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TECHNOLOGY DEVELOPMENT OF KRYVYI RIH IRON ORE OPEN PITS WITHIN ZONES STOPING OF UNDERGROUND MINE WORKINGS

На современном этапе отработка залежей железных руд в Кривбассе ведется в условиях как совместных открытых и подземных горных работ, так и открытой разработки месторождений в зонах, подработанных подземными выработками. Приведенные технология и новые способы и устройства погашения воронок обрушения в пределах рабочей зоны карьеров путем засыпки их рядовыми скальными породами вскрыши непосредственно с поверхности, что позволяет существенно повысить безопасность горных работ в карьере и получить значительную экономию за счет предупреждения нарушения окружающей среды внешними отвалами.

На сучасному етапі відробка покладів залізних руд у Кривбасі ведеться в умовах як сумісних відкритих і підземних гірничих робіт, так і відкритої розробки родовищ у зонах, що підроблені підземними виробками. Наведені технологія й нові способи та пристрої погашення вирв обрушення у межах робочої зони кар'єрів шляхом засипки їх рядовими скельними породами розкриву безпосередньо з поверхні, що дозволяє суттєво підвищити безпеку гірничих робіт у кар'єрі й отримати значну економію за рахунок попередження порушення довкілля зовнішніми відвалами.

At present iron ore in Kryvbas is mined by both a combination of open pit and underground methods and by the open pit method in zones underworked by underground mining. The suggested technology and new methods of backfilling pit craters within the pit working area with rock overburden directly from the surface that enables substantial increase of mine safety and economy due to preventing environmental damage caused by surface dumps.

Introduction. At present within the city of Kryvyi Rih and its suburbs lean iron ore is being mined and concentrated by five powerful ore mining and concentrating combines with the designed annual capacity of 140 mln t. The volume of overburden makes nearly 112 mln m³, the tails volume is 31.4 mln m³. As areas for surface dumps and tailing ponds are limited, use of the mined area of underground mines solves the problem of stockpiling mining wastes. The volume of underground mines is than 1 billion m³ [1].

Problem statement. Open Pit №1 of PJSC CGOK carries out mining in the slide zone within the former Frunze, Komintern and K. Liebnecht underground mines' area. As the pit goes deeper (over 350 m) and wider on the surface, the greater part of its eastern pit edge is entering areas of possible cratering. Under such condi-

tions mining should be carried out with observance of safety rules to prevent sudden caving and bench displacements at the pit. As surface dumps are located far from it, efficiency increase and substantial decrease of crude ore mining costs can be achieved due to stockpiling the greater part of overburden rocks in slide and caving zones.

Problem solving. To prevent accidents drilling is performed to detect underground voids and locate devices for signaling of massif deformations at the beginning of rock slides in the open pit. Detected voids and workings are backfilled with waste rock. From the -110 m horizon mining is carried out with 12 m high benches. Outbreak of secondary pit craters on the surface of the eastern pit edge may cause sudden local deformation of one or several benches. Therefore, such underground voids should be backfilled with overburden rocks to provide stability and stabilize economy of stripping.

Fig.1 shows the technological scheme of forming a non-working pit edge exhausting underground workings, where the cross section marked: non-working pit edge (1), contours of unfinished underground workings (2), roofs of uncaved rocks outside of open pit (3), crater stoping after underground mine of iron ore (4), the first from the surface bench of soft rock (5), contours of protective rock banks from the side of the pit crater 4 and the idle edge 1 (6, 7 respectively), safety areas of the banks from the side of unfinished underground working 2 (8) the idle pit edge 1 (9), the transport line on the first from the surface bench 5 (10), the transport line on the opposite side of the pit crater outbreak 4 (11), auxiliary protective rock banks on the opposite side of the idle pit edge 1 and aside the road 11 (12, 13) and direction of truck movement when filling pit craters 4 (14) are given in plan and section [2].

First, within the idle pit edge 1 from the side of underground mining there are determined contours of unfinished underground workings 2 and the height of roofs of uncaved surface over them on the rocks in the vertical section 3 are determined as well as the height of remaining free space on the rocks h_b (m) over the lower plane of the crater stoping 4 and the soft rock thickness h_m (m). According to the Design Standard, on the upper area of the first from the surface bench along soft rocks 5 there are determined contours of the protective banks 6 from the side of pit crater 4 and the idle pit edge 7 (Fig. 1)

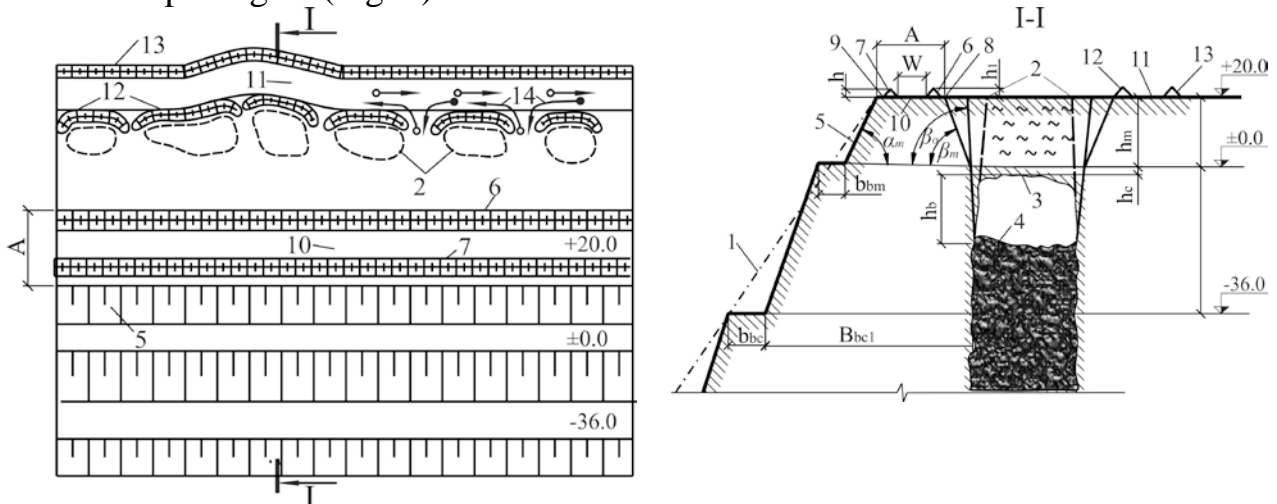


Fig. 1. The method of forming the non-working pit edge

At that, protective banks from the side of the crater stoping 4 are built with safety areas at the distance 8 from the contour of the unfinished underground working 2, and from the side of the idle pit edge – at the distance 9. The width of the transport line 10 is determined considering dimensions mining transport and equipment to be used for repairing protective banks 6 and 7 and for filling the underground area 4. The day surface above the underground areas 4 are backfilled, if necessary, with overburden rock delivered by the transport line 10 or it is placed on the opposite side of the outbreak the crater stoping on surface 11, and the overburden rock is delivered by transport means of the operating open pit. When setting the benches of the non-working edge into the final state they are sloped by common methods until stable values of the slope angle of about 40° and 80° for soft and hard rock respectively. Backfilling of underground areas is carried out continuously with the constant monitoring of the process along the height.

When forming the non-working edge 1 to increase the angle α of the stable slope it is possible to double soft rock benches along height and triple hard rock benches providing their stability by relevant sloping and presplitting methods. This results in pit working zone increase and additional iron ore volumes.

The suggested technology of forming the idle pit edge enables significant material and money savings. So, when operating Pit №3 of PJSC “ArcelorMittal Kryvyi Rih”, the foot-bench technology with locating the non-working eastern edge on the border with the zone of underground mining operations in axes XII and XIVb and keeping only three transport areas at horizons ± 0 m, -120 and -180 m enables temporary conservation of 15 mln m³ of oxidized hornstone till 2030.

The minimal width of the safety area $A = 163$ m on the surface between the edge of the idle bench 5 and the remaining underground area 2 can be determined as follows:

$$A \geq 2(h_1 + h_2) \operatorname{ctg} \alpha_m + W + (h_b + h_c + h_m) \operatorname{ctg} \beta_o + h_m \operatorname{ctg} \beta_m + C_1 + C_2, \quad (1)$$

where h_1 and h_2 – height of the protective rock banks from the side of the pit mined area and the underground area respectively, m; α_m – the rock slope angle in the protective banks, degrees; W – the mining transport line width, m; h_c , h_b – height of the remaining underground uncaved rock and the mined area respectively, m; h_m – thickness of upper bench soft rocks, m; β_o , β_m – the slope angles of caved hard and soft rocks, degrees; C_1 , C_2 – safe distance on the surface between the protective banks from the side of the pit and the underground area respectively, m.

The slope angle of the idle wall $\alpha_n = 50^\circ$ is determined as follows:

$$\alpha_n = \operatorname{arctg} H_{k.o.} / [H_m \operatorname{ctg} \alpha_m + n_m \cdot B_{b,m} + N_c h_c \operatorname{ctg} \alpha_c + n_c B_{bc} + (N - n_c) B_b], \quad (2)$$

where $H_{k.o.}$, H_m – oxidized hornstone and alluvium thickness respectively, m; α_m , α_c – angles of stable slopes of the temporary idle soft and hard rock edges, degrees; n_m , n_c – number of transport areas on soft and hard rocks, items; N_c , n_c – number (items) and height of soft and hard rock benches, m; B_{bc} – safety bench width, m.

Increase of the slope angle along the non-working edge profile up to 50° on the marked site enables additional 100 mln t of magnetite quartzite to be mined. Annual

profitability $E = 95,13$ mln UAH due to temporary retirement of only oxidized hornstone and is determined as follows:

$$E = (V_o \times q_o \times e) / T_o , \quad (3)$$

where V_o – the volume of long-term conservation of oxidized hornstone, mln m^3 ; q_o – cost of mining 1 m^3 of oxidized hornstone, US dollars; e – the factor of conversion into the national currency, UAH/US dollars.; T_o – the oxidized hornstone conservation period, years.

The overall profitability of the suggested method is to be substantiated during the design documentation preparation and will significantly increase due to not only the growth of slope angles of idle benches and the idle wall as a whole but also additional increase of the raw material base of the pit and reduction of the distance of transporting oxidized hornstone to the stockpiling zone in the mined area.

Besides the above method of backfilling crater stoping, it is possible to increase the mining safety through drilling holes on the entire area of the underground working roof. The mined area can be backfilled with the crushed overhanging and overburden rocks, safety of stockpiling can be increased and mining costs can be decreased. At that, according to rock hardness and roof height of the underground working there is determined the size of the drillhole spacing for loosening rock of the first from the surface bench according to rock hardness in the working zone of the pit 1 (Fig. 2), the location of the underground working 2 and the thickness of its roof (3). The working soft rock area 4 is widened sufficiently to form the working area 5 of the first from the surface bench 1 according to rock types where the area of possible outbreak of the roof 2 of the underground working 3 is enclosed with the protective rock banks 6.

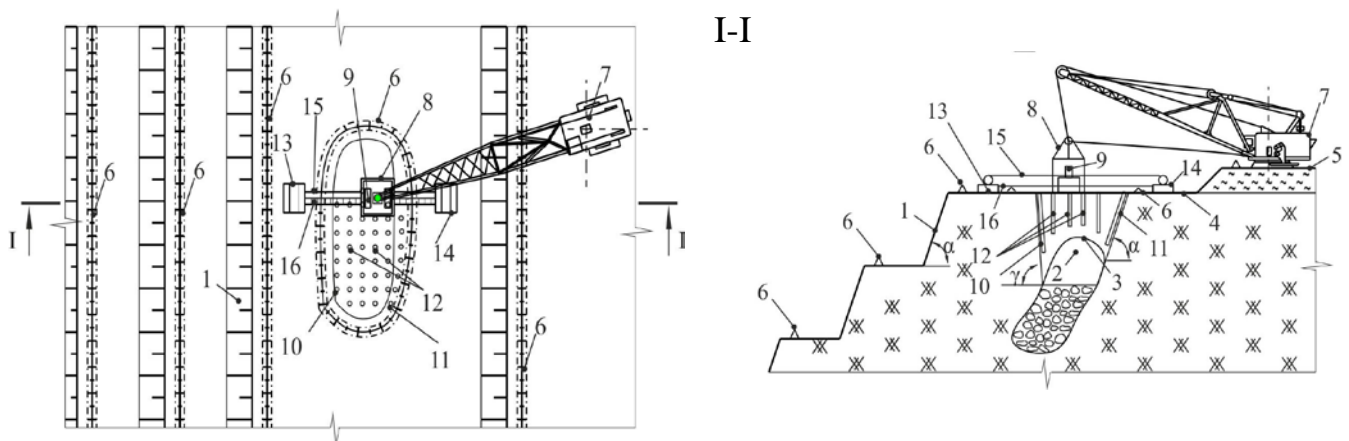


Fig. 2. The method of filling mine working in the massif of working pit edge

The dragline 7 is placed beyond the possible outbreak of the crater stoping of the underground working 2 on the working area 5. Instead of the bucket the dragline 7 is equipped with a suspended platform 8 with a drilling rig 9 on it and put down on the required place. Three sloping holes 10 are drilled sequentially on the outer contour at the angle γ from the side of the working pit edge and sloping 11- at the angle α from the side of the unmined massif. Then, according to the given spacing and depth parameters, loosening holes 12 are drilled.

To service drilling and blasting on a safe distance from the protective banks 6 on the opposite borders of the possible outbreak of the pit crater 2 two reversible winches (13, 14) are placed from the side of the mined area and from the side of the

unmined massif respectively. They are equipped with a suspension cable 15 which conveys explosives and stemming material to the places of hole drilling 12. Personnel use a footbridge on two cables 15 and 16. The suspension cable 15 and the footbridge 16 go across the cross-section of the underground working 2. After delivering explosives and stemming material to holes the equipment is taken beyond the drilling and blasting area. When breaking the roof 2 the loosened rock fills the underground area 3. Slope angles γ and α are made about 85° and 60° . After that the bench in the working zone of the pit 1 is mined in the established order.

The suggested technology of mining the underground area in the massif of the working pit edge allows to receive significant financial and material savings. At Pit№3 of the PJSC “ArcelorMittal Kryvyi Rih” there are several underground workings that interfere with normal development of benches [3]. As their outbreak into the working zone of the pit is planned to be done during a year or a five-year period, the profitability of the claimed technical solution advisable to calculate only investing capital in the acquisition of basic working equipment. In this case the annual profitability E (thousand UAH) is calculated as follows:

$$E = (m_1 - m_2) \cdot K \cdot C \cdot E_n,$$

where m_1, m_2 – working equipment weight, tonnes; in accordance with the technical characteristics and the prototype excavator ЕКГ-5 (248 t), the receiving bunker БС-120 (13 t), the jaw crusher ЩДП-12×15 (115.7 t), the conveyer loader (16 m, 15 t), the blast-hole drill СБШ-250МН (66 t); declared by way of lead dragline ЕШ-6/45МН (295 t), the blast-hole drill 2СБР-125-30 (12 t) which are moved on a platform weighing (5 t); K – cost of 1 t of the equipment, thousand US dollars, $K = 10000$ US dollars; C – the exchange rate of Ukraine’s National Bank as of 2015, $C = 21$ UAH; E_n – the normative factor of capital investment use, $E_n = 0,12$.

According to the given data the annual profitability due to introduction of the new technology equals 3168 thousand UAH.

Conclusion. On the basis of the analyses, observations and geological surveys carried out by the Research Ore Mining Institute the map has been made that systemizes the combined plan of underground craters stoping and anomalous zones, estimates the mined area and its possible outbreak on the surface. Open pit mining has been proved to be impossible without previous backfilling of voids. There has been developed the technology of backfilling craters stoping with rock overburden from the surface with the help of new methods and equipment. The technology enables substantial profitability due to prevention of surface destroying caused by surface dumps.

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ОПРЕДЕЛЕНИЕ ПРОПУСКНОЙ СПОСОБНОСТИ И ЭНЕРГОЕМКОСТИ ТРАНСПОРТИРОВАНИЯ ГОРНОЙ МАССЫ КОНВЕЙЕРНЫМ ТРАНСПОРТОМ УГОЛЬНЫХ ШАХТ

На основании математической модели функционирования систем конвейерного транспорта с последовательным и параллельным соединением конвейеров и бункеров, а также с древовидной веерной и самоподобной структурами получен рекуррентный алгоритм определения их пропускной способности и энергоемкости транспортирования угля. Приведены примеры расчетов при различных значениях грузопотоков и производительностей питателей.

На підставі математичної моделі функціонування систем конвеєрного транспорту з послідовним і паралельним з'єднанням конвеєрів і бункерів, а також з деревовидною віяловою і самоподібною структурами одержано рекуррентний алгоритм визначення їх пропускної спроможності і енергоємності транспортування вугілля. Приведено приклади розрахунків при різних значеннях вантажопотоків і продуктивності живильників.

On the basis of mathematical model functioning of conveyor transport systems with a consistent and parallel connection conveyors and hoppers, and also tree-fan system and a self-similar structure. Receive the recurrent algorithm to determine their capacity and energy coal transport. Illustrates the calculated values for different values of traffic flows and capacity feeders

Вопросами определения пропускной способности систем конвейерного транспорта занимались многие исследователи [1–6].

В работах [5–6] на основании метода динамики средних для марковских процессов получен алгоритм определения средней пропускной способности системы конвейерного транспорта с последовательным и параллельным соединением конвейеров с бункерами и без бункеров, а также алгоритм определения средней пропускной способности системы конвейерного транспорта для древовидной веерной и самоподобной структур соединения конвейеров с бункерами и без бункеров.

В работе [6] приведен алгоритм определения средней энергоемкости транспортирования для систем конвейерного транспорта древовидной самоподобной структуры без бункеров.

В данной работе, на основе разработанных математических моделей функционирования систем конвейерного транспорта с бункерами [7], рассмотрен вопрос определения средней энергоемкости транспортирования системы подзем-