Ground control with backfill and caving in deep-level mining of gently dipping ore bodies

SYu Vasichev
Chinakal Institute of Mining, Siberian Branch, Russian Academy of Sciences, Novosibirsk, Russia
E-mail: s.vasichev@yandex.ru

Abstract. The research aims to solve the relevant problem of ground control through the combination of backfill and caving in complicated geomechanical conditions of stoping in thick and gently dipping ore bodies with reliable support of overlying rocks. This problem is conditioned by the absence of data of ground control using a combination of mining methods with caving and cemented backfill, which attracts scientific interest and needs a deep insight. The problem was solved by means of finding regular patterns in stress distribution in different modulus rock mass and based on the sound selection of parameters for ore and artificial pillars, including shape and size of caving zone, such that to ensure stability of undermined rock strata, using the finite element method. It is found that alteration of trapezoid-shaped areas mined out by backfill with caving, as well as making more accurate dome of natural equilibrium in the roof caving zone improves mining safety.

1. Introduction
Currently underground mining of thick and gently dipping ore bodies features high share of geotechnologies with backfill [1–7]. These geotechnology are safely applicable within the whole range of geological and geomechanical conditions. On the other hand, in low-grade ore bodies, the geotechnologies with backfill become inefficient, and less expensive methods of mining are required. In this connection, it seems to be interesting to estimate conditions and range of for application of a new ground control method with caving and backfill [8-9] which ensures reliable stability of overlying rock strata within the whole period of mining at the considerably reduced cost.

2. Method of ground control with caving and backfill
The new hybrid method of ground control (Figure 1) is based on the mechanism of interaction between undermined strata and artificial and ore pillars, as well as influence of their parameters and order of stoping on mechanical behavior of rock mass.

It is known that the key objective of backfill is to ensure stability, prevent fall and limit displacement velocity of overlying rock mass within a stoping area. For this reason, an artificial mass as a load-bearing structure should meet definite standards strength and deformation. The degree of influence exerted by backfill on stress state of undermined rock mass depends on many factors, one of which is the presence of the backfill-and-rock contact. Given this condition is fulfilled, the interaction mechanism of roof and backfill is governed by the ratio of their physical and mechanical characteristics. The governing characteristic among the latter is combination of deformation characteristics of rock mass and height of pillar (thickness of ore body). Thus, application of mining...
systems with backfill depends first of all on the sound selection of strength and deformation characteristics of backfill material.

![Figure 1. Hybrid method of ground control with caving and backfill](image)

Complete roof caving eliminates high inputs of labor, time and material to make backfill. However, the application field of complete roof caving is limited.

The overlying roof rocks should have such mechanical properties that caving results in no large overhangs above the mined-out area. Formation of large voids under the uncaved roof increases stresses at the rock mass margins, as well as constitutes a hazard of uncontrollable roof collapse, bumps and seismic loads in mines. Induced caving allows formation of a safety cushion composed of caved roof rocks, which reduces ore loss and prevents bumps. On the other hand, as practice shows, caving to mined-out area has a weak influence on concentration of the abutment pressure. Furthermore, application of systems with caving is limited in case of difficult roof.

Another important moment of ground control is deformability of backfill pillars in the course of mining of ore pillars. Extraction of pillars without backfill dictates application of induced roof caving, which is required for reduction in ore loss and for smooth subsidence of overlying strata as the associated swell of rocks creates lateral support to the backfill pillars and improves their stability. The pressure of overlying rocks above the mined-out area is considerably higher than the load-bearing capacity of backfill pillars, which results in compaction of the latter and their transition to the post-limit state. At the same time, owing to lateral support from the side of caved rocks, vertical strains which given roof caving are largely limited. By the effect of displacement limitation, such support of backfill pillars by induced roof caving is comparable with application of loose backfill in open stopes.

In the geotechnology in Figure 1, the first stage is extraction of stopes with trapezoid cross section to the total thickness of ore body. Then, the mined-out stopes are filled with cemented backfill. In this fashion, the structure of backfill pillars and temporal ore pillars is formed. Redistribution of load to the backfill pillars is achieved through partial extraction of marginal reserves in the ore pillars by stopes expanding upward. Then, caving of the overlying rock roof between the backfill pillars is induced. As
a result, the central solid temporal pillar is totally relaxed from stresses. The caved rocks fill the upward expanding stopes and create lateral support for the artificial pillars.

The final stage of stoping is extraction of the massive ore pillar by sublevel caving (block caving). After ore reserves are extracted from this central pillar, in the mined-out void, at the sidewalls of the backfill pillars sloped supports are made. The ground control uses the dome of natural equilibrium supported by the backfill pillars.

The neighbor ore pillars should be extracted so the the upward expanding stopes of the same stage form a bench front of stoping.

3. Stability analysis of structural elements of geotechnology

The technological uncertainty in the mining methods can be eliminated by multi-variate modeling of the structural parameters which govern safety and efficiency of mining. Financial risk is assumed as the main cause of economic hazard which conditions sustainability of a mining method. Evidently, successful introduction of a new method in practical mining requires qualitative and quantitative estimation, as well as geomechanical assessment and feasibility study to be accomplished.

For finding regular patterns in interaction of undermined rock mass with backfill and ore pillars, as well as influence of their parameters on the structural stability of the mining system, a set of problem on stress distribution was solved by the mathematical modeling using the finite element method [10]. The numerical estimates were obtained for three-dimensional elastic compression with tectonic distribution of natural stresses in rock mass (lateral earth pressure coefficient $q = 1.56$ [11–13]) in terms of the Norilsk mines for the mining depth $H$ of 1200 m at the ore body thickness $m$ of 40 and 20 m.

The main equations describing the stress state of the study object were: static relations $\sigma_{ij,j} + pF_i = 0$, geometrical relations $\varepsilon_{ij} = 0.5(u_{i,j} + u_{j,i})$, and physical relations $\sigma_{ij} = 2G\varepsilon_{ij} + \lambda\theta\delta_{ij}$, as well as boundary conditions in the form of compressions and constraint normal displacements at the boundary of the computational domain (where $\sigma_{ij}$ are the stress tensor components; $pF_i = \gamma g\delta_{ij}$ are the bulk forces; $\gamma$ is the density of rocks; $g$ is the acceleration by gravity; $\varepsilon_{ij}$ is the strain tensor; $u_i$ is the displacement vector; $\theta = \varepsilon_x + \varepsilon_y + \varepsilon_z$ is the relative volumetric stain; $G$ and $\lambda$ are the Lamé parameters: $G = \frac{E}{2(1+\mu)}$, $\lambda = \frac{E\mu}{(1-2\mu)(1+\mu)}$; $\delta_{ij}$ is the Kronecker delta; $\mu$ is Poisson’s ratio).

As the obtained stress–strain behavior of rock mass, the stability analysis of the geotechnology elements was performed by the Mohr–Coulomb criterion [13–22]:

$$2C \cos \phi + (\sigma_1 + \sigma_3) \sin \phi \geq (\sigma_1 - \sigma_3),$$

where $\sigma_1$ and $\sigma_3$ are, respectively, the maximal and minimal principal stresses; $C$ is the cohesion of rocks; $\phi$ is the internal friction angle.

The physical and mechanical properties of the model media are described in table 1. Figure 2 presents distribution of the maximal shear stress $\tau_{\text{max}}$ in the structural elements of the hybrid geotechnology during extraction of marginal reserves from the temporal ore pillar and formation of the caving zone above it in the form of the dome of natural equilibrium in terms of the typical conditions of deep-level mining ($H = 1200$ m) in the Oktyabrsky Mine.

| Rocks                  | $\gamma$, kg/m$^3$ | $\sigma_{\text{comp}}$, MPa | $\sigma_{\text{con}}$, MPa | $C$, MPa | $\varphi$, deg | $\mu$ | $E$, GPa |
|-----------------------|--------------------|-----------------------------|-----------------------------|----------|----------------|-------|----------|
| Disseminated ore      | 4000               | 90–140                      | 8–16                        | 15–25    | 35–55          | 0.26  | 40       |
| Gabbro-dolerite, dolomite, limestone | 2700               | 80–160                      | 5–16.5                      | 12–30    | 33–52          | 0.24  | 50       |
| Cemented backfill     | 2000               | 4–6                         | 0.1–0.5                     | 1.4      | 25             | 0.32  | 2.0      |
Figure 2. Distribution of shear stresses $\tau_{\text{max}}$ in the structural elements of the geotechnology with backfill and caving (cross section of filled stopes and ore pillars) at $m = 40 \text{ m}$, $h_r = m$, $B_{c.p} = 80 \text{ m}$, $B_{c.k.v} = 70 \text{ m}$, $B_{k.p} = 30 \text{ m}$, $B_{c.p.v} = 20 \text{ m}$ and $H = 1200 \text{ m}$.

The calculations show that (Figure 3) zones of critical state form in the areas of increased concentrations of shear stresses $\tau_{\text{max}}$. These zones are the roof and floor in the filled trapezoid stopes, edges (through the whole thickness of the ore body) of the ore pillar at the stage of ore extraction from the upward expanding stopes, as well as the roof in the dome of natural equilibrium. In the meanwhile, the zone of probable failure in the overlying strata, owing to the roof made as the stable arch in the stopes, is relatively small, which ensures high safety of operations at the stage of mining of the ore pillar (uncontrollable damage and bumps are eliminated). Thus, the effect of the arch caving zone on the stability of the overlying rock mass is evident.

Figure 3. Stability index $K_s$ in structural elements of mining system (values $< 1$ characterize instability zones).
It is worthy of mentioning that roof caving brings considerable relaxation of the ore pillar from stresses (the pillar is bounded by exposed surfaces on three sides), which also creates favorable conditions for extraction.

While ore reserves are extracted from ore pillars, the geomechanical situation changes. The modeling finds out that the increase in the total span of stoping, i.e. the section of alternate backfill pillars and mined-out ore pillars, results in essential instability and failure of the backfill. The calculations yield that the cemented backfill strength to ensure the rated load-bearing capacity and stability of surround rock mass around the stoping area during extraction of ore reserves from the pillar should be not less than 6.0 MPa. However, as the total stoping area is enlarged, the backfill with the uniaxial compressive strength of 6.0 MPa gradually loses stability (fracture zones grow in the backfill); this results in the smooth subsidence of the overlying rock mass, which eliminates dynamic events due to rock pressure.

The influence of the structural parameters of the geotechnology on the behavior of the temporal ore pillar was also analyzed (Figure 4). At the stage of mining of ore reserves from the marginal areas of the pillars by trapezoid stopes with upward expansion, extra load on its internal area greatly increases with decreasing width. As the width of the artificial support (filled stopes) grows, so does the load on the internal area of the massive ore pillar (its width being fixed). The latter is governed by the increase in the total span of stoping (stope+pillar).

From the obtained results, the safest parameters of the filled stopes and ore pillars, with regard to mining stages and static subsidence of the overlying strata, in mining of ore bodies with thickness $m = 40$ m at great depths (1100–1300) are: $B_{k,p} = 40–70$ m, $B_{c,p} = 80–120$ m.

4. Conclusion

1. The combination of mining systems, formation of artificial support (backfill), caving zone roof in the form of the dome of natural equilibrium, as well as structural features of ore extraction from pillars allows reliable ground control with elimination of dynamic phenomena due to rock pressure.
2. As the total span of stoping grows, the artificial (backfill) pillars together with the caving zone above the ore pillars create conditions for smooth subsidence of the overlying strata.
3. The efficient ratio of the width of ore pillar to the width of the backfilled stope, considering volumes of extraction by the systems of backfill and caving, is 1.6–2.5.

References

[1] Freidin AM, Shalaurov VQA, Eremenko AA et al 1992 Improvement of Efficiency in Underground Ore Mining in Siberia and Far East of Russia Novosibirsk: Nauka (in
(Russian)

[2] Bronnikov DN, Zamesov NF and Bogdanov GI 1982 Deep-Level Ore Mining Moscow: Nedra (in Russian)

[3] Galperin VG, Yukhimov Yal and Borsuk IV 1986 Foreign Experience of Deep-Level Mining Moscow: TSNIIETS (in Russian)

[4] Feng D-K, Zhang J-M and Hou W-J 2018 Three-dimensional direct-shear behaviors of a gravel-structure interface Journal of Geotechnical and Geoenvironmental Engineering 144(12) 04018095

[5] Celleri HM, Sánchez M and Otegui JL 2018 Fracture behavior of transversely isotropic rocks with discrete weak interfaces International Journal for Numerical and Analytical Methods in Geomechanics 42(18) pp 2161–2176

[6] Wei X, Li C, Zhou X, Hu B and Li W 2017 The change laws of strength and selection of cement–sand ratio of cemented backfill Geotechnical Engineering 48(4) pp 144–150

[7] Berkane A and Karech T 2018 Numerical modeling of the pathological case of a damaged tunnel application to Djebel El-Ouahch tunnel (east–west highway) Asian Journal of Civil Engineering 19(8) pp 913–925

[8] Fredin AM, Vasichev DSYU et al 2012 RF Patent No 2454540 Method of ground control Byull. Izobret. No 18

[9] Neverov AA, Neverov SA, Tapsiev AP, Shchukin SA, Vasichev SYu 2019 Substantiation of ore deposit mining geotechnologies based on the development of model representations on changing natural stress field parameters J. Min. Sci. Vol 55 No 4 (in print)

[10] Zienkiewicz OC 1971 The Finite Element Method in Engineering Science McGraw Hill

[11] Neverov SA 2012 Types of orebodies on the basis of the occurrence depth and stress state. Part I: Modern concept of the stress state versus depth J. Min. Sci. Vol 48 No 2 pp 249–259

[12] Neverov SA 2012 Types of orebodies on the basis of the occurrence depth and stress state. Part II: Orebody tectonotypes and geomedium models J. Min. Sci. Vol 48 No 3 pp 421–428

[13] Neverov AA 2014 Geomechanical assessment of combination geotechnology for thick flat-dipping ore bodies J. Min. Sci. Vol 50 No 1 pp 115–125

[14] Neverov AA 2012 Geomechanical substantiation of modified room-work in flat thick deposits with ore drawing under overhang J. Min. Sci. Vol 48 No 6 pp 1016–1024

[15] Freidin AM, Neverov SA and Neverov AA 2016 Geomechanical assessment of compound mining technology with backfilling and caving for thick flat ore bodies J. Min. Sci. Vol 52 No 5 pp 922–942

[16] Özdemir M, Kahraman B, Doğruöz C and Yalçın E 2016 Evaluation of a coal mine located in çan, çanakkale using 3D modeling 6th International Conference on Computer Applications in the Minerals Industries Istanbul Turkey pp 1–4

[17] Wael R Abdellah, Mahrous AM Ali, Gamal Y Boghdady and Mohamed E Ibrahim 2016 Evaluation of the effect of rock joints on the stability of underground tunnels Journal of Civil Engineering and Architecture Research November 25

[18] Neverov AA, Konurin AI, Shaposhnik YuN, Neverov SA and Shaposhnik SN 2016 Geomechanical substantiation of sublevel-chamber system of developing with consolidating stowing 16th International Multidisciplinary Scientific Geoconference–SGEM 2016 Bulgaria: Albena Vol II pp 443–450

[19] Freidin AM, Neverov SA, Neverov AA and Konurin AI 2018 Validation of choice and determination of geotechnology parameters with regard to stress-strain state of rocks IOP Conference Series: Earth and Environmental Science 134(1) 012019

[20] Kazikaev DM 2005 Geomechanics of Underground Ore Minine: University Textbook Moscow: MGU (in Russian)

[21] Bieniawski ZT 1987 Strata Control in Mineral Engineering CRC Press

[22] Turchaninov IA, Iofis MA, Kasparyan EV 1989 Fundamentals of Rock Mechanics Leningrad: Nedra