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IMPROVEMENT OF DRUM SHEARER COAL LOADING PERFORMANCE

The article discusses improvement of broken coal loading on armored face conveyor by tailgate drum of a shearer in the course of fully mechanized coal mining. The scope of the analysis covers coal flow in the zone of coal–pick interaction, axial displacement between vanes, passive removal from the drum to the conveyor as well as coal flow formation on the conveyor. The features and constraints of coal flow in each zone are discussed. It is shown that the highest influence on the coal loading efficiency is exerted by the drum and conveyor spacing, discharge window size and the face side height of the conveyor. The discharge window coefficient is introduced. It is pointed at the required reduction in the coal flow resistance by increasing the cross-section area of the discharge window and the volume of coal loading zone on the conveyor. The discharge window cross-section area is increased by implementing the proposed integrated engineering solutions, in particular, conical drum web, ranging arm frame structure and decreased depth of cross-section of the supporting beam. The accepted new design enables increasing maximum loading capacity of the shearer tailgate drum, decreasing dust formation and coal fineness in the process of coal removal from cutting zone and loading on conveyor, as well as reducing energy consumption.

Key words: *coal, longwall face, shearer, drum, loading, armored face conveyor, efficiency*
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Introduction

On the road of development, thin seam shearers have been advanced from the 1st to 4th generation machines, including layout, energy and structure. New technologies and structural concepts enable high performance operation of coal shearers in more difficult geological conditions. At the same time, there arises a conflict: high shearer performance leads to higher coal output, on the one hand, as well as to excessive fines (up to 40%) in the end product and extreme dust formation, on the other. In particular, shearer loading performance is yet to be optimized.

With increasing rate, capacity of the tailgate drum of the shearer may exceed capacity of a coal discharge window. The discharge window is understood as a conventional cross-section area of the loading channel from the drum and to the armored face conveyor. The coal flow resistance in this area is equal to the real resistance value.

The drum rotation speed governs the axial velocity of coal flow between vanes and, thus, in the discharge window [1–5]. Geometry of the discharge window conditions the rate and completeness of removal of broken coal from the cutting zone [6].

The process of broken coal loading is characterized by:

— variable density of coal flow in different zones of loading channel, amassment of broken coal considerable portion of which remains on the floor (unloaded);

—circulation flow of broken coal, which induces extra crushing, dust formation and energy consumption [7, 8];

—the use of an apron results in over-filling of scrolls between vanes.

It is inadvisable to increase the discharge window capacity by means of speeding up the drum rotation as the cutting capacity of picks can be decreased. For this reason, the drum rotation speed adjustment is usually unprovided; though, sometimes, step-by-step speed change is allowed.

The quality of coal removal from the cutting zone is greatly affected by the nonuniformity of coal inflow to the discharge window due to small number of vanes. The influence of the latter on the shearer loading efficiency is studied insufficiently while the system analysis is absent at all.

Coal loading analysis

At the present time, the double-drum shearer drums can rotate along and across the shearer advance direction [9]. Depending on the rotation direction of the tailgate drum, broken coal fills in a certain manner the space between the drum vanes [10, 11]. Mostly, broken coal falls on the floor and is displaced by the tailgate drum vanes toward the face conveyor for a distance $B_z + L$ (**Fig. 1**). Some coal is left in the layer with a thickness h between the tailgate drum and the conveyor in the longwall face area.

Coal flow in the work area of the drum vanes can be divided into three zones (see Fig. 1): I — discharge (discharge window); II — zone overlapped by the drive motor shell and III — zone overlapped by the face conveyor side at a distance L from the drum vanes.

Coal removal efficiency from the cutting zone is restricted by high resistance of the discharge window (zone I) due to its small dimension. Higher resistance (zones II and III) results in over-filling of the vane-to-vane space and in built-up circulation flow of coal.

Effect of the cross-section area of the discharge window on the loading capacity of drum shearers was proved experimentally in Oktyabr Mine, Kemerovo Region, Russia. When the drums 1600 mm in diameter were replaced by the drums with a diameter of 1400 mm, the volume of unloaded coal left on the floor increased 2.8 times while the discharge window cross-section area was decreased from 0.24 m^2 to 0.17 m^2, or by 30% [12]. This is an evidence of disproportional relationship between the coal discharge completeness and the discharge window cross-section area.

The discharge window capacity can be estimated in terms of a discharge window coefficient given by:

$$K_o = \frac{S_o}{S_h}, \quad (1)$$

where S_o — the discharge window cross-section area, m^2; S_h — the useful area of loading, m^2.

In a general form, the discharge window cross-section area is (**Fig. 2**):

$$S_o = S_1 - S_2 - S_3 - S_4 - S_5, \text{ m}^2, \quad (2)$$

where S_1 — the design cross-section area of the drum bottom part bounded by the center lines of the apron bracket $h_t/2$ and ranging arm $h_r/2$; S_2, S_3, S_4, S_5 are the cross-section areas of the structural elements, namely, apron bracket, conveyor, drum web and ranging arm, respectively.

On the other hand, S_O can be determined in terms of the shearer drum parameters:

$$S_o = \left[\frac{1}{4}\cdot\frac{\pi\varphi}{360}(D+d) - \frac{(h_r+h_t)}{2}\right].(D-d) - \\ -\frac{1}{8}D^2\arcsin\frac{D-2h_k}{D} + \frac{1}{8}D(D-4h_k), \text{ m}^2, \quad (3)$$

where h_r — the depth of cross-section of the ranging arm body, m; h_t — the depth of cross-section of the apron rod, m; h_k — the height of the conveyor side, m.

The useful area of loading is given by:

$$S_h = \frac{\pi}{4}(D^2 - d^2) - \frac{(D-d)\delta_h N_z}{2\sin\alpha_h}, \text{ m}^2, \quad (4)$$

where D — the reduced diameter of the drum, m; d — the drum web diameter, m; d_h — the vane thickness, m; N_z — the number of vanes; α_h — the angle of vanes, deg: $\alpha_h = \text{arctg}\frac{t}{\pi D}$, t — the spacing of vanes, m. Then, the shearer drum coal loading capacity understood as the maximum loading capacity at the preset design parameters and operation conditions is:

$$Q_{\max ho} = K_o Q_h, \text{ t/min}, \quad (5)$$

where Q_h — the drum loading capacity at the constant coal flow, t/min.

The distance between the drum and conveyor, L, has an essential influence on the loading efficiency. According to the experimental research findings by K. N. Belikov [13], an increase in L from 175 to 375 mm more than halves the loading completeness. The modeling results obtained by Kuidong Gao, Changlong Du, Jianghui Dong ands Qingliang Zeng [14, 15] show that with a drum diameter of 600 mm, depth of section of ranging arm of 350 mm, drum rotation speed of 60 rpm and shearer advance rate of 2 m/min, the increased drum–conveyor distance from 150 to 33 mm results in reduction in the loaded coal volume by more than 6 times.

The unloaded coal volume can be found from the expression [16, 17]:

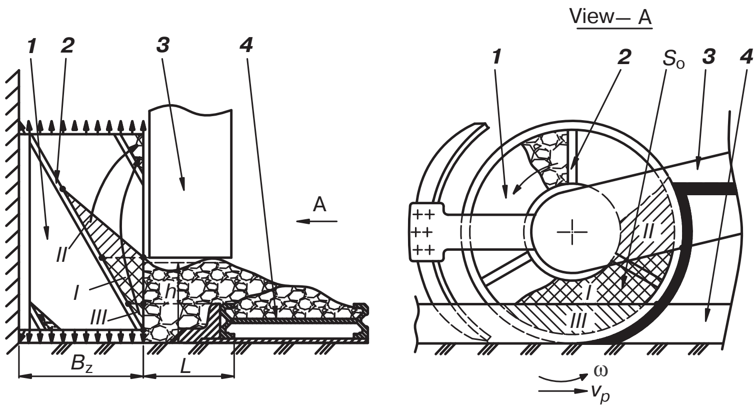

Fig. 1. Coal loading:
I — discharge window; II — overlapping by the drum motor shell; III — overlapping by the conveyor side; *1* — tailgate drum; *2* — vanes; *3* — ranging arm; *4* — conveyor; S_O — discharge window cross-section area

$$Q_{n.p} = v_p L h, \text{ m}^3\text{/min}, \quad (6)$$

where v_p — the shearer advance rate, m/min; L — the distance from the drum to the face conveyor, m; h — the thickness of the coal layer left on the floor between the drum and the face conveyor (see Fig. 1).

It follows from (6) that with shortening haulage distance L to the conveyor, the volume of unloaded coal is decreased and the loading efficiency is higher. Currently, the haulage distance between the shearers and conveyors is mostly longer than 300 mm, and it is difficult to reduce the value due to structural features of their assembly. For this reason, it is still of concern to find engineering solutions to improve coal loading efficiency of the tailgate drum.

Engineering solution

After analysis of the known designs of shearers and units, the proposed engineering solution [18] is aimed to improve efficiency of coal loading to the armored face conveyor by way of enlarging the discharge window cross-section area and the loading zone size.

A shearer *7* (**Fig. 3**) has two symmetrical drums arranged at the head and tail. The unidirectionally rotating headgate drum cuts top coal and loads it to the armored face conveyor *9*. The tailgate drum *1* cuts bottom coal and, born by the apron *8*, also loads coal to AFC.

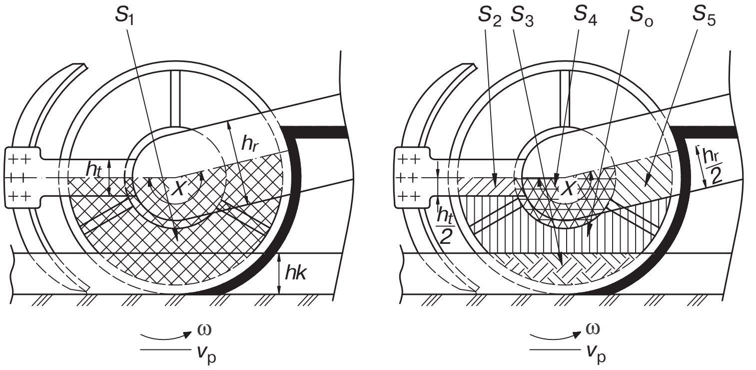

Fig. 2. Calculation scheme for discharge window cross-section area

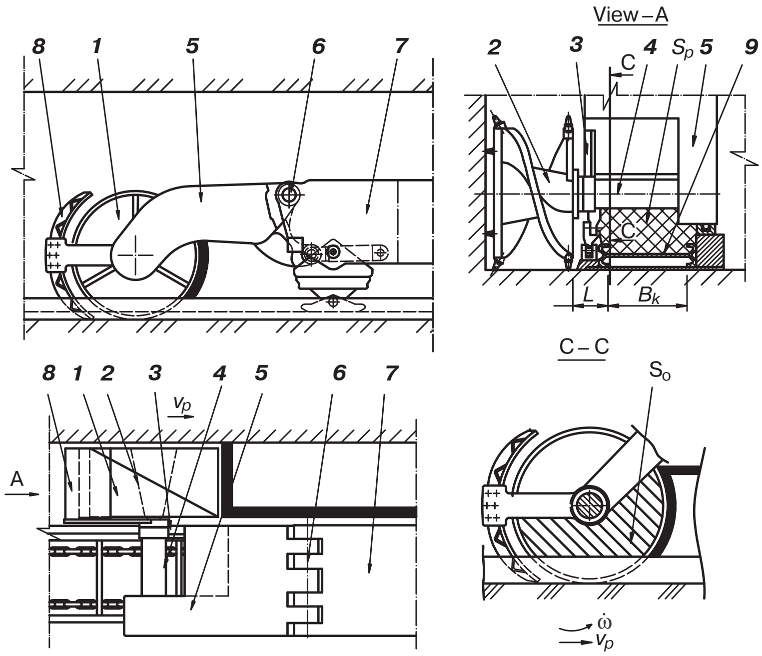

Fig. 3. Shearer loading unit:
***1* — tailgate drum; *2* — drum web; *3* — supporting beam; *4* — bush sleeve; *5* — goaf side beam; *6* — swing joint; *7* — shearer body; *8* — apron; *9* — armored face conveyor**

In this flow chart, the most portion of broken coal is loaded to AFC by the tailgate drum through the discharge window S_o (see Fig. 3) between the bottom surface of the face side beam 3, face side of AFC 9, apron and drum web *2*. The loading zone V_p is formed by the surface of the conveyor *9*, bush sleeve *4*, apron *8* and the tailgate drum-face contact surface. Thus, the loading zone volume is given by:

$$V_p = S_p L_p, \text{ m}^3, \quad (7)$$

where S_p — the coal flow cross-section area on the conveyor, m^2; L_p — the loading zone length, m.

The increase in the cross-section area of the discharge window, S_o, and in the size of the loading zone on the conveyor, V_p (see Fig. 3), is ensured by adding the shearer design with two beams: goaf side beam *5* and face side beam *3*. The goaf side beam *5* made as a whole entity with the ranging arm body and equipped with the reinforcement ribs is arranged on the goaf side of the shearer above the goaf side of the conveyor *9*. The face side beam *3* as an additional bearing assembly of the gear shaft of the drum *1* is arranged above the face side of AFC *9*. Both beams are grippingly engaged with drive body of the drum *1* at the bottom and with the bush sleeve *4* on the opposite side, which improves rigidity of the resultant frame. The lower surfaces of the face side beam *5* and goaf side beam *5* are upward concave, the web *2* is conical with the reducing diameter towards the discharge window S_o, and the face side beam *3* has a smaller cross-section than the goaf side beam *5*. All that allows enlarging the cross-section area of the discharge window, S_o, and the volume of the loading zone, V_p, on armored face conveyor.

The conical shape of the web makes it possible to improve coal loading efficiency as the effective volume of the drum increases toward the discharge point while the the broken coal fineness is reduced. The decrease in the depth of cross-section of the supporting beam makes it possible to improve efficiency of coal removal from cutting zone and loading on AFC, as well as decreases broken coal fineness and energy consumption.

Conclusion

The proposed integrated engineering solution provides:

— the increase in the cross-section area of the discharge window and in the volume of coal loading zone on AFC;

— the decrease in the coal flow resistance;

— the reduced broken coal fineness and energy consumption;

— the higher efficiency of coal loading by the tailgate drum.

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ANALYSIS OF GEARING-UP DEVICES FOR HIGH-SPEED DIAMOND BIT DRILLING OF LONG EXPLORATION HOLES

In long-hole drilling the rate of penetration reduces as in this case much more energy is consumed to rotate drill strings, vibrations and oscillations of drill strings are higher, and drill pipes which often operate at rim of abilities break more frequently. All that implies that a drill string is a low-efficient transmitter of energy from drill rigs to bottomhole, especially in long-hole drilling with diamond bits.

It is possible to intensify rock drilling process by placing mechanical energy generator at bottomhole. To this effect hydraulic motors and electric drills are currently available. However, hydraulic motors need high discharge pressure, while rotation speed of drill bits and drilling fluid flow rate are difficult to control and adjust. In case of electric drills, delivery of electric energy to bottomhole is complicated.

All these constraints are overcome with the help of mechanical transmission represented by a bottomhole multiplying gear capable to speed-up diamond drill bits several times as against the drill string rotation.

Key words: *drilling tool, bottomhole, multiplying gear, hole, diamond bit, rotation speed, core drilling*
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Introductoin

Technologies of hard rock drilling enjoy booming development both in surface and underground mining these days [1]. The growing demand for such technologies is explained by the application of advanced process solutions in surface and underground mining of hard minerals, coal and rare earth metals, in civil construction, etc. [2–11]. Recently diamond bits gain an increasing popularity as they extend service life of rock-breaking tools, increase penetration rates, reduce lowering/lifting operation time and, thus, displace rolling cutter drill bits.

The research in the area of hard rock destruction by diamond tools, as well as the experience gained in drilling and natural stone processing specifies that the potential rise in the penetration rate is first and foremost connected with an increase in the diamond drill bit speed. At the same time, the higher speed of drill bits and, accordingly, drill strings, is restricted by the drilling conditions in reality. For instance, in long-hole drilling (to 1000 m and more) with a diameter of 76 of 59 mm using a diamond rock-breaking tool, the drill string rotations per minute are no more than 350 min^{-1}. In case of the higher rotation speed, the risk of drill string breakage grows, and drilling consumes much more energy to overcome the drill hole wall friction. The linear cutting speed is no more than 1–1.5 m/s, which is known to be insufficient for a diamond tool (the recommended speed for impregnated diamond drill bits is 2–5 m/s) [12].

It is even impossible to solve the problem with the increasingly popular coiled tubing as the use of very long flexible pipes needs a bottomhole hydraulic actuator which requires high discharge pressure; moreover, it is difficult to control rotation speed of drill bit and flow rate of drilling fluid. Electric drilling needs an electric power supply [13, 14].

A way out may be the use of mechanical transmission as a bottomhole speeder capable to increase diamond bit speed several times as compared with the drilling string rotation.

Analysis of gearing-up devices for high-speed diamond

The machine should be reliable and high-performance while low metal- and energy-consuming, and should comply with engineering ergonomics [15]. These require-