Archives of Mining Sciences 45, 3 (2000) 409–426

KRZYSZTOF KRAUZE*

AN ANALYTICAL ASSESSMENT OF LOADING PROCESS FOR CUTTING ELEMENTS OF LONGWALL CUTTER-LOADER

ANALITYCZNA OCENA PROCESU ŁADOWANIA FREZUJĄCYMI ORGANAMI ŚLIMAKOWYMI KOMBAJNU ŚCIANOWEGO

An analysis of available information and design for cutting elements of longwall cutter-loaders has indicated that an analytical analysis of its loading capacity is required. An analytical method for assessment of worm cutting elements has been developed by establishing boundary conditions and appropriate relationships. The method enables an effect of design and kinematic parameters on loading efficiency to be determined and the most favourable parameters for longwall cutter-loader and downstream machinery to be found. Therefore, any problems related to mined coal loading onto longwall conveyors and retarding the mining progress due to longwall chocks can be avoided. By knowing the loading capacity for cutting elements and necessary investment expenditures for longwall equipment the user can choose the most suitable solution.

Key words: cutting elements, loading.

Proces skrawania jest procesem zapoczątkowującym cały proces urabiania (skrawanie, ładowanie) i dlatego ważne jest prawidłowe ustalenie parametrów tego procesu (organu). Proces ładowania zachodzący równolegle z procesem skrawania jest procesem pomocniczym, choć równie ważnym. Postawienie w ten sposób powyższego zagadnienia pozwala ustalić następującą procedurę, w której parametry procesu ładowania wynikają i są ograniczone procesem skrawania. Stwierdzenie to jest tym ważniejsze, że jak to już wspomniano, proces skrawania jest stosunkowo dokładnie opisany w literaturze fachowej. Proces ładowania natomiast traktowany był marginalnie i z niewielkim powiązaniem z procesem skrawania [K r a u z e, 1995].

Podjęte próby opisu procesu ładowania [K r a u z e, 1995] skutkowały opracowaniem opisu tego procesu poprzez teoretyczne związki między parametrami konstrukcyjnymi organu i ruchowymi maszyny urabiającej. Powyższy opis procesu ładowania wymaga jednak

* KATEDRA MASZYN GÓRNICZYCH, PRZERÓBCZYCH I TRANSPORTOWYCH, AKADEMIA GÓRNICZO-HUTNICZA, 30-059 KRAKÓW, AL. MICKIEWICZA 30

wielu uzupełnień, które wprowadzono w niniejszym opracowaniu, przyjmując, że proces ładowania nie może powodować degradacji urobku.

Mając powyższe na uwadze określono związki analityczne pomiędzy parametrami konstrukcyjnymi i ruchowymi organu, kombajnu ścianowego, przenośnika oraz obudowy a:
— wydobyciem dobowym (rozdział 2),
— wpływem parametrów obudowy na prędkość posuwu kombajnu (rozdział 3),
— zdolnością przejmowania urobku przez przenośnik (rozdział 4),
— objętością wewnętrzną organu i jego wydajnością urabiania (rozdział 5),
— wielkością furty ładującej (rozdział 6),
— obciążeniem organu pochodzącym od procesu ładowania (rozdział 7),

tak by wyznaczone wielkości pozwalały uzyskać wymagany stopień załadowania urobku.

Słowa kluczowe: organ urabiający, ładowanie.

1. Introduction

A set of special machinery and equipment for coal longwall mining, called a longwall complex, is generally comprised of three main machines for coal mining, hauling and working face protection. By appropriate sizing of mining machine (cutter-loader, shearer), means of conveyance (longwall alongside conveyor, bottom conveyor) and roof supports (chocks, shields) it is possible to reach a target output (day output) as required by the user, i.e. the coal mine. Therefore, it should be stated that the sizing of machinery and equipment for longwall complex consists in combining these equipment so that the target output is reached at minimum operating costs, while considering both mining and geological as well as organisational conditions.

The selection of mining machine that commencing coal mining is of utmost importance. A mining machine must meet the day output criterion, while taking into account specific working face conditions. The problem mentioned above is reduced to establishing such mining machine movement and power parameter as well as outer dimensions that meet the user's requirements. Since cutter-loaders prevail in Polish coal mining, it is necessary to develop an appropriate method enabling the cutter-loader parameters to be chosen according to specific mining and geological conditions and assumed day output.

It is also important to establish the operation and performance parameters for remaining longwall equipment (conveyor, chocks) so that the day output criterion is fulfilled.

By establishing the parameters of longwall machinery and equipment it is possible to determine the parameters of work cutting elements to meet the day output criterion, while considering all and any factors that have an effect on cutting and loading.

The cutting is the process that begins coal mining (cutting, loading) and therefore its parameters should be established properly. The loading that is carried out in parallel with the cutting is an auxiliary process of great importance too. Such approach to the problem mentioned above leads to a procedure in which the loading process parameters are determined from and constrained with the cutting process. This is of great importance as the cutting process has been described in detail in many papers, while the loading process has been analyzed marginally and separately with only minute references to the cutting process [Krauze, 1995].

The attempts to describe the loading process in [Krauze, 1995] led to theoretical relationships between the design and performance parameters of mining machine. In this paper the description mentioned above has been extended to include an assumption that mined coal cold not be degraded in the loading process.

2. Day output

The longwall mining efficiency is expressed by the day output, i.e. an amount of coal mined during a day. The relationship between the day output and working face parameters can be written as follows [Krauze, 1995]:

$$V_s = \frac{H \gamma L Z T}{L\left(\frac{1}{v_p} + \frac{1}{v_m}\right) + t_p + t_0}, \tag{1}$$

where:

- H — longwall height;
- L — longwall length;
- Z — mining machine web;
- γ — mined rock specific density;
- T — face mining time within a day;
- t_p — flitting time;
- t_0 — remaining time for single cutting;
- v_p — mining machine travelling rate (operational);
- v_m — mining machine handling velocity.

If $v_p = k v_m$ then

$$V_s = H \gamma z T \frac{L v_p}{(1+k)L + (t_p + t_0) v_p}, \tag{2}$$

where:

- $k = 0$, for two-way mining, $v_m = 0$,
- $0 < k \leqslant 1$, for one-way mining, $v_p \leqslant v_m$,
- $k > 1$, for one-way mining, $v_p > v_m$.

It follows from (2) that such parameters as H, γ, Z and T have a proportional effect on day output. In that case the user, i.e. the coal mine, can adjust the day output by changing these parameters.

I should be noted that in many cases it is impossible to change the face height H as it corresponds to the bed thickness. Thus, the day output can be adjusted by changing Z and T. If changing the web Z from 0.65 m to the most common value of 1.0 m then V_s increases by 54%. Similarly, when changing the face mining time T from 18 hours to 24 hours, the output V_s increases by 33%.

The choose of mining method (on-way or two-way) is described by coefficient k that is zero for two-way mining. For one-way mining k is ranging from 0 to 1. This means that the handling speed is equal or greater than the travelling rate. There are no technical and economical reasons for the third case presented above ($v_p > v_m$). In practice, $v_p = v_m$ ($k = 1$) or $v_p = 0.5\ v_m(k = 0.5)$.

The second term of relationship (2) is more complicated. While considering its denominator one can conclude that it reaches the minimum value for $k = 0$ and for the smallest sum of t_p and t_0.

It follows from relationships presented above [Krauze, 1999] that — longwall length L decides on other parameters such as travelling rate v_p, sum of $t_p + t_0$ and mining method:

— an effect of travelling rate v_p is very small for short coal faces (about 100 m in length) and therefore the cutter-loader of travelling rate exceeding 4 m/min should not be used:

— for longer coal faces (about 300 m in length) it is necessary to use high travelling rates (up to 10 m/min), short times t_p and t_0 (up to 20 minutes) and two-way mining:

— if one-way mining is required, the handling speed shall be increase (minimum $v_m = 2\ v_p$), while times t_p and t_0 shall be shortened up to 20 minutes.

The remarks mentioned above shall be considered along with mining, technical and economical conditions specific for the user. This is of crucial importance as it decides on selection of other longwall equipment such as conveyor and roof support.

3. An effect of chock movement parameters on the travelling rate of a longwall cutter-loader [Krauze, 1999]

The selection of longwall supporting equipment depends on the roof and floor conditions as well as technical parameters of working face. In addition, the movement parameters (flitting time) shall be chosen respectively. The problem consists in selecting such a control and supply system so that the flitting time for single section could be shorter than the cutter-loader travelling time on its entire length.

$$t_{st} = \frac{t_{0b}}{v_p}, \tag{3}$$

where:

t_{0b} — pitch (width) of single section.

For $v_p = 5$ m/min and $t_{0b} = 1.5$ m the time t_{st} is 18 seconds. If v_p is increased up to 10 m/min, the time t_{st} is reduced to 9 seconds. However, the shorter flitting time for single section, the higher performance (and much expensive) control and supply system is required.

4. The mined coal collecting capacity for a longwall conveyor [Krauze, 1999]

The conveyor performance, drive power and chain speed (conveyor type) should be selected so that a longwall conveyor could haul all mined coal from the mining machine. The mined coal collecting capacity Q_{zp} is obtained by multiplying the mined coal stream cross-section F_t on the conveyor and the speed sum or difference for the cutter-loader (v_p) and chain (v_t) depending on its relative motion (backward or forward direction).

$$Q_{zp} = F_t(v_t \pm v_p). \tag{4}$$

The cutter-loader bulk performance Q_{kO} is:

$$Q_{kO} = k_L H Z v_p k_r, \tag{5}$$

where:

k_r — mined coal loosening coefficient ($k_r \geqslant 1$),

k_L — a coefficient that includes coal loaded spontaneously due to falling from the side wall under the rock pressure ($k_L \leqslant 1$).

In the case of speed difference the mined coal collecting capacity Q_{zp} should be equal or greater than Q_{kO}, that is,

$$Q_{zp} \geqslant Q_{kO}. \tag{6}$$

By using relationships (4) and (5) the inequality (6) can be written as a function of Q_{kO}, F_t and v_p

$$Q_{tO} \geqslant Q_{kO} + F_t v_p \tag{7}$$

or Q_{kO}, v_t and v_p

$$Q_{tO} \geqslant \frac{Q_{kO}}{1 - \frac{v_p}{v_t}}. \tag{8}$$

In both cases relationships (7) and (8) described the required longwall conveyor bulk capacity Q_{tO}. Since in manufacturers' catalogs the capacity is given in Mg per hours it is necessary to introduce the specific density γ. However, it should be kept in mind that this refers to so called apparent conveyor bulk capacity Q_{tp}, since the coefficient of mined coal loosening k_r has been taken into account in these relationships.

$$Q_{tp} \geqslant (Q_{kO} + F_t v_p)\gamma, \tag{9}$$

$$Q_{tp} \geqslant \frac{\gamma\, Q_{k0}}{1-\dfrac{v_p}{v_t}}. \tag{10}$$

The actual weight of mined coal to be transported on the conveyor (actual capacity Q_{tr}) should be computed by omitting k_r.

$$Q_{tr} \geq \left(\frac{Q_{k0}}{k_r} + F_t v_p\right)\gamma, \tag{11}$$

$$Q_{tr} \geq \frac{\gamma\, Q_{k0}}{\left(1-\dfrac{v_p}{v_t}\right)k_r}. \tag{12}$$

It follows from (9) that a conveyor of given Q_{tp} and F_t (trough width and height) can be chosen. Relationship (10) enables selecting the chain speed v_t or adjusting v_p for assumed capacity Q_{tp}.

Relationship (12) can be transformed to determine the maximum travelling rate v_p for given conveyor capacity Q_t:

$$v_p \leqslant \frac{60\, Q_t\, v_t}{3600\, v_t\, \gamma\, H\, Z\, k_r\, k_L \pm Q_t}, \tag{13}$$

where the sign „—" refers to backward (opposite) relative motion and „+" to forward direction. For the data presented above one can find that $v_{p\max} \leqslant 10$ m/min and $v_{p\max} \leqslant 15$ m/min for forward and backward motions, respectively.

It follows form (13) that:

— various travelling rates shall be used in two-way mining to reach the required day output:

— in one-way mining the cutting or loading process (as required) shall be carried out during movement with lower travelling rate v_p.

5. Inner capacity of cutting element and its mining efficiency

The paper [K r a u z e, 1999; K r a u z e, 1995; K r a u z e, 1994] presenting a model of loading process have indicated that the cutting element inner volume V_0 should be greater than the mined coal volume V_u to ensure proper loading with a worm cutting element, while observing the conditions resulting from experimental investigations. In addition the mined coal stream transferred from the mining machine into the conveyor shall be of such volume that its free flow through so called loading gate is assured. The sufficient condition is an analysis of these phenomena during one turn of cutting element.

The mining efficiency for the front (V_{up}) and rear (V_{ut}) cutting elements have been also found (see Fig. 1).

$$V_{up} = 0.25\, k_r\, Z\, D_s^2 \left[\pi - 2 \ \arccos \frac{v_p}{n\, D_s} + \sin\left(2 \arccos \frac{v_p}{n\, D_s} \right) \right], \tag{14}$$

$$V_{ut} = V_u \left(\frac{H - D_s}{D_s} + 0.3 \right), \tag{15}$$

where:

k_r — mined coal loosening coefficient,
D_s — diameter of cutting element,
n — rotational speed of cutting element.

In the paper [B ę b e n, 1999) field F_4 (see Fig. 1) has been replaced with an equivalent rectangular field to obtain the following simplified relationship describing the mining efficiency V_u:

$$V_u = k_r\, Z \frac{v_p}{n} D_s. \tag{16}$$

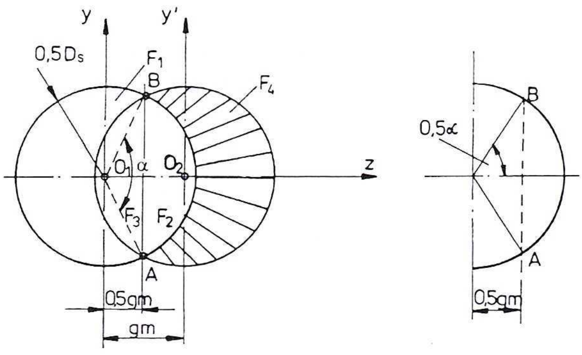

Fig. 1. Worm cutting element cross-section.
α — central angle, g_m — maximum depth of cut

The inner capacity V_0 corresponds to free space between the worm blades that haul mined coal onto the conveyor. This space should allow free movement of transported coal into discharging point. If consider the volume between the worm blades as the difference between the volumes of cylinder of diameter D, hube of diameter d and blades, then for cutting elements with so called normal blades the inner capacity V_0 is:

$$V_0 = 0.25\, \pi \left(D^2 - d^2 \right) \left(Z_u - \frac{b}{\cos \alpha_2} \right) \tag{17}$$

and for cutting elements with overlapped blades:

$$V_0 = 0.25\, Z_u (D^2 - d^2)\left(\pi - \frac{i\, b}{D \sin \alpha_2}\right), \tag{18}$$

where:

Z_u — cutting element web without a cut-off disk (cutting-loading element),
D — cutting element drum diameter,
d — hub diameter,
b — blade thickness,
i — number of blades,
α_2 — helix angle.

As mentioned above, the model is based on the following general assumption:

$$\Delta = k_w V_0 - V_u \geqslant 0, \tag{19}$$

where:

k_w — cutting element filling factor.

Assuming that all the cutting element parameters are of appropriate values in respect of loading efficiency, then if the inequality (19) is not fulfilled the loading capacity is reduced, including a jam on cutting element. Δ depends strongly on the cutting element diameter and speed as well as the cutter-loader travelling rate. Thus, the maximum travelling rate and cutting element speed should be chosen carefully for given face height and cutting element diameter or a possibility to install a loader can be considered.

The model mentioned above has been extended to include the following factors:

— cutting element filling factor,
— direction of rotation,
— mating of the front and rear cutting elements,
— different hub shape,
— use of loaders,
— elimination of coal crushing,
— allowable travelling rate v_p for loading process

have not been considered in detail yet, it is reasonable to include them into the model described above.

At first, relationships (17) and (18) have to be revised to include different hub shape, thus another volume. This has been done by introducing a coefficient k_{kp} defined as the ration of the volume different than cylindrical one to the volume of cylinder of hub diameter d.

The corrected relationships for the inner volume take the form:

— for a cutting element with normal blades

$$V_0 = 0.25\, \pi (D^2 - k_{kp}\, d^2) \cdot \left(Z_u - \frac{b}{\cos \alpha_2}\right), \tag{20}$$

— for a cutting element with overlapped blades

$$V_0 = 0.25\, Z_u (D^2 - k_{kp} d^2) \cdot \left(\pi - \frac{i\,b}{D \sin \alpha_2}\right). \tag{21}$$

For example, for a cutting element with cone hub (truncated cone) k_{kp} is:

$$k_{kp} = \frac{V_n}{V_w} = \frac{\frac{1}{12}\pi\, Z_u (d_1^2 + d^2 + d_1\, d)}{0.25\,\pi\, Z_0\, d^2} = \frac{d_1^2 + d^2 + d_1 d}{3\, d^2}, \tag{22}$$

where:

V_n — cone hub volume,
V_p — cylindrical hub volume,
d_1 — cone base diameter.

If the cutting element diameter $D_s = 2.0$ m, then for $d_1 = 1.6$ m and $d = 0.9$ m coefficient $k_{kp} = 1.9794$ and for $D_s = 1.4$ m, $d_1 = 1.0$ and $d = 0.9$ m the same coefficient is 1.1152.

Similar corrections have been made for relationship (16) by introducing coefficient k_L to take into account the fact that mined coal is loaded in part spontaneously under the rock pressure. By writing this coefficient for the front and rear cutting elements we obtain:

$$V_{up} = \frac{k_r\, k_L\, Z_p\, v_p\, D_{sp}}{n_p} \tag{23}$$

and by eliminating 0.3 from (15) we get:

$$V_{ut} = \frac{k_r\, k_L (H - D_{sp}) Z_t\, v_p}{n_t}. \tag{24}$$

The total efficiency V_u is a sum of efficiencies for both cutting elements, i.e. V_{up} and V_{ut}:

$$V_u = V_{up} + V_{ut} = k_r\, k_L\, v_p \left[\frac{Z_p\, D_{sp}}{n_p} + \frac{(H - D_{sp})\, Z_t}{n_t}\right]. \tag{25}$$

For identical cutting elements relationship (25) takes the form:

$$V_u = \frac{k_r\, k_L\, Z\, v_p\, H}{n}. \tag{26}$$

When the inequality (19) is to be fulfilled, one should consider the use of loaders with or without shields. The necessary and sufficient condition for cutting elements with loaders involves that the mining efficiency per one turn of the front element is less than its loading capacity, that is:

$$\Delta = k_w\, V_{0p} - V_{up} \geqslant 0. \tag{27}$$

Then the allowable travelling rate v_p can be determined from (27):

$$v_p \leqslant \frac{k_w n_p V_{0p}}{k_r k_L Z_p D_{sp}}. \tag{28}$$

Should the mining machine is running without loaders, the following inequality shall be fulfilled:

$$\Delta = k_w (V_{0p} + V_{0t}) - (V_{up} + V_{ut}) \geqslant 0. \tag{29}$$

Similarly, the allowable travelling rate v_p for different cuttin elements can be obtained:

$$v_p = \frac{k_w n_p n_t (V_{0p} + V_{0t})}{k_r k_L \left[\frac{Z_p D_{sp}}{n_t} + \frac{(H - D_{sp}) Z_t}{n_p} \right]} \tag{30}$$

or for elements of the same parameters:

$$v_p \leqslant \frac{2 k_w V_0 n}{k_r k_L Z H}. \tag{31}$$

Obviously, relationships (27) and (29) can be freely modified to include other parameters of cutting elements, but in practice, the only possible change refers to reduced travelling rate.

The value of filling factor $k_w = 0.3$ is adopted from industrial practice for cutter-loader operation without shield loaders both for working to the rise and to the deep. It is possible to increase this coefficient by using loaders (see Fig. 2). However, as mentioned below, the value of $k_w = 0.3$ is recommended at the stage of design and sizing.

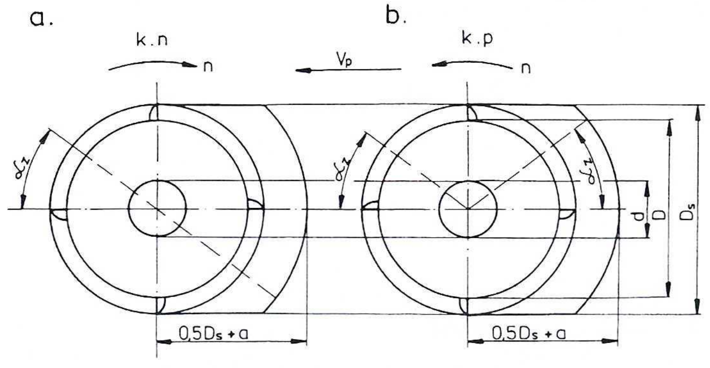

Fig. 2. A model for maximum allowable filling for a cutting element with a shield loader (α_z — angle of natural coal slip).
a — working to the rise, $k_w = 0.5$, b — working to the deep, $k_w = 0.7$

The use of loaders improves the loading efficiency by avoiding mined coal, in particular fine coal, to be left on the floor. If assume that only fine coal is hauled with a loader, then sufficient amount of fines shall be collected to ensure transfer onto the conveyor. Such situation may occur when a cutting elements is running with no loaders. Otherwise, any changes in loading resistance depends on the amount of mined coal collected on a loader. In extreme case, a cutting element can be jammed and mined coal is degraded as a cutting elements becomes to run as a compactor. Such situation should be avoid by establishing proper values of v_p, n, k_w and V_0 as well as by using a loader with a shield inclined at angle α_l (Fig. 3). In such a case the travelling rate component v_{pu} will enhance discharging of mined coal from an area between the cutting element and shield, while keeping the latter fixed.

In addition, it is highly recommended that the distance a between the loader and cutting element is $g_{\max}$ (maximum depth of cut) to avoid mined coal be crushed when loading.

The calculated maximum travelling rates for selected parameters of cutting elements, while taking into account the constrains resulting from loading process, are presented in tables 1 and 2. The computations were made for two values of diameter D_s (1.4 m and 2.0 m) for cutting elements with normal and overlapped blades and for operation with and without loaders and for different values of k_w.

The resulting values of v_p differ from the assumed maximum travelling rate ($v_p = 8$ m/min) and correspond to speeds known in mining practice. This suggests that the loading resistance leads the mining machine automatic control system to decrease the travelling rate, in particular when shield loaders are used. It should be specifically underlined that low values of k_w ensure the minimum recirculation and degradation of mined coal at minimum loading resistance. By increasing k_w it is possible to use higher travelling rates, but at a price of high degradation of mined coal and increased loading resistance.

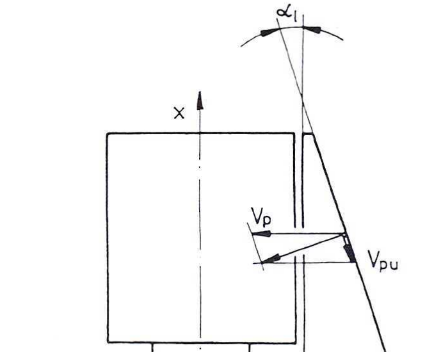

Fig. 3. A loader with inclined shield (diagrammatic drawing)

TABLE 1

Calculated maximum travelling rates v_p due to loading process for $D_s = 1.4$ m

$D_s = 1.4$ m, $v_p = 8$ m/min						
v_s m/s	2.85		4.00		5.50	
n rpm	39		55		75	
g_{max}, mm	205		146		107	
$D = 1.2$ m, $d = 0.9$ m, $b = 40$ mm, $Z = 0.65$, $Z_u = 0.55$ $H = 2$ m, $i = 3$, $\alpha_2 = 20°$, $k_r = 1.4$, $k_L = 0.9$						
	Normal blades			Overlapped blades		
V_0, m^3	0.250935			0.241606		
n, rpm	39	55	75	39	55	75
Cutting elements with no loaders, $k_w = 0.3$						
v_p, m/min	3.59	5.06	6.89	3.45	4.87	6.64
Cutting elements with loaders, working to the deep of front element, $k_w = 0.7$						
v_p, m/min	5.98	8.43	11.49	5.75	8.11	11.06
Cutting elements with loaders, working to the rise of front elements, $k_w = 0.5$						
v_p, m/min	4.27	6.02	8.21	4.11	5.79	7.90
Cutting elements with loaders, $k_w = 0.3$						
v_p, m/min	2.56	3.61	4.93	2.46	3.48	4.74

TABLE 2

Calculated maximum travelling rates v_p due to loading process for $D_s = 2.0$ m

$D_s = 2.0$ m, $v_p = 8$ m/min						
v_s m/s	2.85		4.00		5.50	
n rpm	27		38		53	
g_{max}, mm	296		211		151	
$D = 1.8$ m, $d = 0.9$ m, $b = 40$ mm, $Z = 0.65$, $Z_u = 0.55$ $H = 2$ m, $i = 4$, $\alpha_2 = 20°$, $k_r = 1.4$, $k_L = 0.9$						
	Normal blades			Overlapped blades		
V_0, m^3	0.967891			0.892849		
n, rpm	27	38	53	27	38	53
Cutting elements with no loaders, $k_w = 0.3$						
v_p, m/min	6.38	8.98	12.53	5.89	8.29	11.56
Cutting elements with loaders, working to the deep of front element, $k_w = 0.7$						
v_p, m/min	11.17	15.72	21.92	10.30	13.98	19.50
Cutting elements with loaders, working to the rise of front elements, $k_w = 0.5$						
v_p, m/min	7.98	11.23	15.66	7.36	9.99	13.93
Cutting elements with loaders, $k_w = 0.3$						
v_p, m/min	4.79	6.74	9.39	4.41	5.99	8.36

6. Loading gate

The loading gate has been defined as a free space through which mined coal is discharged from a cutting element. The problems connected with setting up an appropriate loading gate were known for long time (wide arms), but became much more complicated when narrow arms and loaders were used. This resulted from the fact that the space where mined coal could be discharge was partially obscured by the cutter-loader (arms, loaders) and chain conveyor. Thus, the reduced travelling rate changes of rotation result from the lack of required loading gate to avoid mined coal degradation and cutting element jamming.

A diagrammatic drawing of cutter-loader arms at normal operating position is presented in Fig. 4. Such position enables the loading gate to be determined for the front and rear cutting elements. The operation with and without loaders and at different direction of rotation has been considered.

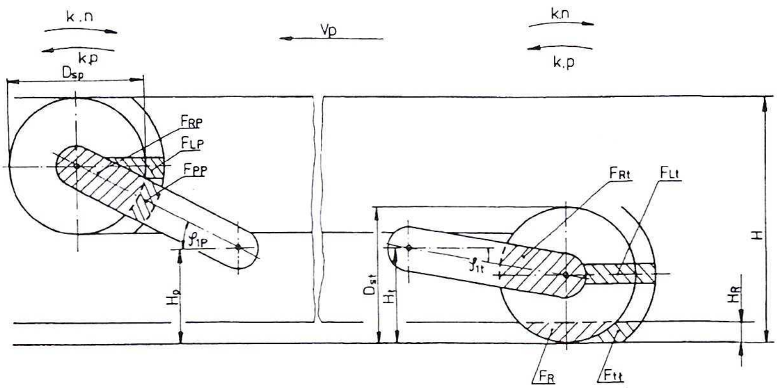

Fig. 4. A position of cutter-loader arms enabling the loading gate to be established for the front and rear cutting elements (diagrammatic drawing)

The expressions describing the loading gate take the following form:

— front cutting element with no loader,

$$F_p = 0.25\,\pi\,k_{wp}\left(D_{sp}^2 - d_p^2\right) - F_{RP} \tag{32}$$

— rear cutting element with no loader,

$$F_t = 0.25\,\pi\,k_{wt}\left(D_{st}^2 - d_t^2\right) - \left(F_{Rt} + F_R\right) \tag{33}$$

— front cutting element with a loader,

$$F_p = 0.25\,\pi\,k_{wp}\left(D_{sp}^2 - d_p^2\right) - \left(F_{Rp} + F_{pp} + F_{Lp}\right) \tag{34}$$

— rear cutting element with a loader,

$$F_t = 0.25\,\pi\, k_{wt}(D_{st}^2 - d_t^2) - (F_{Rt} + F_{Lt} + F_R + F_{tt}), \tag{35}$$

where:

k_{wp}, k_{wt} — coefficients to include the fact that only part of cutting element cross-section takes part in loading process depending on direction of rotation.

The value of k_{wp} or k_{wt} can be adopted depending on direction of rotation and use of shield loaders. It follows from Fig. 4 that the maximum value of k_{wt} is 0.75 for cutting elements with loaders, regardless of direction of rotation and work area. In the case that loaders are missing, $k_{wp} = k_{wt} = 0.5$ working to the rise and $k_{wp} = k_{wt} = 0.75$ for working to the deep.

The mined coal discharge speed on one hand and cutting element mining efficiency on the other hand should ensure that the following relation is fulfilled:

$$V_{u(p,\,t)} \leqslant F_{(p,\,t)} v_{w(p,\,t)} i_{(p,\,t)}, \tag{36}$$

where:

p, t — index of cutting element under consideration (p — front, t — rear),

v_w — longitudinal speed of mined coal.

To express v_w as a function of step S and number of blades i and rotational speed n, the relation (36) can be written as:

$$V_{u(p,\,t)} \leqslant F_{(p,\,t)}\, S_{(p,\,t)}\, n_{(p,\,t)}\, i_{(p,\,t)} \tag{37}$$

and then after omitting the indices p and t, the surface area F can be expressed as:

$$F \geqslant \frac{V_u}{S n i}. \tag{38}$$

By using the data presented above the surface area F was calculated for a cutting element of diameter $D_s = 1.4$ m, $n = 39$ rpm and $D_s = 2.0$ m, $n = 27$ rpm and $v_w = 8$ m/min:

— front cutting element,

$D_s = 1.4$ m $F \geqslant 21.07$ cm^2
$D_s = 2.0$ m $F \geqslant 29.12$ cm^2

— rear cutting element,

$D_s = 1.4$ m $F \geqslant 9.03$ cm^2
$D_s = 2.0$ m $F \geqslant 14.56$ cm^2

It seems that the resulting values of F should be larger due to coal granulation. Thus, coal must be transported through this surface area without additional crushing. This effect shall be verified through experimental investigations.

7. Cutting element load resulting from loading process

The loading process with worm cutting element consists of periodic three dimensional discharge of collected coal between the blades (see Fig. 5). The cyclical nature of the process depends on the number of blades and spiral lead. Obviously, the condition $P_{yl} = P_{zl} = 0$ describes the most favourable arrangement of forces accompanying the loading process. In that case the entire motor output is transmitted through force P_{xl} onto mined coal that is moved along the axis of cutting element towards discharging point. However, due to cutting element design all three components of loading resistance are always present.

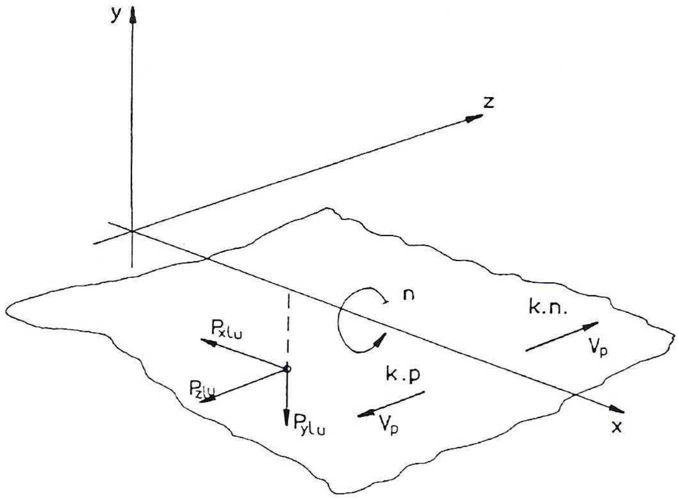

Fig. 5. Resolution of the loading resistance force

Assuming that the loading resistance force P_l results from the weight of mined coal G_w collected inside a cutting element and the coefficient of loading resistance μ_s that depends upon friction between coal and coal, coal and floor, coal and worm blades and, in addition, between coal and loader, if used, the equation describing this force takes the form:

$$P_l = G_w \mu_s \cos\beta \cos\alpha = V_u \gamma\, \mu_s \cos\beta \cos\alpha, \tag{39}$$

where:

α — longwall longitudinal inclination angle,

β — longwall traverse inclination angle.

Thus, the components P_l depend on μ_s and G_w as well as direction of rotation.

$$P_{xl} = G_w(K2 \sin\beta \cos\alpha - \mu_s \cos\beta \cos\alpha, \tag{40}$$

$$P_{yl} = -Kier.\mu_s G_w \cos\beta \cos\alpha, \tag{41}$$

$$P_{zl} = G_w \cos\beta \,(\mathrm{K}1 \sin\alpha - \mu_s Kier. \cos\alpha, \tag{42}$$

where:

Kier. — direction of rotation
for working to the rise $Kier. = -1$
for working to the deep $Kier. = 1$

$K1$ — longitudinal mining
for mining to the rise $K1 = -1$
for mining to the deep $K1 = 1$

$K2$ — traverse mining
for mining to the rise $K2 = 1$
for mining to the deep $K2 = -1$.

The moment of loading resistance force depends on diameter D and force P_{zl}:

$$M_l = 0.5\, D\, P_{zl}. \tag{43}$$

As known, the amount of mined coal inside the cutting element is increasing over the length of worm blades and reach its maximum at blade edges. Therefore, a linear increase of mined coal weight along the worm blades can be safely adopted. The linear distributions of mined coal weight inside a cutting element with normal blades and overlapped blades are shown in Fig. 6. Such nature of mined coal collecting combines the cyclical process with the number of blades and spiral lead. An example of computations for the moment of loading resistance is presented in table 3.

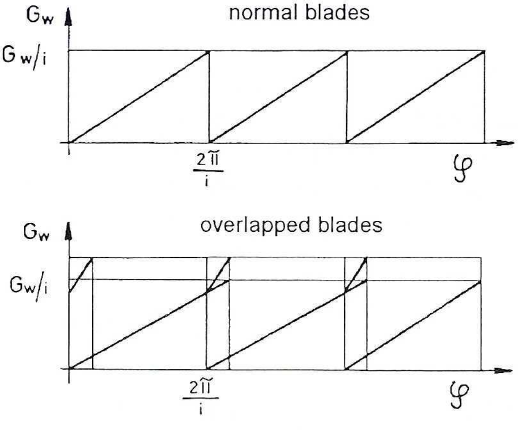

Fig. 6. Mined coal collecting between the worm blades

The results given in table 3 have been obtained by taking into account all remarks and conclusions presented in previous chapters.

TABLE 3

The moments of loading resitance force depending on D_s, Z, n and v_p for $\mu_s = 0.8$, $\gamma = 13.2$ kN/m^3, $\alpha = \beta = 0$

Z, m	n, rpm	v_p, m/min	D_s, m	
			1.4	1.8
			M_l, kNm	
0.63	30	3	0.67	1.05
		5	1.11	1.81
	60	3	0.30	0.48
		5	0.59	0.86
1.00	30	3	1.11	1.81
		5	1.85	2.85
	60	3	0.44	0.86
		5	0.89	1.43

8. Summary

The verification of mining process presented above and related mainly to loading with a worm cutting element has led to an analytical approach to the design and kinematic parameters deciding on mined coal loading. The obtained analytical relationships enable the loading process to be modeled in order to achieve the required filling factor for the given design and kinematic parameters as well as downstream longwall machinery and equipment. The requirements related to the assumed day output, loading process, mined coal collecting capacity and chock travelling speed allows to size the longwall complex machines and equipment and to estimate the investment expenditures (K r a u z e, F l a g a, 1999).

REFERENCES

B ę b e n A., 1999. Nowe ujęcie określenia wydajności urabiania dla bębnowego kombajnu ścianowego. Mechanizacja i Automatyzacja Górnictwa, nr 3, Katowice.

K r a u z e K., 1994. Objętość wewnętrzna organu a jego wydajność urabiania. Mechanizacja i Automatyzacja Górnictwa, nr 2, Katowice.

K r a u z e K., 1995. Teoria procesu roboczego frezujących organów ślimakowych w skałach średnio zwięzłych. Rozprawy i Monografie 36, Wydawnictwo AGH, Kraków.

K r a u z e K., 1999. Dobór ścianowych kombajnów węglowych w aspekcie uzyskania założonej wydajności. Mechanizacja i Automatyzacja Górnictwa, nr 6, Katowice.

K r a u z e K., F l a g a S., 1999. Dobór maszyn kompleksu ścianowego pod względem realizacji założonego wydobycia i minimalizacji nakładów na uzbrojenie ściany. Międzynarodowa Konferencja Naukowo-Techniczna. CMG Komag, KBN, PARGWK, Szczyrk.

REVIEW BY: PROF. DR HAB. INŻ. ADAM KLICH, KRAKÓW

Received: 09 May 2000.