering, the case where the holes are drilled fan, the specific consumption of explosive is given by: $W = 29d\sqrt{\frac{\rho}{q.Kr}}$; With ρ: loading density in a borehole; d: diameter of a borehole; q: the specific consumption of explosive kg / m^3; Kr: coefficient of approximation of the holes. We take 0.8 to 1.2 underground mines; W: the line of least resistance.

2.1.1.2. Calculating the amount of explosive per firing pattern.

Q = q.Vr ; Vr = S.Lf ; S = L.h ; $q=\frac{K_\circ.p.Nt}{S.Nu}$; With Q: consumption per firing pattern (kg); Q: specific explosive consumption (kg / m^3); Vr: volume of the rock to be felled (m^3); Lf: drilled length (m^2); L: width of the gallery (m); h: height of the gallery (m); $K_\circ$: blast hole coefficient per explosive, which varies from 0.3 to 0.8; P: quantity of explosive per hole according to the type of explosive and the loading method (1.3kg / m or 1.4kg / m; Nt: number of boreholes in the diagram; Nu: effective use coefficient per short mine, which varies between 0.75 and 0.85.

2.1.2. Presentation and analysis of schemas in tracing

We will present the diagrams applied for the digging of the access, the hunting and the drifts. Each diagram has a given number of holes, depending on the section of the book. The characteristics of the rock (ore or sterile) thus the holding and the particle size, whose admissible dimensions are those of blocks of diameter lower than 1 m.

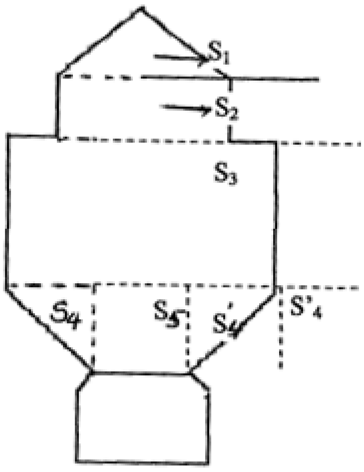

Figure 1: Schéma du volume en place

2.1.3. Presentation and analysis of depilage schemes

The opening angle is 300 °; the spacing between the front and the first ring: 5 m to allow an angle of 45 to 50 °; Line spacing: r = 1.80 m as line of least resistance; Distance between mines of a ring: amin = 0,5m; amax = 1.50m; Drift height: 3.80 m.

NB: We will do an analysis of the 9, 11,13 and 15 hole patterns. The determination of the slaughtered volume results from the subdivision of the schema into geometrical parts: S1, S2, S3, S4, S5, S6.

2.2. Analytical study of caving techniques.

2.2.1. Dimensions of structures

The solution will be to reduce the section of the drifts in order to have a smaller number of holes. The studies will then be oriented towards a reduction of the section going from 4.5 x 3.80 m to 4 x 3 m and this in the optics to preserve the equilibrium and the evaluation of the load to support.

2.2.1.1. Determination of the number of holes

$N = 2{,}7\sqrt{\frac{f}{s}}$ [hole / m^2] or again N = 0.17 f - S (0.003 f + 0.027) [hole / m2]. With f: coefficient of hardness of the rock according to the Protodiakonov scale.S: Cross section (m^2) = 12m^2; $f = \frac{Rc}{100}$; Rc: rock resistance to understanding.

2.2.1.2. Operating height

It will be determined according to the height of the sub level and it depends on the dimensions of the pillars. Analytically, it is given by: HE = Hs + 2Hp. With: Hp: Height of the pillar.Hs: Height of the sub-level.Or still: HE = Hs + Hd; With: Hd: Height of the drift.

2.2.1.2.1. Height of sub level Hs

To simplify the problem, we will adopt only the geometrical principle according to which: the flow of the sub level is possible when: Td ≤ Hs; With Td: distance between axes of two drifts arranged side by side.

2.2.1.2.2. The optimum thickness of the slice to be slaughtered: D

- Slices of height of 12.5 m: These slices are those which fix the height of the sub level to 12,5m.
- The extraction height HE: It is the sum of the heights of the sub level and the pillar.Slices 17.5m high. : We are referring to the same analytical reasoning developed for the 12.5m slices.
- The drilling pattern: For the drilling scheme, we will consider the same characteristics of the 12.5m subshell drill pattern; Horizontal spacing between drift axes: 5.5 m; the distance of the rings: 1.80m, which is the line of least resistance; the number of holes per ring; the interring = 1,80m; the interdrift = 5.5m; Hs = 17.5m. The Drift Section: (4x3) m^2 = 12m^2; the distance between mines of a ring "a". (a_{min} = 0.7W, a_{max} = 1.7W, with $W = 1.13d\sqrt{\frac{\Delta Kf}{g.Kr}}$; $0{,}80 \leq a_{min} \leq 1{,}00m$; $1{,}00m \leq a_{max} \leq 1{,}00m$).

3. Results

3.1. Determination of the amount of explosive by firing patterns in Sublevel caving.

Considering: S = 16.65m2; Lf = 3.20m; P = 1.3kg / m; $k_\circ$ = 0.3; Nt = 57-1 = 56 holes; Nu = 0.75Vr = 47.952 m^3.

- Specific drilling (Fsp); $Fsp = \frac{Tmf}{Vr} = \frac{Xtrous\ .3.20}{47.952}$ = 3.8mf/m^3.
- Specific Charge (ChSp); $q = \frac{0.3(1.3x2.75)x56}{16.65x0.75}$ = 4.8kg/m^3
- Amount of explosive per firing pattern, Q = q. Vr = 230kg.
- Number of bags = Q / P, with P: the weight of a bag of ammonium nitrate, Ns = 230/50 = 5 bags.
- Quantity per hole (Qtr) = Q / Nt, with Qtr: average consumption per hole in kg; Qtr = 410 kg per hole.
- Hole diameter: Ǿ = 45mm.
- weight of the cartridge: 150gr.
- Primer charge: 0.150kgx56 = 8.4kg.
- Coefficient of equivalence: 1.01.
- Weight of a solid explosive card: 25kg.
- Weight of ANFO bag: 53kg.
- Density of the gas oil: 0.85

ANFO: 8.4kgx1.01 = 8.48kg equivalent ANFO.
Number of carton: 8.48 / 25 = 0.33 ~ 1 carton
ANFO load: 4.1x56 = 230kg;

Considering 50kg, the weight of a bag of ammonium nitrate, and 3kg the weight of the fui oïl: 3.5x0.85 = 2.97 = 3kg.

- The total charge: 8.48 + 230 = 238.48 kg equivalent to ANFO.
- Number of bags: 238.48 / 53 = 5 bags.

Fire yield (Rd): This is the ratio between the advancement and the drilled length.Advancement = 57x2.88 = 164.16m; Lft = 57x3.20 = 182,4m. $Rd = \frac{Avancement}{TotalLengthfor\ é}$ x100

$Rd = \frac{164{,}16}{182.4}$x100=90%. Ratio de consommation : $FSp = \frac{Tmf}{Vr}$=182,4/47,952=3.8mf/m^3

ChSp=1.32kg/ m^3

3.2. Presentation of the ratio in tracing

Considering that: $Q = q.Vr$; $Vr = S.Lf$; $S = L.h$; $q=\frac{K_{\circ}.p.Nt}{S.Nu}$

Table 1: Presentation of the ratio in tracing

Indication	FSP (mf/ m^3)	CSP (kg/ m^3)	Rd (%)	Tmf (m)
a. Access 6x5m				
80 holes	2.29	0.78	90	256
75 holes	2.77	0.73	90	240
70 holes	2.59	1.213	90	224
b. Chassage 5x4m				
75 holes	4.16	1.95	90	240
70 holes	3.88	1.82	90	224
56 holes	3.61	1.69	90	208
c. Drift Nord 4x4 m				
60 holes	4.16	1.95	90	192
55 holes	3.81	1.78	90	176
50 holes	3.472	1.62	90	160
d. Drift Sud 4.5x3.7 m				
65 holes	4.33	3.03	90	208
57 holes	3.80	1.78	90	182.4
50 holes	3.30	1.56	90	160

3.3. Presentation of the ratio in unstacking

Calculates shooting elements. Number of meters drilled (mf): 114mf; Explosive consumption (Q):

- Section of the hole (S) (m3): Q = (mf - Lb) x S = 0.25755m^2. In ANFO equivalent: 0.25755 x 1.01 x 1000 = 260.1255kg / TS (Kilogram / Dry Ton).
- Volume in place: $S=\frac{(B+b)H}{2}$ with S: trapezium area (m2); B: large base (m); b: small base (m); H: height (m) $S=\frac{(13.5+6.5)21}{2}= 81.75m^2$
- Vr + Ls x S; With Vr: volume of the rock fell (m3); Ls: Length of section (m); Vr = 327m^3.
- Tonnage in place: 327 x 2.85 = 931.95 TS;
- Specific Drilling Fsp : $Fsp=\frac{Tmf}{Vrtot}$ With Tmf: total to put drilled; Total Vr: Total rock volume (m3): Fsp = 0.348mf / m^3.
- Specific explosive consumption: $Csp=\frac{Q}{Vr.ab.total}$ = 0.279kg/Ts

Cross section for slot opening
Meter drilled: 216mf;

- Explosive consumption (Q): Q= (mf-Lb).S=$(216-15)\frac{3,14.(0,057)3}{4}$ =0,5126 m^3
  In equivalent Anfo: 0.5126x1, 01x1000 = 517.726kg / Ts.
  The cubage or (the volume shot down)
  Referring to Figure 1: ST = S1 + S2 + S3 + (S4 + S5 + S6): ST = 195.675 m2; Vr = 782.7 m^3
  Tonnage = 782, 7 x 2, 85 = 2230,695 Ts.
  Specific drilling (mf / m3): Fsp $=\frac{Tmf}{Vrtot}=\frac{216}{782,7}=$ 0,275mf/ m^3
  $Csp=\frac{Q}{Vr.ab.total}$
  = 0.232kg/Ts

Table 2: Unstacked ratios for opening slots

Indication	Fsp (mf/ m^3)	Csp (kg/m^3)	Rd (%)	Tmf (m)
Slots Suedois	0,348	0,279	100	114
Cross section	0,275	0,661	100	216

Table 3: Presentation of the opening rings

Ring	Evantail	INCL. (°)	Length EV (m)	r (m)	a(en m) a_{min}	a_{max}	Volume
1	1	45	5,50	1,80	0,5	1,50	1544,13
	2	55	8				
	3	62	10				
	4	69	14				
2	5	69	15,20				
	6	74	15,60				
	7	80	14,60				
3	8	80	14,60				
	9	89	14,40				
	10	90	14,40				

Table 4: Unstacked ratios for fans

Indication	Fsp (mf/ m^3)	Csp (kg/m^3)	Rd (%)	Tmf (m)
EVANTAILS				
9 HOLES	0,371	1,059	100	84,65
11 HOLES	0,409	0,866	100	62,6
13 HOLES	0,43	1,051	100	131
15 HOLES	0,477	1,114	100	164,2

Starting from the results above on the determination of the ratios, we can conclude:

Table 5: Tracing

Indication	Fsp (mf/ m^3)	Csp (kg/m^3)	Tmf (m)	Number holes
ACCESS 6X5m	2,77	0,73	240	75
CHASSAGE 5x4m	3,88	1,83	244	70
DRIFT 4,5x3, 7m	3,80	1,78	182,4	57

For the opening of slots: We retain the diagram of 15 holes: Fsp: 0,275mf / m^3; Specific load: 0.661kg / m^3

For unstacking: Number of holes: 11; Fsp: 0.409 mf / m^3; Specific load: 0.866 kg / m^3.

Table 6: Schemes retained

Indication	Fsp (mf/ m^3)	Csp (kg/m^3)	Rd (%)	Tmf (m)
Unstacking				
Slots Swedois	0,343	0,279	100	114
EVANTAILS OU RING				
11 HOLES	0,409	0,866	100	62,6

3.4. Schema design

Two possibilities are presented respecting the parameters the scheme of which the first is to have 15 holes and the second 16 holes.

Table 7: Diagram for 15 Holes

N° Holes	Tilt (°)	Length (m)	Rods of 4 inch	Rods of 6 inch
1-15	40	4	5,20	2,20
2-14	50	6,20	5	3,5
3-13	61	9	7,20	5
4-12	68	12,70	10	7
5-11	74	15	12	8,30
6-10	80	17,50	14	10
7- 9	86	19	15	10,5
8	90	21,6	17,30	12
Total ring 188,4				

Table 8: Diagram for 16 Holes

N° Holes	Tilt (°)	Length (m)	Rods of 4 inch	Rods of 6 inch
1-16	35	4	3	22
2-15	45	4,80	4	7
3-14	56	6	5	4
4-13	65	10	8	6
5-12	70	14,70	12	8
6-11	77	14,60	12	8
7-10	84	18	14,4	10
8-9	87	19,60	16	11
Total ring 183,4				

4. Conclusion

The search for a rational solution for primary mining in sublevel caving is one of the concerns of the Kipushi underground mine and this for reasons related to the volume of work on the rock and the reduction of mining costs. The choice of a drilling pattern in tracing or unstacking balancing all the economic and technical requirements with a sub-level of 12.50m, requiring the evaluation of the work to be done, and the passage of the sub-level of 12.5 m to a higher sub-level requires increasing the number of holes to avoid large blocks because the ideal is to "undermine: not very strong, not very weak". The analyzes and technical solutions envisaged to rationalize the operations of primary mining in sublevel caving, prove that one will invest less for a sub-level of 17,50 m instead of two sub-levels of low height of 12.50 m. The advantage of increasing the height of the sub-levels results in a gain in dry ton ore per unit of time and a decrease in the cost of implantation of the sub level. There is a decrease in the load to be mined per dry tonne, which creates a saving in explosives. After optimum study of existing fire patterns, and finally, the result confirms that that of 11 holes for the sub-level of 12.5m and 16 holes for the sub-level of 17.5 m are rational schemes. This will not prevent the operator from choosing one or the other in the choice of the operating sub-level (12.5 m or 17.5 m).

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