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Review

Review of Underground Mining Methods in World-Class Base Metal Deposits: Experiences from Poland and Chile

Krzysztof Skrzypkowski [1,*], René Gómez [2,3], Krzysztof Zagórski [4], Anna Zagórska [5] and Roberto Gómez-Espina [2]

1 Faculty of Civil Engineering and Resource Management, AGH University of Science and Technology, Mickiewicza 30 Av., 30-059 Kraków, Poland
2 Faculty of Engineering, Universidad de Concepción, Concepción 4030000, Chile
3 Advanced Mining Technology Center, Santiago 4070371, Chile
4 Faculty of Mechanical Engineering and Robotics, AGH University of Science and Technology, Mickiewicza 30 Av., 30-059 Kraków, Poland
5 Research Centre in Kraków, Institute of Geological Sciences, Polish Academy of Science, Senacka 1, 31-002 Kraków, Poland
* Correspondence: skrzypko@agh.edu.pl

Abstract: There are several massive deposits around the world with different geological characteristics. Thus, different mining methods and strategies are applied based on the particularity of each method and mine experience. Particularly, in this work, we review and summarize the underground exploitation of some world-class base metal deposits based on Poland and Chilean experiences. Here, the main geological and mining parameters of Poland and Chilean mines applied in massive deposits are reported and analyzed. In Poland, mainly room and pillar methods (and variants) have been applied in massive deposits. Here, back-filling is required to maintain the mine's stability due to the large deposit size and open areas. In Chile, the block caving method is commonly used in massive underground deposits where less development is required. Here, the cave is naturally filled with broken material and a large subsidence zone is generated. In this review, it has been observed that different underground methods and strategies can be effectively used in massive deposits. Some parameters that influenced the method selection are mainly related to rock mechanics, ore recovery and dilution, subsidence zone, extraction rate, and mining experience. Here, key mining variables and parameters such as productivity, support, and equipment, as well as various issues related to the world-class deposit are studied. Additionally, a comparison between both experiences is presented, highlighting the main geological and mining parameters. This study can be used as a reference to evaluate the different option of underground mining methods to be applied in future massive mine projects with similar geological characteristics.

Keywords: block caving; massive deposit; mine design; room and pillar; underground mining; world-class deposit

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1. Introduction

The technical and economic progress in the exploitation of ore deposits consists of the introduction of modern technologies, mechanization, and automation of processes while meeting the requirements of occupational safety and protection of the deposit and the human environment. In the mining process, in terms of deposit protection, underground mining methods are of particular importance, for which the increase in extraction, labor efficiency, and lower extraction costs should be accompanied by a reduction in operating losses. Mass mining is frequently applied worldwide, related commonly to low-grade deposits, large depths, water tables, and high capital costs. In particular, in underground mining, these deposits are related to several challenges that have to do with the mining method, ventilation, and drainage requirements, mine support, and mine planning. Mass

mining is defined as that with production greater than 10 kt/day or 3 Mt/year [1]. Mass-mining methods are commonly applied to low-grade, high-tonnage deposits, so the size and shape of the deposit are relevant characteristics in determining the suitability of a mass-mining deposit [2].

The characteristics of the rock mass are of vital importance to determine the most suitable mining method based on the stress deformations that occur according to the chosen method [3]. In addition, it is necessary to select the extractive method considering the operational costs [4]. To face these challenges, a methodology is proposed based on an algorithm to choose the optimal method for the development of deposits, considering the geological, technical, and operational factors of the deposit [5].

Ladinig et al. [2] mainly identify two typologies of deposits favorable for these exploitation techniques: narrow tabular deposits of large area extent, as might be the case with some coal deposits and thick tabular deposits and massive deposits, characterized by a large extension in all three spatial directions. In general, this last category adjusts quite well to the so-called world-class deposits. Although the typologies and formation processes of the deposit that this term encompasses are varied, these are characterized by being large-tonnage and volumetrically extensive deposits. These characteristics are more remarkable in the deposits of base metals compared to those of precious metals, presenting, in addition to higher tonnages, higher grades.

Singer [6] defined world-class mineral deposits as the upper 10% of all deposits in terms of the content of a given metal. The term world-class deposit is an informal term applied to ore deposits with an exceptionally large tonnage of economically recoverable ore [7]. This author points out that world-class deposits can be divided into giant and supergiant, with an ore metal content in a deposit/metal greater than 1×10^{11} t and 10^{12} t of average crust material, respectively. For example, in the case of gold, the resource in a giant deposit is equal to or greater than 100 t Au, and supergiants are deposits that contain equal to or greater than 1200 t [8]. For copper, supergiant deposits are defined as those with more than 24 Mt Cu, and giants as having more than 2 Mt Cu; 2400 t of silver, 1.7 Mt of zinc, or 1 Mt of lead are required to be considered a world-class deposit [6]. Although this definition and the limits it uses are relatively old, taking into account the advancement of exploration techniques and the continuous discovery of new large, mineralized ore bodies, these ranges are still used today to categorize deposits as world-class, for example in the works of [9–12], among others. Among the world-class deposits of base metals, there are different typologies according to their genesis, ores, lithologies, etc. Porphyry copper deposits stand out for their tonnage, these being the world's principal source of Cu and Mo [13]. Another important type of deposit that occasionally gives rise to world-class mineralization is a sediment-hosted stratiform copper deposit. There is a low number of known stratiform copper deposits, but with very attractive Cu and Ag grades and tonnages [14]. Other types of deposits such as the manto type, Sedimentary Exhalative deposits (SEDEX), Volcanogenic Massive Sulphide (VMS), iron oxide copper gold (IOCG), and Mississippi Valley Type (MVT) are also main sources of base metals that occasionally generate massive ore classifiable as world-class [15].

Different criteria can be used to define a world-class deposit. However, in this work, we focus on the underground mine activities applied in large deposits using Poland and Chilean large deposits as the example. A review of current underground mines within massive deposits is a useful tool to identify the main methodologies used nowadays that have shown good results and to identify the lessons learned in these environments. Therefore, the purpose of this study is to analyze different underground mines in Poland and Chile focusing on key parameters such as productivity, support, and equipment, as well as main issues.

2. Chilean and Polish World-Class Base Metal Deposits

In Chile, the most abundant deposits and the largest tonnages correspond to porphyry copper deposits. Porphyry deposits occur in arc-related settings of various ages throughout the world. However, giant systems are restricted to only a few mineral provinces and periods [16]. In accordance with reference [11] among the 31 largest porphyry copper deposits discovered to date, 14 fall under the supergiant class (Chuquicamata, El Teniente, Los Bronces, Escondida, Los Pelambres, Collahuasi, Pebble Copper, Safford, Morenci/Metcalf, Continental/Butte, Almalyk, Grasberg, Oyu Tolgoi, and Bingham) of which 6 are in Chile. The porphyry deposits are characterized by low-grade copper, gold, and/or molybdenum mineralization developed within and around a porphyritic intrusive complex, where vein stockworks and hydrothermal breccias are common [16].

The iron oxide copper gold deposits (IOCG) in north-central Chile form part of the Andean IOA-IOCG belt, which extends from immediately north of Santiago to north of Antofagasta. These IOCG deposits are commonly spatially associated with or hosted in faults that form part of the Atacama fault system [17]. Among all the deposits associated with this belt (El Espino, Montecristo, Mantoverde, etc.), the Candelaria–Punta del Cobre district is the most important due to its size. It is within the range of world-class deposits. The majority of the IOCG mineralization in the district is hosted in the upper part of the lower andesite unit and the overlying volcanic-sedimentary and dacite units, all within the Punta del Cobre Formation [17]. Mineralization is hosted in fault zones, breccias, and specific lithologies.

Another type of common deposit in Chile that can reach large tonnages, although they do not reach the magnitudes of porphyries, are the stratabound deposits, known as manto type. These deposits are the third source of Chilean copper production after the porphyry copper deposits and the IOCG deposits [18]. Similar deposits in North America are named "volcanic red-bed" [19]. Within this manto-type located in Chile are the deposits of Papomomo [20], Mantos Blancos, Cerro Negro, and El Soldado, among others. They are mainly Cu–Ag mineralized bodies that are hosted in Upper Jurassic to Lower Cretaceous volcano sedimentary sequences along the Chilean coast from center to north [21]. They originate in back-arc extensional basins associated with calc–alkaline volcanic belts in the continental crust on an active convergent margin. The manto-type deposits in Chile present very similar characteristics, with comparable geometries, mineralization controls, and mineral paragenesis. They are considered epigenetic, hydrothermal, or metamorphic fluid origins are suggested, and albite is the most common alteration. The main controls are lithologic (permeable strata), structural (extensional faults), or intrusion-related [22]. Ore appears in andesitic lavas and occasionally in volcanoclastic and sedimentary rocks, where it is commonly disseminated and associated with zones with organic matter. Pyrobitumen has been described in stratabound deposits in Chile, such as El Soldado [23] and Papomomo [22]. These manto-type Cu deposits tend to display relatively high grades with variable quantities of Ag and Au as a by-product [24].

Poland is a country with abundant mineral resources ranging from coal and copper to other raw materials, such as rock salt or zinc, and lead ores. Regarding copper, the most important source of this metal produced in the world in addition to porphyry copper deposits involves sediment-hosted stratiform copper deposits [25,26]. These copper deposits are an important, economically attractive, world-class mineral deposit type, traditionally represented by supergiants such as the Kupferschiefer of north-central Europe and the Copperbelt of Central Africa [14]. In the New Copper district, located on the Polish side of Kupferschiefer, resources were identified in 2018 totaling 32.62 Mt Cu and 97,938 kt Ag [27]. The Lubin, Polkowice-Sieroszowice and Rudna mines are in this district. The principal host rock in the Kupferschiefer deposit is shale, with 2 to 10% Cu content, and a thickness of up to 20 m [28]. Structurally it is located on a monocline dipping gently to the north-east (less than 12°).

The morphology and distribution of mineralization in these deposits are markedly different from that of the porphyries. Brown [14] affirms that sediment-hosted stratiform copper deposits tend to be high-tonnage deposits because of their wide lateral extents

along preferred stratigraphic units, and their copper grades frequently surpass those of porphyry copper ores. In addition, they may contain significant amounts of other highly desirable metals such as silver and cobalt.

The Upper Silesian ore district in south-central Poland is an important producer of zinc, lead, and silver [28]. Five clusters of orebodies have been discovered: the eastern Olkusz area (Pomorzany and Olkusz), the southern Chrzanów area (Trzebionka mine), the western Bytom area, and two northern clusters in the Zawiercie and Klucze area [29]. These deposits have been classified as Mississippi Valley Type [22,29,30]. The ore forms replacements, cavity fillings, linings, veins, and mineralized breccias [31]. The lower and upper units of the sequence enclosing the ores are marly or argillaceous sediments. The main ore minerals include sphalerite, galena, pyrite, and marcasite, accompanied by the gangue minerals dolomite, calcite, barite, chalcedony, and quartz. According to Sass-Gustkiewicz and Kwiecinska [29], the host rock of the sulfide ores is a coarse-crystalline, ore-bearing dolomite that tends to occur in the form of extensive, roughly tabular bodies in Triassic sequences or bodies of various shapes in Devonian carbonates. The orebodies in Triassic rocks occupy various positions within the ore-bearing dolomite and are tabular, lenticular, or nestlike, except for a single, large, chimney-like body hosted in Roethian carbonates. The Devonian orebodies occur as vertical, sub-vertical, and/or horizontal forms. The vertical bodies contain abundant mineralized breccias, and higher in the sequence, grade into horizontally disposed ores in both the Paleozoic and Triassic rocks [32]. The total extractable metal content of the district is estimated at 30 Mt [28].

3. Chilean Study Cases

In this section, three Chilean underground mines that are currently in different stages are reviewed. These mines are exploited using caving methods (block and inclined). The caving methods are commonly applied in massive underground deposits, highlighting the use of block and sub-level caving. Block caving is an underground method commonly applied in Chile [33], used in important copper deposits such as the El Teniente mine, Salvador mine, Andina mine, and currently in transition from open pit mining, the Chuquicamata underground mine is also included [34]. The Chuquicamata underground mine transition to block cave mining is a consequence of the deepening of reserves [35], which also could occur in other Chilean massive deposits (actually open pit mines), such as in the Escondida mine, Collahuasi mine, Los Bronces mine, and Radomiro Tomic mine. Additionally, the Papomono mine is a new underground mine project that will be exploited by the Inclined Caving method [36].

Block caving is an underground method applied in massive ore deposits. This method relies on gravity and induced mining stresses to cave the orebody which is known as caved rock [37,38]. The caved or fragmented rocks are extracted using gravity by drawing them from drawpoints located at the production level. The current challenges of this method, as well as other underground methods, are related to deeper deposits, hard rock, high-stress fields, higher production requirements, and uncertainties in ground conditions [39–41].

3.1. Copper Exploitation in the Coquimbo Region

The Papomono mine is in the Coquimbo Region (north-central, Chile). The Deposit consists mainly of a sequence of andesite interbedded with pyroclastic rocks of the Quebrada Marquesa Formation, N 40° W strike, dipping 15° to the SW. The deposit is 450 m long and about 80 m wide.

Papomono mine reserves were calculated to be 2.68 Mt at 1.22% of recoverable Cu [36]. In Papomono, the favorable lithological horizons that served as fluid conduits were strongly affected by a pervasive to sparse albitic alteration [22]. The lithologies have a thickness that ranges from 2 to 50 m, up to 500 m strike extension and length along dip reaching 600 m. In addition to the most common stratiform geometry, these deposits can present lenses, breccia pipes, veins, or irregular shapes, sometimes all together in the same deposit [22].

3.1.1. Underground Mining Method

The underground mine has two main access points, one located at 1429 m.a.s.l. and another located at 1315 m.a.s.l. The selected mining method is Inclined Caving, which is currently under construction. The deposit inclination and poor rock strength are some of the main reasons for the selected underground method. The production level is divided into four sub-level spaced 11 m in height and 13 m in width, with extraction drifts of 4 × 4 m. Here, a total of 97 drawpoints were defined. A total of 5173 m of horizontal development and 182 m of vertical excavations (including ventilation and ore pass requirements) are required. Figure 1 shows an example of an inclined (front) caving design.

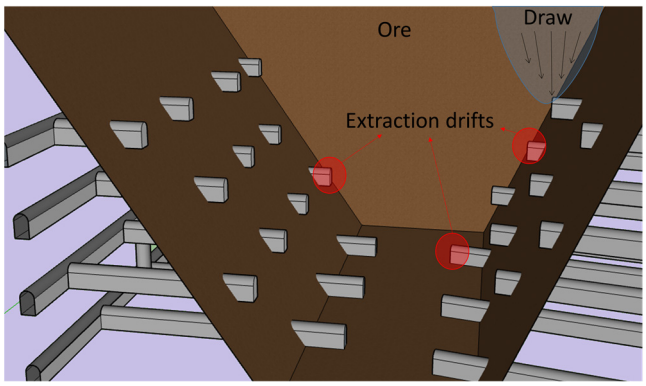


Figure 1. Example of inclined caving design, isometric view.

In this mine, there is no undercut level. Then, drawbells are drilled and blasted to start the caving. A small hydraulic radius of 9 m is estimated to start the caving process. This parameter was estimated using an MRMR of 20. In this method, the draw strategy is key to ensuring interaction between drawpoints, increase ore recovery, and delay dilution. Here, the drawpoint spacing is a critical parameter to ensure drawpoint interaction (flow zones) and production pillar stability.

3.1.2. Mining Support

The support requirements used in this project included the following: helical bolts (2.6 m length; 22 mm diameter), self-drilling bolts, electro-welded mesh (C-196) and brained steel mesh, internal anchoring with grouting, resin and expansion pin, shotcrete with and without fiber (150 mm), and reticulated frames. The support design varies depending on the rock mass quality of each sector. Figure 2 shows an example of the support design applied in a drift.

3.1.3. Equipment

In the current construction phase, there has been different mining equipment such as one dumper, two arm jumbos, one arm jumbo, one Simba, four mixer trucks, two LHD of 6 yd^3, two robotshots (equipment used for shotcrete projection), and one backhoe with a hammer for scaling. The extraction equipment requirement is currently under study. These probably will include one or two LHD due to the low production rate and the trucking system to move the ore out of the mine.

3.2. Copper and Molybdenum Exploitation in the Antofagasta Region

The Chuquicamata deposit is in the foothills of the Atacama Desert, west of the modern volcanic arc of the Andes Mountains. The deposit is of the porphyry copper type and is related to intrusive magmatism of Eocene—Oligocene age (41–31 Ma), a porphyry copper belt that extends for about 1400 km, from 18° S to 31° S [42,43]. The orebody geometry is 3000 m north–south, 350 to 800 m east–west and the drilling campaign suggested the existence of more than 900 m of mineable ore from the final pit bottom [44]. An important

geological feature is the west fault observed in Figure 3. This fault separates the waste on the western side from the ore body.

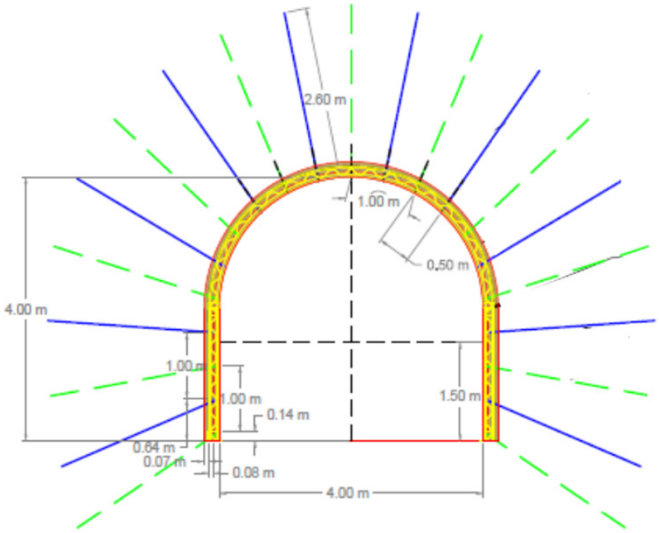


Figure 2. Example of drift support used in Papomono mine, including Helical bolts (blue), Shotcrete (red), and frames (yellow).

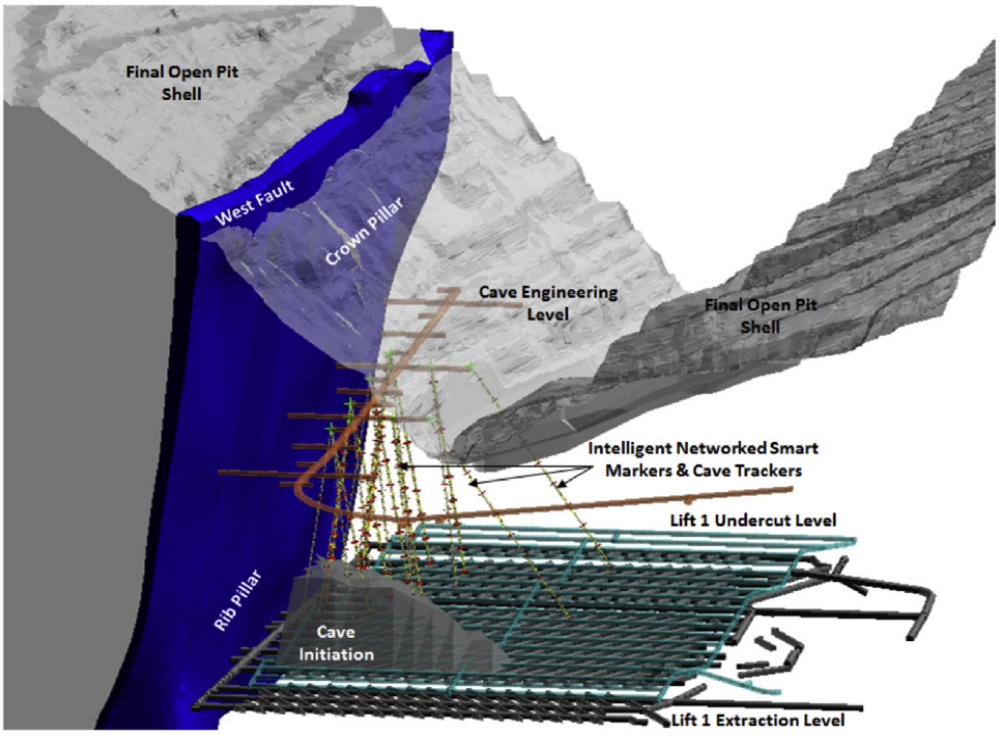


Figure 3. Chuquicamata transition project and west fault [44], isometric view.

3.2.1. Underground Mining Method

This project is a transition from an open pit project that was transitioned to an underground mine due to the open pit depth (1.1 km). The underground method selection was block caving with macroblocks applying the El Teniente extraction level layout [44,45]. The ore column heights are 216 m for level 1, 432 m for level 2, and 216 m for level 3 [44]. The ore extraction already started at the first macroblocks (in the center of level 1) while level 1 continues to be under development, at the time of this writing.

In this project, preconditioning through hydraulic fractures and confined blasting is applied in the rock mass over the production level to improve the caving process. The holes were drilled from some drawbells, the undercut level, and a hydro-fracturing level. The undercutting level is west–east oriented, with galleries of 4.2 m × 4.4 m. This size is defined by the driller equipment and ventilation ducts.

In Figure 4, the production level development is shown where three macroblocks are completed (N2N3, N1S1, and S2S3). Different extraction layout spacing is used based on the macroblocks (32 × 16 m and 32 × 20 m [46]). For example, two macroblocks (N2/N3) include the following: 300 confinement walls, 278 drawpoints, 1278 Norwegian frames, 15 IE frames, 9689 m of horizontal development at the production level, and 4694 m of development at the undercut level. The production drifts are 5.0 × 4.5 m, by applying the El Teniente layout.

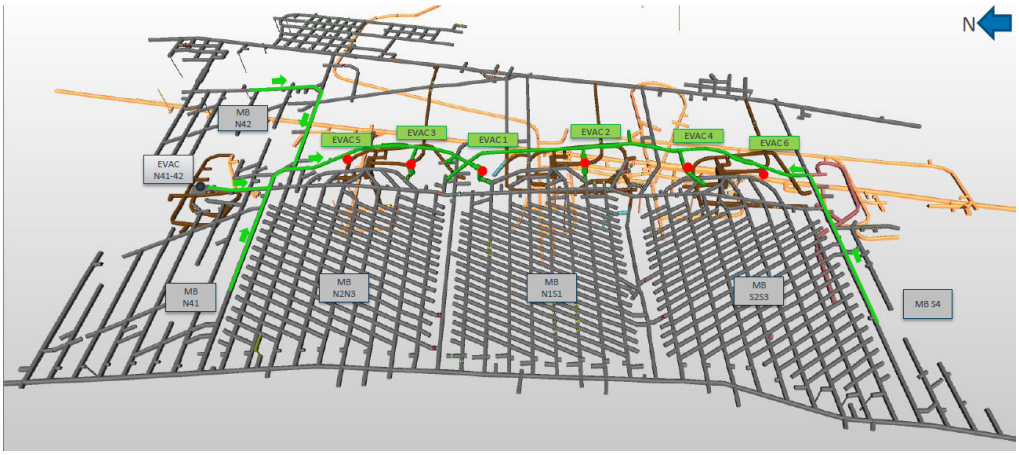


Figure 4. Macroblocks development in level 1, Chuquicamata underground mine [47].

3.2.2. Mining Support

The supporting element and structures used in this project are helical bolts, cable bolts, shotcrete, steel frames (H type), and confinement walls. Confinement walls are used to reinforce pillars, which consist of a group of concrete blocks reinforced with steel bars with the purpose of maintaining the pillar shape and assisting with the resistance of load above the production level [46]. Figure 5 shows examples of support construction at the production level (level 1).



Figure 5. Production and extraction drift support constructions [47].

3.2.3. Equipment

Construction equipment used includes Jumbo and Simba drillers, backhoes, excavators, blind hole, raise borer, loaders, scoops (LHD), dumpers, mixer trucks, robotshot, trucks of explosives, and forklifts. In production, LHD of 7 yd^3 that dump in a primary crusher connected to an ore pass are currently being used.

3.3. Copper and Molybdenum Exploitation in the O'Higgins Region

The El Teniente mine is one of the largest underground copper mines in the world with more than 3000 km of drifts. It is in the Andes range (Farellones formation) in the central zone of Chile, about 70 km from Santiago, Chile. El Teniente is a porphyry Cu–Mo deposit, anomalous in size, with over 94 million tonnes (Mt) of contained fine copper, where the principal host rocks are andesites [48]. The primary copper ore is very competent and massive, with almost no open discontinuities [49]. The mine is subdivided into smaller production units that are located around a sterile pipe. The deposit is formed by three main lithologies called El Teniente Mafic Complex (CMET), Felsic Complex (SC), and Brecha Braden Complex (Pipe). Figure 6 shows the sterile pipe (green) around which different exploitation mines are located.

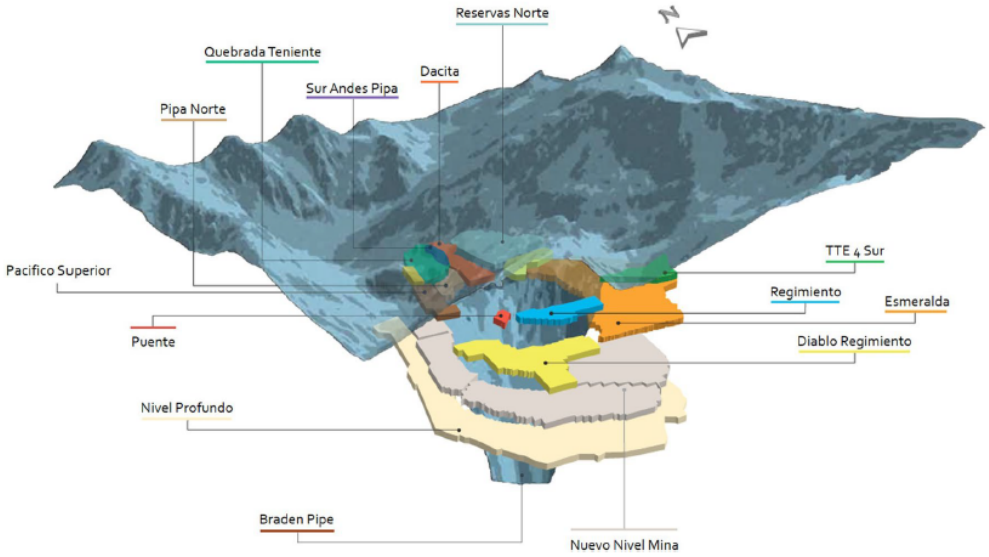


Figure 6. Location of different exploitation levels around Pipa Braden [50].

3.3.1. Underground Mining Method

The underground mines or sectors are exploited using the panel caving method with conventional undercutting and advanced undercutting using crinkle cut, among other techniques, to initiate the caving. The ore is collected at drawpoints by LHDs, which dump the ore in ore passes to an intermediate transport level operated by trucks system. El Teniente also has an open pit operation called Rajo Sur, located between 2730 and 3240 m.a.s.l. Nowadays, the new mine level (Figure 6 bottom) has been constructed and divided into different mines or sectors. In the El Teniente mine, there are different extraction layouts geometries, and dimensions. However, the El Teniente layout type is currently used in all sectors.

Figure 7 shows an example of different levels required to apply a panel caving method in a large deposit. The first level is used to blast the deposit base and start the caving. At the production level, the ore is drawn using LHD that dumps the material to ore passes connected with the haulage level. From the undercutting and production levels, preconditioning in the rock mass is commonly applied. At the haulage level, the ore is commonly transported using underground mining trucks. The ventilation level consists of a layout of galleries connected by chimneys to the other levels.

3.3.2. Mining Support

During the construction of the (originally called) new mine level in El Teniente, several rock bursts occurred due to the competent rock and high-stress field. Thus, a precondition was applied during drifts development using hydraulic fracturing and distress blasting to decrease stress on the working front. In addition, a strong fortification was considered, highlighting a minor use of shotcrete (due to the fall of concrete blocks) and an intensive used of bolts, cable bolts, double galvanized mesh, shotcrete with fiber, and confining

walls in production level pillars. Additionally, the use of mechanized and tele-commanded equipment increased during development. These measures increased the safety during development but also the cycle time.

Figure 8 shows an example of the support used during the new mine level construction. The bolt layout is shown in blue, while the cable bolt layout is shown in pink. Bolts and cable bolt adhered to rock mass with concrete and additives. Two shotcrete layers are applied by dividing by two meshes. The bolts are pressed between both shotcrete layers.

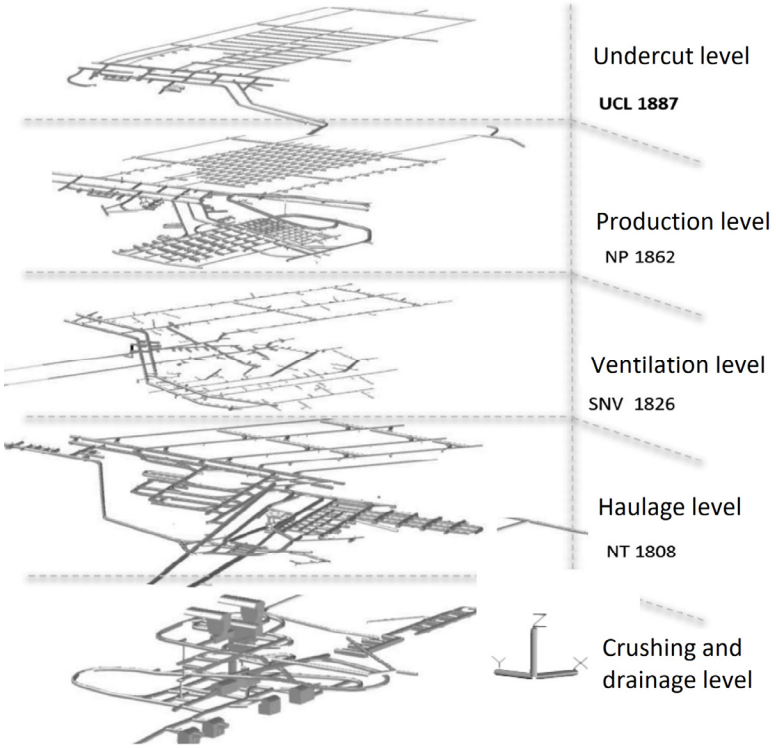


Figure 7. Isometric view of the initial new level mine project, El Teniente [51].

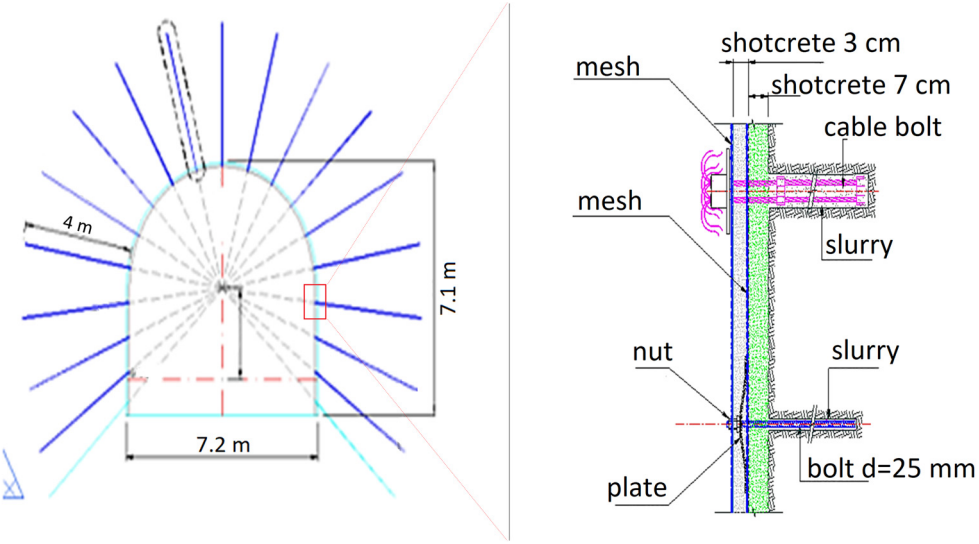


Figure 8. Example of support used in high seismic risk [52].

3.3.3. Equipment

Diverse mining equipment is used in this mine during construction and ore extraction. At the production level, LHD of 7 yd^3 (~9.7 tonnes) CAT R1600H are commonly used, with a productivity of around of 3000 tpd. In addition, other LHD models are used such as the Sandvik lh410 and lh517 models, and the Caterpillar R1600H model. The haulage level (Teniente 7) is an intermediate haulage level where underground trucks are used. These trucks have commonly 40 t or 80 t of capacity (Volvo and Supra 012H model, respectively). Then, the ore is transported by ore pass to the railway located in El Teniente 8 level. In the new levels, conveyor belt will also be used in the new haulage level.

4. Poland Study Cases

Deposits with a thickness of more than 7 m are considered thick, this applies to deposits located in Poland. Currently, the exploitation of thick copper ore deposits is carried out only in the Legnica–Głogów Copper district at Rudna mine owned by KGHM Polska Miedź S.A. A characteristic feature of the deposit is its horizontal deposition with a small angle of inclination, about 8°. Exploitation is carried out at a depth of more than 1200 m in the sediment-hosted stratiform copper deposits, Kupferschiefer-type.

The article also presents the methods of exploitation of thick zinc and lead ore deposits, which were used in the Trzebionka mine (Chrzanów region) until 2009 and in the Olkusz mine (Olkusz region) until 2020. The deposits in the form of nests and lenses were deposited at shallow depths of 80 to 200 m. Both mines are already closed due to resource depletion. Nevertheless, the presented mining methods indicate their evolution, especially in terms of the future exploitation of zinc and lead ore deposits in the Zawiercie region, which were identified by drilling holes from the ground surface and for which it is possible to protect buildings with the use of hydraulic backfilling.

Due to the fact that copper ore deposits occur at a depth of less than 1000 m, they are accessible only by means of vertical shafts. On the other hand, deposits of zinc and lead ores lying at a small depth (about 100 m) are accessible by means of a decline and vertical shafts. Regardless of the access method, one excavation is used to supply fresh air and the other to discharge used air. The output from the exploitation field was transported to the surface by means of a vertical shaft. An exemplary transport route is shown in Figure 9.

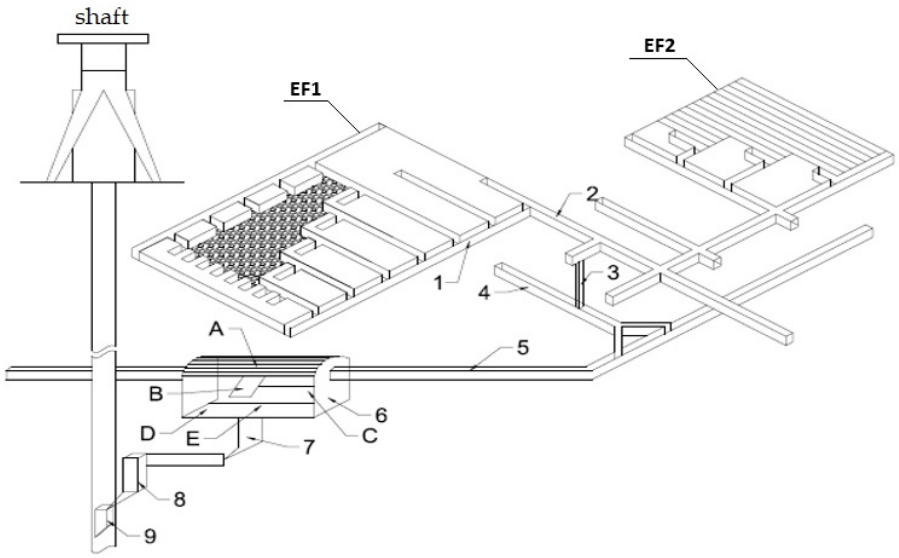


Figure 9. The way of transporting the excavated material from the exploitation field to the surface: EF1—exploitation field no 1; EF2—exploitation field no 2; 1—haulage room; 2—transportation drift; 3—discharge (chute) small shaft; 4—field rock drift; 5—main rock drift; 6—dump's room; 7—storage reservoir; 8—measure reservoir; 9—skip hoist; A—passable dump; B—inclined plane; C—steel membered conveyor; D—jaw crusher; E—belt conveyors.

4.1. Zinc and Lead Exploitation in the Chrzanów Region

In Poland, in the Chrzanów region, until 2009, various variants of room and pillar mining methods were used to select thick deposits, with the mining of the deposit with rooms to the full thickness and the liquidation of the room with the use of hydraulic backfilling. The contoured area with exploration workings was divided into two mining fields (Figure 10).

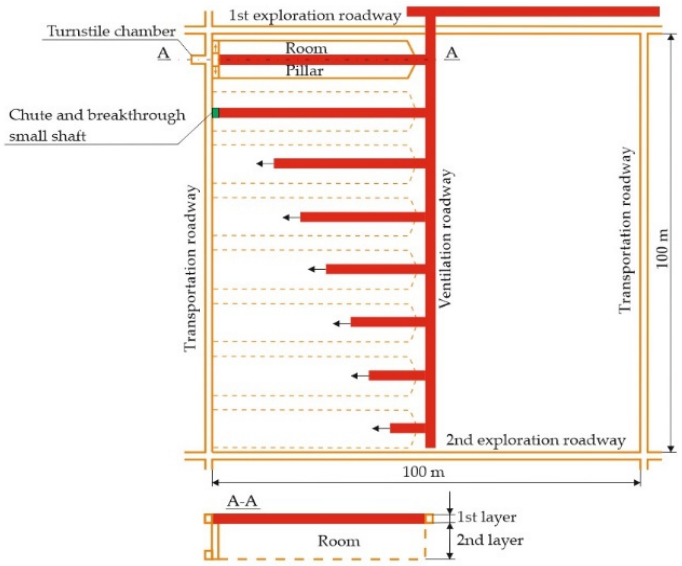


Figure 10. Scheme of preparation of a thick orebody for exploitation.

In the mining field, rooms with a width of 10 m to 16 m and a length of up to 70 m were designed, as well as inter-room pillars with a width of 3 m to 4 m. In the deposit up to 10 m thick, the deposit with the contour of the designed room was selected by dividing it into two layers. First, the 3 m thick roofing layer was exploited, and then the bottom layer was as well. The preparation of the rooms for exploitation consisted of the construction of a transportation roadway on the bottom of the deposit, located under the exploration roadway, over-room roadways located in the axes of the designed rooms under the roof of the deposit, and chute and breakthrough small shafts connecting the transporting roadway with the over-room roadway (Figure 11).

In the first stage of exploitation, the over-room roadway was widened to the width of the designed room. The deposit in the first layer was made with short blast holes. The output was placed in the chute and breakthrough small shaft. The roof of the workings was secured along the entire length with the rock bolt support with expansion heads or with an application of resin cartridges. The length of the bolt support was at least 1.6 m. The distance between the bolts did not exceed 1.2 m. The mining of the second layer began with widening the small shaft chute to the width of the future room. The widening was performed with short blast holes, leaving a 3 m wide retaining barrier pillar from the side of the transporting roadway. Then the bottom layer was selected with the use of long blast holes drilled from the top layer. During the exploitation of the bottom layer, 4 m wide and 4 m deep cut-rooms were made in the sidewall of the room from the side of the orebody. The distance between the cut-rooms was 8 m. The cut-rooms were crossed by the designed inter-room pillar, reducing operational losses. Finally, there were unselected rectangular pillars with dimensions of 4 × 8 m between the rooms.

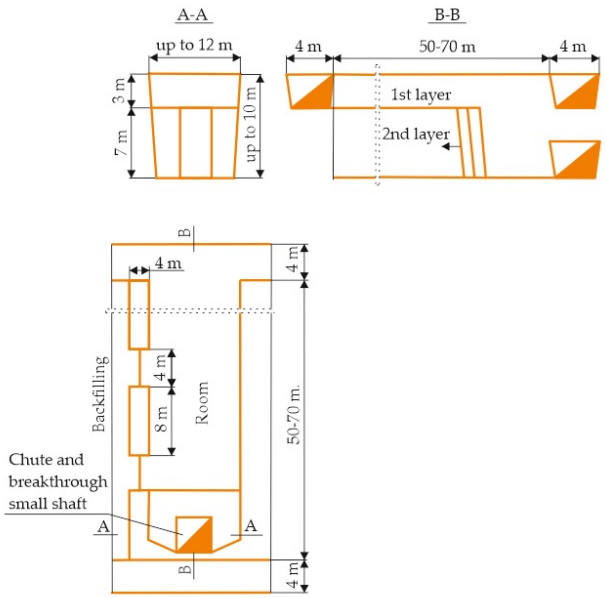


Figure 11. Room and pillar mining method in an orebody with a thickness of up to 10 m.

Exploitation works were carried out simultaneously in several rooms, usually in three rooms: the first room—excavating the roof layer; the second room—selecting the bottom layer and cut-rooms in the pillar; the third room—building backfilling wooden dams in the cut-rooms in the pillar. The rooms were shaped into a trapezoid as a result of the deviation of the sidewalls towards the orebody at an angle of about 5°. The deposit was mined with the use of explosives such as dynamite, ammonite, and saletrol. The haulage of the excavated material was carried out with the use of loading and hauling trucks. Wooden dams and backfilling cloth were used to enclose the room. The room was supplemented in two stages. In the first stage, the room was extended to half its height. Complementing the filling to the roof took place while filling the next room. In the ore deposits up to 15 m thick, layers up to 6 m thick were separated in the room, which were mined from the room to the bottom of the room (in the case of stable rooms) or in the direction from the floor to the roof (in the case of less stable roofs). In the case of mining layers in the top-down direction, the filling of the rooms took place after selecting the orebody in the room [53]. In the case of the reverse order of selecting the layers, after mining the layer, backfilling was performed, and the next higher layer was selected along the backfilling. The scheme of mining a deposit in a 15 m high room using three layers in the top-down direction is shown in Figure 12.

4.2. Zinc and Lead Exploitation in the Olkusz Region

Room and pillar mining methods with the bottom discharge of the output was used in the Olkusz region in the 1970s to extract a thick deposit of zinc and lead ores with a thickness of 9 m to 15 m [54]. The exploitation was conducted in two stages. In the first stage, rooms with a width of 10 m to 12 m were selected in the mining field and continuous pillars 7 m wide were left between the rooms (Figure 13). In the second stage, the inter-room pillars in the mining field were mined. It was assumed that after the simultaneous mining of the pillars in the selection field, the excavated material would be ejected to the rooms and the roof rocks would fall. The material thrown into the rooms was to be recovered through discharge openings made under the rooms. The elementary field was contoured with exploration roadways in a 100 × 100 m grid, which was divided into two fields, and the rooms were perpendicular to the longer edge of the field. Preparatory works in the field were performed on two or three levels: under the orebody, under the roof of the orebody and above the orebody. Above the roof of the orebody (about 5 m), over-pillar roadways were made along the axis of the designed pillars. They were intended for drilling blast

holes in the pillars and mining the pillars between the rooms. This concept was quickly abandoned and the pillar roadways were replaced with pillar roadways made at the same level as the haulage roadways. Under the roof of the orebody, over-room roadways were driven, located in the axis of the designed rooms or the corner of the room. These roadways were connected by a system of preparatory workings with a ventilation level.

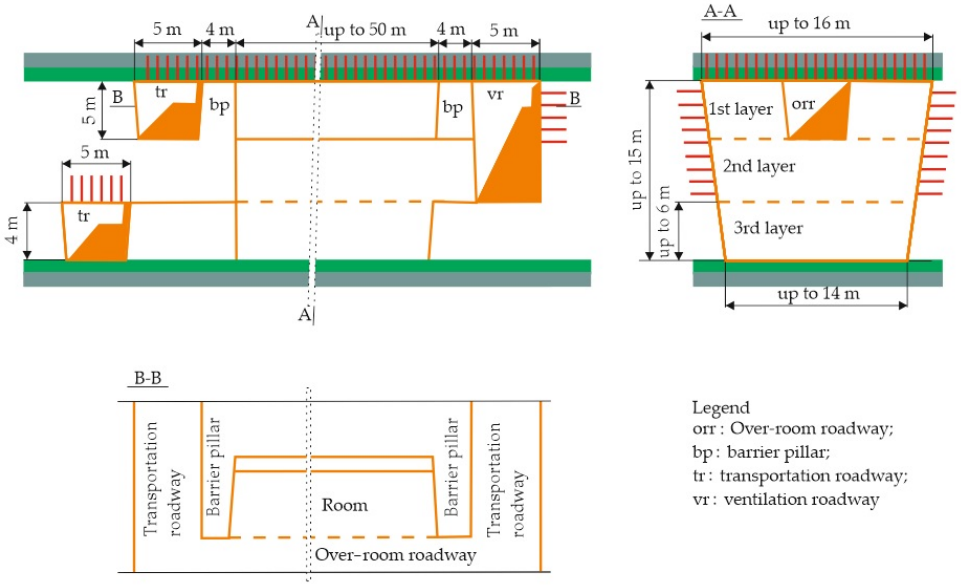


Figure 12. Room and pillar mining method in an orebody with a thickness of more than 10 m.

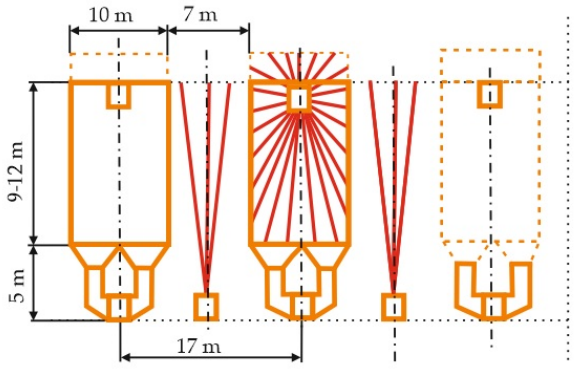


Figure 13. Room and pillar mining method with bottom discharge. Blasting holes in red.

They were used to drill long cutting holes and discharge the used air. About 5.5 m below the bottom of the orebody, there was a transporting level. At this level, in the axes of the future rooms, haulage roadways were driven, and in the axes of the pillars, roadways for excavating the pillars were also driven. The haulage roadways were driven at intervals of about 17 m. From the haulage roadways, the excavations were driven and used to discharge the output. These were funnels made at intervals of 5.5 m, arranged in a checkerboard pattern. The diameter of the funnels in the upper part was 5 m and they undercut the bottom of the room on about 70% of the surface. A breakthrough small shaft was constructed at the beginning of each room. It was an extension of the first funnel and was connected with the over-room roadway using a short cut-off roadway, made perpendicularly along the axis of the room.

In the haulage roadways, scraper loaders were installed in order to place the output pouring out of the openings on the transporting roadway. The mining in the room was started until the breakthrough was completed, i.e., the small shaft was extended to the full width of the room. The breakthrough was made with long blast holes and it was a

compensation space in the case of the material blasted from the first advance (the distance between the fans of the holes, was 2.5 m). The diameter of the blast hole in the fan was 70 mm, and the length of the holes was from 6 m to 15 m. The total length of the holes in the fan was from 96 m to 115 m.

For the blasting, saletrol was used and injected into the holes using pneumatic injectors. The maximum mass of the explosive in the fan was 350 kg. One or two fans were blasted simultaneously. The output rolled down to the discharged chutes, which undercut the bottom of the room. The first stage was related to the mining of the orebody in the rooms. After mining the orebody in the selection field, the mining of the inter-room pillars was started, which was the second stage. For this purpose, one row of long blast holes was drilled from the over-pillar roadways and all the pillars were mined at the same time. After blasting, only part of the excavated material was thrown toward the rooms. Most of the output remained between the rooms and it could not be recovered through the discharge openings made for the rooms.

In addition, the planned caving of the roof rocks did not take place. Maintaining such a large underground void was dangerous. The aforementioned technological defects, compared to the planned design assumptions, showed that the operating losses in this method were very high—up to 40% and due to the second stage of operation, the system was uneconomical. In the following fields, various attempts were made to improve the technology in the second stage of operation. In some fields, all the pillars were abandoned and only every second pillar in the field was mined. The attempts at improvement did not bring the expected economic effects, which prompted the discontinuation of this exploitation mining method.

Later, successful attempts were made with the help of a sublevel caving method. This method was used in the Olkusz region in Poland to extract zinc and lead ore deposits, the roof of which was disturbed by the earlier exploitation of the higher layers, and to extract the remains of the ore [55]. The sublevel caving method could also be used in a deposit that had not been affected by previous mining operations when the roof rocks showed a high tendency to collapse. The method could not be used in the case of watered layers in the overburden or the upper layers. In the case of a thick orebody, 7 m thick layers were separated in horizontal planes, and 8 m wide blocks in vertical planes (Figure 14).

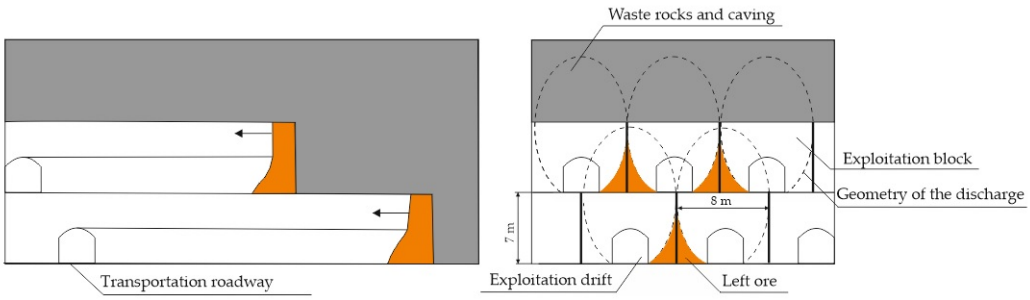


Figure 14. Sublevel caving used in the Olkusz–Pomorzany mine in Poland.

The layers were mined with blocks/stopes exploited from top to bottom. In the axis of each block, mining drifts with dimensions of 3 × 3 m were driven on the floor of the layer, and they were protected using rock bolt support. In the lower layers, the blocks were moved by half their length between the axes of the drifts, which ensured the recovery of the ore left in the leftovers between the mining drifts in the upper layer. The length of the exploitation blocks did not exceed 100 m.

The blast holes were drilled from the exploitation drift in a fan arrangement. The diameter of the holes ranged from 50 mm to 70 mm, and the distance between the fans ranged from 1 m to 1.8 m. About 12 blast holes were drilled in one fan. The ore was blasted with saletrol, which was loaded into the holes using a pneumatic injector. The blasted material rolled down to the roadways and was loaded and set aside with self-propelled machines—loaders. The main advantages of the method were the ability to safely select the

deposit under the roof of rocks damaged by previous mining; the possibility of recovering the ore left in the collapsed space of the upper layers; the ability to safely extract the ore left in the inter-room pillars. The disadvantages of the method were the dilution of the ore with rocks permeating from the caving space; enormous devastation of the surface, as the exploitation was carried out at shallow depths, about 60 m below the ground surface. The above-mentioned disadvantages resulted in abandoning the exploitation of zinc and lead ore deposits using the sublevel caving method. From 1990 to 2020, thick deposits were mined from the bottom up, with horizontal layers 5–6 m high. In the first layer, a room and pillar method was always used, while the subsequent layers were exploited with a strip method with the liquidation of post-mining space with the use of hydraulic backfilling [56,57].

4.3. Copper Exploitation in the Legnica–Głogów Copper District

Diversified strength parameters of rocks in the mining gate led to the development of a two-layer method of extracting thick copper ore deposits in the Legnica–Głogów Copper district in Poland, with the use of hydraulic backfilling. In the thick orebody, two layers are separated—the roofing layer with a thickness of 4 m to 5 m and the bottom layer with the remaining thickness of the deposit. The preparatory workings for both layers are made in the roofing layer, in which the roof of the workings is made of carbonate layers with high strength parameters. This enables the stability of these excavations to be maintained despite the large depth of exploitation and the occurrence of induced seismicity. Attempts to make and maintain the preparatory workings in the bottom layer, which are usually sandstones with low strength parameters, failed. Both layers are mined at a certain time in advance of each other. One preparatory excavation made in the bottom layer is the inclined drift (ramp). This inclined drift connects the preparatory workings in the roofing layer with the bottom of the rooms in the lower layer. It is used to haul the output from the rooms of the lower layer [58].

The room and pillar method with a two-layer deposit extraction (proper name Rudna 5) was used in an orebody with a thickness of up to 15 m and an inclination angle of up to 8°. The preparatory works consisted of the construction of double roadways and inclined drifts in the roof layer separating the exploitation fields with the following dimensions: field width from 150 m to 200 m; the length of the field was about 800 m, depending on the geological and mining conditions. At the same time, mining was carried out in at least two fields of exploitation (Figure 15).

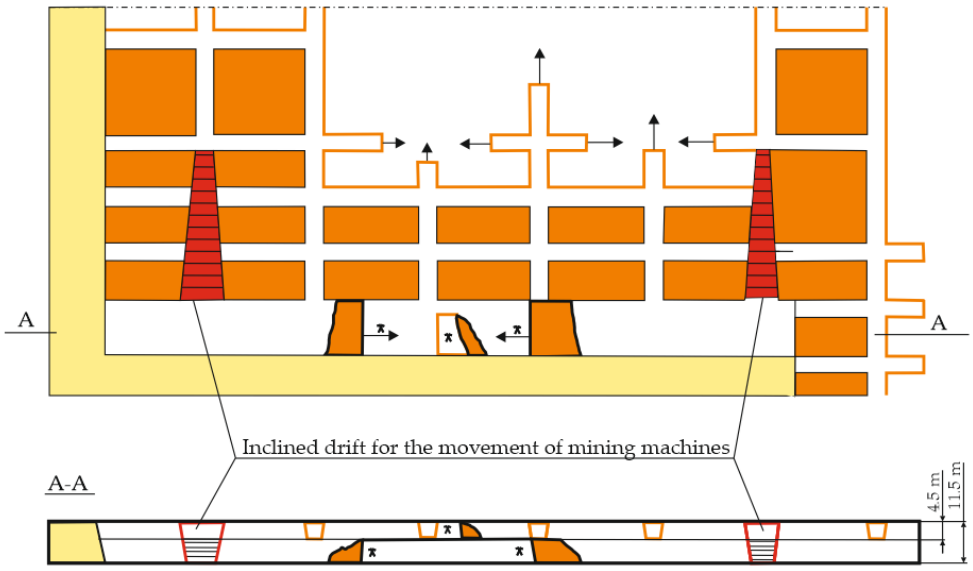


Figure 15. Room and pillar mining method using two-layer orebody exploitation.

First, the roof layer with a thickness of about 4.5 m was selected [59]. It included dolomites, shales, and sandstones. In this layer, 28 × 10.5 m pillars were separated by room workings. The longer side of the pillar was parallel to the goal line. The rooms were about 7 m wide at the roof, and their sidewalls were inclined towards the whole body at an angle of about 10°. There were three rows of pillars ahead of time. Preparation for the exploitation of the bottom layer consisted of making inclined drifts from the top layer, giving them a slope that enabled the movement of mobile machines (from 5° to 8°). Flooring down the inclined drifts was performed continuously, as the work front progresses. The selection of the orebody in both layers took place simultaneously with the maintenance of appropriate advance. In the top layer, the last row of pillars was selected and the maximum width of the exposed roof was 17.5 m.

The selection of pillars proceeded on both sides from the inclined drifts toward the center of the field. The mining of the bottom layer followed the selected pillars. A distance of 12 m to 36 m was maintained between the face in both layers due to the blasting works and the movement of machines. The lower layer was cut with long blast holes drilled from the upper layer. After mining both layers, a room about 15 m high, 17.5 m wide at the roof, and 150 m to 200 m long was created. The drilling of blast holes and loading and hauling of the excavated material were performed with typical self-propelled machines. Full column resin bolt support (2.6 m) was used, installed in a 1 × 1 m net bolting. In the upper layer, the workings had a trapezoidal shape, while the inclined drifts and room workings oval shape. The rooms in the upper layer were ventilated until they were connected using duct ventilation. In the perpendicular rooms of the upper layer, backfilling wooden dams were built at the edge of the room. In the inclined drifts for the movement of the mining machines, wooden dams were built on the sand, after the room was backfilled up to the floor of the upper layer (Figure 16). When designing the excavation at a distance of 15 m from the stream of air flow generated by the main fan, they are ventilated by diffusion. Excavations above 15 m are ventilated with separate fans. The amount of fresh air introduced must ensure the appropriate composition, such as O_2—min. 19%; CO_2—max. 1.0% (10,000 ppm); CO—max. 0.0026% (26 ppm); H_2S—max. 0.0007% (7 ppm); SO_2—max. 0.000075% (0.75 ppm); NO_2—max. 0.00026% (2.6 ppm).

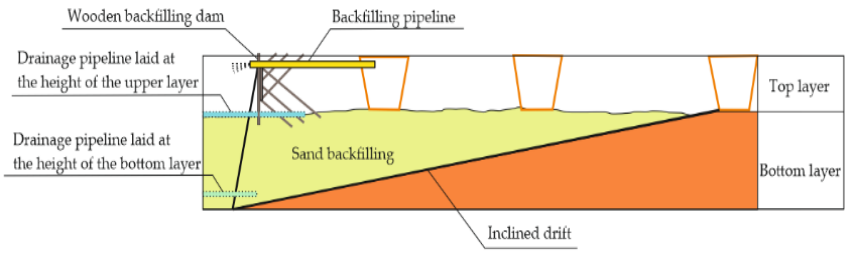


Figure 16. Backfilling in the inclined drift.

Mining of copper ore deposits in the area of LGOM is carried out in conditions of high seismic activity of the rock mass. Therefore, in the blasting metric, apart from breakage, mining, and contouring holes, destress relief holes are often made, the length of which are usually twice as long as the mining holes (Figure 17). Moreover, the diameter of the destress blasting holes is almost twice as large. For example, for 30 holes, the explosive load is 84 kg, and with destress holes with a diameter of 0.089 m, the maximum explosive load is 116 kg.

The backfilling water was discharged to the field sedimentation reservoirs through pipe channels (worn pipes with a diameter of 500 mm) laid in the backfilling at the level of the bottom of the upper layer and through perforated tubing and wrapped with a backfilling cloth laid on the bottom of the lower layer. Pipe channels were arranged in each upper layer room and the end layer rooms of the bottom layer. The backfilling pipeline was installed in the rooms of the upper layer. After backfilling the post-exploitation area, the sand was removed from the inclined drifts (picked up with loading trucks) and the cycle

was repeated in the field. Exploitation was performed simultaneously in at least two fields of exploitation. The times of selection, liquidation, and preparatory works in both fields were selected in such a way as to ensure the continuity of the mining process.

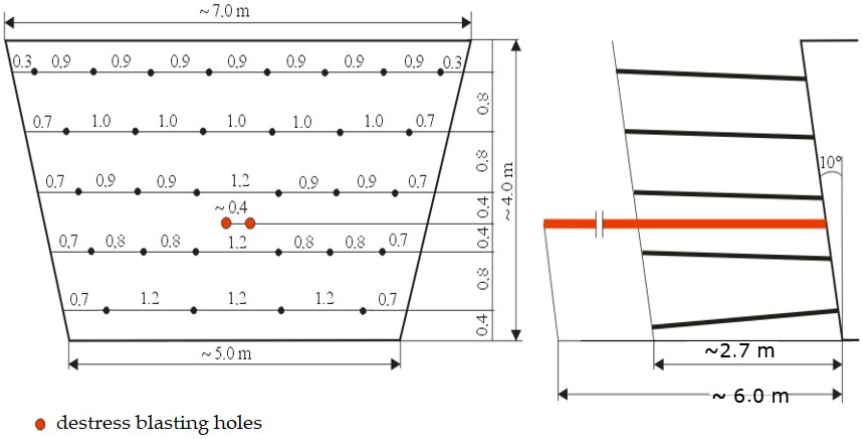


Figure 17. Arrangement of blasting holes in the ore forehead.

The backfilling that is used in mining excavations is either dry or hydraulic. In the case of dry backfilling, the rock material comes from preparatory and mining excavations and is delivered to the space selected by mine loaders. The hydraulic backfilling, on the other hand, is transported using pipelines with a diameter of 0.150 m and 0.185 m. Backfilling material may be non-combustible and non-toxic solids. Filling material can also be used: sand; gravel; waste: slags; gang; industrial waste. It is worth noting that the maximum dimension of grains should not be greater than 0.06 m; the content of particles of size less than 0.1 mm are at most 20%; and the compressibility at pressure 15 MPa is at most 15%.

Water flowing into mine workings is discharged to the surface through drainage devices and special workings included in the mine drainage. In the vast majority of cases, water runoff occurs automatically, by gravity, through transport headings in the direction from the mining fields to the main mine drainage system. Due to the drainage of water, it is advisable to drive a drift and roadways with an inclination of up to 3‰ towards the shafts, and in the case of more polluted water up to 5‰. The whole of the main drainage workings (Figure 18) consists of water reservoirs at the pump chamber; drifts and inclined drifts supplying water to reservoirs; pump chambers; suction wells and water channels—connecting water reservoirs with water wells; an access roadway connecting the pump chamber with the bottom shaft; a pipe channel connecting the pump chamber with the shaft and used to accommodate water pipelines and to ventilate the chamber.

Water reservoirs are mining excavations in the vicinity of the pump chamber of the main drainage, designed to collect underground water from the natural inflow and from the hydraulic backfilling. In Poland, the capacity of active water reservoirs should be sufficient for at least a 12-h supply of water from natural inflow and backfilling to the excavations, where main drainage systems are capable of discharging the highest daily water inflow in less than 20 h. The pump chamber is a mining excavation with built-in pump units for pumping water directly to the surface or to a higher level of the mine. The chamber is located parallel to water reservoirs and most often between water reservoirs. The size of the pump chamber depends primarily on the number of installed pump units. In Poland, pump chambers of the main drainage at levels with water inflow over 1 m^3/min are equipped with at least three pumps. If the pump chamber is equipped with pump units, the number of pumps in these units shall be at least: five pumps with two pumps working in a group; seven pumps with three pumps working in a group. The following numbers of pumps should be in full operational readiness at all times in the pump chamber of the main drainage: two pumps, in chambers equipped with three pumps; two sets of pumps, in chambers equipped with sets of pumps. The devices, together with the main drainage systems, should enable the discharge of the highest daily water supply in no more than 20 h.

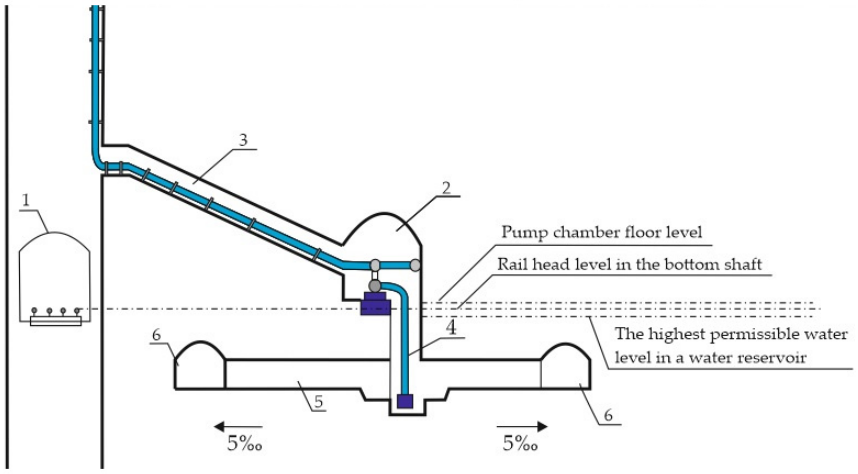


Figure 18. Main drainage workings: 1—bottom shaft; 2—pump chamber; 3—pipe channel; 4—pump suction well; 5—water channels; 6—water reservoirs.

During the 1990s, due to the increasing seismic risk and rock bursts, attempts were made to select a thick deposit with the use of pillars operating in the post-destruction phase, located perpendicular to the operational front line. The positive results of industrial research confirmed the legitimacy of the introduced geometric changes and made it possible to develop a one-stage method with the liquidation of the space selected with a hydraulic filling (the proper name of the mining system "Rudna 7") [60]. With this method, ore deposits with a thickness of up to 18 m and an inclination angle of up to 8° can be mined. The roof conditions should allow the use of the rock bolt support. It is recommended that this method be used when rocks with increased strength parameters and high rock burst tendency are present in the exploitation gate. In the first phase of excavation works in the mining field, a 6 m to 7 m wide room is cut into the upper layer, up to 5 m high, separating technological pillars with dimensions of 7–9 × 16–38 m (Figure 19).

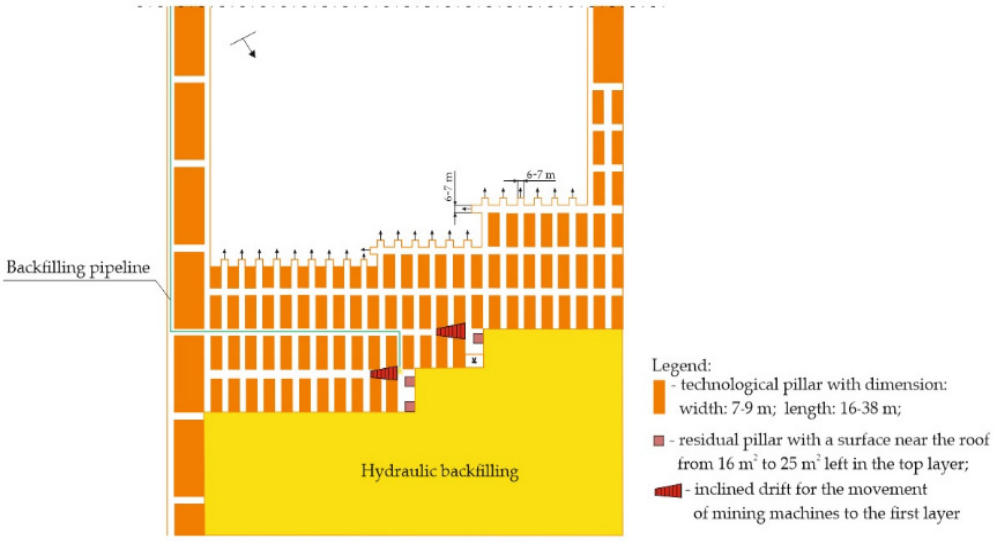


Figure 19. A two-layer room and pillar mining method with a hydraulic backfill in the case of the exploitation gate conditions made of rocks strongly prone to rock bursts.

There are no less than two rows of technological pillars in the front. The width of the rooms in the upper layer may be increased to 10 m as a result of induced seismicity and falling rocks from the sidewalls. Before mining the bottom layer, a pillar adjacent to the backfill is partially mined, leaving residual pillars with a roof area ranging from 16 m^2 to 25 m^2. Under favorable roof conditions, it is permissible to select the pillar completely,

without leaving any residual pillars. To the bottom of the layer in an elementary field with one room and one technological pillar, there are inclined drifts with an inclination from 5° to 8° that allow the movement of mobile machines. The inclined drifts are used in successive elementary fields, but this requires constant flooring as the front of works progresses. Then the bottom layer in the field is mined. The orebody is processed with explosives. Blasting holes in the lower layer are drilled from the floor of the lower layer (horizontal holes) or the floor of the upper layer (vertical holes).

The roofs of the excavations and the sidewalls are secured using rock bolt support. After the bottom layer is mined in the elementary field, the filling dams are built in the room leading to the field and the inclined drift (after partial backfilling), and the post-exploitation space is filled with hydraulic backfill. The backfill water is discharged through pipe collectors or workings fenced off in the filling to field settlers. The mining of the deposit in the next field may take place after the entire scope of liquidation works in the previous field has been completed. All these works are performed with the use of self-propelled machines: blast holes are drilled with self-propelled drill trucks; loading explosives such as saletrol or emulsion is carried out with blasting vehicles, while the loading of the cartridge materials is carried out manually; the output is loaded and set aside from the fronts with the use of loading and hauling trucks or cooperating with box haulage vehicles; the filling material brought in during the selection works is loaded and put away into the post-exploitation voids using loading and hauling trucks; mechanical and resin rock bolts with a length of 1.6 m to 2.6 m are made with the use of self-propelled machines; ripping off the roof and sidewalls is performed mechanically with the use of trucks equipped with cutters.

5. Discussion

Mining in massive deposits has various challenges, mainly related to great depth, hard rock, in-rush related to water, and mine stability. Additionally, the mine development in these environments requires the planning and optimization of several activities to improve the construction times, mining times, and support requirements. This work has summarized different underground mining methods applied to massive deposits located in Chile and Poland.

Underground mining of ore deposits in Chile and Poland is diversified due to the geological conditions and the exploitation methods used. For the ore deposits located in the Legnica–Głogów Copper district in Poland, copper and silver are mined together, as well as zinc and lead (Chrzanow and Olkuski regions). A comparison of the basic geological and mining parameters is presented in Table 1. Currently, the only exploitation of ore deposits in Poland is carried out in the Legnica–Głogów Copper district. Mining of Cu–Ag ores in this region in 2021 amounted to 30 million tonnes of ore with a content of 1.48% Cu and 50.73 g/t Ag, containing 0.443 million tonnes of metallic copper and 1.522 million tonnes of silver. Exploitation of zinc and lead ore deposits in the Chrzanów region (Trzebionka mine) and in the Olkusz region (Olkusz–Pomorzany mine) was completed in 2009 and 2020, respectively. The annual output of zinc and lead ores in 2009 amounted to 2.349 million tonnes, including 0.089 million tonnes of zinc and 0.034 million tonnes of lead. In 2020, it was 1.435 million tonnes, including 0.043 million tonnes of zinc and 0.018 million tonnes of lead. The content of zinc and lead in the ore was 2.6% and 1.3%, respectively.

On the other hand, the deposit in Chine are in different stages. The deposit located in the Coquimbo region, is under development, and is being designed to mine 2000 tpd of copper ore with a content of 1.22% Cu containing 2680 million tonnes of cooper. The deposit located in the Antofagasta region is an open pit to underground transition project (Table 1). Mining of Cu–Mo ores in this region has 1700 million tonnes of ore with a content of 0.7% Cu and 450 ppm Mo. The deposit located in the O'Higgins region is a large block caving project (Table 1) with more than 4500 km of drifts mined from 1905 to date. Mining of Cu–Mo ores in this region has 2020 million tonnes of ore with a content of 0.86% Cu, containing several underground exploitation areas including an open pit.

According to reference [2], mass mining exploitation methods must consider the mechanical behavior of the rocks and the depth of the deposits. Caving is more difficult to initiate in competent rock masses and cave progression is typically slower. Furthermore, the rock mechanics aspects in deep cave mines are more relevant due to the increasing primary stress magnitudes. There is no common characteristic regarding the quality of the rock in world-class deposits since their typology and genesis are varied and, therefore, the host rocks and their characteristics are also varied. Particularly, the porphyry deposits present their ore generally associated with granodiorites and andesites, on which different types of alteration due to hydrothermal fluids are superimposed. This alteration implies the presence of abundant clays, which decrease the resistance of the rock. The depth at which the ore appears depends largely on the degree of erosion suffered by the rock mass, although the location of the magmas that give rise to the hydrothermal fluids is around 2–3 km and even more superficial. In any case, given the large dimensions of the mineralized bodies, it is common for at least part of them to be found at a considerable depth for their exploitation.

The mine development in these environments required the planning and optimization of several activities to improve the construction times and support requirements. This work summarized some of the activities for different mine deposits and underground mine methods and designs. In particular, the caving method required minor development compared with the deposit volume to be mined. However, the caving process implies high induced stress that must be included in stability. Additionally, the crater of subsidence generated in the surface due to caving involves an extensive zone. In Chile, a few studies have evaluated the impact of refilling this zone [61], but to date they have not been applied. Some of the current tendencies in the Block Caving method are focused on decreasing the development required and proposed improvement in mine design [62–64]. For example, the last caving projects used wide space extraction level layouts as in the Chuquicamata underground mine [44] and the Diamante sector of the El Teniente mine [65]. Additionally, in Cadia East in Australia, the undercut level has been omitted, so the caving was started directly from the extraction level [62]. On the other hand, the Polish mines described in massive deposits here use more selective underground methods. In these methods, the development and mining activities are commonly carried out at the same time (as well as caving propagation and undercutting). The mine planning then involved several challenges. Additionally, a back-filling process must be used to control the mine stability in thick deposits with these selective methods due to the large open areas.

For the underground copper ore mines in Poland in the area of LGOM in the coming years, three fundamental issues will arise with which the mining staff will have to look for and improve solutions. The first refers to the climatic hazard associated with the primary temperature of the rock mass. The primary temperature of the rock mass at the level of the mining excavations (Zechstein bottom level) ranges from 34 °C to 47 °C, with an average of 39 °C. On the other hand, the geothermal gradient ranges from 2.5 °C to 3 °C for every 100 m. Considering the greater depth of exploitation, the primary temperature of the rocks will increase, which will make it necessary to ensure the appropriate temperature for the mining crew by cooling the air temperature. The second issue is closely related to the decreasing thickness of the selected deposit. For thin deposits below 2 m, mines currently use split mining methods, trying to minimize dilution and operational losses. Further reduction of the exploitation gate will be related to the introduction of low-profile machines while maintaining the current level of extraction. The third issue relates to the risk of rock bursts. The occurring tremors have a value of 10^9 J. Therefore, mines are constantly improving the methods of rock bolt support, capable of absorbing as much dynamic load energy as possible. In addition, in order to minimize the risk of the crew staying in hazardous zones, methods for the remote control of mining operations are being developed, including blasting and bolt hole drilling, as well as loading and haulage of ore output. Further research works will be closely related to innovative mining technologies for the increasing level of exploitation.

Table 1. Comparison of mining and geological parameters for underground mining of selected ore deposits (copper and silver, zinc and lead) in Chile and Poland.

Parameter	Unit	Chile		Poland	
		Copper + Molybdenum (Chuquicamata)	**Copper + Molybdenum (El Teniente)**	**Copper + Silver**	**Zinc + Lead**
Depth of exploitation	(m)	100–650 (current level 1)	~1150	800–1300	80–120
Annual production	(mln tonnes)	50.4 (in regimen)	48.6	30	1.435
Metal content	(%); (g/Mg for silver)	0.77 (copper); 450 (ppm Mo)	0.86 (copper)	1.28–2.3 (copper); 54.5–62.5 (silver)	2.6 (zinc); 1.3 (lead)
Deposit thickness	(m)	250	~1200 (porphyry diameter in surface)	Thin up to 2 m; medium form 2 to 7 m; Thick more than 7 m	up to 30 m
The form of the deposit		Massive	Massive with mineralized veins.	The deposit belongs to the type of stratoid deposits (pseudo-seam)	Lenses and nests
Deposit inclination	(°)	-	-	about 8	about 5
Access		Tunnels	Tunnels	Vertical shafts	Decline and vertical shafts
The shape of preparatory and mining excavations		Horseshoe	Horseshoe	Trapezoidal	Rectangular with a flat or oval roof
Natural hazards		Rock burst; collapses; spalling	Rck burst; collapses; mudrushes	Rock burst; water; rock and gas outburst; roof falls; climatic (primary temperature of rock mass)	Water
Mining methods		Block caving (Macroblocks)	Block/panel caving	Room and Pillar	Room and Pillar
Liquidation of post-mining space		Subsidence	Subsidence	Bending of roor layers with further caving; hydraulic and dry backfilling	Hydraulic backfilling
Type of explosives		ANFO and Emulsion	ANFO and emulsion	Cartridge (manually loaded); emulsion materials (mechanically loaded)	
Support type		Shotcrete, cement grout rock bolt support, mesh	Shotcrete, cement grout rock bolt support (helical and split set), mesh	Mechanical and resin (full column) rock bolt support.	Resin (full column) rock bolt support.
Additional support		Shotcrete with syntetic fiber, concrete walls, cable bolts, steel frames	Shotcrete over mesh, shotcrete with fibet, concrete walls, cable bolts, steel frames	Cable up to 8 m; wooden cribs; arch yieldig support; steel and wooden props; geosynthetic and steel grids	Wooden cribs; arch yieldig support; wooden props; steel grids

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