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Substantiation of rational parameters of open-pit development of a two-layer deposit

Umidjon Yermekbayev*, *Shadiboy* Khakimov, *Zoir* Nazarov, *Otabek* Giyazov, and *Azizjon* Nematov

Navoi State University of Mining and Technologies, Navoi, 210100, Uzbekistan

Abstract. Technological schemes for open pit mining of mineral deposits show the reduce of costs by diminishing the distance of transportation. The development of fundamentally new types of technological schemes includes the division of the mining front into sections, the mining of sections with overburden benches that wed out to the mineral layer, and mining benches with the placement of rocks in the tiers of internal dumps and the advancement of the dump front after the mining of the benches.

1 Introduction

The efficiency of open-pit mining of stratified mineral deposits mostly depends on the size of the quarry field.

When working out horizontal deposits in an open way, the movement of the working zone occurs over the area of the deposit, and with the specified size of the quarry field, the parameters of this zone change only in height when the overburden capacity changes. The change in the size of the quarry field leads to a corresponding change in mining capital and operating costs [1,2] associated with the construction of a quarry, transportation of rock mass, stages of mining equipment, maintenance of transport communications, transfers of power lines (transmission lines) and others. The size of quarry fields under various technological schemes should be investigated taking into account all factors affecting the efficiency of mining operations and is an urgent task. The purpose of this study: - development of a methodology based on a minimum of costs and determination of rational sizes of quarry fields in technological schemes of transport systems for the development of horizontal and gently falling reservoir deposits.

2 Materials and methods

When justifying the minimum length of the mining front, the condition for providing overburden equipment with exploded overburden rock, drilling and prepared work front is taken as a basis [3-5].

To justify the minimum length of mining operations in dynamics, the graph $L_{a \cdot h}$=f(t) is used, taking into account the following fixed parameters:

* Corresponding author: Bobakulov_umidjon.uzb@mail.ru

- safety zone between overburden and mining equipment J_b=30 m;
- the length of the formation preparation block for excavation ($L_{b.p.}$) (stripping of the safety jacket) according to practice data is assumed to be 100 m;
- the length of the mining block ($L_{d.b}$) is assumed to be 300 m (according to previous studies);
- the length of the drilling block ($L_{b.b}$) is determined taking into account the organization of drilling and blasting operations at the quarry according to the formula

$$L_{b.b} = V_{min} = \upsilon_p N_{exp} \tag{1}$$

where υ_p is the rate of development of stripping operations per day, m / day; N_{exp} is the frequency of explosion, N_{exp} = 7 days.

The rate of development of stripping operations along the length of the front is determined by the production capacity of the quarry according to the formulas:

- for transport schemes of development with longitudinal approaches

$$\upsilon_p = \frac{P_d}{\Sigma h_l * A_z * K_r} \tag{2}$$

- with the panel method of excavation

$$\upsilon_p = \frac{P_d}{\Sigma h_l \cdot S_w \cdot K_r} \tag{3}$$

where Π_d is the daily productivity of the quarry by loosened mass, m^3/day

$$P_d = P_v \cdot n_{sm}, \tag{4}$$

A_z - the width of the excavator entry, m; $n_{sm}$the number of working shifts per day; Σh_l-the total height of overburden ledges, m; K_r - the coefficient of loosening of rocks; S_p - the width of the panel, m; P_v - the replaceable productivity of the quarry overburden;

– when excavating the overburden with one longitudinal entry

$$P_v = Q_1 + Q_2, \tag{5}$$

Q_1, Q_2 - replaceable operational productivity of working excavators along the horizons, thousand m3/cm.

– when conducting mining operations in a panel way with several excavator approaches within the panel

$$P_v = \frac{n_y \cdot S_w \cdot Q \cdot K_{b.z}}{A_z}, \tag{6}$$

where n_y is the number of overburden horizons; $K_{r.z}$ is the coefficient of safe gaps between adjacent excavators, m ($K_{r.z}$ = 0.8=1).

Figure 1 shows a graph for determining the capacity of a quarry by stripping, depending on the performance of excavators and the width of the panel, performed according to calculations in accordance with the methodology.

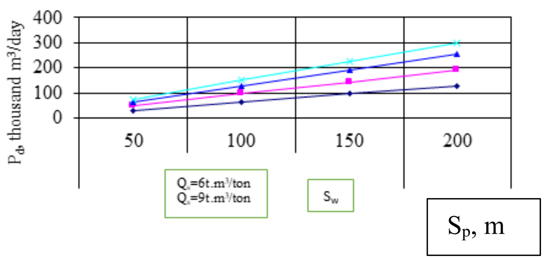

Fig. 1. The graph of determining the production capacity of the quarry (P_d) depending on the width of the panel (S_p) with different equipment performance (Q_d) according to technological schemes with excavator-automobile complexes.

In accordance with the schedules, the possible production capacity of the quarry was determined with the productivity of the excavator RH-200 Q_d = 9 thousand m^3 per day, the results of which are shown in Table 1.

Table 1. Production capacity of the quarry.

Technological schemes	Panel width, m	Production capacity of the overburden quarry, thousand m^3/day
With parallel arrangement of overburden, mining and dump work fronts	70	70
With the sequential arrangement of the overburden, mining and dump work fronts	20	25

The layout of the overburden, mining and dump work fronts in transport technological schemes can be parallel or sequential. When developing two overburden ledges by panel method and parallel arrangement of work fronts, the minimum length of the work front is 200-250m. adopted for road transport. With the sequential arrangement of the work fronts in accordance with fixed parameters and the dynamics of the development of mining operations, the minimum length of the work front at the quarry can be determined by the formula (Figure 2):

$$L_{fr} = 2 \cdot (L_{bb} + L_{gm} + V_p + R + L_{or} + L_{bp}) \quad (7)$$

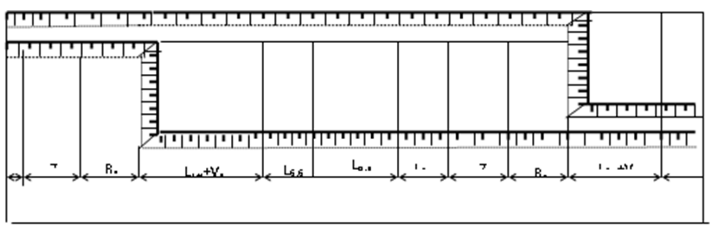

Fig. 2. Scheme for choosing the length of the mining front in technological schemes with sequential arrangement of the work fronts.

Figure 3 shows graphs of the dependences of the length of the mining front depending on the width of the panel and the average daily productivity of the quarry for stripping at different power ratios of external and internal stripping.

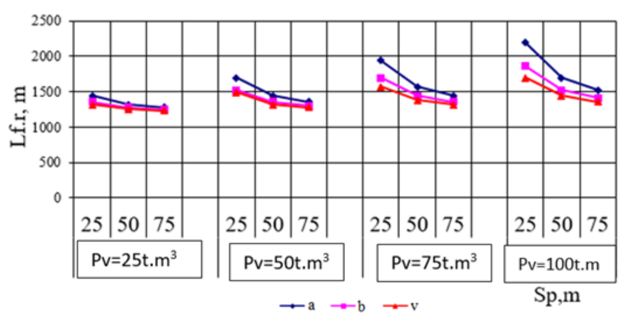

Fig. 3. Graph of the change in the length of the mining front under the schemes depending on the daily productivity of the quarry (Pv) and the width of the panel (Sw) with the ratios of the height of the overburden ledges Hb = 10 / Hn = 10 m (a), Hb = 10 m/ Hn = 20 m (b) and Hb = 20 m/Hn=20 m (in).

Figures 4 and 5 show schemes for determining the cross-sections of the working areas of the quarry, according to which the average width of the working areas of the quarry (B_{mid}) is determined by the formula

$$B_{mid} = (B_o + B_п)/2 \tag{8}$$

where B_o is the width of the base of the working area of the quarry, m; $B_п$ is the width of the surface of the quarry.

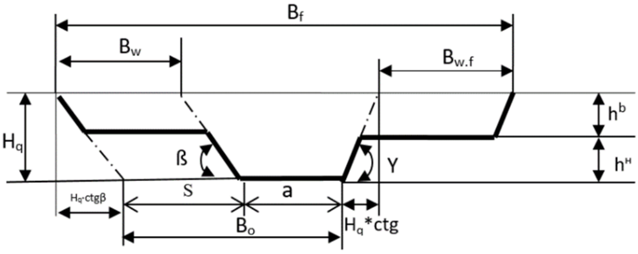

Fig. 4. Scheme for determining the transverse dimensions of quarry fields in a technological scheme with a sequential arrangement of mining fronts.

3 Results

The value of He and Vp according to technological schemes is determined by the formulas:

1. With technological schemes with sequential arrangement of mining fronts.

$$B_o=S_w+a \tag{9}$$

$$B_f = B_o + H_q * (ctg\beta + ctg\gamma) + B_w + B_{wf}, \tag{10}$$

Where, S_w -is the width of the panel, m; a -is the gap between the lower eyebrows of the inner blade and the mining block, m; B_w -is the width of the unloading area, m; B_{wf} -is the width of the working area, m.

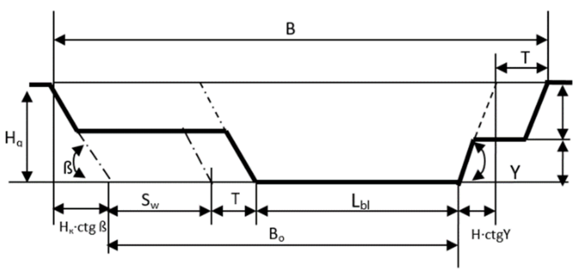

Fig. 5. Scheme for determining the transverse dimensions of a quarry with a technological scheme with a parallel arrangement of work fronts.

If there is a separate strip in the working area of the quarry for the excavation of the lower overburden ledge

$$B_{wf}=T+S_w$$

Also in the absence of a separate notch strip

$$B_{wf}=T,$$

T -is the width of the transport lane, m; β -is the slope angle of the blade, deg.; γ –is the slope angle of the overburden ledge, deg.

2. With a technological scheme with a parallel arrangement of work fronts

$$B_o = S_w + T + L_{bl}, \tag{11}$$

$$B_п = B_o + H_q * (ctg\beta + ctg\gamma) + T, \tag{12}$$

Where, L_{bl} -is the length of the lower block, m; S_w -is the width of the unloading area (panel), m.

Calculations according to the above methodology are performed in the panel width range of 20-200 m, since it is optimal within these limits. The calculation results are shown in the form of graphs in Figure 6, which allow for a given panel width to determine the average width of the working area of the quarry.

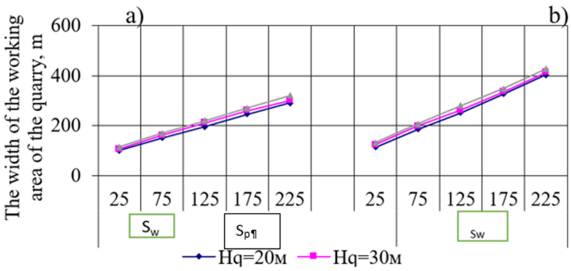

Fig. 6. Change in the weighted average transverse width of the working area of the quarry (S_{mw}) depending on the width of the entry (A_z) at a depth of the quarry from 20 to 40 m. a, b – technological schemes, respectively, in the absence and presence of a reserve strip of excavation of overburden ledges.

It can be seen from the graphs that the width of the working area of the quarry increases proportionally as the panel width increases and in the range from 100 m to 350 m, which can lead to different economic indicators and therefore an additional economic assessment is required. It should be noted here that the minimum dimensions of the working area of the quarry under various technological schemes are the main criterion for determining the size of the deposit to be worked out by one quarry field.

To analyze the patterns of changes in economic indicators from changes in the size of a career and, ultimately, to develop a methodology for choosing rational parameters of the career field, costs are considered in aggregate, since they are interconnected, and changes in one type of costs entail changes in another type of costs.

The rational dimensions of the quarry field for various technological schemes are determined by searching for the minimum value of the sum of all interrelated costs per unit volume of overburden

$$\sum Z_{so} = Z_s + Z_{m.v} + Z_o + Z_{t.r} + Z_{k.r} + Z_{p.e} + Z_{k.} \rightarrow min \qquad (13)$$

where Zs, Zm.v, Zo, Zt.r, Zk.r, Zpe, Zk – respectively, the specific operating costs for the construction of exits, the costs of overburden transportation, dumping, ore transportation, the costs of capital overburden during the construction of a quarry, the costs of distillation of mining equipment and transmission lines and the costs of recultivation of the quarry at the end of mining.

The costs of construction of exits, transportation of overburden and minerals, as well as the costs of dumping during transport technological schemes with the use of exits, are determined by formulas 14-15.

The specific operating costs for the intra - barrier transportation of ore and the protective layer of overburden in transport technological schemes without the use of exits can be determined by the formula

$$Z_{o.c(bt)} = \frac{h_{p(pr)}}{h_o} \cdot C_{tom} \cdot L_{lf} \cdot \gamma, \qquad (14)$$

where $h_{c(sj)}$ -is the capacity of the mineral (safety jacket), m; h_o -is the capacity of overburden rocks, m; C_{tom} -is the cost of a ton of kilometer of transportation of overburden and mineral, cu/m3; $L_{l.f}$ -is the length of the mining front, km; γ -is the density of rock mass, t /m3.

It is established that the structure of mining and capital costs in all technological schemes of development is the same, but depending on the change in the parameters of the quarry, their number changes. In transport technological schemes, the costs associated with the arrangement of exit trenches (exits) at the construction stage of the quarry are taken into account when justifying the parameters of the development system. At this stage of construction, the costs of overburden transportation are higher than during the operation of the quarry due to the additional lifting of overburden into external dumps, which can be determined by the formula

$$Z_{s.c.} = L_c \cdot C_{ckc} \cdot \gamma / 1000 = \frac{(H_q - h_p) \cdot C_{ckc} \cdot \gamma}{i_p \cdot 1000}, \quad (15)$$

where L_c -is the length of overburden transportation associated with additional lifts during external dumping; m; C_{ckc} -is the cost of transporting a ton of kilometer of cargo, cu/t km; H_q -is the depth of the quarry (ledge), m; h_c -is the capacity of the ore formation (layers), m; i_p -is the value of the guiding slope, dol. units.

Thus, the capital unit costs for the considered technological schemes can be determined by the formula

$$Z_{c.u.} = Z_{s.c.} \cdot \frac{V_{q.w.}}{V_{v.t.}} = \frac{L_{lf.} \cdot S_{mid} \cdot H_q}{L_{lf.} \cdot S_z \cdot H_q} \cdot Z_{s.c.} = \frac{S_{mid}}{S_z} \cdot Z_{s.c.}$$

or

$$Z_{c.u.} = \frac{S_{mid} \cdot (H_q - h_p) \cdot C_{ckc} \cdot \gamma}{S_z \cdot i_p \cdot 1000}, \quad (16)$$

where S_z -is the length of the deposit in the direction of the mining front (along the short axis of the quarry), m.

Graphs of the dependencies of mining and capital unit costs on the depth of the quarry, the length of the deposit and the width of the panel according to the considered technological schemes, performed on the basis of the described methodology, are shown in Figure 7.

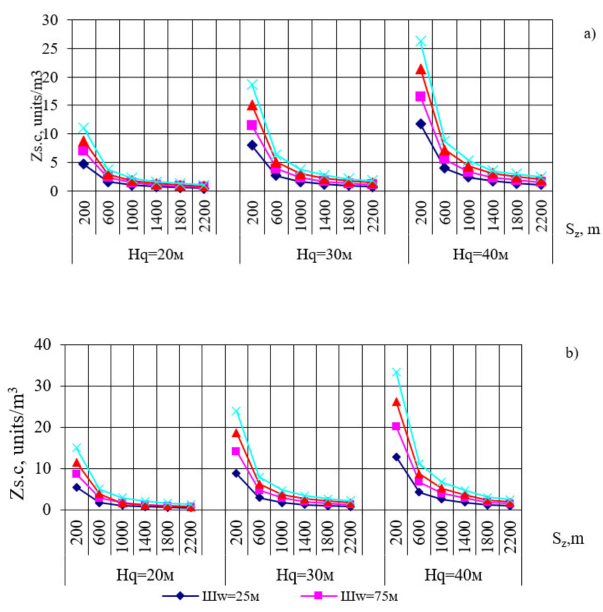

Fig. 7. Graphs of changes in specific mining and capital expenditures ($Z_{c.u.}$) depending on the depth (H_q), panel width (S_w) and length of the deposit (S_z). a, b - respectively, in the absence and presence of a separate strip of recess for the lower overburden ledge.

The graphs show that with an increase in the depth of the quarry and the width of the panel, the capital unit costs increase and decrease as the length of the deposit allocated to one quarry increases. At the same time, the rational length of the deposit developed by one quarry field in order to reduce capital unit costs should be taken within 1400-2000 m (depending on the depth of the quarry).

Studies have found that with a decrease in the length and width of the quarry field, operating costs increase due to downtime of mining and transport equipment after testing each entry or panel for their transfer to a new entry.

According to practice data, these costs are differentiated and amount to about 1.5-2 thousand units per 1 length (stage) of the mining front. The value of these costs decreases proportionally as the width of the developed panel ($Ш_w$) and the depth of mining (H_q) increase during the life of the quarry

$$Z_p = \frac{NpL_{lf}.\text{Ce.h}}{V_{c.d}} = \frac{\frac{S_Z}{S_W}L_{lf}.\text{Ce.h}}{V_{c.d}} = \frac{S_Z L_{\text{ф.р.}}\text{Ce.h}}{S_W S_Z L_{\text{ф.р.}} H_q} = \frac{\text{Ce.h}}{S_W H_q}, \tag{17}$$

where $C_{e.h}$ -is the unit cost per 1 m of equipment haul, cu/m^3; N_p -is the number of equipment runs; $V_{c.d}$ -is the volume of overburden in the contour of the deposit, m^3.

The graphs shown in Figure 8 are based on calculations, and show that the unit costs of moving equipment depend on the depth of the quarry and the width of the entry or panel being worked out. The unit cost of distillation is significantly lower with a panel width of $Ш_w$ >150 m.

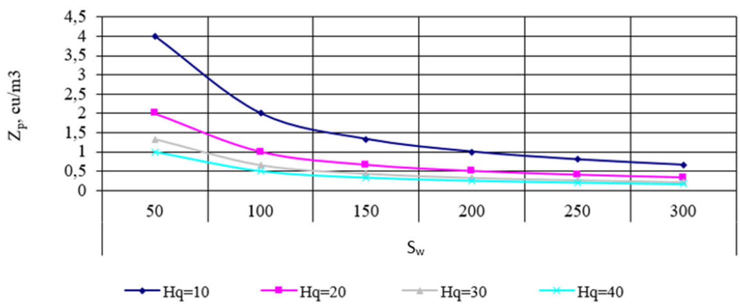

Fig. 8. Graph of changes in unit costs (Z_p,) for the distillation of equipment depending on the width of the panel (S_w or A_z) and the depth of development (H_q)

4 Conclusion

Thus, methods have been developed for engineering calculations to determine the size of the working area of quarry fields with various technological schemes for the development of low-power two-layer deposits of horizontal and shallow occurrence. At the same time, it is established that:

- The minimum sizes of quarry fields under various technological schemes are determined by the length of the work front, the width of the panels, the productivity of the quarry and the speed of development of mining operations selected for the specified mining and geological conditions.
- A method of engineering calculation of the length of the mining front has been developed and experimentally tested,
- The specific weight of capital and operating costs is directly proportional to the length, width and depth of the quarry, determined, in turn, by the parameters of the development system and the characteristics of the equipment used.

– Capital and operating costs are reduced as the length of the deposit increases and increase with increasing depth of mining and the width of the working area of the quarry. The minimum length of the deposit allocated for the development of one complex of mining and transport equipment under various technological schemes, where a minimum of costs is achieved, ranges from 2000-2500 m.

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