Contents

Evaluation of mining companies by investors

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The present time of economic crisis overcoming is influencing also mining industry when companies are fighting with high indebtedness. High level of debt means a number of mining companies started to have big problems, a decrease of production and a single recession. One of the possible ways to solve the situation is private equity, which has a single impact especially to the mining industry. Contribution provides an evaluation in the area of PE investors in the mining industry with the aim to compare the performance of the mining industry with PE participation, attractiveness and total impact to the mining companies. The goal was to find out if PE means benefit for the mining industry and if so, what are benefits, reflecting in the performance of the company. The determined goal was achieved by a so-called micro index that could according to analytical results and evaluation of mining companies present certain informative structure about individual parts of the companies and to which measure these parts are qualitatively managed from the view of PE investor. Results of the analysis show that PE in the mining industry records better results than companies without PE participation. Through analysis of the mining industry, we found out the mining industry achieved better performance than the whole European industry. PE acting in the industry had a positive impact on the mining companies. The situation can be solved through simple micro index and summary of individual risks, which mining company should avoid with the aim to increase attractiveness for PE investors and at the same time to increase the stability of the company.

Keywords: *mining industry, investments, private equity, business risk, investor, gross domestic product*

Introduction

The economic crisis, rising in 2007, caused a number of industrial sectors, mainly mining industry, had to borrow financial sources. High level of debt means a number of mining companies started to have big problems, a decrease of production and a single recession. Results of research by Axelson et al. (2009) speak about debt financing that is still greatly influenced by debt conditions that have bigger power to influence a single structure of the industry and companies. The present economic decrease in the frame of national economies provides a broad spectrum of evidence, according to which we can search and validate various alternative economic scenarios. Dvořáček et al. (2012) calculate financial bankruptcy models in order to determine the ratio of non-bankrupt and bankrupt industrial firms. Also, the solution could be private equity (further PE), since it reacts to the economic decrease more intensively than the single business model without the involvement of leverage (Hege et al., 2018). It means that during economic decrease companies, supported by PE can achieve better results than their competition at the market, mainly due to the stable core of the company and the broader structure of financing. PE could present better and more prompt reaction to the changes in the frame of industries.

Private equity (further PE) is a type of investment concept, presenting the selling process of the firms (Fidrmuc et al., 2012). It presents alternative to various other types of breakup transactions and buyouts, as divestitures, spin-offs, equity carve-outs, tracking stock and strategic buyouts (Betton et al., 2007, Boone and Mulherin, 2007; Straka et al, 2018, Khouri et al 2017). PE has the economic role of mergers by performing a comparative study of mergers and internal corporate investment at the industry and firm levels. Andrade and Stafford (2004) found strong evidence that merger activity clusters through time by industry, whereas internal investment does not. Research by Jensen (1989) proved that PE positively improves the operating performance of the industrial companies. John et al. (1992) provided successful empirical evidence that fear of sale or takeover by other company increased motivation and performance of the given company. The result of research by Kaplan (1989) states the fact that PE companies help to increase the profit of target companies; it can improve cash flow and also other financial indexes. Similar results have also been achieved by other studies, for example, the study by Guo et al. (2009), and study by Muscarell and Vetsuypens (1990). Results of mentioned studies showed similar results in the frame of various PE segments, which emphasises still the more total opinion of the positive effect of PE investments. Another study that had been orientated to the PE is the study by Bloom et al. (2009) that studied managerial contribution to total PE position and also PE benefits from the management point

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of view. Results showed that PE management is more successful than management in other companies in the frame of industries. Also, some cases exist that emphasise the opposite effect of PE investments, while the company is still achieving significant profits (Bargeron et al., 2008). Such a phenomenon was presented in a study by Bernstein et al. (2010) and Rasmussen (2008). A number of authors studied the influence of PE in various countries (e.g. Fang et al., 2018, Saini and Singhania, 2018). From mentioned researches there is obvious, PE using is influenced by different economic conditions.

Methodology

The goal of the contribution is to make an evaluation of private equity with impact to the mining industry and to evaluate by way of comparison and analytical methods performance of mining industry with PE participation, the attractiveness of total PE impact of an investor in the frame of mining companies.

Solving is possible by a so-called micro index that could according to analytical results and evaluation of mining companies present certain informative structure about individual parts of the companies and to which measure these parts are qualitatively managed from the view of PE investor. Micro index of private equity and risk capital are presented by the research of Groh et al. (2008). Main factors, influencing the index are:

- economic activity,
- capital market,
- taxes,
- protection of investor and corporate management,
- working and social environment and
- business possibilities (Čulková et al., 2015).

Economic activity belongs among the most important factors and partial components of the total index. It is obvious that in a number of dynamic environment segments with private equity operation and risk capital, the economy will grow rapidly, providing more possibilities (Gompers, Lerner, 1998).

Total goal is to find when PE brings benefits for the mining industry and what benefits are reflecting in performance and management of mining companies. The goal is achieved by EVCA data, data presented for individual industries with the acting of PE investors. The more exact interpretation of results at various levels of activities is divided to four quarters, but during analysis deep analysis from the view of quarters had been not done due to the necessary information about a number of employees that were not provided from publicly available sources.

During the research, there was necessary to adapt the total GDP per inhabitant when the method of data adaptation had been used (Groh et al., 2008), mainly adaptation of:

- total GDP annual change,
- a measure of unemployment,
- emission of new shares,
- the capitalisation of share market of the country,
- value of the paid dividend of the share market,
- the market of acquisitions and mergers,
- a discount rate of the central bank,
- private capital and other financial institutions,
- the activity of private equity and risk capital,
- state expenses to education,
- a number of university students, etc.

Partial role during index construction also has education, working laws, index of bribes, corruption and crimes. From the view of business possibilities following indexes are favourable - general index of innovation, expenses on development and research, level of company restructuring, activity on share market and obstacles in the frame of new companies rising. Evaluation of total conditions of the economy is very important, since the level of conditions from the view of GDP, capital accumulation for investments, start-up companies and their financing has a considerable noticeable value of attractiveness (Romain 2004).

There are three methods for calculation of individual indicators in the frame of the total index. The first method is simple, consisting of a process to add the same weight to every partial indicator. The second method is orientated to the indicator analysis, and the third method is orientated to the various multiplication analyses of the indicators. Nicolleti et al. (2000) mention the division of used indicators to three levels of the index. Such an approach will enable better and easier interpretation of achieved results and provide easy identified evidence regarding the weaknesses and strengths of individual countries. According to this single division process of indexes, aggregation will be simpler due to the massive number of data. For example in the case of the indicator, mentioning the human and social environment, the indicator has four partial indexes, but according to the used methods, the indicator will be presented by a single value in the main index.

Certain deflation was made at chosen data with the aim of heterogeneity of data providing due to the removing of the influence of the main index by improper direction. Due to the maintaining of differences between countries and observing proportional trend view to the development of individual countries we used deflator on GDP or number of inhabitants in the country. The reason for the mentioned was removing of vast difference, based on the size of individual countries.

Results of the presented analysis are illustrated by tables and by the application in the mining industry. Graph lines show the percentage change in performance trend observed in productions, and the dotted lines within the graph indicate on a four-year trend in these productions (World-Mining-Data, 2017 and Invest Europe, 2017). Given study is dealing in the frame of a chosen mining company that is in the stage of bankruptcy, or recession, and the goal is to show how PE investor could revive such company.

Data normalisation

The primary goal of data normalisation is to normalise data for further analysis and index construction, which can be done by various methods. Methods used by Freudenberg (2003), present standardisation and change of scales level. So-called z-score standardisation means conversion of normal distribution data, where mean value is given by zero or standard deviation 1.

$$z = \frac{x-\mu}{\sigma} \tag{1}$$

where:

x - value of single factor,

μ - mean value of evaluated factors,

σ - standard deviation of evaluated factors.

Because some of the indicators were in values that had very different scales, we used methods of scales change and format of values due to the normalisation by a linear transformation. Further analysis showed if such methods can be used for all indicators since time difference included a massive decrease in share market in Europe, as well as in the whole world. Method of scales change and harmonisation is rather not effective in case of transactions, which are presented by extreme values. In our case, it is very frequent.

$$y = \frac{x-\min(x)}{\max(x)-\min(x)} \tag{2}$$

where:

y – linear data normalization,

x – value of single factor,

min x) – minimal value of given factor,

max x) – maximal value of given factor.

The z-score method is broadly used method during the analysis, orientated to the accumulation of data or indexes. We choose to use the method due to the chosen data, presenting useful data files, including significant „gap" among individual years of single crisis and since the z-score method is able to eliminate such shortages.

Results

Different accesses during evaluation of mining companies result from the complicated character of evaluation in commodity companies, which value is greatly influenced by high cyclicality of the mining industry. In the mining industry, there are two main cycles: the price of commodities and the economic cycle (Csikosova et al., 2016). A number of projects in the frame of the company and single companies operate during various cycles of their existence and in the frame of evaluation and management of the company they must take into consideration various risks. Most important risks are: financial risk, the risk to obtain permission for mining and its lengths, risk connected with geological tasks, risks regarding metallurgy, economic risks, and risks of individual countries, political risks, geographical and social risks (Blistan et al. 2012; Blistan et al. 2015; Cehlár et al 2011; Cehlár et al 2016).

Among the most important evaluation processes of mining companies (investments) are such processes that deal with the evaluation of incomes and cash-flow, market and costs. Given evaluation is distinguished according to the level of company development and according to what type of company is doing deposits finding, if the company is dealing with preparation of deposit for single mining or if the company is making mining activity (Taušová et al., 2017).

A short analysis of the situation in the frame of Europe and mining industry presents an economic situation and PE acting in a given region during 2007-2011. Table 1 illustrates the economic situation from the view of production in the mining industry. There is a decrease in gross domestic product GDP in Europe and the annual production of the mining industry.

Tab. 1. Percentage change of production in the mining industry.

World mining production
2017 16,9 billion metric tons
2000 11,3 billion metric tons
1985 9,7 billion metric tons

Source: World-Mining-Data, Vienna 2017

Production of the mining industry did not correlate directly with production in the frame of European industries. The single analysis shows that mining industry showed with previous year GDP decrease in the frame of the whole European economy (Fig. 1).

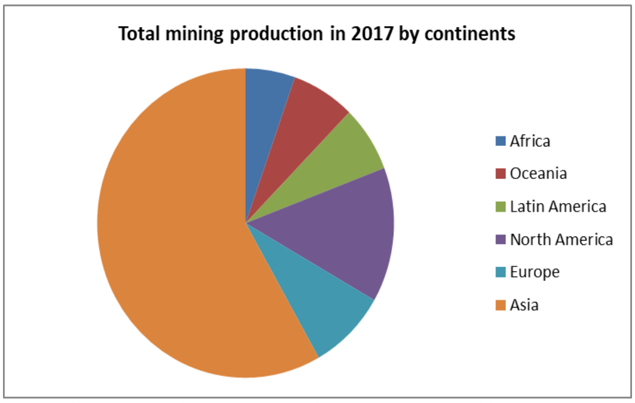

Fig. 1. Total mining production in 2017 by continents (World-Mining-Data, 2017).

Visual illustration of PE acting of investors in the frame of Europe and concretely in the mining industry in comparing with other sectors is illustrated in Figure 2. We can see that PE activity in companies is expressed by the volume of investments. PE activity of investments in the mining industry in Europe recorded a single decrease, or negative change of investments volume, which was similar in all industrial sectors. As mentioned, PE reacts more flexible and promptly to the changes in world markets, which means that also a change of investments volume in the frame of European industries is more rapid (Invest Europe, 2017).

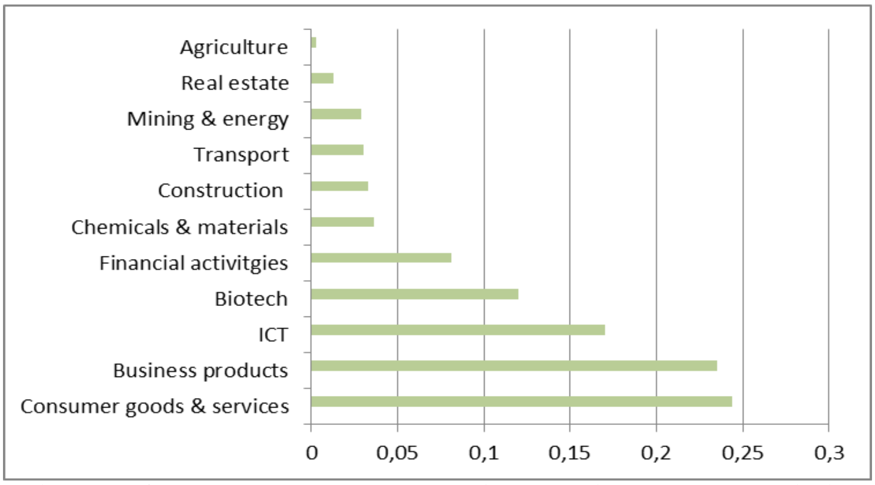

Fig. 2. The activity of Private equity investments in 2017 in the mining industry in comparing with other sectors (Invest Europe, 2017).

Absolute growth of the industry with PE participation of investor and industry without PE investor is illustrated in Table 2. As presented in the mentioned table, the average growth of total production in industries with PE investor participation is 36,6 %, and industries without PE mean 4,25 %. Results speak about very strong average growth of industries with PE participation.

Tab. 2. Comparing of industry growth.

Industrial sector	**Average growth without PE**	**Average growth with PE**
Agriculture	5.80 %	-0.94 %
Business and industrial products	8.29 %	61.22 %
Business and industrial products	2.11 %	44.79 %
Communications	-46.84 %	34.24 %
IT and electronics	17.31 %	17.91 %
Constructions	13.13 %	103.94 %
Retail	8.36 %	48.87 %
Consumers services	7.9 7%	107.34 %
Energetics, mining and living environment	**3.45 %**	**23.90 %**
Financial services	7.21 %	-7.30 %
High-tech	15.27 %	11.09 %
Chemical industry	11.31 %	1.67 %
Research and development	3.69 %	40.26 %
Real estate market	5.00 %	29.42 %
Transport	1.72 %	28.01 %
Average	4.25%	36.30 %

Source: own calculation according to data from EVCA (2017)

Our view and analysis are orientated to the evaluation of mining industry rate at the total GDP in Europe and analysis what is the rate of PE at investments and production of the given industry. Consequently, we analysed what is growth potential in the mining industry with PE investor participation comparing without PE investor. The rate of the mining industry in GDP in Europe has prevailingly stable or slightly growing trend during the analysed 9 years period recorded except 2009 when production of the mining industry was higher than the previous year. The single decrease in 2009 was more or less expected since industrial production across the whole European industrial market slowed down. Given the level of industrial production at GDP in Europe is moving averagely around 20 %. In the following Figure 4, we can see what the rate of the industry is during the analysed period at total GDP in the European countries, where we can see the level of PE investments, directly orientated to the mining industry and what is the level of the investments achieved with a time of global crisis outbreak.

PE investment deals and amount deployed in 2017 was an encouraging year for investments by mining private equity firms with $2.3bn invested across 60 deals. Both the number of deals and the amount invested were up on 2016 levels but down from 2015 peak activity of $3.2bn across 119 deals. 2015 peak activity was driven by increased stakes, often to protect investment in distress situations. Over the last four years, investments by way of strategic stakes have nearly doubled, and this reflects the recovery in the sector generally as well as the second wave of fundraising by the mining private equity funds themselves (Tab. 3).

Tab. 3. Private equity investments in the mining industry.

	2015	2016	2017
PE volume	3,2 bn$	1,75 bn$	2,3 bn$
number of deals	119	30	60

(own processing according to Keepin, 2018)

Mentioned Table 3 illustrates the level of PE investments to the mining industry in the frame of Europe, which more or less follows the changes in the amount of PE investments in the European market. Given development reflects that the mining industry is attractive from the view of PE investors and immediately after the first year of crisis there was the growth of investments to the mining industry.

Development of investments to the mining industry and mainly its rate at total GDP could be influence also to a certain level by a single rate of individual industries at total GDP. Interesting analysis proved the trend of the industry at total GDP and trend of PE investments as the rate of GDP. Following Figure 3 illustrates trends of the percentage rate of mining on GDP, which speaks about the level of mining industry development against GDP.

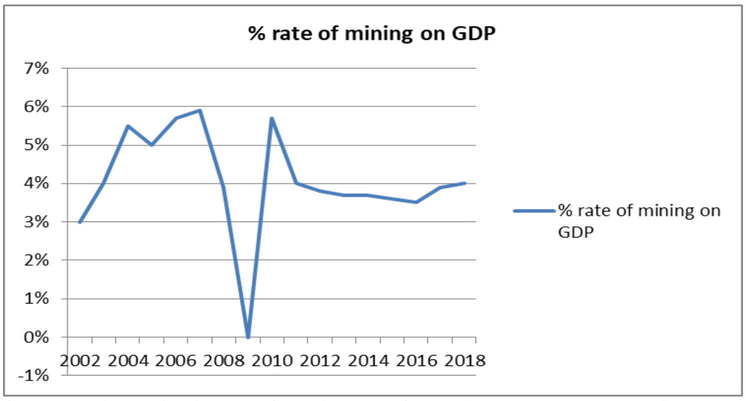

Fig. 3. The trend of mining industry development from the view of GDP (own processing according to www.pwc.com/mining.

As we can see, the activity of the mining industry is less expressive than the activity of PE investments in the mining industry. The single trend of PE investments to European industry recorded a growing trend, and the trend was in correlation with the mentioned trend in the Figure. PE investments have a growing trend in most years not only in comparing with the rate of industries at GDP but also in comparing with a single rate of mining industry at total GDP in Europe.

According to given findings, we can state that the mining industry has better performance during analysed 11 years in comparing with the whole development of the industry. Further, we can state according to the analysis that the mining industry with the influence of PE investors has higher performance than a single mining industry. The trend of growth of the mining industry is more expressive from the view of all evaluated factors. The last comparison of PE investors' contribution from the view of the mining industry is comparing of production and performance of the mining industry from the view of PE investors acting and without PE investors. Given comparing is illustrated in Table 4.

Tab. 4. The activity of the mining industry in Europe.

Year / Analysis	The rate of the mining industry on GDP	PE investment % GDP	PE investment in the mining industry (Mld. Eur)	PE investment to mining industry % GDP
2008	3,9 %	0,38 %	840 403	0,0081 %
2009	-,01 %	0,39 %	911 155	0,0084 %

2010	5,7 %	0,50 %	1 331 658	0,0118 %
2011	4,0 %	0,84 %	2 008 666	0,0168 %
2012	3,8 %	0,75 %	1 980 663	0,0157 %
2013	3,7 %	0,65 %	1 424 994	0,0112 %
2014	3,7 %	0,60 %	638 823	0,0053 %
2015	3,6 %	0,29 %	1 165 885	0,0093 %
2016	3,5 %	0,39 %	1 243 258	0,0095 %
2017	3,9 %	0,44 %	1 250 360	0,0098 %

Source: own calculation according to database EVCA (2017) and www.pwc.com/mining

Development, production and value added of the mining industry are significantly different in case of PE investors acting and in the case when PE investor is not part of single companies in the frame of the industry. Companies without PE investor during 11 analysed years achieved very stable, but stagnating development. In one year when companies, supported by PE investor achieved lower performance than in 2002, companies without PE achieved slightly higher performance in 2002. From the view of performance and growth of the company, there is obvious that company with PE investor would achieve higher performance and growth, which in the long term also enables the growth of the company and creation of new working posts. Since the growth of the company during more than 10 years is unchanging, the company could not provide sufficient possibilities for higher employment or increasing value added.

According to mentioned mining industry has investment potential from the view of PE investors, which is reflected in 10 years growth in companies, supported by PE investors, which was also reflected in the whole European mining industry. In comparing with companies without PE investor, the companies are stable but did not achieve any significant growth from the European view. Given analysis proved the mining industry has significant potential for attracting of PE investors.

Discussions and Conclusions

Among the most important approaches for evaluation of mining companies (investments) belong such approaches, dealing with evaluation of incomes and cash flow, market and costs. For example, well-known method Balance scorecard is described by Antošová et al. (2014), who states that the BSC is not meeting the management expectations to the required level it was implemented into practice, yielding limited contribution, or reduced functionality. Given evaluation should be distinguished according to the level of social development, if the company is in the frame of deposit finding, or if the company is already dealing with deposit preparation for mining, yet the company, making mining activity.

Mining companies are acting in the whole world, and single knowledge, obtained from individual continents can be used in case of any company. It is obvious that any mining company would be in a different stage and different history. Strong prices of commodities and trust in long term base of mining sector have to revise investments to mining and to develop new projects or to start up the production. Such increased investments are connected with risk company could slow down the trend and increase costs. Important risks, connected with lack of skills influence the production, late completion of projects and growing costs per working power. Single risks in mining sector developed from two main factors: low level of commodity prices, which led to higher risk in mining companies from short time view. The second factor is changes in capacities in the sense of skills and infrastructure that had a direct impact on short term liabilities of the company from the view of capital projections. Among most important risks belong lack of qualitative working power, access to necessary infrastructure, cost inflation, capital projection, currency and price volatility, management of accesses and capital, corruption and stolen property and political risk.

Due to the creation of the value of the mining company, there are important in the following areas:

1. Optimising of business strategy in the frame of growing and operative goals with a primary orientation to development of strategic possibilities, capital effectiveness, radical improving business, the creation of growing power, using of advisory services in the frame of Technologies, following of possibilities in the area of financing.

2. Cost savings, improving services through outsourcing with an orientation to sharing of individual mining services, using of outsourcing of employees.

3. Management of sustainability with an orientation to the following aspects: sustainable mining, security analysis, investments to the corporate social environment.

4. Effective using of available technologies, with a primary orientation to the following aspects: reconfigured software solutions, the integrity of usable applications, application of managerial services, IT, risk following and consequent value creation.

5. Optimising of tax and legal position, with an orientation to the following: using of tax advisory services, tax effective net, transfer evaluation, using of advisory, agreements optimising, optimising of managerial evidence.

6. Proper addressing of talent challenges, with a primary orientation to the development of talents and evaluation possibilities, the design of individual services, orientated to the employee's evaluation, various e-learning possibilities.

The basic model for micro index derived from attractiveness and extended by necessary factors that are necessary to regard in case of mining companies. The result presents a simple micro index and summary of individual risks, which mining company should avoid with the aim to increase attractiveness for PE investors and at the same time to increase the stability of the company. The process of the model is illustrated in Figure 4.

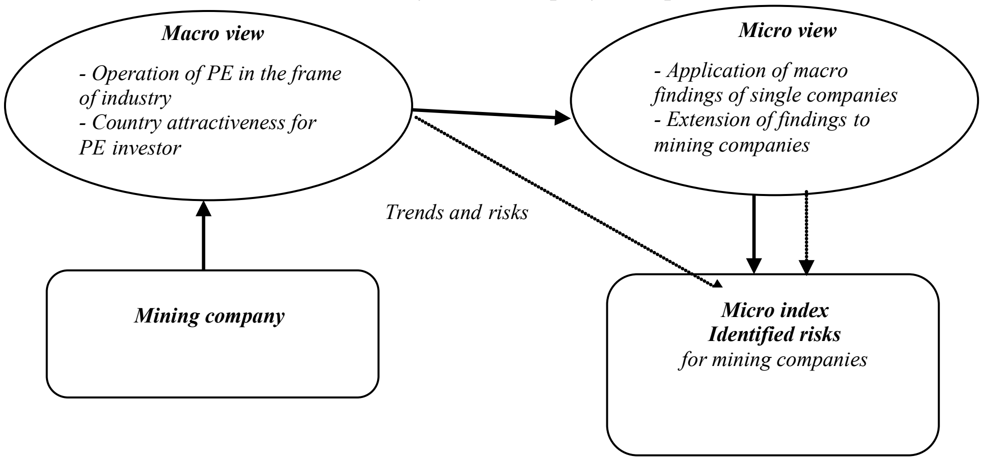

Fig. 4. The process of model creation.
Source: own processing

Increasing of the company value can be done through determination of mentioned areas. It was applied in a mining company that is bankruptcy proceedings and presenting materials for PE investors. After some discussions with PE investors, the company found out the situation is so negative that single decision to participate at PE investments must be compensated with a considerable measure of return of investment. As modern portfolio theory presents, growing risk demands also increasing measure of expected return. Such a problem can be overcome by transformation from a micro view to the mining company in the level of PE to a macro view of the mining company in the level of European PE investors. According to the obtained information, we made extrapolation of proper indexes and factors for individual companies in the mining industry. The obtained information is applicable to single mining companies. Moreover, the competitiveness of the mining company is determined by long term developments of national policies in the energy and mining industry (Madzík et al., 2016). Companies that have innovation activity have a chance to pass through a period of slow growth without substantial problems (Pawliczek et al., 2015).

The process of PE establishment to the mining company should be as follows: (Gadiesh and MacArthur, 2008)

- To define total potential: with the aim to have a higher value of equity. A strategic audit can help to determine a proper number of increasing cash flow to equity.
- To construct the plan: which will be the map for filling the potential – who, what, when, how.
- To speed up performance: running of an organisation must be adapted to the plan and key initiatives. It means the creation of a precise program – consisting of concrete tools, processes and merits for achievements of the highest possible performance.
- Engaging of talents: it means to create proper offers with the aim to obtain and motivate most talented people that they behave in the same way as investors and owners.

- Maximal use of the capital: it means to understand principles of the leveraged buyout – aggressive management of the working capital, discipline in the frame of capital expenses and emphasise to the balance.
- Orientation to the results: the goal is to impress PE discipline that would become part of the company culture and to create a repeatable model for results achievement.

Summary

The present time of economic crisis overcoming is influencing also mining industry when companies are fighting with high indebtedness due to the lack of own and cheap financial means, necessary of financing of mining activity. Mainly private equity presents one of the possible ways how to solve indebtedness in mining companies and to have easy access to low-cost financial sources. Such new sources of debt financing and cheap money support massive growth mainly in industrial sectors of individual countries, and it has a single impact especially to the mining industry since private equity supports mining industry by its presence, which could influence single mining companies by practical contributions.

Contribution provides results of evaluation in the area of PE investors in the frame of the mining industry with the aim to compare the performance of the mining industry with PE participation, attractiveness and total impact to the mining companies. The goal was to find out if PE means benefit for the mining industry and if so, what are benefits, reflecting in the performance of the company. Results of the analysis show that PE in the mining industry records better results than companies without PE participation.

Through analysis of the mining industry, we found out the mining industry achieved better performance than the whole European industry. PE acting in the industry had a positive impact on the mining companies. According to mentioned, we can state that PE participation in the frame of the mining industry from European view achieved interesting position and trend of performance, recorded in the mining industry with PE investors was significantly better than without PE participation.

Total results prove PE provides higher value added to invested companies, increases employment and there is a difference among industries with high or low PE participation in industries. Private equity investor has better management and mainly due to the mentioned we did not find higher vulnerability of the companies with PE investors against aggregated economic shocks.

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Plastic Flow Modeling in Rock Fracture

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Borehole methods and methods of mining using combines and ploughs will be widely used in Karaganda (Kazakhstan) in lavas and short faces. Therefore, special requirements are applied to the modelling of rock destruction processes. The results should show the possibilities of improving the strength of the tool, as well as the processes that occur under pressure on the rock. The specifics of solids fracturing by a tool are described. These processes are accompanied by the formation of a plastic wedge and a thin, curved body between surfaces of a cleaved element and destructed rock. The surfaces of dislodged elements have alternating rough and smooth areas. Such zones are formed by low-frequency acoustic radiation, which can be observed during fracturing of rock cores by a punch and optically active, organic glass with a controlled speed of crack motion. The fracturing direction is predicted along paths of principal stresses and can have poorly predictable areas in zones of isoclines convergence leading to the intersection of surfaces. The element models that extend the capabilities of research, taking into account a plastic flow of rock and drilling tool for boreholes, were developed. In these studies, there can be taken into account various factors: the structure of the rocks in the face and the scheme of their collapse behind the face, the depth of the work. This is especially important for lavas and short faces. The use of 2-3 research methods (destruction of rock samples and other solid materials, studying stress-strain state (SSS) by optical and finite element modelling) is a prerequisite for obtaining accurate results.

Key words: *plastic wedge, focusing, thin body, crack path, finite element, optical modelling*

Introduction

As the analysis of mining technologies shows, the borehole methods for hydrocarbon development will be widely used in the development of oil, gas and coal fields. The depreciation of drilling bits sharply reduces the effectiveness and increases the length of drilling due to the need to extract the whole drilling assembly from the depths of up to 3000 m. So the special requirements are imposed on modelling of rock fracture by a tool (Pinka et al., 2007; Bujok et al., 2013; Ġofranko et al., 2014; Flegner et al., 2016; Wittenberger et al., 2015; Wittenberger et al., 2017). Rocks are plastically deformed under pressure of tools during drilling, but this process has not been explored. In the process of fracture of equivalent materials with tools of different forms, the best conditions for observing the plastic zones appeared for rolling tools and cylindrical punches.

For rocks near the faces (mudstones, siltstones, sandstones) and partly coal beds D6 and K12 of the Karaganda coal basin, these processes are similar. In new technologies for the development of coal seams, degassing wells are to be drilled from the earth surface, as well as from mine workings. The use of tunnelling combines with power capacity will be 2-3 times greater than now. Technologies of coal mining by short faces and angular conveyors will gain development. On thin layers of "Kazakhstanskaya", "Tentekskaya" mines there will continue introducing plough lavas. In this case, the rock pressure and the depth of the work will affect the processes of destruction of rocks and coal. Mining pressure also depends on the patterns of rock formation and the features of their collapse behind the lava. The layers of rocks hang over the face, collapse or smoothly drop on the collapsed pieces of rock and the length of layers depends on the pressure on the layer where there works a combine or a drill.

Modelling such schemes is possible on the basis of finite element technologies, and such models are used here (Ansys). Their general principles in the transition to details are presented in (Grzejda, 2014; Dodagoudar et al., 2015; Witek et al., 2016; Ğazuka et al., 2016) where for the accuracy of the results, the methods of constructing the grid, especially for contact problems are important (Grzejda, 2014). In (Hosseini, 2014) the use of these methods to the technical schemes of excavation is presented. Typically, there is carried out the interaction of CAD/CAM/CAE systems. In Ansys, we often solve static problems, and in Adams, we first determine the possible dynamic component that is then used in Ansys. However, even licensed software tools

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have errors, and especially in accurate studies, so additional techniques are needed. They are often used in models where there are used the fragments of the general technological scheme that makes it possible to clarify the processes that are not visible on the general scheme. In them, there are also can be used finite element modelling. For example, considering the interaction of tools and rocks with a highly deformed zone where special requirements are imposed on the construction of the grid. There can be effectively and quickly carried out the study based on optical modelling (organic glass of SD type), as well as in the destruction of rock samples, organic glass and other solid materials. The use of SD makes it possible to visually observe the process of plastic deformation and the development of cracks. In this work, we are more interested in the plastic flow that is clearly visible for the main rocks of the Karaganda coal seams.

Material and Methods

The use of methods (Yang et al., 2017) which discuss the energy principles of loading the face of lava layers of the roof is more complicated. Our schemes are different and require forming for each individual scheme of new conditions and new solutions, and this leads to large formulas. A universal solution of the plane problem given in also provides for finite element modelling. However, we build models in 3D and take into account various schemes of rock collapse down to the earth surface. This makes it possible to determine the nature of rock loading at the tool, affects the force and the direction of failure. This data can then be used to model individual fragments.

At optical model operation geometrical and power scale was kept (Eq.1; Eq.2):

$$\mu_g = L_n/L_m \tag{1}$$

$$\mu_f = F_n/F_m \tag{2}$$

where: L_n, L_m ï the sizes in nature and model, F_n, F_m ï forces in nature and model.

Scale for the relation (ɽ ʤ/ɽ ʥ) can be chosen randomly (ɽ ʤʒ ɽ ʥ - elastic modulus of model and nature). Scale of stress (Eq.3) (Borissov, 2005):

$$\mu_{\mathring{u}} = \mu_f \,.\, \mu_g^{\,2} \tag{3}$$

The traditional formula for modelling stress multiplying the geometric by the square of the power scale (Trumbachev et al., 1962).

The plastic layer was modelled and as a viscous liquid. Material characteristics are in Table 1.

Tab. 1. Characteristics of materials.

	Materials	**Elastic modulus, ɽ [ʃ pʘ]**			**Poissonôs ratio**
Optical model operation	**Organic glass - block polystyrene**	ɽ = 2,700*10^4	Density, p = 1050, kg.m^{-3}	Breaking point: ů = 42 ʃ Pʘ Softening point: ʊfŋ= 100°	0, 35
Rock failure	**Aleurolite**	9,5*10^4			0,43
	Soapstone	6,5*10^4			0,47
	Sandstone	2,2*10^4			0,24
Finite element method	**Tool**	2*10^5			0,25
	Rock	0,18*10^4			0,35
	Plastic layer (viscous liquid)	1,6*10^3			0,45

In numerical modelling, in order to predict and to transfer the data to new schemes, it is necessary to determine the modulus of deformation (elasticity) and other characteristics of materials. However, the use of cores does not always give accurate results since after extraction from wells they are not in real conditions (Koneļný et al., 2015; Chalupa et al., 2017). Therefore, methods based on the comparison of static and dynamic elastic parameters can be used. In the study, there is used elastic velocity propagation of longitudinal and transverse waves performed in various conditions including reservoirs (Fei et al., 2016). There is also used the technology of analysis of T-matrices, and as a result, the system approach of the studies will be provided, the effects of separate factors having a sensitive impact on the process are considered. Knowing the modules makes it possible to achieve a sufficient correspondence between the results of the calculations and the data obtained from the measurements in the mines.

The above references permit to increase the accuracy of modelling taking into account porosity, damages and cracks in the rocks that leads to reducing the modules that were not previously taken into account. For example, when calculating stresses in a strongly deformed zone, the elastic moduli (Young moduli) and the Poisson coefficients vary with the tool, and their relationships are refined in the indicated sources. Moreover, the reliability of the model was tested on limestone samples from a core of a borehole (Chalupa, 2017).

When carrying out the studies, there were considered the elastic modulus (deformation) and Poisson coefficients for the conditions of the Karaganda mines. In the transition to theoretical prediction taking into account porosity and cracks in the material, the values of the modules, respectively, with the above references are reduced (Beissembayev, 2010). This is also permissible when calculating the rock pressure on the bed at the face and with the tool of the executive organs of combines, ploughs and drilling machines in the zone of plastic deformation.

The solution by the finite elements method (FEM) is made by compiling the equations of equilibrium of the model nodes. They are based on the matrix rigidity equations for the elements (Eq.4):

$$[K]_{\mathbf{j}}\ \{U\}_{\mathbf{j}} = \{F\}_e + \{P\}_e^q + \{P\}_e^g \tag{4}$$

Where: $[K]_e$ is the matrix of the model rigidity; $\{U\}_e$ is the vector of the nodal displacements; $\{F\}_e$ is the vector of the nodal forces; $\{P\}e^q + \{P\}_e^g$ is the vector of mass and surface forces.

Then there is compiled the general system of equations for the entire model (Eq.5):

$$[K]\ \{U\} = \{F\} + \{P\}^q + \{P\}^g \tag{5}$$

The vector of the stated external nodal forces. $F_i = \{F_i\}$ ï is the submatrix of the n_i force components applied at the node i.

$$\text{[illegible]} = \begin{Bmatrix} \text{[illegible]}1 \\ \vdots \\ \text{[illegible]} \\ \vdots \\ \text{[illegible]} \end{Bmatrix} = \begin{Bmatrix} \text{[illegible]}1 \\ \cdots \\ \text{[illegible]} \\ \cdots \\ \text{[illegible]} \end{Bmatrix}$$

Results and Discussion

Research in optically active materials

When a cutter or punch is moving in rock, a thickened zone appears in the central zone or in front of them. It is less thickened at the outer edges of the contact zone, as the material behind it is displaced into a less dense zone. The central zone is narrowed, and the material compressed by the punch forms a plastic wedge (a cone for cylindrical punch), the material of which is thickened. This zone is sometimes called as the core. This is proved both by modelling in optically active materials when the wedge is visible, and by experimental fracture of rocks and coals by a tool (Khesin et al., 1969; Vasilyev, 1976). However, the wedge formation during coal cutting was not further explored, although, in the (Čazuka et al., 2016) by-step calculation of nonlinearly decreasing angle of the wedge thinning point, the experimental equations actually representing the beginning of a process similar to self-focusing shown in (Askaryan, 1973) were obtained.

In particular, G. Askaryan considered cavitation in rotating blades during operation of pumping units, which significantly damaged them. Gas in bubbles collapses during cavitations, and a cumulative jet with the pressure of more than 1000 MPa at the temperature up to 10000ęC is formed. The process is accompanied by thickening of particles in a narrowing jet when energy breaking the particles into smaller ones is reached. The resistance of the medium is characterised by jet slowdown and the formation of a typical wedge. The especially stressed zone will appear in the wedge edge where the maximum mechanical stress occurs. Along the axis, the wedge in the area of its sharp edge is stretched out and thickened, and the apex angle is close to zero, and, respectively, stresses sharply rise theoretically approaching to singularity what is known even from the classical task of the wedge elasticity theory. The process is non-linear and called as the wedge thinning. These processes are described as wave processes, but they are associated with the interaction of particles with the medium. So, for the acoustic self-focusing during cavitation, such particles are gas molecules; however, in accordance with (Askaryan, 1973), one can also calculate the material fracture energy.

The angle of the wedge point thinning increases during fracture of rocks with a non-linear increase in stresses in the experiment and a quite thin zone with properties different from that of the massif plastic wedge material is formed at the edge, (Fig.1, Fig.2), (Beisembayev, 2010; Khesin et al., 1969). Following the results of the destructed zone opening, the shape and properties of the material of the thin area at the wedge edge at the fracture moment are close to the state of fluidity. At the same time, it is well known that the fracture of rocks is accompanied by low-frequency radiation recorded by acoustic systems (Beisembayev et al., 2011; Bombizov, 2014; Borissov, 2005).

The pulses of a similar type are also recorded in the mines at the centres of fracture that, in particular, is used to predict areas of sudden coal and gas releases or earthquakes, since low-frequency radiation with a characteristic long wave easily penetrates through the layers of rock. It is also common to the process of cavitation.

To clarify the characteristics of such zones, fracture of a model made of optically active material with a trapezoidal hole was additionally done. It simulates production in underground. Organic glass is chosen as it is homogeneous, and it is easier to observe the laws of fracture in it. This gave an opportunity to find the characteristic features in an ideal environment, and then examine them in rocks. A goal was set to get a crack going from an upper angle. A modelling method with crack deceleration was used. A model of rock made of the optically active material was gradually loaded by metal platforms across the whole surface. They were approaching by screws through special springs where elastic deformation energy was accumulating, (Fig. 1).

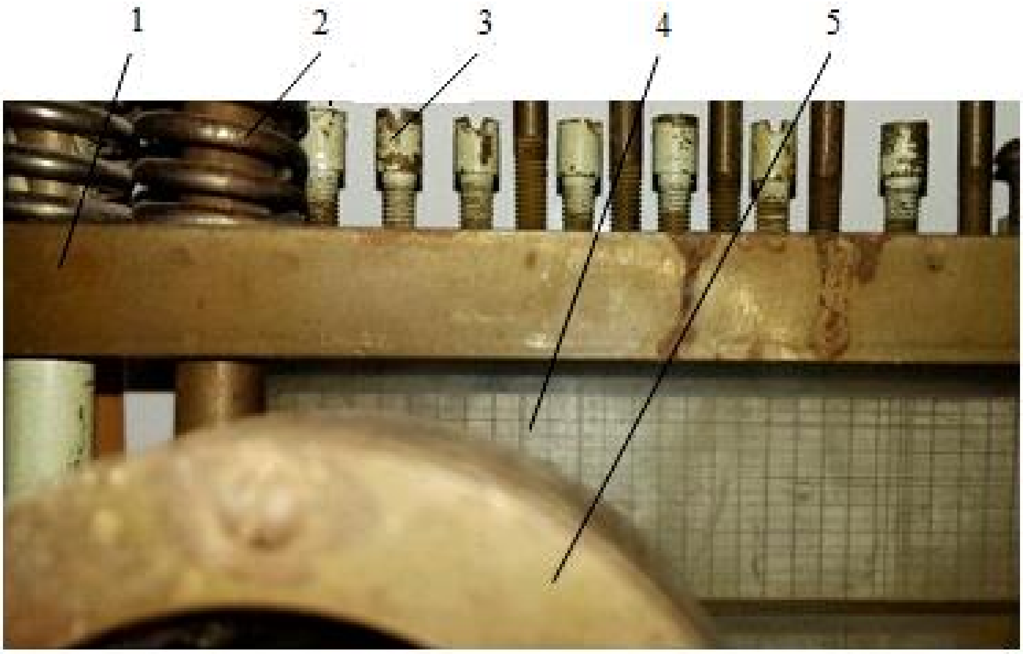

Fig. 1. Stand elements: 1 - platform; 2 - springs with screws; 3 - low-pitch screws; 4 - optically active model; 5 - polarizer.

The load on the surface in the contact zone of the upper layer and the platform was transferred by low-pitch screws through balls in holes.

So, the even load on the layer was put. After reaching the load for the model material close to the limit, gradual unloading of the low-pitch screw located above the working angle was done. This allowed making transfers of the stress-strain state (SSS) in a small area of the layer with a sharply reduced gradient of a stress change in the unloading area. As a result, the stress ratio necessary to begin fracture was reached, and the crack growth speed was reduced. At the same time, a cyclic glow was fixed in the crack area in the low light conditions, and the periodic switch-on of polarisation devices allowed measuring the changing picture of the stresses.

Research in the rock samples

The formation of a plastic wedge and fluid body is scrutinised in more detail for cases of fracturing rock cores loaded by punches when cleaving of a part of the material is possible (Fig.2). At the time of cleaving, a thin petal as a half disc is formed from the plastic wedge (cone) edge, which can be treated as an analogue of a cumulative jet during cavitation. As an extension of the wedge zone (core), it follows the shape of the cleaved element surface at the initial moment of its formation and is a thin body curved in the space between adjacent surfaces of the cleaved element and the surface of fracture on the core. Its shape reminds a fluid body thinning while moving away from the main core and ñfrozenò as a result of a sharp drop in pressure at the moment of separation of the cleaved material. The structure of the body and surfaces that it separates are different, and the petals are therefore easily separated from it, while the body and the plastic wedge (cone) are clearly different in shape.

In contrast to the conditions (Vasilyev, 1976; Askaryan, 1973; Beisembayev et al., 2011), petals are formed in a confined space of the rock capable to completely split along its curved surface. The incurvation of the fracture surface of the wedge and thin body is determined by vectors of stresses active in the body under the punch load given the area of its proximity to the core. The structure of such material is capable to ñfreezeò when removing pressure reminds famous in physics particles orderly shifted from each other under pressure that acquired an unstable (flowing) state. If the pressure is removed during fracturing, the residual linkages in particles return them to a stable ópackagedô state. However, one more possible explanation is that the petals structure quickly formed due to non-linear rise in stresses at every point of a petal as a result of the wedge narrowing without significant movement of the material in the zone. In this case, each petal part is formed of the same material, which was in that zone before thinning. In each case, the material in this zone will have different properties: a coefficient of elasticity goes sharply down, and a Poissonôs ratio goes up approaching a liquid value.

When studying the zone, the plastic wedge (cone) with petals is easily separated from the rock core, and the petal tips may fall and have notches due to their thinning. A clear wedge zone with growing petals can be achieved during rock fracture by a punch with a diameter of up to 10 mm and rock cores with a diameter of up to 50 - 60 mm, since, in such a case, the foundation of the wedge-shaped core has the same dimensions and is visually easily observable, it is also easily separated with a petal using the ordinary tweezers.

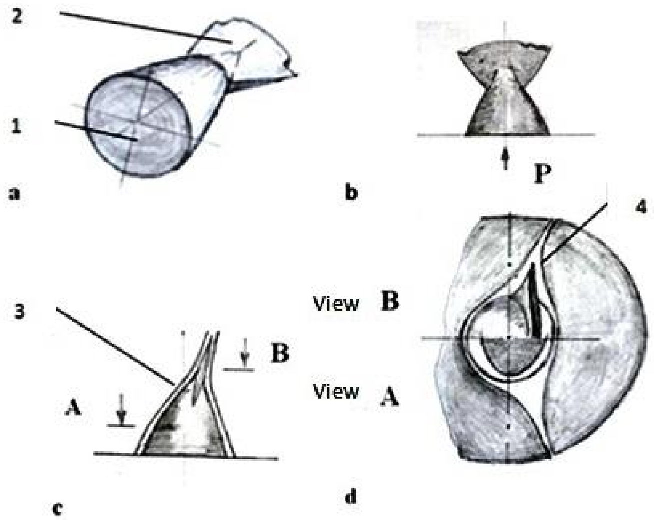

Fig. 2. Formation of ña thin fluid bodyò in a plastic core: Ѳ) formation of thin structures in a wedge-shaped core zone ; b) view from above with a symmetric sample split; fŋ curved side view of a wedge during separation of a cleaved part from a core; d) view from above in arrows ɸ and Ɉ. 1 ï plastic wedge; 2 ï thin fluid body; 3,4 ï contours of separation surfaces; ʈ ï load direction.

The petals appear before a destructive crack separating the cleaved rock part, and the pressure in a petal is normally directed to its surface. The process of fracturing is therefore represented as follows: at a certain moment, the pressure reaches a value when a crack from stretching possibly appears on the petal contour as the massif on each side of the petals is stretched in opposite directions. The fact of cut and stretching during rock fracturing by cutters is well known, but the reasons for the first type of fracturing and formation of extensions were not previously set (Khesin et al., 1969). This is likely due to the experimental material (coal) chosen for fracturing which has a lot of cracks (Vasilyev, 1976).

That is why, the intensive chaotic cracking and brittle fracture of the material happened when the load was applied that did not allow to notice the classic petals, although the focusing process is theoretically described (a sharp increase in stresses in the zone of thinning) (Fig.2). The rock is more monolithic that allowed a clear recording of all stages of the process.

Fracture surfaces and modelling by the finite element method

The surfaces stretchable along the petals are broken in the joining point forming a corrugated surface of type 2 or 6 in Figure 3. In terms of the special features of surfaces, the fracture, in some cases, could be of such types as shift - sliding and separation (Khesin et al., 1969; Vasilyev, 1976). It is assumed that shifting surfaces are smooth 1, 3, while separating surfaces are rough 2, 4. They are interleaved in the fractured samples made of optically active material.

The reason for that is that the pressure formed by the petal surfaces creates another compression zone and another plastic wedge area, in which the secondary focusing is done. Then, it all repeats. It is difficult to fix the secondary petals in the rock due to the smaller size of the wedge zone and material crumbling. However, during fracture of the models made of optically active material, they are identified by fine dust partly found on smooth surfaces, as well as by the view of the surface itself, a perfectly smooth, and the fact of their alternation while remoting from the tool. For example, they are repeated in 2-3 sections in Figure 3a.

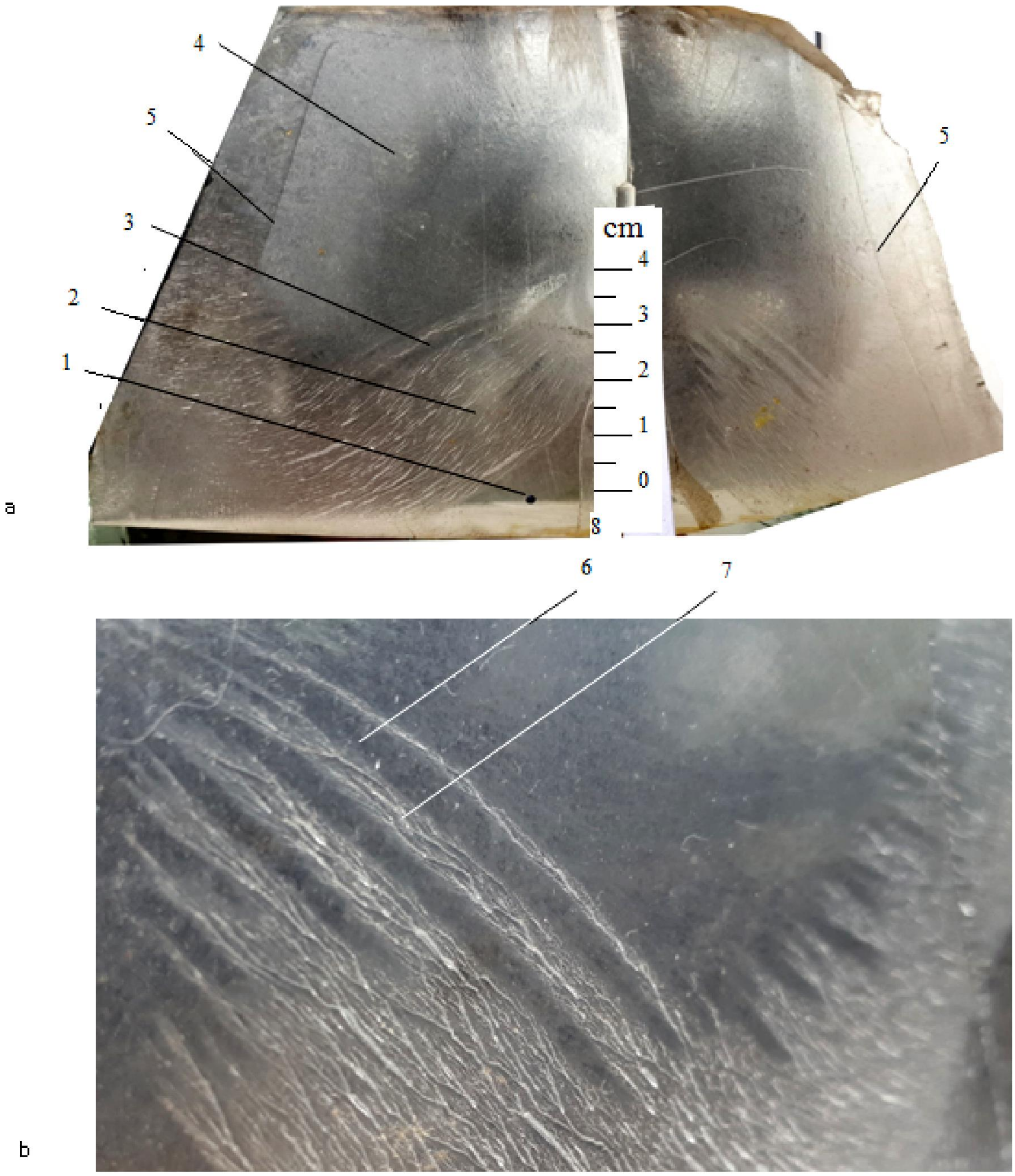

Fig. 3. Surfaces of separation of a cleaved element (organic glass of SD type): a) general view: 1,2,3,4 - alternation of macro smoothed and rough surfaces; 5 - natural line dividing surfaces repeating a contour of a cleaved element; b) increased left side of a cleaved element: 6 - micro smoothed surface; 7- rough surface; 8 - zone of even pressure for model fracture.

We call these repeating pictures as macro fracturing, where the area of one section is comparable with the area of a sample (surfaces 1-4). Micro-fracturing covers the processes inside one line of a section, as shown in 7, for example. The rough macro surfaces 2 (Fig. 3a) are formed from a variety of other surfaces having areas being smaller in tens of times, but there are also smaller macro formations of type 7 consisting of grooves reminding fractals, as well as smooth formations 6. Fracture of the optical material blocks (cross-linked polystyrene SD-3) was chosen because, in the beginning, one could identify their SSS under the load applied that made it possible to predict the paths of fracturing by isostatics (Beissembayev, 2010).

The proximity to the isostatics paths was confirmed by comparison with the real path during fracture of rocks, a block of coal and cement and optical material. The evidence of matching paths with isostatics is confirmed by the creation of artificial conditions of fracture altering the paths, for example, by additional baring of a sample, holes, etc. Analysing the features of the cleaved surface, it could be assumed that lines 5 appear in the zone of ñspecial pointsò, where the isoclines meet as the isostatics paths close to them abruptly change their direction, for which 5 are formed. This line fully repeats the contour of the separated element and is located at a distance of ~ 10 mm from all borders of the cleaved element outlining the separation border throughout the cleaved element. The surface finishing the fracture on line 5 is above the initial one by ~ 1 mm.

The location of special points is shown for two interacting tools set with pacet, (two between the tools symmetrically to 0ڳ- 0ڳaxis, and one on the left side on the bottom sample border, the free surface of the explored model, where a crack usually goes to) (Fig. 4a).

The isoclines with the parameters specified in degrees converge at these points. The left special point, out of the two between the tools, is located in the area of sharply curving isostatics, while the right one describes the isoclines. Though the isostatics at the special point between the tools sharply go to the opposite sides, the crack, as the fracturing shows, is located between the tools since the energy consumption rate here is much lower.

For a majority of lateral paths to the left of the tool going under the slope to the bottom surface slightly above the special point, the difference in energy consumption is small, but the distance to the free surface is much shorter, and the crack therefore abruptly changes its direction going to the near plane of the sample that is in the intersection of the fracture surfaces and defines the line 5 (Fig.3a).

The special points are well observed during the study of optically active thin samples (the task is close to plane). It is rather difficult to observe in the conditions of the general volumetric layout due to the ñdispersal through thicknessò, and the computational modelling based on the finite element method does not always help. However, as in the small sample, these points should be distributed near the cleaved element separation borders, as evidenced by the location of line 5.

To clarify the peculiarities of rock fracture in Ansys package, the modelling of 3D fracture process using a tool with cone-type, trihedral or tetrahedral sharpening was held. Two or one tools were installed into the prepared holes in a rectangular block. The tool was installed close to one of the free surfaces of the block so that it could be possible to cut an element from it. So, the conditions used for optically active models were repeated, and a plastic zone of the tools was modelled with its shape following the shape of the tool and the volume being chosen on the basis of stresses on the core surface equal to a limit value. The linear and non-linear solutions were reviewed.

In the second case, Solid 92 volumetric finite elements and CONTA - type contact elements were applied (Grzejda, 2014). They were introduced after the construction of an extremely fine grid in a contact zone and ensured sliding of the tool with a preset friction coefficient (0,05-0,3). The hard-pliant contact task was resolved: target-hard, contact-pliant surface choosing a tool surface for a target surface and a rock surface for a contact surface, since the deformation of the tool made of carbon steel was less than the rock deformation.

The stress analysis shows that there are two peaks of surge **ů**$_x$: the first is at the joint point of the tool with rock, the second is at some distance, though it could be either with the plastic zone or without it, (Fig. 4b). So, the cracks can form in at least two zones. The curves show the need for grid correction, for example, by comparison with data obtained on optically active samples. So, peaks **ů**ڳ in Figure 4b must be identical, while the grid symmetrical and smaller than in the example given. The curves in Figure 4c were obtained with high accuracy by optical modelling, and can also serve as a sample for grid optimisation. The grid model can be used afterwards for other forecasts as well. It was confirmed that crack paths are not stable and easily change their direction when the direction of load application is changed, and in particular, they depend on compliance with the modelling accuracy. So, the grid distortion by Ansys processor, that is adequate with load deviation from symmetry, can cause an abrupt change of ñcrack pathsò.

This is especially evident in plane modelling or thin destructible block (organic glass SD3), (Fig. 5). At the same time, the rule applies to both modelling and direct fracture of samples. Please note that the processes of plastic wedge formation in organic glass are visually easily observable. The whole area is crossed by a grid of mutually orthogonal ñslip linesò (Beissembayev, 2010).

The zone of plastic deformation is considered as conventionally liquid, (Fig. 6); the mechanical parameters are given in Table 1. When the tool moves, it expands until the ultimate stress for the core destroying is reached. When being destroyed, the core is divided into two parts, (Fig.7a). On the surface of core 3 in zone 4, according to the optical and finite element modelling data, there is a zone of increased tensile stresses. The rock does not withstand even a small stretch and therefore there is a crack that moves to the tool. Two lateral cracks move from the tool.

When there is destroyed plexiglas, the initial crack on the surface is not formed, and one part breaks from the block, (Fig.3).

Figure 7b shows the picture of lateral crack 2. It has been obtained by breaking a block of coal and cement imitating the formation. Its trajectory closely coincides with the isostatic curve of the principal stresses for optical models.

It was confirmed that crack paths are not stable and easily change their direction when the direction of load application is changed, and in particular, they depend on compliance with the modelling accuracy.

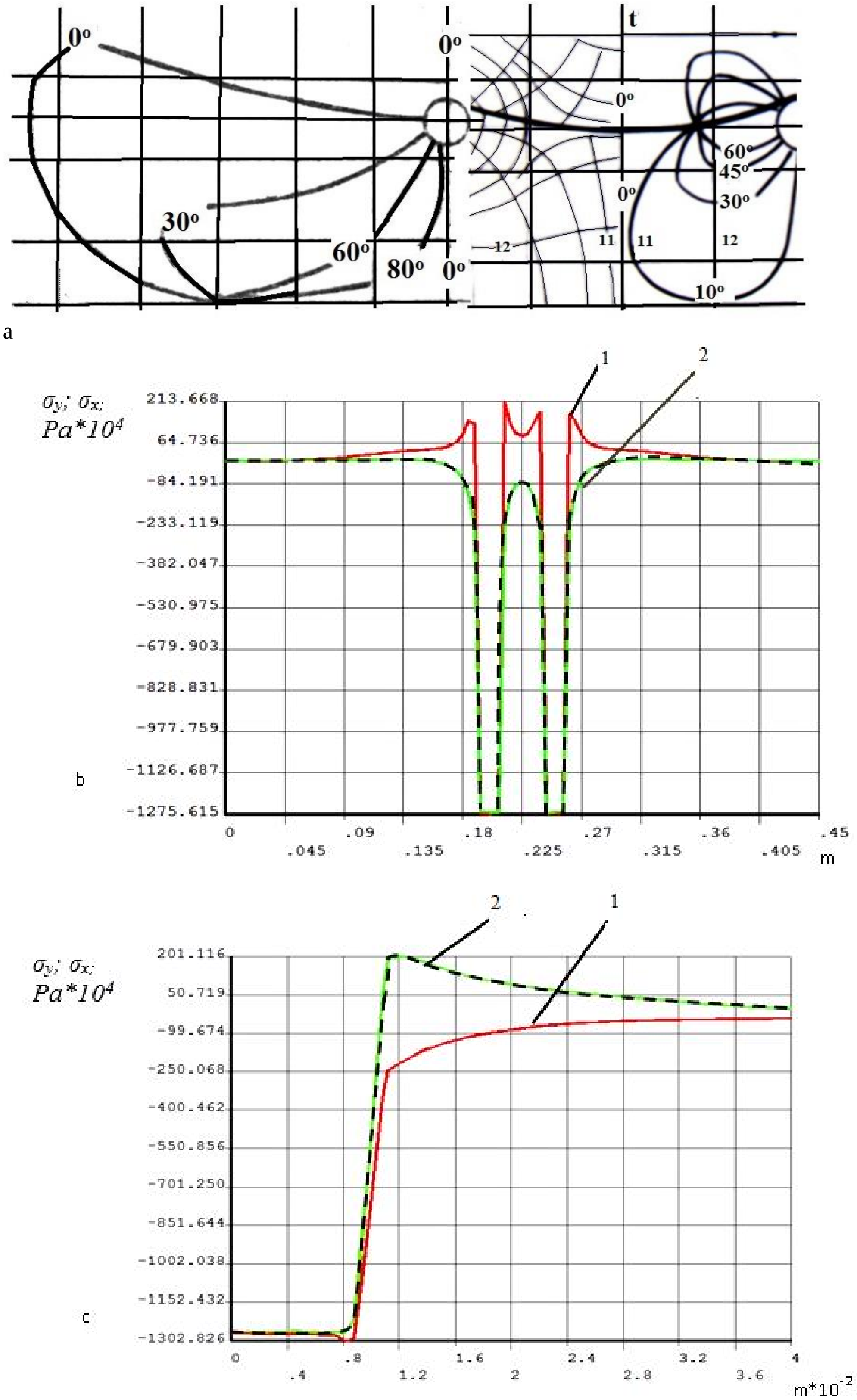

Fig. 4. Screenshot, the effect of two tools on a rock: ʘ) field of isostatics and isoclines; stress σ_x - 1 and σ_y - 2 along the line between tools (b) and along a line from tool centre down to the surface (c).

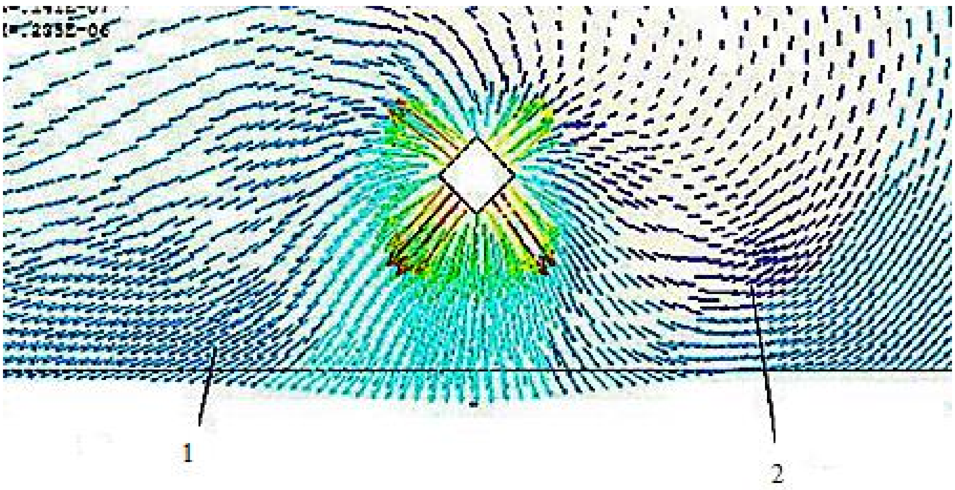

Fig. 5. Screenshot: paths of principal stresses: 1- isoclines convergence zone (optical modelling); 2 ï artificial distortion zone.

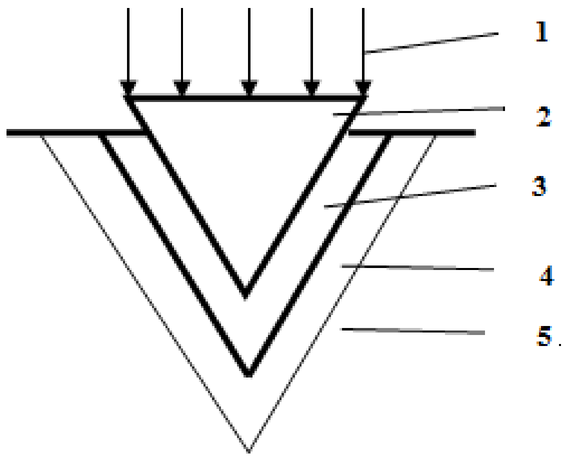

Fig. 6. Model diagram: 1- pressure; 2- tool; 3, 4 - positions of the plastic zone; 5 ï core.

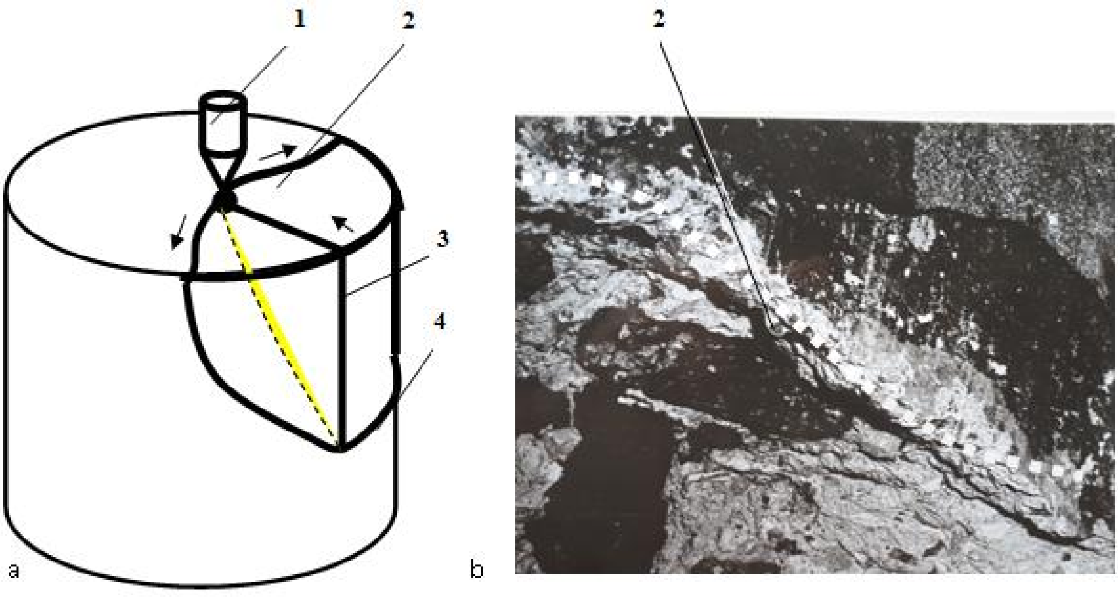

Fig. 7. Features of the crack trajectories: Θ- in the core; 1- tool; 2 ï lateral crack; 3 - initial crack on the surface; 4 ï core; b ï for the block of coal and cement.

Conclusions

Destruction in rocks is accompanied by low-frequency radiation detected by acoustic systems. In recent years they have been widely developed and permit to predict these processes in the subsoil. For example, they are necessary to identify the destruction of rocks over lavas in order to refine the design scheme for predicting the maximum pressure on the face. The load from the hanging layers is determined by the length and height of the layers in the worked space behind the lava. Moreover, accordingly by the load from them, one can judge the possibility of destruction that must be confirmed by detecting radiation. The obtained facts make it possible to obtain visual confirmation of these factors, to reveal the mechanism of destruction that in the future will permit to clarify both the energy of the processes and the working conditions of the tool. It is also important to predict the direction of the cracks development.

The experiments displayed good relevance between the forecast for isostatics and real results. At the same time, there are methodologies that allow building a path step by step, i.e. first a small section is built, then the next section gave the changed SSS, etc. (Beisembayev et al., 2011). The authors proceed from the fact that the appearance of a crack zone radically changes SSS. However, the methodology stemming from the initial SSS is confirmed in practice. Moreover, the loading put on massif reveals violations and consolidates a path by the creation of microzones that have reached the limit state providing a predictable path. These factors may be more important. It is necessary to take into account the time of crack growth as well: whether it is able to exert a fatal effect on dislocation so that it changes its path.

So, the analysis shows the validity of the two seemingly opposite approaches to the prediction of a crack path. The first approach is applicable for environments with high speed of crack growth, while the second for the disturbed environment. Please note that with a stepwise path building, one should proceed from identification of a new SSS at each step, within which the same isostatics hypothesis can be applied.

It can be assumed that the effects of cavitation with the formation of a cumulative jet and mechanical fracturing of solid rocks are grounded in the principles of focusing, which in addition to general patterns have own characteristics defined by specifics of environments of micro and macro levels, in which they occur. So, the studies helped clarify the mechanism of cyclic rock fracture that allows clarifying the calculation of energetic of a fracturing process and explain the mechanisms of its identification. Some of its provisions can be applied in modelling, as well as choosing a drilling tool for holes and analysing the processes of bottom rock fracture.

Some provisions of the studies can be applied in the methodology of modelling and analysing mining equipment structural destruction processes, as well as in selecting drilling tools for making wells.

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The Model of Direct Dumping Technology Implementation for Open Pit Coal Mining by High Benches

***Maxim Tyulenev*[1], *Sergey Markov*[2], *Michal Cehlar*[3], *Sergey Zhironkin*[4]** *and* ***Magerram Gasanov*[5]**

The article describes a problem of coal open pit sides designing with combined mining method (transport and direct dumping by high benches). The architecture of open pit sides implies solving the problems of decreasing coal open pits land capacity, application of direct dumping technology with internal dumping which increases coal mining economic and environmental efficiency. The main criteria defining the architecture of quarry sides for flat coal seams strata open-pit mining are general slope angle of internal multi-tier dump, the height of transport and direct dumping sub benches on the face side, the number of tiers and their height, the width of the berms on the dump site. The increase of benchesô height at open pits developing deposits with flat coal seams entails the use of a combined technology: transport and direct dumping to develop the upper and lower sub benches respectively. Therefore, it is necessary to advance methodology for calculating the parameters and indices of direct dumping technological schemes and find ways to increase the capacity of internal dumps and provide safety for open pit mining operations. As a result of research, the model of direct dumping technology implementation for open pits mining coal by high benches is presented in the article. The research was based on the data from Southern Kuzbass (Kemerovo region, Russia) coal deposits being developed by large open pits.

Key words: *design of quarry sides, open pit mine, direct dumping, high bench, dragline, internal dumps*

Introduction

At present, in Russia, in particular in Kuzbass (the main coal-mining region located in Western Siberia), large open pits develop coal deposits with flat seams (Agafonov, 2017; Gvozdkova, 2017; Tyurin, 2017). There is a trend towards increasing the height of the benches (so-called ñhigh benchesò) followed by their development with combined transport and direct dumping technology by two sub-branches. The upper sub bench is worked out by transport technology with loading rock for motor transport, the lower one ï by direct dumping, with rock transfer to internal dumps by a dragline (Alarie, 2002).

This allows, firstly, using equipment with a large unit capacity (dump trucks with carrying capacity up to 460 tons, draglines with a bucket capacity of more than 40 m^3). Secondly, the growth of annual coal production in Kuzbass from 100 to 220 million tons over the past decade has been accompanied by an increase in the number of sections, from 42 to 66. This means the need to improve open coal mining technologies that allow minimising the use of land resources and, thus, reduces withdrawal of agricultural land from farming and allocation of dust from the dumps (Cehlár, 2017).

In this sense, direct dumping technologies at open coal pits, tending to horizontal and flat coal seams extraction, are preferable to transport ones especially considering innovations in high benches drilling and blasting (Hrehová, 2012). However, it is critically important for direct dumping effective application that as the number of dump tiers placed inside open and the volumes of overburden piled in them increase, the requirements to the accuracy of calculating the main parameters face and dumping sides are repeatedly increased. This is stimulated mainly by the requirements for the stability of internal dumps in order to avoid landslides (Tyulenev, 2018).

Thus the increase of the benchesô height changes the profile of open pit and raises the issues of architecting its face and dump sides. In this regard, three key tasks of open pit sides architecting are considered to be relevant:

1. Determination of the optimal height of transport and direct dumping components of the high bench, which makes it possible to use draglines with maximum effectiveness to move the overburden into internal dumps.
2. The choice of the best direct dumping scheme with different heights of direct sub bench, which makes it possible to fully use the capacity of internal dumps.
3. Determination of general angle of the slope of the internal dump, guaranteeing the absence of landslides if the technology of its backfilling is observed.

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Materials and Methods

Analysis of the state of mining at Southern Kuzbass open pits showed that at present, using direct dumping technology, strata of two coal seams with 49-52 m of the total thickness of rock formation is being worked out. The overburden is piled into three-tier dumps of 73-78 m high. The cost of the direct dumping of overburden is 30-40 % lower than the transport cost. In addition, there is an excess in the speed of the direct dumping front over the transport one, which leads to a decrease in the volumes of overburden rock developed by direct dumping.

Further, the potential capacity of the internal dump is underused, which can be judged by the ratio of the actual heights of dumps and allowed by the dump slope stability condition (110-150 m).

Considering the above, in order to reduce the cost of overburden removing from the whole open pit, it seems appropriate to partially redistribute the rock volumes between the transport and direct dumping technologies by increasing the volumes of rock placed in the internal dumps. To verify the effectiveness of this proposal, it is necessary to solve a number of technological and technical-and-economic tasks related to ensuring the reception of additional overburden by internal dumps and to define the economically acceptable height of the cut rock layer. When solving these tasks, the conditions for maintaining the achieved indicators of mining operations intensification and reducing the cost of overburden should be met. In this context, the use of high benches allows optimising the parameters of open pit mining of coal using direct dumping technology.

During the period of the 1960s-70s in the former USSR, the areas and boundaries of direct dumping implementation were significantly expanded, thanks to an increase in the height of the rock benches worked out by direct dumping technology, according to the condition of economic efficiency. In addition, at present, combined transport and direct dumping methods of overburden removing are widely used at Kuzbass open pits. Obviously, the minimum costs for the quarry development can only be achieved by direct dumping technology, since the cost of 1 m^3 of overburden for transport technology is almost constant, and the cost of 1 m^3 of overburden for direct dumping varies depending on the thickness of the overburden, which determines the re-excavation rate. This provision is taken into account in this article.

The task to optimise the height of the bench for direct dumping which influences on the whole architecture of open pit sides is very important because the technology with rock transfer by dragline directly into the dump has a substantially lower cost of processing of 1 m^3 of overburden (Cehlár, 2007; Demirel, 2011; Ċimková, 2016; Vukotic, 2013). The solution to this problem involves drawing a number of direct dumping schemes of excavation with different heights of dragline sub bench (Karpuz, 1990). Therefore, an appropriate methodological approach to the solution of the problem is necessary.

At present, the researchers, defining the parameters of mining technological processes use grapho-analytical modelling with sufficient confidence (Baafi, 1997; Erdem, 1998; Katsubin, 2018; Malli 2015; Markov, 2017; Martyanov, 2018; Mirabediny, 1998; Prakash, 2013). Therefore, in this article, we present an ideal grapho-analytical model for high bench development by two sub benches (transport and direct dumping) to achieve the aim of parameterisation.

Theoretical provisions of a grapho-analytical model for high bench working by a combined method are given below.

Grapho-analytical model of direct dumping excavation schemes is acceptable for their use in any conditions of coal seams occurrence. Based on the development of the theory of blasting downhole charges and its experimental verification, a computational method was developed for determining the parameters of the collapse of overburden benches with direct dumping technology, which applies to the development by high benches (Martyanov, 2018). In connection with the complication of direct dumping excavation schemes, some authors determined the rational length of the mining front, organising equipment operation for the interconnection of overburden and coal mining operations and its influence on the mining schedule (Tyulenev, 2017; Gvozdkova, 2017). In particular, the following methods of increasing the capacity of the internal dump for direct dumping technology were considered:

- increase the number of tiers of overburden dump;
- an additional charge of the coal seam;
- reduction of the width of the stope, when due to more complete use of the unloading parameter of the excavator, it is possible to put a larger volume of overburden rock in the dump;
- the curvature of mining front works to increase the length of the dump front and, therefore, to increase the capacity of dump;
- development of overburden stope with the shift of draglineõs axis in a plan, which allows, as it were, to increase the size of the dragline unloading parameter and, consequently, the dump capacity.

The grapho-analytical model considered in the article implies that the additional charge of the coal seam is usually used in simple excavation schemes, but at the same time, the coal losses increase. This method is not spread in Kuzbass open pit mines due to significant loss of high-quality coal.

Reducing the width of the stope is usually recommended for local mining areas along the front of work with increased overburden thickness. Therefore, this method cannot be applied in the geological conditions considered in the article, characterised by sustained overburden thickness.

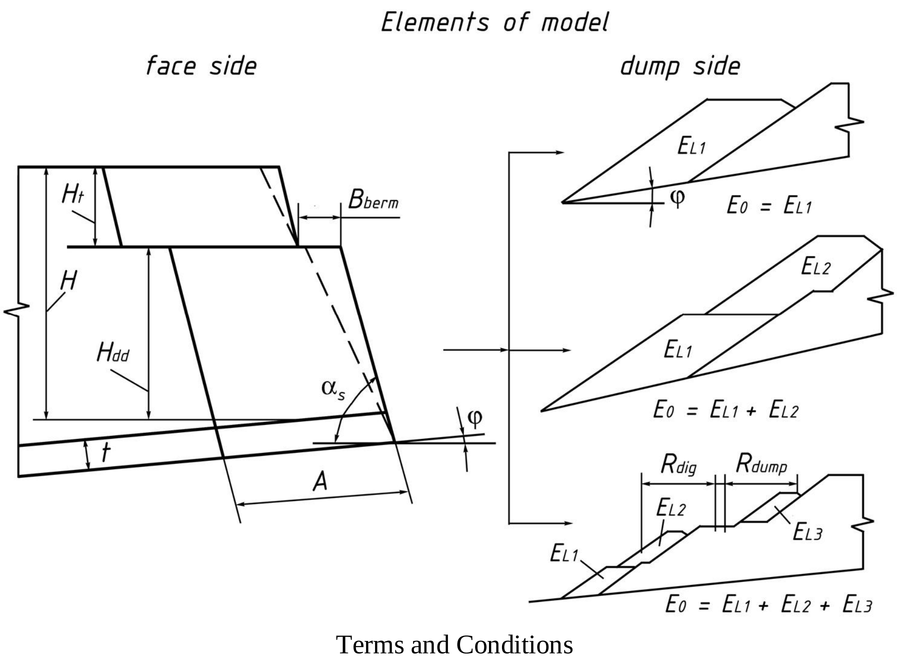
The curvature of the work front changes the capacity of the internal dumps, especially with a small radius of curvature, but this method can be applied with relatively small fluctuations in overburden thickness and long quarry fields. In the conditions of the combined technology (transport and direct dumping) with significant dip angles of coal seams suite, this method has very limited application.

The shift of draglineôs axis is widely used in the quarries of Russia to increase the capacity of the second tier of the dump. Therefore, this method is used in the study, reflected in this article, devoted to the design of excavation schemes.

When analysing the position of surface mining at open pit mines of Kuzbass, it was concluded that at present mainly three-tier dumps are piled. To accommodate an additional volume of overburden rock in the internal dump, the piling of dumps with a number of tiers of more than three is needed. The real question is about the number of tiers: four, five or more.

The Kuzbass coal open pit mines have experience in piling four-tier dumps, and it has shown a certain complication in the organisation of work of draglines on the dump and an increase in the total re-excavation rate due to the moving of additional rock volumes during its secondary displacement. When dumping five-tier dumps, a significant increase in the total re-excavation rate and complication of the organisation of draglinesô work on the dump are expected.

Considering above-mentioned, we define the main part of a grapho-analytical model for high bench working by a combined method as it is given in Figure 1. The initial position of the model consists of two elements: the face and the dump sides.



Terms and Conditions

$$H = H_t + H_{dd};\begin{cases} H = const; H_t \; and \; H_{dd} - \text{variables} \; ; \\ H_t \neq 0; H_{dd} \neq 0 \end{cases}$$

$$\begin{cases} H_{\max} = \dfrac{E_0}{A \times K_d}; H^{t}_{\min} = 5 \; m; H^{dd}_{\min} = 10 \; m; H_{\min} = 15 \; m \\ H^{dd}_{\max} = H_{\max} - H^{t}_{\min}; H^{t}_{\max} = H_{\max} - H^{dd}_{\max} \end{cases}$$

$$\varphi = 3 \div 12^{o}; \alpha_s = 75°; \boldsymbol{\Phi} = 30 \div 50 \; meters$$

Fig. 1. The structure of grapho-analytical model for high bench working by a combined method.

The face side of the model includes the structure of rock formation processing. The dump side of the model includes three schemes of dumping: a single-tier dump, a two-tier dump and a three-tier dump.

The following legend was adopted in the model:

H ï the height of rock formation, m;

H^t, H^{dd} ï respectively, the height of the transport and direct dumping benches, m;

H_{max}, H^t_{max}, H^{dd}_{max} and H_{min}, H^t_{min}, H^{dd}_{min} ï respectively, maximum and minimum heights of rock formation, transport and direct dumping benches, m;

A ï the width of the stope, m;

t ï coal seam thickness, m;

ű ï dip angle of the coal seam, degree;

Ǚ ï the angle of rock slope, degree

K_d ï coefficient of rock disintegration in a shotpile and in a dump;

E_0 ï the capacity of an internal dump (per meter of the working front), m.

The development of transport sub bench is carried out according to the classical technological scheme characterised by rock excavation and loading by shovel into a dump truck with rock transportation to external or internal dump.

For this model, the schemes of piling the dumps differing in the preparation of additional capacity for any tier have less capacity than the schemes without capacity preparation. Also the width of drilling-and-blasting stopes, the minimum value of which is determined by the condition of the turn of the coal-carrying trucks in the mining faces (35-40 m), does not allow to fully use the discharge radius of the draglines currently used in piling the dump layers.

The use of this model in the design of direct dumping using high benches on the face side of quarry includes the formation of an excavation scheme ï a graphic description of the overburden excavation and its transfer to the internal dump. Excavation schemes (in different or in the same geological conditions) when changing the dragline locations in a profile of a stope differ in their structure, that is, in the number of stages of development and displacement of the rock, as well as in the technological interconnectivity between them. If the number of stages and the technological interconnectivity between them are the same, but the geometric shape of the stages and their sizes can differ, then such excavation schemes can be defined as one-type. According to the proposed structure of the model, each element of the face side (the initial stage of the excavation scheme) can be docked with any element of the dump side (the final stage of the excavation scheme). In addition, by changing the position of the middle seam on the geological structure of the face side, it is possible to reveal the interrelation between the conditions of seamsô bedding in the formation and the method of its development by high benches, taking into account any of the dump piling schemes.

Results and Discussion

The idealisation of the model for direct dumping bench working is the following. The full use of dragline parameters for piling the dumping layers is accepted. In the proposed model, the direct dumping bench on the face side (the initial stage of the excavation scheme) can be joined to any element of the dump side (the final stage of excavation scheme). Thus, the model allows assigning any number of variants of excavation schemes. An example of a typical scheme with the dumping of a two-tier dump is shown in Fig. 2.

The procedure for calculating the parameters and indicators for high bench processing for transport and direct dumping technologies consist of the following.

First, the parameters and indices of the transportless technological scheme must be determined because the development of direct dumping bench causes the preparation of the seam for excavation and determines the speed of working front movement (Tyulenev et al., 2017).

Then the indicators of intensification of transport sub bench working should ensure non-stop operation of the direct dumping sub bench, i.e. to avoid dragline downtime. Consequently, the condition $V^t_f = V^{dd}_f$ (where V^t_f and V^{dd}_f ï the speed of moving the work front on transport and direct dumping technology, respectively) must be observed.

Calculation of the parameters and indices of each stage of high bench working can be done in seven steps. Of them, five steps relate to the calculation of parameters and indicators of direct dumping technology and two ï to the transport one.

Considering these steps in details could help to find the following:

1. For a dragline, considered in a specific variant, the maximum capacity of internal dump (E_0) (one-tier, two-tier and three-tier) is calculated. The maximum equivalent height of strata H^{dd}_{max} (accordingly dumpôs capacity) is determined by the height of overburden layer, which must be located in the dump with the dumpôs width equal to A. This task is called ***ñdirectò*** and consists in determining the maximum capacity of the internal dump.

The second stage of calculations is related to the order of assignment variants of direct dumping bench height H_{dd}.

First (the first option), minimum height of the direct dumping bench H^{dd}_{min} is assumed, based on the condition of its scooping by dragline.

Each next variant is characterised by an increase in the height of direct dumping bench by $\boldsymbol{\varphi} H^{dd}$, which can take values of 1, 2, 3 or more meters depending on the required accuracy of calculations.

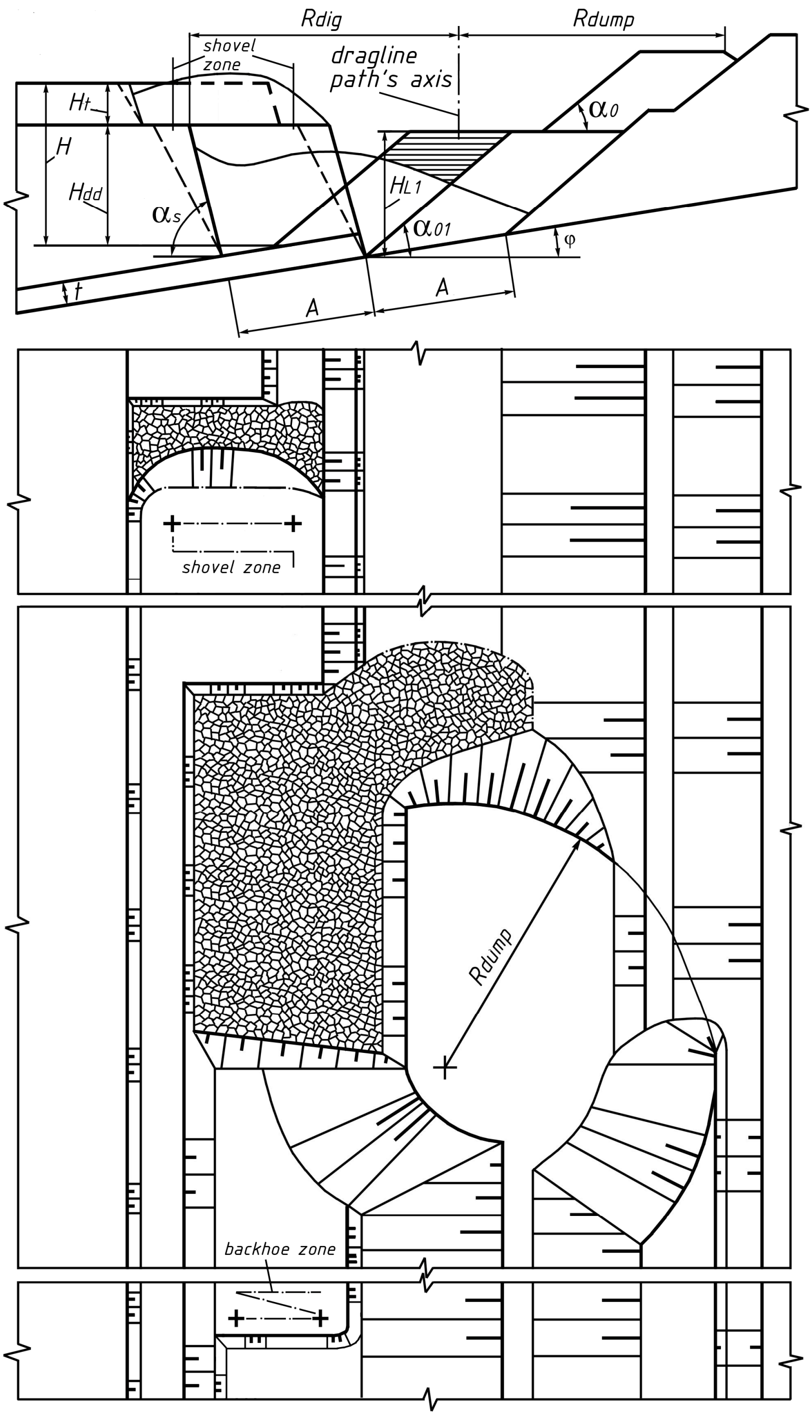


Fig. 2. The typical scheme of excavation with backfilling of a two-tier dump.

2. For each variant under consideration, the rock volume on the face side in the disintegrated state is determined.

Further, by comparing the maximum capacity of the dump and the actual rock volume of the face side, the number of dump tiers and the actual height of one-tier dump, the second tier of the two-tier dump and the third tier of the three-tier dump are determined.

Thus, at the second stage, the ***ñreverseò*** task is fulfilled: according to known rock volume on the face side, the number of dumpôs tier and its parameters in the final (static) position are determined.

3. The parameters of drilling and blasting operations, the parameters of the collapse are calculated, and the coefficient of explosive rock discharge is determined.

4. The dynamics of rock movement from the face side to the dump is projected, for which the structure of the excavation scheme and its parameters are determined by the grapho-analytical method. The schemeôs re-excavation rate in mining profile and annual effective dragline productivity are determined.

5. The main technical and economic indicators (volumes of mined coal, stripping ratio, the productivity of draglines, direct costs of stripping and coal mining, etc.) of direct dumping technology are determined.

6. For already established annual speed of mining front moving for direct dumping technology (V^{dd}_{f}), the transport technology indicators are calculated for a predetermined overburden excavator. The annual economic indicators and the necessary number of excavators are calculated or adopted according to the practice data.

7. The technical and economic indicators of the combined high bench working and the magnitude of the evaluation criterion are calculated. Further, these steps of calculations are repeated for each variant, which parameters are chosen in advance.

As it was mentioned above, the solution of the ñdirectò task involves determining the maximum capacity of the internal dump. For given dragline, first, the maximum capacity of the internal dump (of one-, two- and three-tier), as well as the maximum equivalent height of the strata H^{dd}_{max}, is calculated. One of the most important indicators for this is the **general angle of the internal dumpôs slope**. Fig. 3 (a, b, c) shows the scheme for calculating parameters and maximum capacity of internal dumps.

The following legend is used on the schemes in Fig. 3:

1. Working parameters of the dragline, piling dumping layers, and the parameters of its safe installation:

R_{dig}, R_{dump}, H_{dump}, H_{dump}, R_r, W_{pass} ï respectively: digging radius, dumping radius, dumping height, digging depth, the radius of rotation and width of excavatorôs path (for dragline engaged in the formation of the first and second tiers), m;

B ï minimum distance from the axis of draglineôs path to the upper edge of the underlying tier, m;

b_n ï minimum distance between the draglineôs shoe when working and walking and the upper edge of the underlying tier, m (3-4 m).

2. General parameters of all dumping schemes:

α_0 ï the natural angle of slope for rock in the dump, degrees (37°);

γ_g ï the general angle of internal dumpôs slope, degrees;

B_{berm} ï the estimated width of the berm between the upper edge of the first tier and the lower edge of the second tier, m;

$h_{ű}$ ï the value of lowering of the dump layer between two adjacent stopes, m;

A_g ï horizontal width of dumpôs tires, m;

S_{pr} ï the volume of rock in the second (third) tier, placed there during piling of the first (second) tier of the previous dump, m^3;

ʗ$_{L1}^{max}$, ʗ$_{L2}^{max}$, ʗ$_{L3}$ ï respectively: the capacity of the first, second and third tier per one meter of working front (specific capacity), m^3;

H_{SL1}, H_{SL2} ï the height of tiersô first and second slopes, m;

H_{t1}, H_{t2}, H_{t3} ï respectively: heights of 1st, 2nd and 3rd dump tiers;

H_D ï height of whole internal dump (H_{D1}, H_{D2}, H_{D3} ï heights of single-tier, two-tier and three-tier internal dump);

ʗ$_{\varphi}$ ï empty capacity in the dump tier due to the insufficient value of draglineôs dumping radius (Fig. 3- b, c);

φ ï the width of empty capacity in the upper part of dump tier (Fig.3-b, c), left unoccupied during its piling due to the discrepancy of draglineôs dumping radius R_{dig} and the horizontal width of the dumping tier A_g, m.

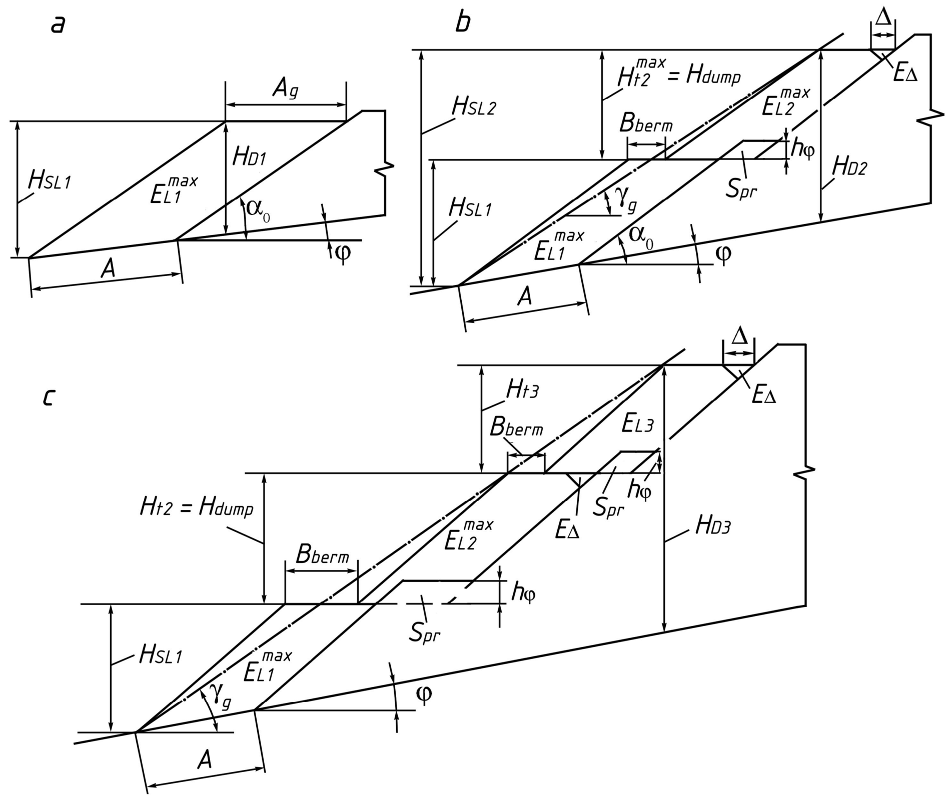


Fig. 3. The schemes for calculating parameters and maximum capacity of the dumps: a - single-tier, b - two-tier, ɘ - three-tier (the solution of ñdirectò task).

The value of parameter ***φ*** is determined on the analysis of the plot of empty capacity $ʈ_φ$ ($E_φ = 0.25 \cdot φ^2 \cdot \tan \alpha_0$) on the ratio $K = φ / A_g$ (Fig. 4).

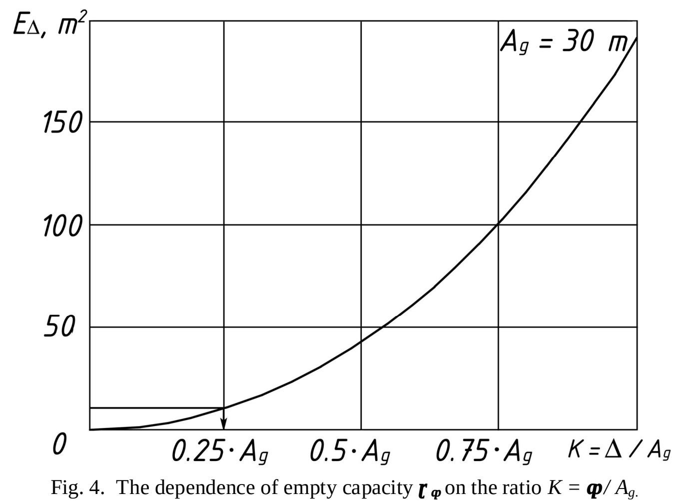


Fig. 4. The dependence of empty capacity $ʈ_φ$ on the ratio $K = φ / A_g$.

As can be seen from the plot in Fig. 4, the empty capacity $E_φ$ increases in quadratic dependence on the width of the cut-off layer A_g. From the point of view of minimising empty capacity $E_φ$, the value of ***φ*** parameter can be taken to be equal to the fourth part of A_g ($φ = 0.25 \cdot A_g$). In this case, the smallest capacitance value $E_φ$ takes place with sufficient use of the dumping radius of the dragline.

The definition of the general angle of dumpôs slope is made in the following order:

1. When dumping a single-tier dump (for variants of excavation schemes with a small height of direct dumping bench ï 10-15 m), the general angle of dumpôs slope $ɔ_g$ is equal to the natural angle of slope $Ǔ_s$ = 37°. The height of the slope of the first tier is assumed to equal to 25 m (H_{SL1} = 25 m), according to the recommendations of Kuzbass State Technical University (Kemerovo, Western Siberia, Russian Federation).

2. When dumping two- and three-tier dumps, the angle $ɔ_g$ is determined using the dependences $ɔ_g = f(H_D; ű)$.

$$V_{\text{int}}^{dump} = H_{dd} \cdot A \cdot \left(\frac{\sin\varphi}{\tan\alpha_0} + \cos\varphi \right) \cdot K_d, \tag{1}$$

then:

$$H_D = \frac{V_{\text{int}}^{dump} + 0.5 \cdot A \cdot A_g \cdot \sin\varphi}{A_g \cdot \left(1 + \frac{\cot\alpha_0}{\cot\varphi - \cot\alpha_0} \right)}, \tag{2}$$

where:

$$A_g = \frac{A \cdot \sin(\alpha_0 - \varphi)}{\sin\alpha_0}. \tag{3}$$

Further, according to the dependence $\gamma_g = f\,(H_D;\ \varphi)$ (Fig. 5), for known φ we find γ_g and the result of the calculation is expressed in round numbers.

Table 1 gives the recommendations for defining the general slope angle of internal multi-tier dumps depending on the total height of the dump and the angle of inclination of its base φ.

Tab. 1. The general slope angle of internal multi-tier dump.

Soft ground (coal clay, carbon-bearing mudstone, soft mudstone and siltstone)	The angle of inclination of dump's base φ, degrees	The general slope angle of internal multi-tier dump γ_g, degrees, for its total height H_D, m [x] – maximum values of slop angle					
		20	30	40	60	80	100
	3	32-37	26-37	22-34	20-31	29[x]	–
	6	28-37	21-37	32[x]	29[x]	27[x]	–
	9	23-35	31[x]	28[x]	25[x]	22[x]	–
	12	32[x]	27[x]	24[x]	–	–	–

According to Tab. 1, a family of dependences $\gamma_g = f\,(H_D;\ \varphi)$ was obtained (Fig. 5). These dependencies are used in calculating the parameters of excavation schemes with different heights of direct dumping sub bench and, consequently, with the piling of internal heaps of a different number of tiers. The results allow keeping further calculations to determine the capacity of single- and multi-tier dumps.

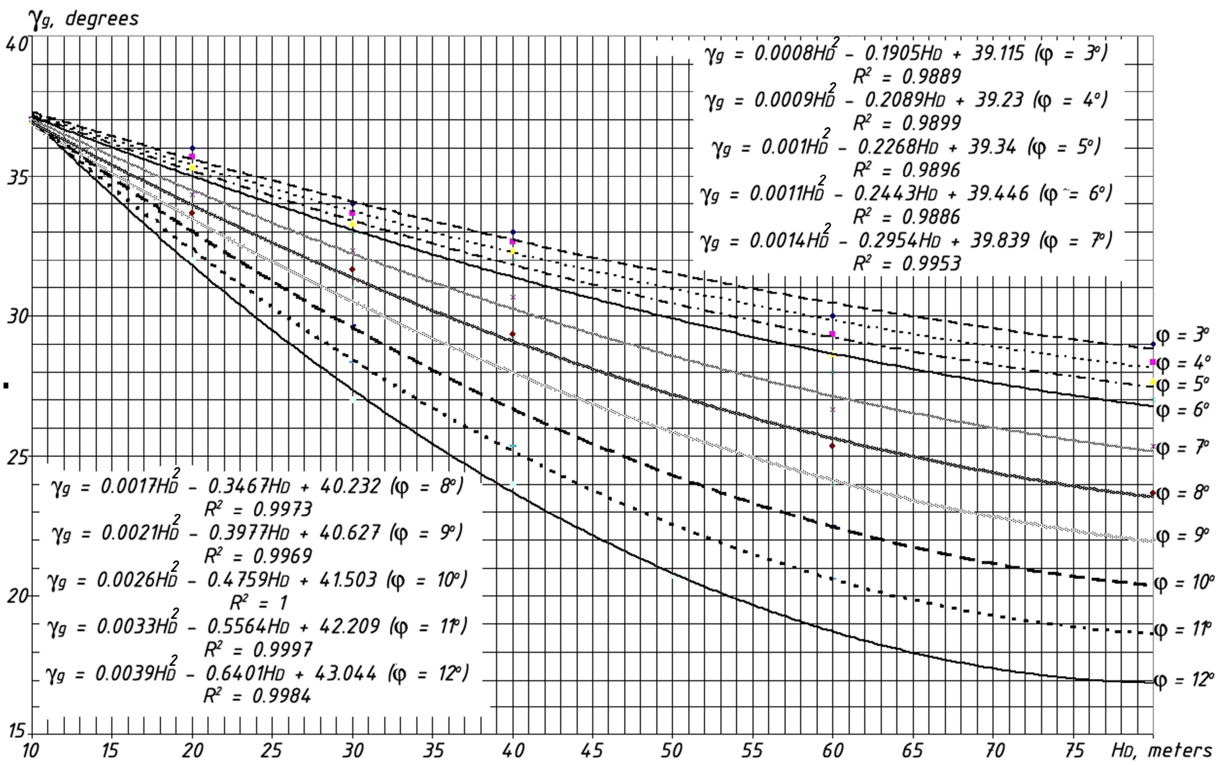


Fig. 5. The dependences of general slope angle of internal multi-tier dump (γ_g) on its height (H_D) and angle of inclination of its base (φ).

Conclusions

1. The architecture of face and dump sides of open coal pit implies modelling of the key parameters of direct dumping technology and defining general angles of internal multi-tier dumps for landslides avoiding and safety providing. Therefore, we recommend determining the minimum height of the direct dumping bench based on the comparison of dumpôs maximum capacity and the actual volume of the face side, taking into account the number of dump tiers and their actual height. Such a calculation must be iterated for a single-tier dump, the second tier of a two-tier dump and the third tier of a three-tier dump. Based on the analysis of the results obtained, the optimal version of the high bench development technology that meets the stated requirements or has the best technical and economic indices can be selected.
2. Based on the studies and calculations carried out, we concluded the following: the general slope angles of multi-tiered internal dumps, which base is composed of soft rocks, should not exceed 24-29º at the height of 40-60 m, and the height of the slope of the first tier (the natural slope angle is 37º) should not be more than 20-25 m.
3. Using the method of direct dumping technology parameterisation at open coal pits developed by high benches allows optimising draglines work and avoiding rock slide on the dumping side of the pit.

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Application of Tecnomatix Process Simulate for optimisation of logistics flows

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In today's digital world, there is a need to apply advanced simulation tools in business practice in order to create value-chain within the product lifecycle. Simulation programs are important, especially in the process of design, analysis and optimisation of production and logistics systems. They enable changes to existing production systems by way of experimenting, which is it is a great advantage. One of a number of simulation applications are modules from portfolio Siemens PLM Software. The present article deals with the optimisation of logistics flow through the application of the selected module Tecnomatix Process Simulate, with an effective use especially in the automotive industry. The importance of such an application is mainly in the execution and evaluation of specific changes in production systems in a virtual environment without any impact on actual production. In the present article, the case study identifies the practical use and possibilities of the simulation tool Tecnomatix Process Simulate for material flow optimisation, inter-operational transport and material handling in company oriented in leather processing for the automotive industry. The result is a simulation of proposed changes which increase the efficiency of logistic flows, the creation of a 3D model of the proposed solution, and quantification of achieved results of optimisation. Application of applied module represents one of the first steps of transformation the selected company to a digital company.

Key words: *analysis, optimisation, material flow, modelling, simulation.*

Introduction

In today's digital world, advanced simulation software is used to design, analyse, and optimise production and logistics systems. These make it possible to implement changes in existing production systems without intervention in ongoing production. However, the simulation itself cannot solve the problem. It allows performing experiments and choosing an optimal solution for the defined conditions. For this reason, continuous interaction with the real environment is important. However, it is a key intermediate step for building a digital company (Pekarcikova et al., 2015; Negagban et al., 2014; Edl et al., 2013; Straka, 2005; Straka et al. 2009). Modelling and simulation are tools for effective verifying of the feasibility of individual production and logistics processes and procedures prior to their implementation into a real environment. The basic task of simulation is verification of spatial constraints, assembly and robotic route planning, static and dynamic collision detection (Straka, 2010; Lenort et al., 2012; Malindģák et al., 2017, Rosová, 2010).

The case study below was realised in Tecnomatix/Tx Process Simulate software from Siemens PLM Software, which constantly evolving portfolio of PLM/Product lifecycle management to meet the needs of the mechanical engineering, automotive, aerospace, defence industry, etc. In the PLM market, Siemens Industry Software belongs to the most powerful companies. In the automotive industry, more than 80% of automakers worldwide use solutions from its product portfolio (Gupta et al., 2016; Hsieh et al., 2018; TrebuŔa et al., 2015; Saniuk et al. 2014).

Material and Methods

1. Possibilities of use the Tecnomatix Process Simulate

The Tx Process Simulate software module was applied to the case study. It is one module from the Tx products package (Plant Simulation, Process Designer, RobCAD, FactoryCAD, FactoryFlow, Jack and Jill). It enables verification of virtual proposals (process, manufacturing and logistics activities) even before they are actually implemented, what significantly reduces the risk of failure. It is also a platform for the deployment of Virtual Commissioning, that allows optimising automation systems and their components in a virtual environment, enables fast system startup, and make changes in production lines without any need to stop the whole unit completely.

Another use of the Tx Process Simulate is that it is fully integrated with the Teamcenter platform, which allows engineers and designers interactively use data throughout the life cycle of a particular product/project. By using Tx Process Simulate, the creation of models is perceived as creating of 3D objects (machinery, equipment,

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products) in CAD software and their implementation into software environment. Logistics flows are created by definition material and information interconnection between individual workplaces and workers (Siemens, 2018; Siemens, 2017). In this context, it was understood modelling and simulation in dealing with the present case study.

The whole simulation process was carried out in the following steps, which follow in time and logically:

1. problem definition, goals formulation, system analysis,
2. collection and processing of all relevant information,
3. creation models of objects using CAD support and selection objects from the Tx Process Simulate library, resp. by 3D scanning,
4. importation the models and creation of a simulation model in the Tx Process Simulate environment (creation logistics connections between individual elements of the system),
5. verification and testing the simulation model,
6. the realisation of simulation experiments,
7. evaluation and processing of experimental results,
8. acceptation and application of simulation results to the real system.

A generalised optimisation model of logistic flows through modelling and simulation is shown in Fig. 1. For this case study, due to the fact that the company does not yet have 3D laser scanning of production halls and equipment, the lamination workplace was created by 3D modeling using CAD software and then imported into the Tx Process Simulate software environment, as well as using software library objects (Dragic et al., 2016; Xiao et al., 2017; Centobelli et al., 2016).

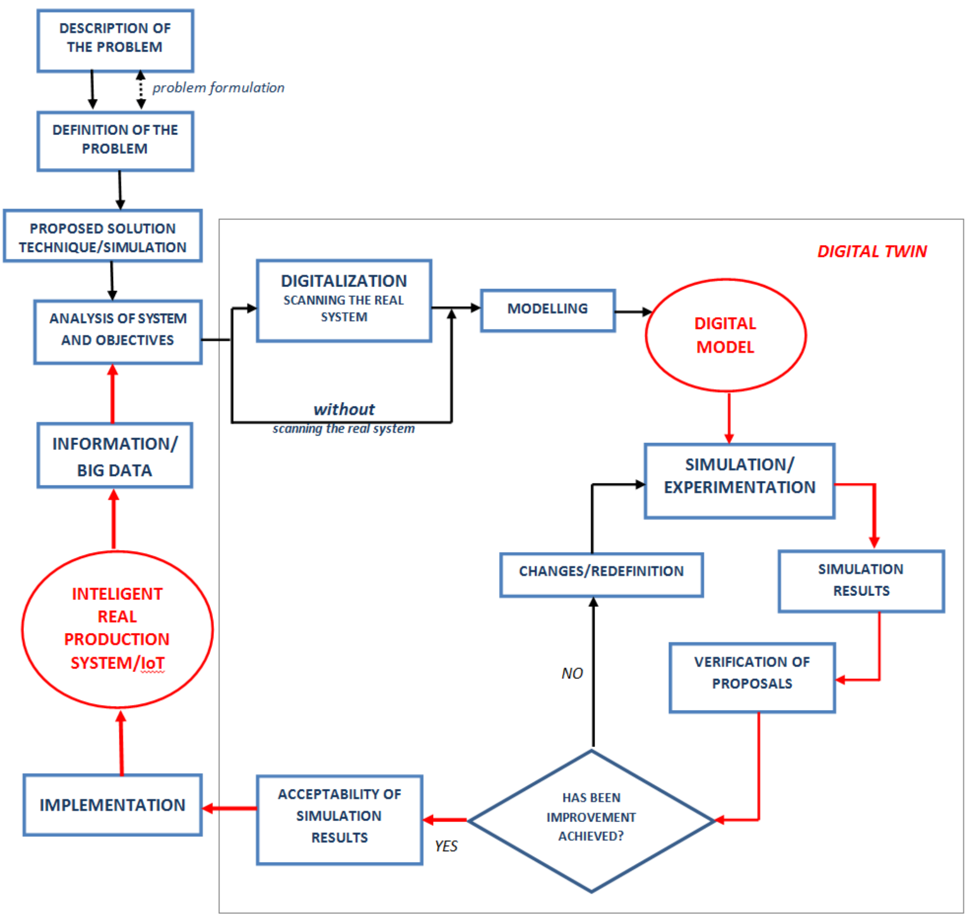


Fig. 1. Flow-chart of the methodology.

Analysis and optimisation of logistics processes took place in the leather processing company for the automotive industry at the lamination site of leather cutouts. The optimisation was focused on logistics processes - inter-operational transport, material handling and material flow. Individual suggestions for changes in logistics processes were verified on the 3D model of lamination workplace in the Tx Process Simulate software module. The process of making leather cutouts is processed in Table 1.

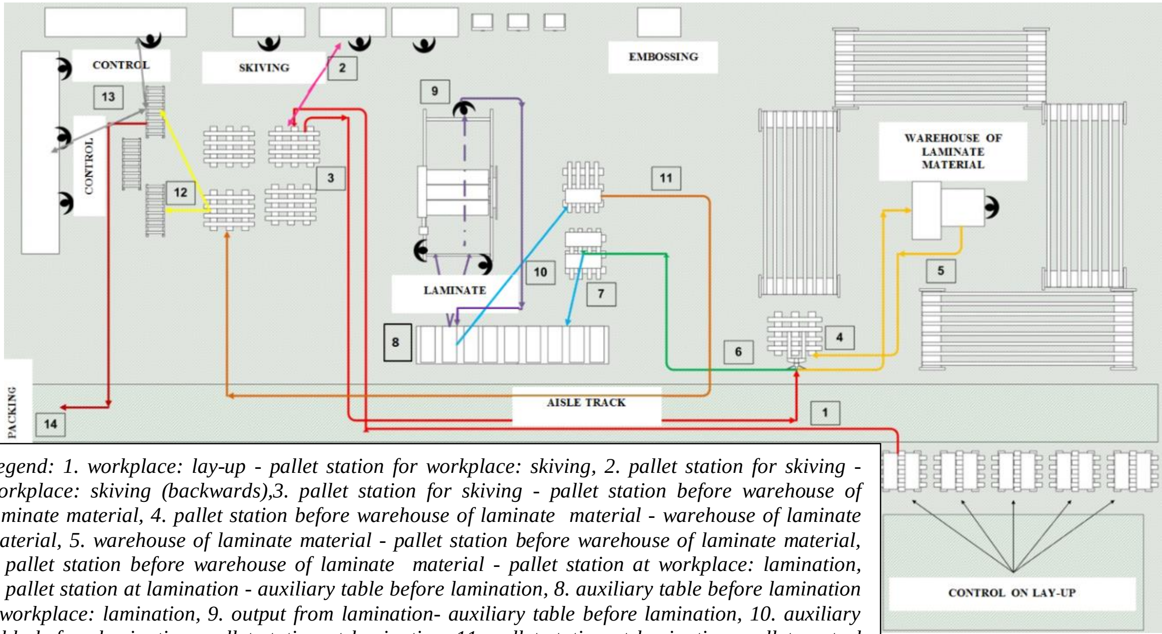
Tab. 1. Technological process of leather processing [Howe, 2016].

Technological process		
1.	Control the skin before cutting 	The basis of the daily production plan are required types and quantities of the leather which are moved from the warehouse to the control workplace for control the skin thickness, drawing, and colour. Damaged pieces are marked with chalk - traces of mosquito stab, lacerations on the skin and others.
2.	Cutting  	On the leather laid on the work table are stowed cut knives with maximum utilisation of the area of the skin, depending on the flaws marked with chalk. The prepared work table is moved to the press for the cutting process (the cutting tolerance is + - 0.5 mm). For the production of leather cutting for the interior of one car is used about 120 - 150 knives are needed. The individual knives are colour-coded depending on the required quality of the cut, 3 types of colours (green - highest quality, yellow - medium quality, blue - lower quality for perforation). Subsequently, the cutting are moved to the control process, where they are sorted into boxes according to the needs of the next process, resp. modifications.
3.	Grinding 	Reducing excessive skin thickness if it exceeds the customer-specified thickness tolerance.
4.	Perforation of cutting 	If the cutting has to be perforated, it proceeds to the workplace: perforation.
5.	Control cutting on lay-up 	Classification of cuttings into cardboard boxes and their control by lay-up - a paper model with the exact specification of the shape and dimensions of leather cuttings at a scale of 1:1. It is control of shape, dimensions, and notches.
6.	Skiving 	The need for skiving, i.e. reducing its thickness is illustrated by a green crosshatch on lay-up. It is realised in order to allow suturing multiple cuttings together when sewing the seat coats resp. head rests
7.	Laminate 	Then follows lamination, i.e., connection, resp. gluing the laminating material to a leather cutting.
8.	Embossing 	Stamping of the logo or other symbols and symbols as required by the customer is done on the embossing press.
9.	Output quality control of leather cutting 	Output control of shape, dimensional accuracy, precision and lamination design and control of other cuttings errors. The output of the manufacturing process is a leather part meeting the above mentioned required quality features.
10.	Product Audit	Random control of cutting, i.e. control of shape, lamination, grinding and natural characteristics of leather parts.
11.	Packaging	Store and package cuttings into boxes and calculation number of individual leather parts. Labels with prescribed data are put on the box and moved to the exit warehouse where they are waiting for export.
12.	Expedition	After the packing process, individual packages are dispatched from the exit warehouse via an external shipping company. Outputs are leather cuttings designed for sewing car covers.

2. Analysis of material flow, transport and material handling at the workplace of lamination

In the case of the analysed workplace of lamination, the input material are leather cutouts embedded in a box and mounted on a pallet. One palette with leather cutouts placed in boxes is considered to be 1 order. A standard representative was chosen for analysis of logistics processes; it means 1 order consisting of 600 pieces of leather cutouts stored in 30 boxes (stored in 20 cutouts):1 order = 1 palette = 30 boxes = 600 pieces of leather cutouts.

As shown in Fig. 2 the material flow is unnecessarily complex, and the material has to overcome unnecessarily long distances, such as in the case of moving the material between the workplace of inspection of cutouts at the lay-up and the skiving workplace, whereby there is a change to reverse the material flow. Based on the above, the table (Table II) lists all non-productive activities performed at the workplace of lamination. These values function as a representative example (1 job = 30 boxes = 600 pieces of leather cutouts).



Legend: 1. workplace: lay-up - pallet station for workplace: skiving, 2. pallet station for skiving - workplace: skiving (backwards),3. pallet station for skiving - pallet station before warehouse of laminate material, 4. pallet station before warehouse of laminate material - warehouse of laminate material, 5. warehouse of laminate material - pallet station before warehouse of laminate material, 6. pallet station before warehouse of laminate material - pallet station at workplace: lamination, 7. pallet station at lamination - auxiliary table before lamination, 8. auxiliary table before lamination - workplace: lamination, 9. output from lamination- auxiliary table before lamination, 10. auxiliary table before lamination- pallet station at lamination, 11. pallet station at lamination- pallet control station for workplace: control of cuttings, 12. cuttings control pallet station - carriage rack, 13. carriage rack - control room (backwards), 14. carriage rack - workplace: packing.

Fig. 2. Material flow on the workplace of lamination.

Tab. 2. Non-productive activities.

Activity	Measured	
	distance [meters]	**total time** [seconds]
move palettes: workplace: control of cutting on lay-up - workplace: skiving	1 x 14.5	45.23
move palettes: workplace: skiving - pallet location for a stock of laminate material	1 x 11.1	39.27
transferring of boxes: pallets location for stock of laminate material - stock of laminate material	30 x 4.3	240.0
transferring of boxes: the stock of laminate material - pallets location for stock of laminate material	30 x 4.3	240.0
transferring of pallets: pallets location for stock of laminate material - pallets location for lamination	1 x 5.2	15.21
transferring of boxes: pallets location for lamination ï table front of lamination	30 x 3.3	164.26
transferring of cutting: output from the lamination - table front of lamination	120 x 6.2	1,130.88
transferring of boxes: table front of lamination - pallets location for lamination	30 x 3.3	164.26
transferring of pallets: pallets location for lamination - pallets location for control	14.7	48.22
relocation of boxes: pallets location for control - carriage rack on workplace: control	60 x 0.82	320
summary	**1,294.7**	

All of these time values represent a non-productive time - transport, material transfer and walking. At the same time, the workers that have to process 1 contract (600 pieces of cutouts), cross with the material 1,294.7 meters. Some of the non-productive activities are performed during the manufacturing operation; it is not possible to consider them in the total processing time of one contract. It is possible to consider only activities that prolong the total processing time of one contract (Tab. 3).

Tab. 3. Non-productive activities prolonging the processing of one contract.

Activity	**Measured total time** [seconds]
move palettes: workplace: control of cutting on lay-up - workplace: skiving	45.23
move palettes: workplace: skiving - pallet location for a stock of laminate material	39.27
transferring of boxes: pallets location for stock of laminate material - stock of laminate material	240.00
transferring of boxes: the stock of laminate material - pallets location for stock of laminate material	240.00
transferring of pallets: pallets location for stock of laminate material - pallets location for lamination	15.21
transferring of the first box from pallet: pallets location for lamination ï table front of lamination	5.48
transferring of the last box from pallet: pallets location for lamination ï table front of lamination	5.48
transferring of pallets: pallets location for lamination - pallets location for control	48.22
relocation of boxes: pallets location for control - carriage rack on the workplace: control	320
summary	**958.89**

Due to the duration mentioned above of non-productive activities, processing of the contract is prolonged by 958.89 seconds (15.98 minutes - non-productive activities, i.e. translating, transporting, handling and transport of material). For calculation of the total processing time of 1 order, the average measured values of the individual production operations for processing 20 pieces of cutouts (1 box) are shown in the table (Tab. 4).

Tab. 4. Duration of production activities for processing of 20 pieces of cutting.

Activity	**Time duration**
skiving of cutting (20 pieces)	4.5 minutes / 1 worker
replenishment of one box with laminating material (20 pieces)	2 minutes / 1 worker
lamination of cutting	427 pieces / 60 minutes / 4 workers
control of cutting from one box (20 pieces)	6.0 minutes / 1 worker

When calculating the duration of individual operations for a representative case (30 boxes = 600 cutouts) and taking into account the number of workers operating, the values are given in Table 5. The scheme for calculating the total processing time of 1 contract is shown in Figure 3.

Tab. 5. Duration of production activities in the processing of cutting from one contract.

Activity	**Time duration**
skiving of cutting	67.5 minutes / 2 workers
replenishment of laminating material	60 minutes / 1 worker
lamination of cutting	84.6 minutes / 4 workers
control of cutting	90 minutes / 2 workers
summary	**302.1** minutes

The main problems resulted from the analysis of logistics processes in the company (material flow, transport and material handling at the workplace of lamination):

1. large transport distances between workplaces,
2. complex material flow (reversion),
3. unnecessary walking and handling of material,
4. the performance of non-value-producing activities,
5. the absence of the possibility to enter the pallet truck into the LM warehouse,
6. poorly placed entrance to the warehouse, resulting in dangerous collisions.

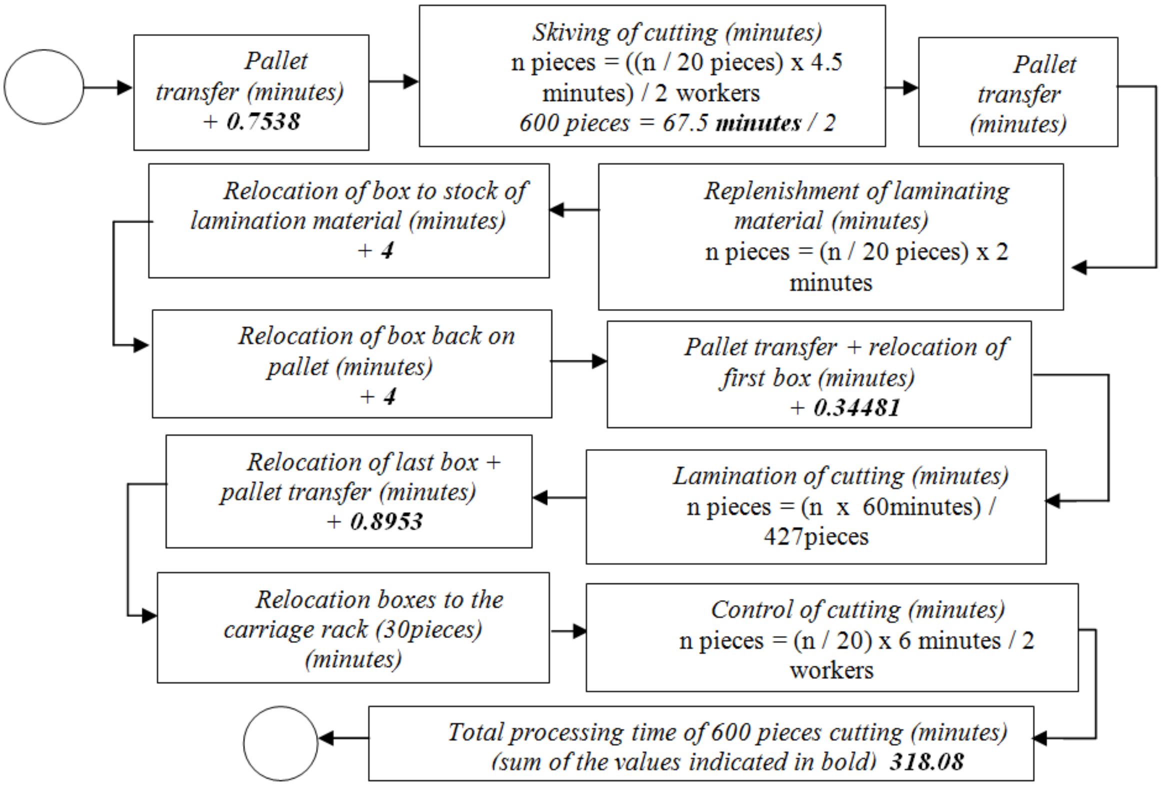


Fig. 3. The processing time of one contract - the current layout of workplaces.

3. The digitisation of the lamination process

The purpose is to analyse the current state of the logistics processes at the workplace of lamination, i.e., to create a digital 3D model at a 1:1 scale and simulate individual operations in Process Simulate software. On this model, it is possible to experiment and look for optimal solutions to the above-defined problems.

After inserting all objects, workers, and simulating all the activities performed, a 3D model of the present state of the lamination work is created at a scale of 1:1, shown in Figure 4.

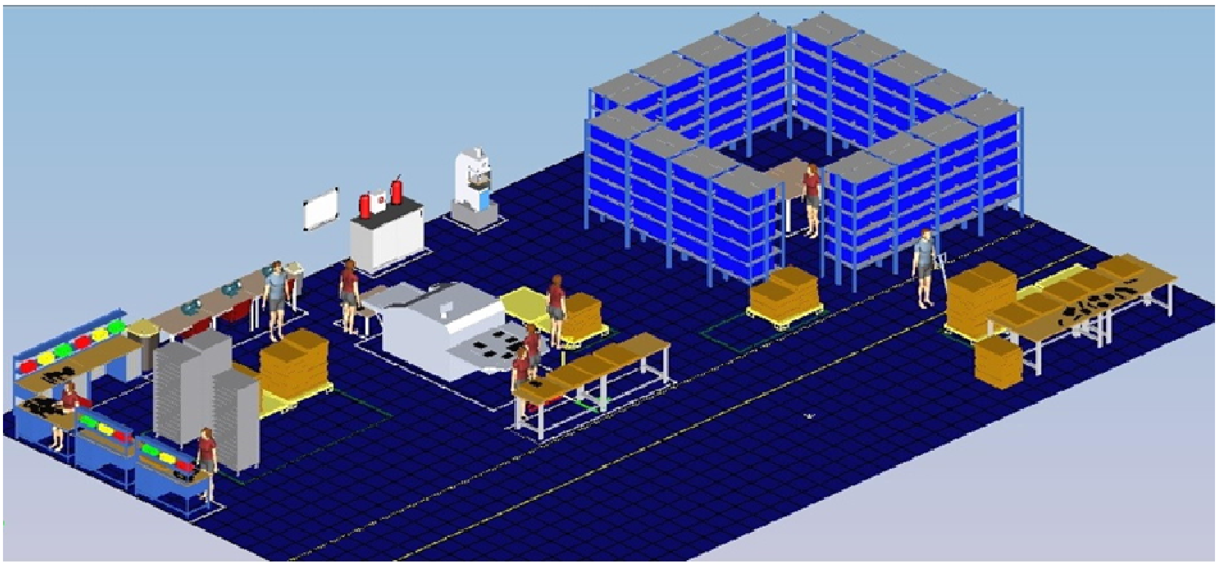


Fig. 4. 3D model of workplace lamination - current state.

Results

1. Design of new lamination workplace in Tx Process Simulate

The failings found by analysis led to the optimisation of the material flow through a new layout of the lamination workplace (Figure 5). A new 3D model of the lamination workplace was created to verify the proposed changes (Figure 6).

In the created 3D model, the proposed changes of the layout were taken into account, as well as the simulated individual activities that resulted from the new layout of the workplace.

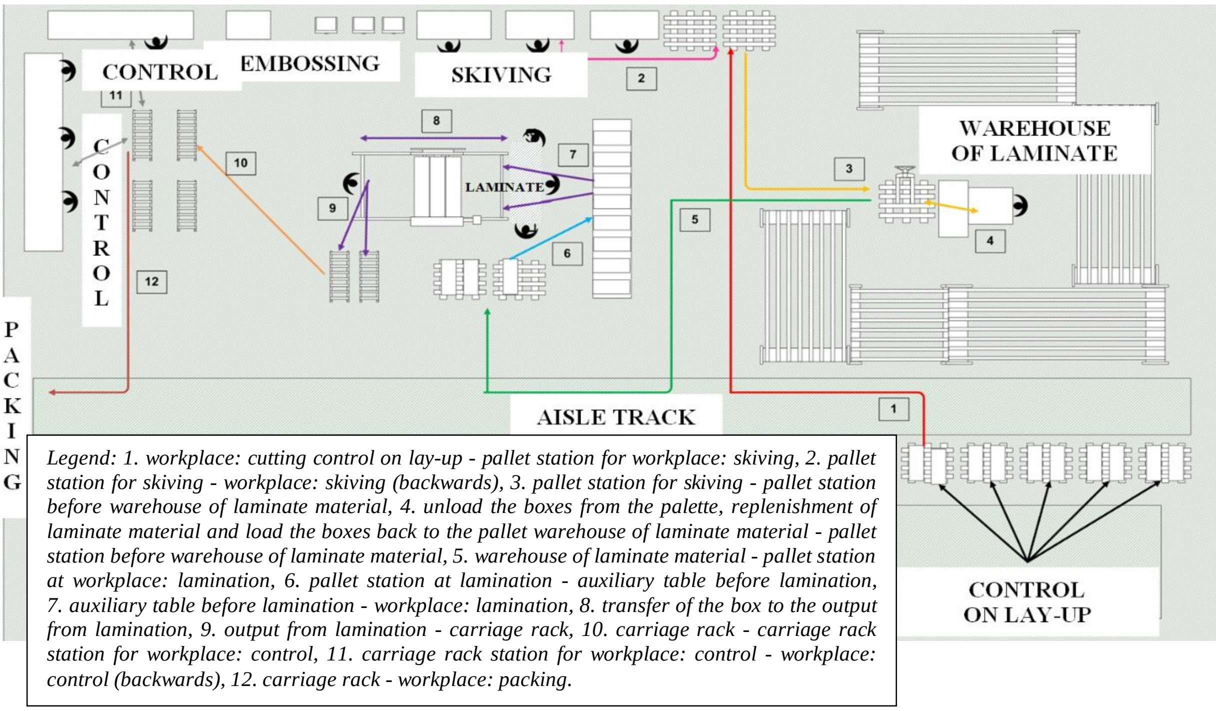


Fig. 5. Material flow for the new layout of workplace lamination.

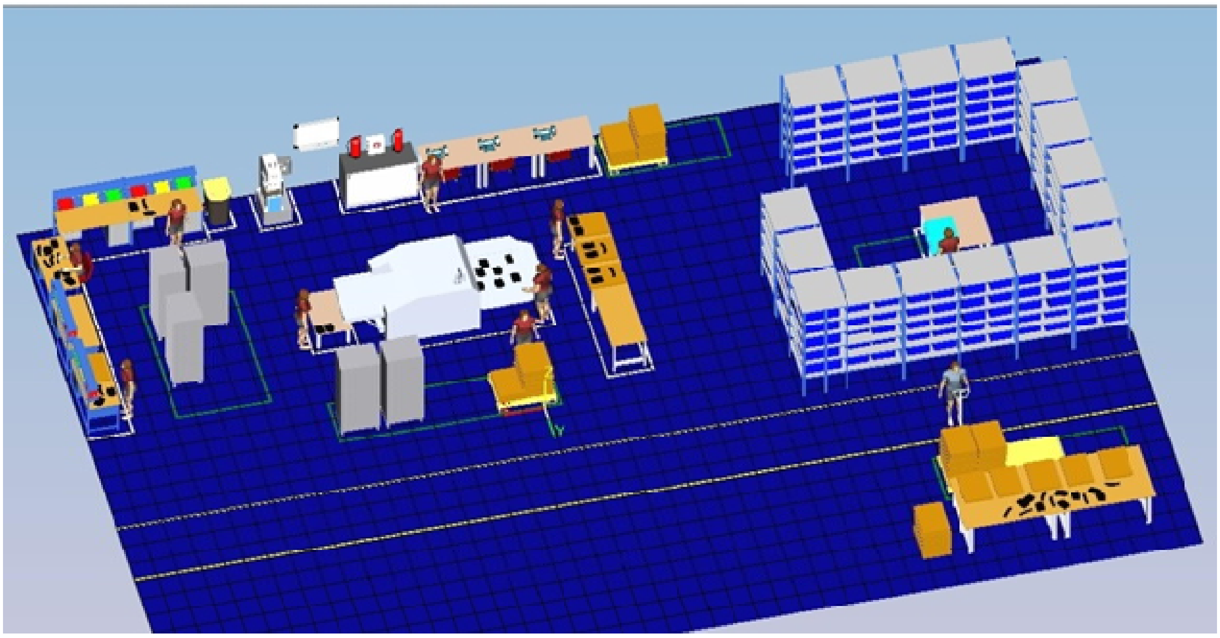


Fig. 6. 3D model of the new layout of the lamination workplace.

2. Achieved effects from simulation in Tx Process Simulate

By changing the layout of the workplace, by simplifying the material flow and by improving the activities of the organisation, the material flow has been optimised in terms of time and distance. Table 6 shows a comparison of the length and the number of repetition of distance passed by workers during their activity. By altering the layout of the lamination workplace, the trajectory was shortened from the original 1,277.37 meters to 578.52 meters. The shortening of the duration of non-productive activities resulting from the new layout was calculated on the basis of shortened transport distances. The duration of individual activities regarding shortening the length of transport distances and the complete elimination of some non-productive activities is included in Table 6.

Tab. 6. Comparison of distance travelled when performing individual activities.

Activity	**Total time** [seconds]	
	current state	proposal
move palettes: workplace: control of cutting on lay-up - workplace: skiving	45.23	34.75
move palettes: workplace: skiving - pallet location for a stock of laminate material	39.27	18.3
transferring of boxes (30ks): pallets location for stock of laminate material - stock of laminate material	240.0	0
transferring of boxes (30ks): the stock of laminate material - pallets location for stock of laminate material	240.0	0
transferring of pallets: pallets location for stock of laminate material - pallets location for lamination	15.21	34.98
transferring the first box from a pallet: pallets location for lamination ï table front of lamination	5.48	4.55
transferring the last box on a pallet: table front of lamination - pallets location for lamination	5.48	3.80
transferring of pallets (carriage rack): pallets location for lamination - pallets location for control	48.22	6.17
relocation of boxes (30ks): pallets location for control - carriage rack on the workplace: control	320	0
summary	**958.89**	**102.58**

To calculate the time savings that can be achieved; it is only necessary to consider activities that prolong the processing of one order. The duration of the activities that prolong the processing of 1 order (600 pieces of cutouts) is shown in the table (Tab. 7).

Tab. 7. Activities prolonging processing of one order.

Activity	**3D model**		
	current state	**proposal**	
	distance [meters]	**distance** [meters]	**time** [seconds]
move palettes: workplace: control of cutting on lay-up - workplace: skiving	1 x 14.02	1 x 11.14	34.75
move palettes: workplace: skiving - pallet location for a stock of laminate material	1 x 10.59	1 x 5.18	18.3
transferring of boxes: pallets location for stock of laminate material - stock of laminate material	30 x 4.42	x	0
transferring of boxes: the stock of laminate material - pallets location for stock of laminate material	30 x 4.42	x	0
transferring of pallets: pallets location for stock of laminate material - pallets location for lamination	1 x 5.18	1 x 11.96	34.98
transferring of boxes: pallets location for lamination ï table front of lamination	30 x 53.04	30 x 2.74	136.39
transferring of cutting: output from the lamination - table front of lamination	120 x 6.15	60 x 5.37	489.74
transferring of boxes: table front of lamination - pallets location for lamination	30 x 3.04	x	0
relocation of boxes: output from the lamination - carriage rack	x	60 x 2.29	227.97
transferring of pallets (carriage rack): pallets location for lamination - pallets location for control	1 x 14.58	4 x 2.11	24.68
relocation of boxes: pallets location for control - carriage rack on workplace: control	60 x 0.79	x	0
summary	**1,277.37**	**578.52**	**966.81**

A comparison of the total duration of non-productive activities at the present state and the proposed state is shown in the figure (Fig. 7).

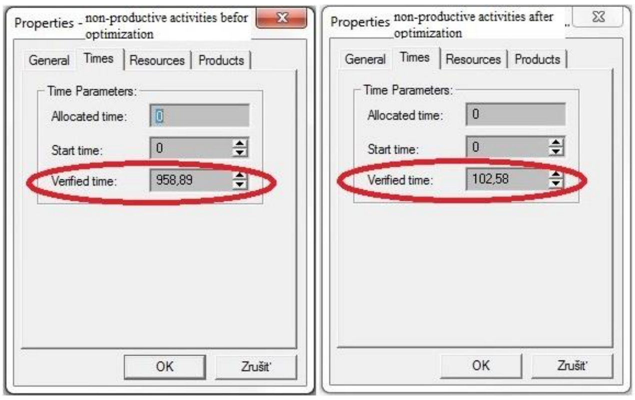


Fig. 7. A comparison of the total duration of non-productive activities.

The best time saving was achieved by removing the non-productive activities of translating boxes from the pallet into the LM warehouse and back, and also by removing the transfer of boxes from the pallet to the transport rack at the workplace of control. Another time saving was achieved by shortening the traffic distance between the workplaces. The time saving for 1 job is 14.27 minutes.

Another time saving is achieved parallel execution of activities, which requires allocation of 1 order into 2 production batches. It means that immediately after loading the rack filled with boxes with laminated blankets, this rack is moved to the station for the transport shelves at the workplace of control. Such a method is not expected to shift the entire order (1 palette) to the control site until all the blank frames have been laminated from one production batch. For comparison: The original total processing time of 1 order (600 pieces of cutouts) is 318.08 minutes. By changing the layout of lamination workplace rationalising the handling operations and dividing the production batch in the lamination process, it is possible to shorten the processing time of 1 order (600 pieces) to 261.34 minutes (Fig. 8).

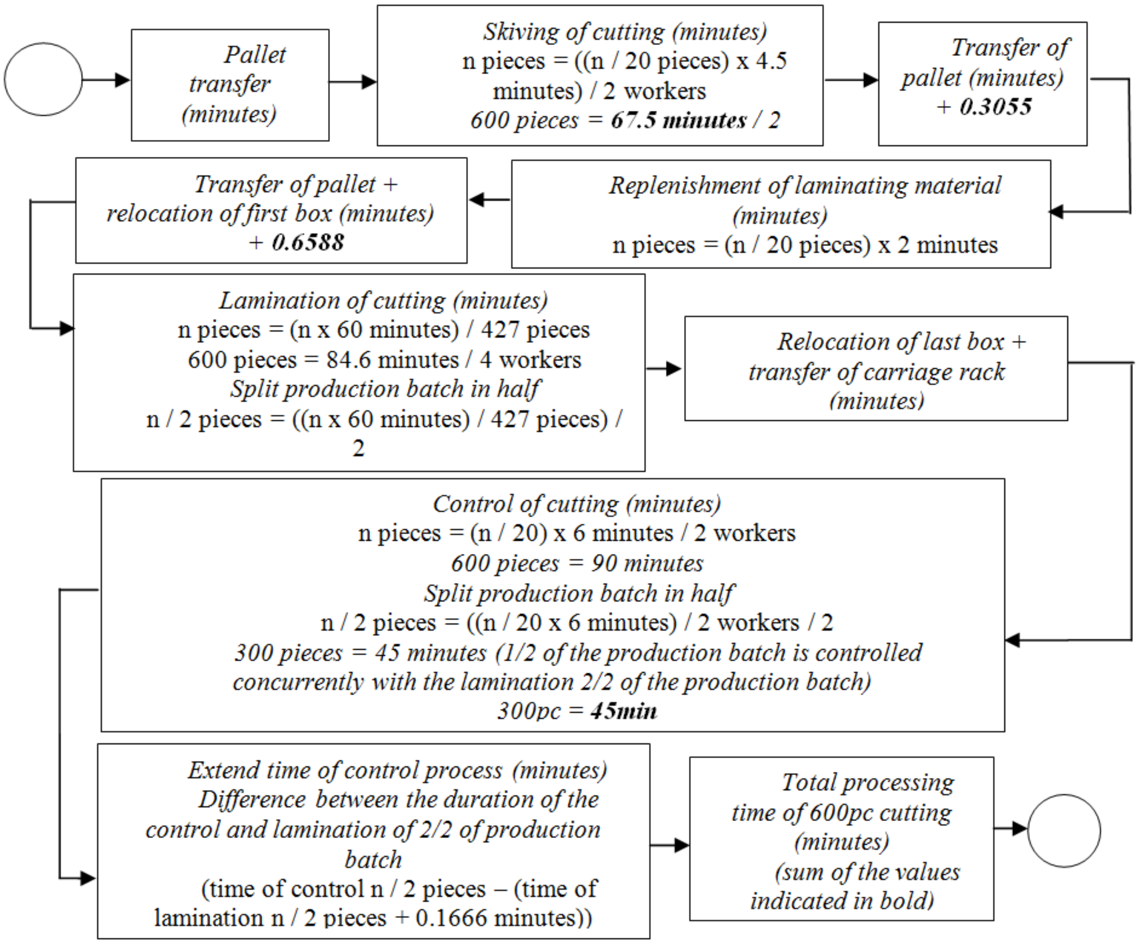


Fig. 8. The processing time of one contract - a new layout of workplaces.

In the original layout, 1 order (600 pieces) is processed in 318.08 minutes. With a duration of 7.5 hours = 450 minutes, 848 pieces are processed, it means 1.413 order. After applying the proposed changes, 1 order (600 pieces) is processed in 261.34 minutes. With a change duration of 7.5 hours = 450 minutes, 1033 pieces would be processed, it means 1722 contracts, what is an increase in productivity by about 20 %. The proposed changes in the layout of the lamination work, the rationalisation of the inter-operational transport, the manipulation activities and the division of the contract for two production batches, it is possible to increase the number of processed leather cutouts by 1 shift about 185 pieces. When considering two working shifts (7.5 hours) during one week (105 working hours) at the size of the order considered (30 boxes = 600 pieces of cutouts), then 11872 pieces would be processed (848 pieces/working shift x 14 working shifts). The same amount of cutouts 11872 pieces would be processed on the basis of the proposal for 86.20 hours. ((11.872 pieces x 7.5 hours) / 1.033 pieces). This means that 11872 pieces would be processed during the calendar week on Saturday, 3:80 p.m. before the end of the second working shift. In total, during 1 week period, there would be processed more about 2590 pieces of cutouts, totally 14462 pieces.

Discussion

Tx Process Simulate software module from Siemens was used for removal detected limitations. By proposing the changes for the layout of the lamination workplace and also by rationalisation of handling activities with the material, the following improvements were achieved:

- simplification of material flow,
- shortening transport distances between workplaces,
- eliminating unnecessary walking of workers,
- removing unproductive activities,
- reduce the total processing time of 1 job in the lamination workplace.

In the quantitative expression of the individual improvements, it was possible to shorten the distance passed by workers in the processing of one contract within the lamination workplace by 698.85 meters. By eliminating unproductive activities and a better organisation of work, the processing time of one contract was shortened from the original 318.08 minutes to 261.34 minutes. Based on the achieved time savings, it could be processed about 185 pieces of leather cutouts more during one working shift (7,5 hours).

Proposals resulting from the Tx Process Simulate application, presented in this case study, have been successfully applied in the selected company. Major importance for the company was the possibility of implementation and evaluation of proposals without interfering the ongoing production.

As the desired results were achieved at the lamination workplace, the possibility of using the module in connection with the digitisation of the entire logistics chain was created.

Hypothetically, reducing the processing time of the cutouts is also possible to achieve with any order size, which will help to increase competitiveness in the context of a flexible response to customer demands. In addition to the aforementioned improvements in the amount of processed cutouts and reducing the processing time, was achieved shortening the distance traveled by workers by handling, moving and transport of material at the intended size of the contract (600 pieces) from the initial 1277.37 m to 578.52 m, it is shortening the route by 698.85 m. Such a significant reduction in the distance travelled by workers will contribute to a decrease in their physical burden, thereby improving the working conditions of workers and it has a positive impact on productivity and work efficiency.

Conclusion

There are many reasons for the implementation of PLM systems into companies. The most important include increasing competitiveness, increasing efficiency, globalisation, innovation, the complexity of products, cost reduction, flexibility, etc. Implementation of PLM systems into companies requires the creation of new algorithms, that is needed to process on the base of experimental verification of the effectiveness by using individual PLM modules on practical examples and then generalize them for wider use in industrial practice (Micieta et al, 2016; Magvasi et al., 2013; Stanek et al., 2016).

Today's world of digitisation requires advanced modelling and simulation tools to design, analyse and optimise production and logistics systems. Modern systems no longer work on the principle of incremental improvements, but they are part of comprehensive quality management of the production system as a whole. It is necessary to ask how to react to this phenomenon and how to become a part of modern technological changes? How to prepare? New generation companies will become intelligent production systems based on the core pillars of Industry 4.0.

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Limestone in V4 countries

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The article is dealing with the issue of limestone in V4 countries, in the countries, Slovakia, the Czech Republic, Poland and Hungary. The limestone is intensely mined because it has a multilateral use: it serves as gravel, building stone, peculiarly coloured varieties are used as a decorative stone; the largest quantity is consumed in the production of cement and lime. In the metallurgy, limestone (high percentile) is an important slag-forming additive; in confectionery is the source of necessary CO_2. At the beginning of the article, are analysed the general properties, formation and composition of limestone. It is followed by mining and preparation and processing of limestone. In the part of the article dealing with production, are parts focused on the production of V4 countries, but also part dealing with the production of concrete products and their use in the industrial sectors. Data collected about export and import of the limestone led to the acquisition of business analyses and the creation of maps for better visualisation of acquired knowledge. In the article are processed data of limestone, its production in V4 countries and export and import, which were statistically processed and evaluated by the test Wilcoxon / Kruskal-Walis. As can be seen from the collected data and its processing, the business has a local character. The biggest producer is Poland, followed by the Czech Republic, Slovakia and Hungary. Biggest exporter of the limestone is Poland; the biggest importer is the Czech Republic.

Keywords: *limestone, V4 countries, production of limestone, export, import, cartographer, Wilcoxon / Kruskal-Walis Tests*

Introduction

Limestone is sedimentary carbonate rock of the pre-amber and recent age, which forms about 15% of the sedimentary lithosphere. Limestone is present in almost all sedimentary geological formations around the world. The major rock component is calcium carbonate (CaCO3) - most often as calcite, rarely aragonite. Limes are often coloured with various admixtures (limonite, hematite, serpentine, organic material, clay minerals). Depending on the method of formation, limestone deposits are divided into sedimentary sea deposits (detrimental, chemogenic, organogenic limestone) and freshwater sedimentary deposits (travertines and sinter). The limestone often occurs along with the dolomite, and it can be a part of this rock. Based on the ratio of calcite and dolomite minerals, respectively clay is classified as limestone, dolomitic limestone or clay-limestone (Petránek, 2018). We analysed limestone in the countries of V4, which are Slovakia, Czech Republic, Poland and Hungary. We took into consideration the geology of limestone in countries, production of limestone, and exploitation of limestone, processing and trade of limestone in the V4 countries. We have been studied trends and possibilities of using raw materials as it is mentioned by (Košťo et al., 2017) and analysed the potential deficit of minerals as it can be seen in (Witkowska-Kita et al., 2017). Mineral industries are well described by (Steblez, 2004).

It is important to take in consideration that non-metallic raw materials are mainly local players, as for example kaolin, like (Lopes et al., 2018) wrote. The theory is support by other authors, (McDaniel et al.,2015).

The utilisation of limestone is in several industries. With industries are connected economic activities, as it was written by (Dikshit, 2014). Processing and finishing of the goods are well described by the authors (Hoda et al.,2013).

As we mentioned above, limestone is used mainly in countries where it is mined or is exported or imported for short distances (Straka et al, 2018). With a similar topic is dealing (Soleimani, 2018). We are in the article dealing with the V4 countries, but good inspiration for us was (Dolia, 2017), describing the situation about raw materials and economy in Ukraine.

The processing of limestone and its utilisation in industries are well written by the author (Baláž, 2006) and (Baláž et al., 2014).

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Material and Methods

We have proceeded in this chapter as follows. We have divided the methodology into the following processes, as can be seen in Figure 1.

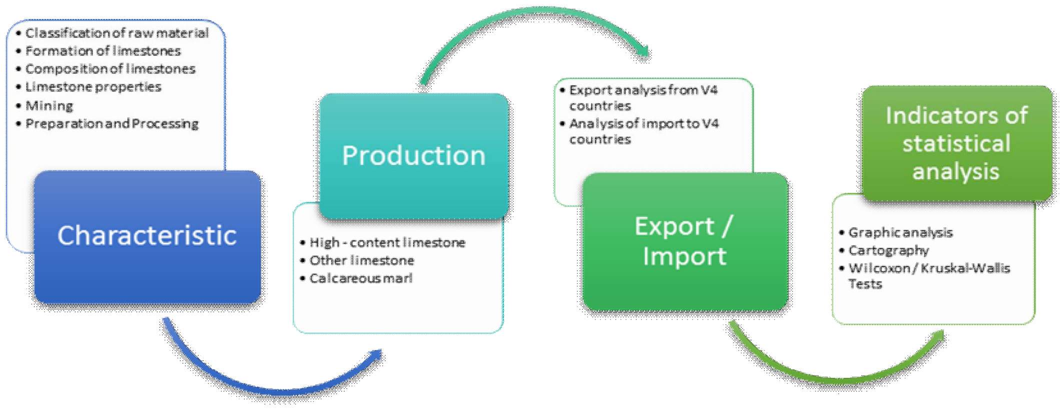


Fig. 1. Methodology and its visualisation (own processing).

We have described individual variables, characteristics, production, export/import and indicators of statistical analyses of the raw material, which is in our case the limestone, through their most important features and processes.

In the text below one can find the facts. Data were processed by statistics methods. We have made nonparametric tests. Nonparametric tests are useful when the usual analysis of variance assumption of normality is not viable. The Nonparametric options provide several methods for testing the hypothesis of equal means or medians across groups. Nonparametric multiple comparison procedures are also available to control the overall error rate for pairwise comparisons. Nonparametric tests use functions of the response ranks, called rank scores. See (Hajek, 1969) and (SAS Institute Inc., 2017).

Note the following:

For the Wilcoxon, test, if the X factor has more than two levels, a chi-square approximation to the one-way test is performed. Performs a test based on Wilcoxon rank scores. The Wilcoxon rank scores are the simple ranks of the data. The Wilcoxon test is the most powerful rank test for errors with logistic distributions. If the factor has more than two levels, the Kruskal-Wallis test is performed. The Wilcoxon test is also called the Mann-Whitney test.

Classification of limestone

Limestone is sedimentary rock forming together with dolomites four-fifths of all sediments on the Earth's surface. It consists of calcium carbonate ($CaCO_3$) (over 80 %), whether in the form of calcite or aragonite. Admixtures are dolomite, siderite, quartz, clay minerals and fragments of fossils. Pure limestone is white (called chalk), various admixtures colours them in grey, red.

According to the place of origin, limestone is divided into:

- Shallow sea limestone occurs rarely. Sediments are composed of limestone sand formed from fragments of shells, which are associated with limestone mud.
- Deep-water limestone is divided into two groups:
  1. Turbid limestone
  2. Pelagic limestone

Turbid limestone is not very widespread; they are known from older formations. They occur at the mainland coasts, where the undersea sediments are merged into the sea and processing by the turbulent current.

The sedimentary environment of pool limestone is in deep ocean basins, where limestone sludge is deposited in the depth of 2,000 to 3,600 m. At greater depths (over 4,000 m) the lime sludge dissolves again.

Limes associated with evaporating - small accumulations of limestone also occur when evaporating temporary lakes in desert areas. The layers of these rough, uneven layers are called caliber and calcretes. The auxiliary minerals are evaporated, especially gypsum and anhydrite.

Freshwater limestone is created in lakes in the form of lake chalk, which is actually a clay lime sediment. Another type of freshwater limestone is oncoids produced by algae with a strong porous texture. The last type is travertine, which is formed by precipitation from thermal sources.

Eolic limestone is created when developing humid fragments from coral reefs on the shores of the seas and subsequent formation of solid layers, which are also called eolianites.

The process of limestone formation is multiple and takes place in different environments from the surface to the depth. During the conversion of mud to limestone, six basic processes are in progress: cementation, micritisation, neomorphosis, dissolution, compacting and dolomitization.

The formation of limestone

Limestone is an organogenic sedimentary rock formed from calcareous shells of living organisms. The shells of marine animals and their skeletons contain calcium that has been extracted from the seawater during their lifetime.

Limestone was created from:

- shellfish,
- the shells of other microscopic organisms,
- Coral shelves.

Shells of dead corals create under living corals layers of hundreds of meters. The shells of dead molluscs are gathered in the shallow seas. The shells of unicellular organisms (plankton) accumulate at the bottom of deep seas.

Composition of limestone

Limestone is a solid sedimentary rock, which is predominantly composed of calcium carbonate. It accounts for more than 80 %; the remaining part is less than 20 %, for example, dolomite, siderite, quartz, clay minerals and fragments of fossils. Limestone rock is forming together with dolomites four-fifths of all sediments on the Earth's surface.

Limestone contains calcite grains and admixtures of clay or organic substances. We know more species of limestone according to the composition. The limestone can be, for example:

- Lumachel limestone - formed mainly from the shells of molluscs,
- Limestone - formed from microscopic organisms,
- Coral limestone - formed from coral.

Changes begin immediately after storage or even during storage, and may take place both in the marine and terrestrial environment. The absolute majority of limestone is of biochemical or organic origin.

Cementation and micritisation occur on the seabed, where cementation is more characteristic of the active environment and micritisation for the seawater. At higher depths, dissolution of calcite occurs. Dissolution depends on the degree of water saturation by calcite, while in cold waters this saturation is smaller. Thus, with decreasing depth, it decreases. It is caused by fresh water, while dissolvation, cementation and soil producing happen. Overlapping with other sediments - compacting (limiting) of the limestone layers happens by mechanical grain saturation in the structure or by chemical reactions. The limes are predominantly composed of calcite, alternatively of aragonite. Calcite is involved in the composition of primary and secondary limestone components.

If the rock-forming organisms make their shells of calcite, it is quite pure, and in the case of aragonite, there are magnesium compounds. If they are high (more than 4-mole percentage $MgCO_3$), we talk about high-magnesia calcite. However, high Mg content is unstable in calcite, therefore older, low algal limestone has low magnesium content. High-magnesia calcites are used for shells, for example, cephalopods and corals.

Aragonite is also unstable, changing to calcite, so it only contains recent limestone. Dolomite in limestone is not primarily present; it is only a result of later metasomatic processes. Siderite is a rare addition to limestone. It occurs in formations in which limestone and siderite iron ores are associated. Of the accessory minerals, which is present in the form of chalcedony spherolites, is of greater importance.

Kaolinite and feldspar are presented from other silicates. Other minerals (phosphates, glauconitic, gypsum) are not a common accessory, and their occurrence is due to specific sedimentation conditions.

Features of limestone

Limestone is polygenetic rocks. A large part was originally mechanically moved and deposited similar to other classical sediments; others were chemically precipitated from water. That is why their textural characters are different. The first group has hydrodynamic textures, while the other is very specific and depends on the environment.

The textures formed by the hydrodynamic model are the same as in the classical sediments - sloping layers, gradation layering, and features on the surface. Paleocarous surfaces are formed by dissolution with bottom water. Irregular shapes, cavity surfaces with cavities are formed. The fenestral texture is a generic name for carbonate cavities, filled with sparites).

Stromatakis is formed in shallow waters and has large irregular cavities. Tuber textures are created in shallow water limestone - they are actually traces of animal bruising. The limestone has approximately the same hardness, good acid solubility in the release of CO_2.

It also dissolves in water, which contains CO_2 to form calcium bicarbonate ($Ca(HCO_3)_2$). This phenomenon is the cause of karst phenomena such as caves, slopes and others. Karst is the name for a limestone area, where erosion factors have occurred to remove the humidified shell and depletion of rock massifs. Due to the chemical exfoliation, the original geological structure and the formation of typical karst formations are disturbed here. These are divided into primary and secondary.

The quality of the limestone found in the deposit depends on its long-term development and the surrounding environment. These features affect the amount of non-carbonate minerals (impurities such as clay, quartz sand, etc.) in the sediment. When the limestone reaches the earth's surface, it can contain impurities from rainwater. Earthquakes and other natural phenomena cause rifts in the limestone and by these can get inappropriate minerals into it and remain there forever.

Within a single quarry, limestone layers may have very different properties and changes in the course of the deposit.

Colour: Limestone can be white, grey, dark, and sometimes red.

Exploitation of limestone

There are several activities at the same time.

Blasting works are planned on the basis of a mining plan - it determines where, when and what methods can be used. If necessary, it is possible to carry out selective blasting or, conversely, mix the material from the blasts in different areas. It is necessary to ensure responsible use of inventories and at the same time satisfy customer requirements.

Mining usually begins with drilling and blasting, followed by loading and transporting the ground rock into the primary crusher. The quality and the enormous amount of stone needed for the treatment in the processing plant is ensured by a carefully designed blasting program. This is an essential step in the process.

In the quarries where blasting is used, the drilling rig drills holes for explosives. Non-crushed stone is commercially utilised in another way or used to fill the pit and reclamation the site after the quarry has been closed.

Preparing and processing of limestone

Stone adjustment consists of primary and secondary crushing. The product is then classified by a fraction. If necessary, washing of the stone can be done with recycled water, usually obtained from settling tanks. After crushing, the stone is placed on the deposition area to ensure a constant supply of stone for subsequent processes.

For lime and dolomite lime production, rock fractions greater than 20 mm are usually sent to blast furnaces. Limestone which is not intended for industrial calcination is usually modified for further applications.

During the production of limestone with a high content of calcium for the chemical industry or dolomitic limestone for industrial use, the rock either mills or dries and mills, thus obtaining the purity and desirable characteristics of the markets, whether in the form of stones or milled products.

The core of the transformation is the kiln, that can transform the crude carbonate rock on lime oxide or dolomite lime, which offers a wide range of usable properties.

Chemical transformation requires a considerable amount of energy and these large industrial facilities. Different types of technologies are used especially rotary and vertical furnaces.

Rotary furnaces produce lime with controlled reactivity and provide complete decarbonisation with very low residual CO_2. These furnaces are capable of providing products with the particular specification, for example, lime with low sulfur or special reactive lime and also dolomite lime. This helps improve resource efficiency.

Once the lime is pulled out from the kilns, it should be sorted and stored according to the particle size, residual CO_2 content and other physical and chemical properties determined by the laboratories. Classification is carried out using conveyor belts, elevators, crushers and sorters. According to particle size, lime is divided into three main categories: pieces crushed and ground lime.

Lime can be further processed to obtain new features and meet the needs of the latest applications. There are three main processes for lime processing:

1. CRUSHING AND MILLING - Lime is crushed in the following types of mills equipped with graders: hammer mills, rotary ball mills, vertical pendulum mills and pelvic mills.
2. HYDRATION - forms dry powder - calcium hydroxide $Ca(OH)_2$. The hydrator consists of one, two or three hydrating chambers and a mixer where the water starts to react with the starting line. Excavated lime is distributed by particle size and stored in strength.
3. PRODUCTION OF LIME MASH - Lime milk is industrially produced in stirred reactors, either from burned or slaked lime, in the presence of more water. The final product is slurry of lime.
4. According to usability is limestone and cement materials divided into:
   - High-content limestone ($CaCO_3$ content> 97%), or high-grade limestone
   - Another limestone,
   - Calcareous marl.

High-content limestone is a material used in:

- Metallurgy (agglomeration);
- Chemical industry (production of cellulose, chlorine lime, soda, carbide),
- Rubber industry,
- Food industry,
- Glass and ceramics industry (filler, enamel flux, glaze preparation)
- Building industry (production of lime and some types of building materials).

Limestone with less quality is used in:

- Agriculture (soil limestone - a reduction of acidity, fertilisation, production of compound feed)
- Building industry (building and decorative stone, building materials).

Limestone is mainly used for the production of a cement material (Petránek, 2007).

The limestone is intensely mined because it has a multilateral use: it serves as gravel, building stone, peculiarly coloured varieties are used as a decorative stone; the largest quantity is consumed in the production of cement and lime. In the metallurgy, limestone (high percentile) is an important slag-forming additive, in confectionery is the source of necessary CO_2 etc.

Limestone material is not recycled, respectively. Is recycled secondary to some products (glassmaking, building industry, etc.). in agriculture limestone can be replaced by dolomites, burnt lime, etc. Various carbonates and their mixtures can be used in the case of desulphurization of the gases.

Limestone and dolomites replace each other by neutralising acidic waters, soils, gases, or they can be replaced by natural and synthetic zeolites or anaerobic bacteria (biological technologies).

In some industries, however, there is no adequate substitution for limestone (production of cement, lime, production of blast furnace raw iron) (Petránek, 2007).

Results

Production in the Slovak Republic

Limestone belongs to the most used raw materials in Slovakia. The consumption of limestone is covered entirely by domestic mining, which exceeds 7 million tons per year, of which about 40% are high-percentage limestone. The most important producers of high-grade limestone are Holcim, JSC (Deposit Vajarská) and Carmeuse Slovakia, Ltd. (deposit Slavec - Gombasek). In the mining of another limestone, the leading position belongs to Carmeuse Slovakia, Ltd. (deposit Vļeláre). Other important producers are also Dobývanie, Ltd. (StráŔavy) and Povaģská cementárŔa, JSC (Ladce). Limestone is mainly used as a material for the production of cement and lime. Cement is the most important Slovak export commodity based on a mineral base. Limestone mining is maintained at a relatively stable level (Baláģ, 2006).

In 2016, the limestone material, cement materials and dolomite reached the second highest revenue per employee. Labour productivity and efficiency are one-third higher than the average for total mining. The share of enterprises classified in the CZ-NACE B division, Mining and quarrying of raw materials is about 40% in this material measured by the share of revenues. According to the assessment of absolute indicators, it is a medium significant material in the number of organisations and revenues. According to other absolute indicators, limestone is less important raw material. Between 2012 and 2016, most of the absolute indicators have improved to added value.

According to relative indicators, limestone is rather a low evaluated material. However, the value of the revenue per employee indicator, due to the link with non-taxed businesses is significantly above average. In terms of dynamics between 2012 and 2016, relative indicators are declining in practice. In 2013, there was a significant decline and growth in 2014 (Starý J. et al., 2017).

Production in Poland

Limestone with a content of $CaCO_3$ above 90% is a mineral used in lime. Variants that have other criteria are used in the chemical industry, metallurgy, sugar industry for lime meal production, including sorbents for flue gas desulphurization. Limestone as a material is used on a cement clinker product that requires the addition of raw materials from the clay. Limestone and spittle are minerals that are only useful in the cement industry. The $CaCO_3$ content may be significantly lower in this case (less than 80 %). However, the content of other chemical constituents and the proportion of their percentages is important (Brzezinski D., 2013).

Production in the Czech Republic

Limestone in the Czech Republic is industrially very important, whose reserves seems to be large, but in fact they are heavily limited by the fact that a significant part of their reserves are located in protected areas (for example the Bohemian Karst, Moravian Karst) or areas thanks to large morphological clusters of landscape, botanical or zoologically attractive. Sources of limestone can be found in many places in the Czech Republic. The most significant areas are Devon Barrandien, the most important and largest bearing area of the Czech Republic with deposits, Konoprusy, Kozolupy-stains in the structure, Kosor-HvízŅalka, further Moravian devon, as the most important area in Moravia with deposits, for example, Mokrá pri Brne, Hranice-Ļernotín; other important deposits are Prachovice, Vitoġov-Lesnice, Vtramberk and Úpohlavy (Starý J., Kavina P., Vaneļek M., 2009).

Resources of limestone in the category of industrial sources reach about 150 yearly life at the current level of mining, which traditionally represents about 10 to 11 million tons per year, resulting in that limestone is being the most widely used raw material. The mining of limestone and cement raw materials are generally directly linked to the processing industry, which has a long tradition in the territory of the Czech Republic (lime and cement in particular). These investment units have been in the past an important item of Czech foreign trade and have been successfully delivered to many Third World countries.

Due to this tradition, the early 1990s, the rapid and successful privatisation of Czech prosperous businesses happened, which were also taken by German and French investors as well as a well-studied raw material base. Also, a significant Swiss investment was made in the Czech Republic for the mining and processing of super pure calcite suitable for microfilm for the production of fillers. In the last decade, the decline in the production of ground limestone for agricultural purposes has been observed, due to a lack of funds for agro-technical purposes (MPaO, 2012).

Production in Hungary

Hungary is also known for its limestone quarries, most of which are located in mountainous areas like in the Bükk and Upponyi Mountains, in Bélapátfalva, Bükkzsérc, Cserépfalu, Eger, Egerszalók, FelsŖtárkány, Miskolc-DiósgyŖr, Miskolc-Omassa, Miskolc -Tapolca, Mónosbél and Szilvásvárad, in the Pilis and Budai Mountains (Budlakalász, Budapest III.ker., Kesztölc, Pilisjászfalu, Pilisszántó, RemeteszŖlŖs, Sóskút), Velencei and Szabadbattyáni-rög (Polgárdi) (Csövár, Keszeg, Vác), Gerecse (Bajna, Csolnok, Dorog, Héreg, Lábatlan, Tardosbánya, Tatabánya), Mátra (Eplény, Hárskút, Sümeg, TapolcafŖ, Ugod) Sirok), Mecsek (Pécs, Pusztakisfalu, Versend), Villányi (Beremend, Csarnóta, Nagyharsány, Siklós - MáriagyŤd, Villány) (Nagy, 2018).

The Komarnian region is known mainly for the exploitation of red marble, basalt, coal mines and limestone quarries. Marble and limestone mines on the western side of the Gerecse Mountains in the eastern Komarno are known for centuries. Limestone material extracted from this area is a solid and hard limestone that is suitable for treating. Between years 1850-1855, the stone material was transported to the fortification of the Komárom castle; and later in the imperial palace in Vienna, where the entire basement from top to bottom consists of stones carved in Hungary. Marble and limestone materials were also transported to state buildings in the countryside; specifically for the construction of the church of St. Stephen in Budapest, New House, Harbor Bridge, the church of Budin and the building of the Ministry of Finance. In the area of Tardos and Dunaaaslm, there are limestone quarries where there is an average of 200 workers and where the production is about 1500-1800 m^3 per year (Borovszky, S., 2010).

Limestone is used in various forms in the metallurgical, agricultural and sugar industries. The limestone meal is used in the agricultural industry and contains minerals that improve soil properties (Dömsödi, 2010).

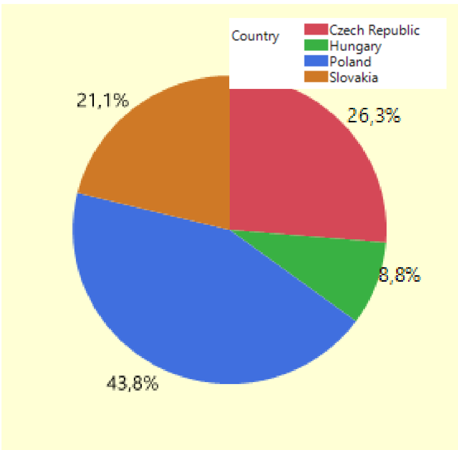
The limestone mining in Hungary from 2003 to 2013 had a decreasing trend. While in the year 2003 ï 2008 was mined 500 thousand tons of limestone, which represented a constant value. However, since 2009, mining has declined. In 2010 mining fell to almost half (less than 260 thousand tones), in 2013 it was less than 200 thousand tons per year.

One of the reasons for the decline in mining and production is the non-renewable of limestone deposits, as well as the replacement of limestone with other materials. Hungary has high-quality mineral resources, mainly building materials, but the building industry has not experienced any major boom in recent years, and for this reason, also the mining of construction materials has declined significantly in recent years (Horn, J., 2013).

Production in V4 countries

By analyzing the production of limestone in the V4 countries in 10 years it is possible to sort the countries according to the importance of the rate, the biggest producer is Poland, which produces 43 % of the raw material, the Czech Republic and Slovakia produce over 20 % of the raw material and the lowest volume of production up to 10 % is Hungary, see Figure 2. Within 10 years, there may be a slight decline in production in the V4 countries (Fig. 3), but this does not affect the long-term.

Structure of V4 production



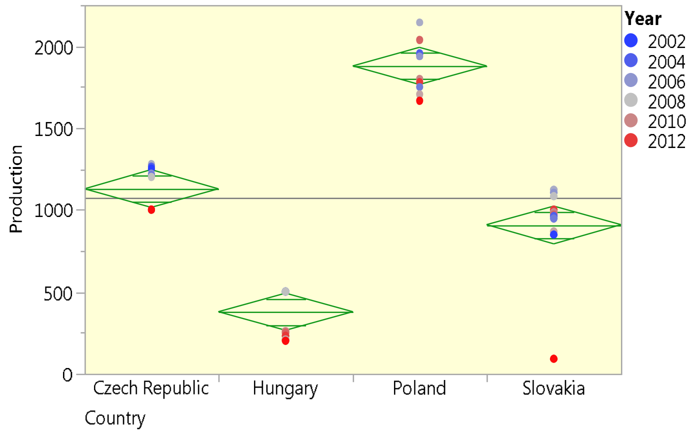


Fig. 2. Structure of limestone production in V4 countries. *Fig. 3. One-way analysis of limestone production for 10 years.*

(Own processing according to www.indexmundi.com, both figures Fig. 2 and 3.)

Analysis of limestone sales prices

Following the analysis of the production of limestone in the V4 countries and then the analysis of import and export in volume units (tones), the development of these indicators was evaluated with regard to the sales prices of the raw material, which has an impact on the economic balance of individual countries.

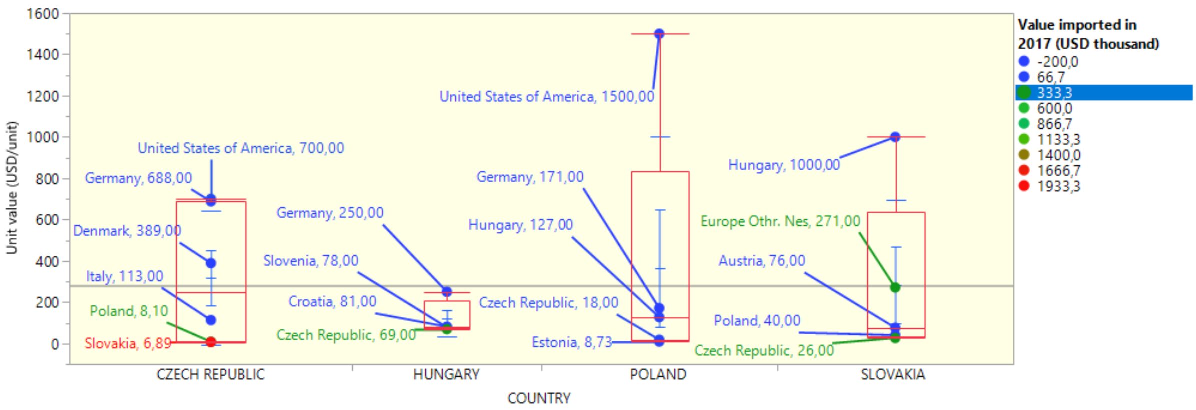
- The sale and purchase price of limestone has a large spread, ranging from \$ 6.89 / ton up to \$ 1,500 / ton for imports and a range of \$ 5.9 to \$ 3000 / ton for exports (see Table 3). The price level of this raw material is derived from several variables, e.g.:
- quality
- utilisation,
- quantity.

Tab. 1. Overview of limestone prices in the international trade for 2017 in the V4 countries (own processing according to www.indexmundi.com and www.trademap.org)

Country	Export		Import	
	Min	Max	Min	Max
Czech Republic	19	667	6,89	700
Hungary	186	214	69	250

Country	Export		Import	
	Min	Max	Min	Max
Poland	6,78	1000	8,73	1500
Slovakia	5,9	3000	26	1000

The observed variance of values in the raw material price levels was analysed by the Wilcoxon / Kruskal-Walis Tests, where we examined the price variability of the importing country. The test results did not confirm the statistically significant variability in import prices due to the landscape (Fig. 4).

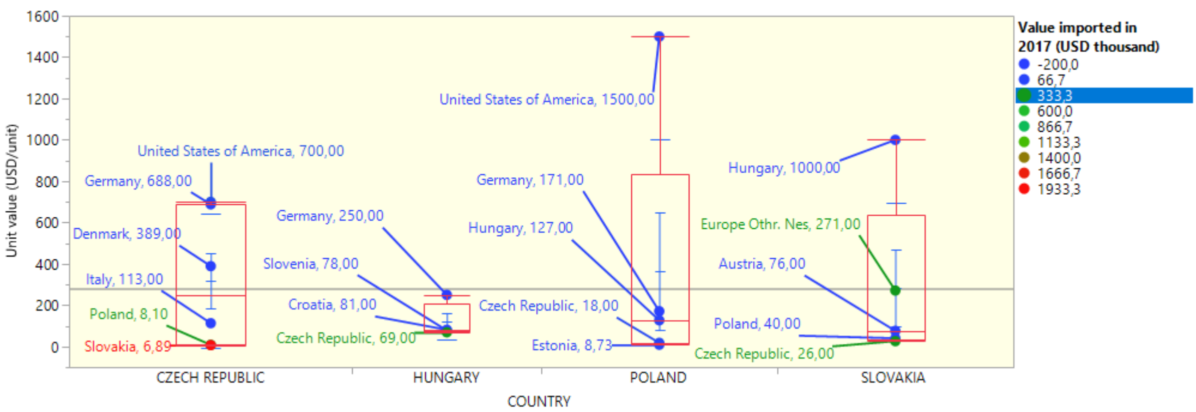


Level	Count	Score Sum	Expected Score	Score Mean	(Mean-Mean0)/Std0
ČR	6	66,000	60,000	11,0000	0,482
HUNGARY	2	31,000	20,000	15,5000	1,395
POLAND	7	61,000	70,000	8,7143	-0,718
SLOVAKIA	4	32,000	40,000	8,0000	-0,750

1-Way Test, ChiSquare Approximation

ChiSquare	DF	Prob>ChiSq
2,9707	3	0,3962

Fig. 4. Analysis of Import Price Variability in V4 Countries (own processing according to www.indexmundi.com and www.trademap.org).



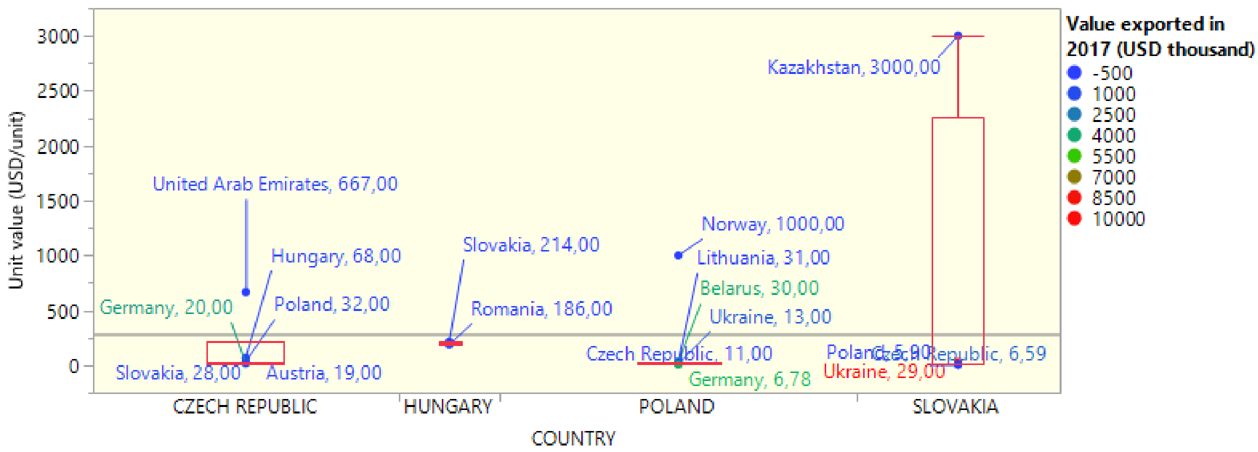
Level	Count	Score Sum	Expected Score	Score Mean	(Mean-Mean0)/Std0
CZECH REPUBLIC	6	65,000	63,000	10,8333	0,124
HUNGARY	4	40,000	42,000	10,0000	-0,142
POLAND	5	52,000	52,500	10,4000	0,000
SLOVAKIA	5	53,000	52,500	10,6000	-0,000

1-Way Test, ChiSquare Approximation

ChiSquare	DF	Prob>ChiSq
0,0505	3	0,9970

Fig. 5. Analysis of export price variability in V4 countries (own processing according to www.indexmundi.com and www.trademap.org).

Through the Wilcoxon / Kruskal-Walis Tests, price variability in the exporting country was also analysed. The test results, as in the previous case, did not confirm the statistically significant variability in export prices due to the landscape (see Figure 5). It cannot be said that any of the countries would have significantly lower / higher prices for exporting/importing limestone.



Level	Count	Score Sum	Expected Score	Score Mean	(Mean-Mean0)/Std0
CZECH REPUBLIC	6	66,000	60,000	11,0000	0,482
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SLOVAKIA	4	32,000	40,000	8,0000	-0,750

1-Way Test, ChiSquare Approximation

ChiSquare	DF	Prob>ChiSq
2,9707	3	0,3962

Fig. 6. Analysis of import price variability in V4 countries (own processing according to www.indexmundi.com and www.trademap.org).

Foreign trade - limestone and cement

The groups of non-metallic materials, which are imported and exported in significant quantities, also include limestone and semi-products made of limestone - cement and lime. One of the main reasons for the import and export of these commodities in recent years has not been such a shortage or surplus of commodities, rather than economic reasons. In the case of limestone, cement and lime, to a certain degree of simplification, it was true, that cheaper Slovak or Polish production (alternative) was delivered to the Czech market, while somewhat more expensive Czech production was offered to German, the Austrian market.

The quantity of limestone import shows significant fluctuations in recent years - only between 2001 and 2011 ranged from 170 to 570 kt per year.

The characteristic feature of imports of raw materials is that 99.9 % come from Slovakia and the average import price is significantly lower than the prices for which limestone from the Czech Republic are exported. Exports of limestone have been long in the range of 85 to 270 kt. per year with a value of 40 to 110 million CZK and are mainly exported to Germany, Poland, Austria and other countries.

The quantity is an even more significant item of cement, whose import reach an average of 0.7 to 1.3 million tons worth of 1.3 - 1.9 billion CZK per year. Until 2002, imports of cement from Slovakia were completely dominated; in recent years imports of cement from other neighbouring countries - Germany and Poland - have also increased. Cement exports from the Czech Republic are currently about 0.6 to 1.7 million ton per year, which is significantly less than in 1999 when exports reached three times more. Cement exports are mainly to Germany, less to Poland, Slovakia and Austria. In the case of cement, it is not surprising that the import prices of cement are lower than export prices, due to the fact, that the German import prices of German cement are relatively high.

The default lime trade has similar properties to limestone - most of the lime is imported from neighbouring Slovakia in volumes ranging from 100 to 120 kt. per year for an average of 10 to 20 % below the average export prices of 150 to 200 kt. limes that are used in the Czech Republic. Export is mainly directed to Germany and in a significantly smaller amount back to Slovakia. In financial terms, the export volume is approximately 250-300 million CZK per year. In 2009, foreign trade of lime was lost on both sides (imports of about 90 kt, export of about 125 kt.), which indicates a decrease in demand. In the years 2010-2011, the volume of LU with lime returned to the range usual in recent years (2010: about 105 kt., 150 kt., Exports: 105 kt., 180 kt. (MPaO, 2012).

The result from analysing of limestone import to the V4 countries

In the analysis of the imported volume of limestone in tones, the quantity of imported limestone was assessed in the countries of V4 - Czech Republic, Slovakia, Poland, and Hungary in 2017. We recorded the largest volume of imports in the Czech Republic with a volume of 708 231 tones and the smallest in Hungary with a volume of 12 742 tones (Tab. 2). The import of this raw material in the V4 countries is provided in the closer territory mostly from the neighbouring countries as seen in Fig. 7. In the Czech Republic, 70 % of imports are provided by Slovakia, followed by Poland, Germany and Italy. The Hungarian, Polish and Slovak imports mostly cover the Czech Republic, which covers 50-90 % of these countries' imports, followed by Germany, Austria and other countries.

Tab. 2. Overview of Limestone Import in the V4 countries in 2017 (own processing according to www.indexmundi.com and www.trademap.org).

COUNTRY	Imported volume of limestone in tones 2017			
	Sum	Mean	Min	Max
CZECH REPUBLIC	354116,00	59019,33	54,00	293110,00
HUNGARY	6371,00	1592,75	12,00	5951,00
POLAND	14734,00	2455,67	0,00	7249,00
SLOVAKIA	22482,00	4496,40	13,00	20527,00

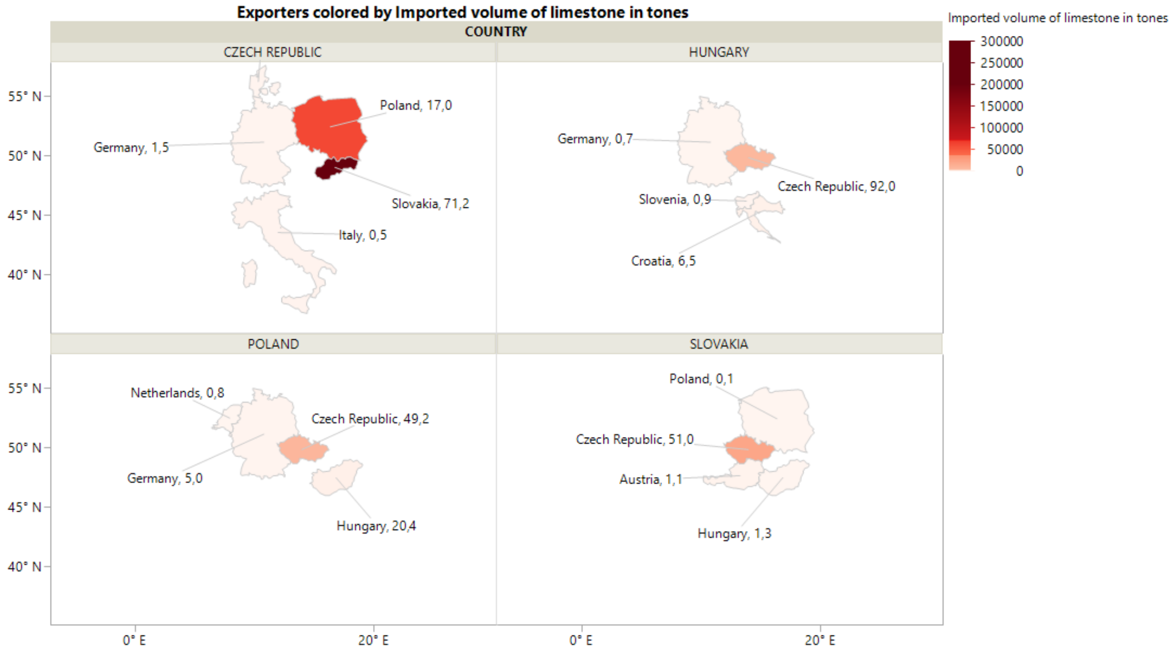

Fig. 7. Cartographer of the Limestone Exporter to the V4 countries in 2017 (own processing according to (www.trademap.org)).

The result from Analysing of limestone export from V4 countries

In the analysis of the Limestone export in tones, the quantity of exported limestone from the V4 countries in 2017 and the routing of this raw material was assessed. The largest volume of exports was recorded in Poland with a volume of 1 830 404 tones and the smallest in Hungary with a volume of 265 tones (Tab. 3). Exports of this raw material from the V4 countries are re-directed to nearby locations mostly in neighbouring countries as shown in Figure 8. The Czech Republic exported almost 70 % to Germany, the rest of the exports to Slovakia, Poland, Hungary and Austria. Poland has two major importers of limestone Germany and Belarus, which covers 80 % of Poland's exports. Slovakia is the country with the 2 largest exports of limestone in V4 at 1 289 589 tones, with up to 84 % of this raw material being directed to Ukraine and 16 % by the Czech Republic.

Tab. 3. Overview of limestone exports from the V4 countries in 2017 ((own processing according to www.indexmundi.com and www.trademap.org).

COUNTRY	Exports of limestone in tonnes			
	Sum	Mean	Min	Max
CZECH REPUBLIC	233168,00	38861,33	3,00	182733,00
HUNGARY	132,00	66,00	14,00	118,00
POLAND	915202,00	130743,14	16,00	627135,00
SLOVAKIA	644795,00	161198,75	1,00	352970,00

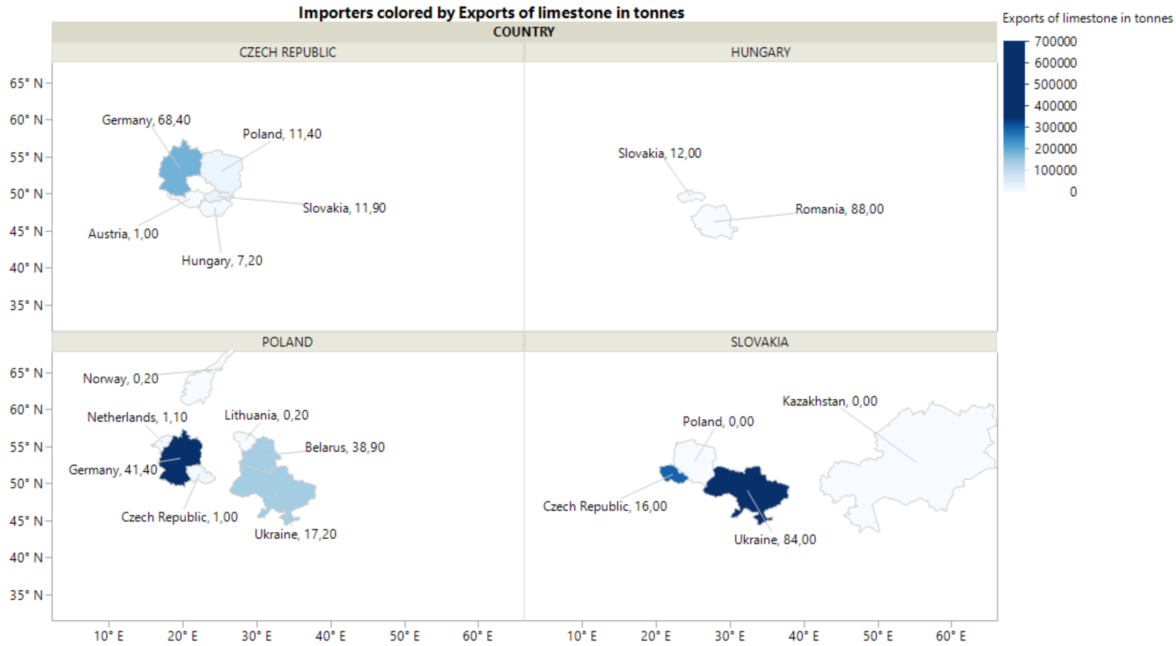

Fig. 8. Cartographer of limestone importers from the V4 countries in 2017 own processing according to (www.trademap.org).

Conclusion

We have chosen the limestone issue in V4 because it is the raw material that is used in many industries and as such is irrecoverable in many processes.

It is also a mineral raw material which is mined and processed in all four of V4 countries.

For analysis through production and export and import, we have decided for data availability and the possibility of comparing them. However, we also consider the processing of the problem in terms of four key factors, characteristic of limestone, limestone production, limestone import and export, and the development of indicators for statistical analysis.

Analysis of the production of limestone in the V4 countries and then the analysis of import and export in volume units (tones), the development of these indicators was evaluated with regard to the sales prices of the raw material, which has an impact on the economic balance of individual countries. By analyzing the production of limestone in the V4 countries in 10 years it is possible to sort the countries according to the importance of the rate, the biggest producer is Poland, which produces 43 % of the raw material, the Czech Republic and Slovakia produce over 20 % of the raw material and the lowest volume of production up to 10 % is Hungary.

The groups of non-metallic materials, which are imported and exported in significant quantities, also include limestone and semi-products made of limestone - cement and lime. One of the main reasons for the import and export of these commodities in recent years has not been such a shortage or surplus of commodities, rather than economic reasons. In the case of limestone, cement and lime, to a certain degree of simplification, it was true, that cheaper Slovak or Polish production (alternative) was delivered to the Czech market, while somewhat more expensive Czech production was offered to German, the Austrian market.

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Complex Photogrammetry Analysis of The Muck Pile Fragmentation Obtained in Russian Ore Mines

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The application of photogrammetry analysis in the evaluation of muck pile fragmentation of Russian diamond and copper-nickel ore mines was presented in the article. The evaluation was performed in the AutoCAD and Split Desktop 2.0 software based on the photographic documentation of three different muck piles. Moreover, calculation of Hazenôs index which may be applied in Russia as an additional tool to evaluate blasting works were included. According to performed analysis, it can be observed that the weighted average grain diameters for Underground A and B were in the optimum range. Furthermore, Hazenôs index shows that all analysed muck piles should be evaluated as non-homogenous. This could be explained by the type of exploitation system which was used during ore extraction.

Key words: *fragmentation, cumulative size distribution, sublevel caving, block caving mining, blasting works*

Introduction

One of the most important parameters which allow evaluating the quality of blasting works is the muck pile fragmentation. Proper evaluation of the muck pile fragmentation not only allows to examine the effectiveness of blasting works, but also affects the gravity movement of blasted ore during rock caving method, selection of loading, hauling and processing equipment, as well as the synchronization of the whole technical system at the mine (Biessikirski and Biessikirski, 2012; Sofranko et al., 2012; Sofranko et al., 2015; Terpák, 2010). Increased content of thick fractions in the loosened ore results in good movement of the material throughout the exhausted draw points, which positively affects the number of rock losses and ore dilution. In the case of a high number of oversized rocks, the exhausted draw points and grizzly level may be clogged or suspend by the broken ore (Terpák, 2016). Because of these observations, it can be stated that proper assessment of broken ore fragmentation should be made in order to validate blasting works or to make a necessary correction of blasting patterns or parameters.

Nowadays in order to assess muck pile fragmentation direct and indirect analysis is performed. Direct analysis (sieve analysis) is considered to be the most accurate method for evaluating actual fragmentation of the material, but given the amount of required material, time which is required to make the assessment and the economic factor, the method becomes less practical (Biessikirski, 2016a; Esen and Bilgin, 2010). On the other hand, indirect methods, including empirical methods, for example, Kuznetsov's and Cunningham's equation, as well as computer support in such methods as photogrammetry and laser technique, are widely used (Cunningham, 1983; Farmarzi et al., 2013; Kuznetsov, 1973; Krawczykowski et al., 2012). However, the photogrammetric and laser techniques are mainly used in opencast mining plants, and they have been the subject of numerous articles (Aler et al., 1996; Batko and Soğys, 2007; Biessikirski et al., 2016a; Biessikirski et al., 2016b; Farmer et al., 1991; Maerz et al., 1987). In case of underground mining, the photometric technique was mainly used in studies on the influence of joint spacing on rock fragmentation under TBN cutter or numerical simulation of rock fragmentation mechanism subjected to wedge penetrations for TBMs (Li et al., 2016; Yin et al., 2016).

This paper aims to present the possible application of the photogrammetric method and Hazenôs index to evaluate the fragmentation of broken ore in underground ore mines. Hazenôs index is mainly applied in geotechnics, but it was also adopted in Russia as one of the factors of blast works evaluation.

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Characteristics of the exploitation system in an underground mine

The analysis of muck pile fragmentation was performed for two Russian ore mines (Underground mine A and Underground mine B) located on the Kola Peninsula, and a diamond mine (Underground mine C) located in Yakutia. Underground mine A and B are located at the Severny-Gluboky deposit. The northern part of the deposit is characterised by the Talnakh and Kharaclakh intrusion. Both intrusions show differences between the hanging wall and footwall of the fault (Lightfoot, 2017).

The extraction of copper-nickel ore (Ni å 0.48% in 1 ton of ore, Cuå 1.94% in 1 ton of ore) is conducted by sublevel caving (Fig. 1).

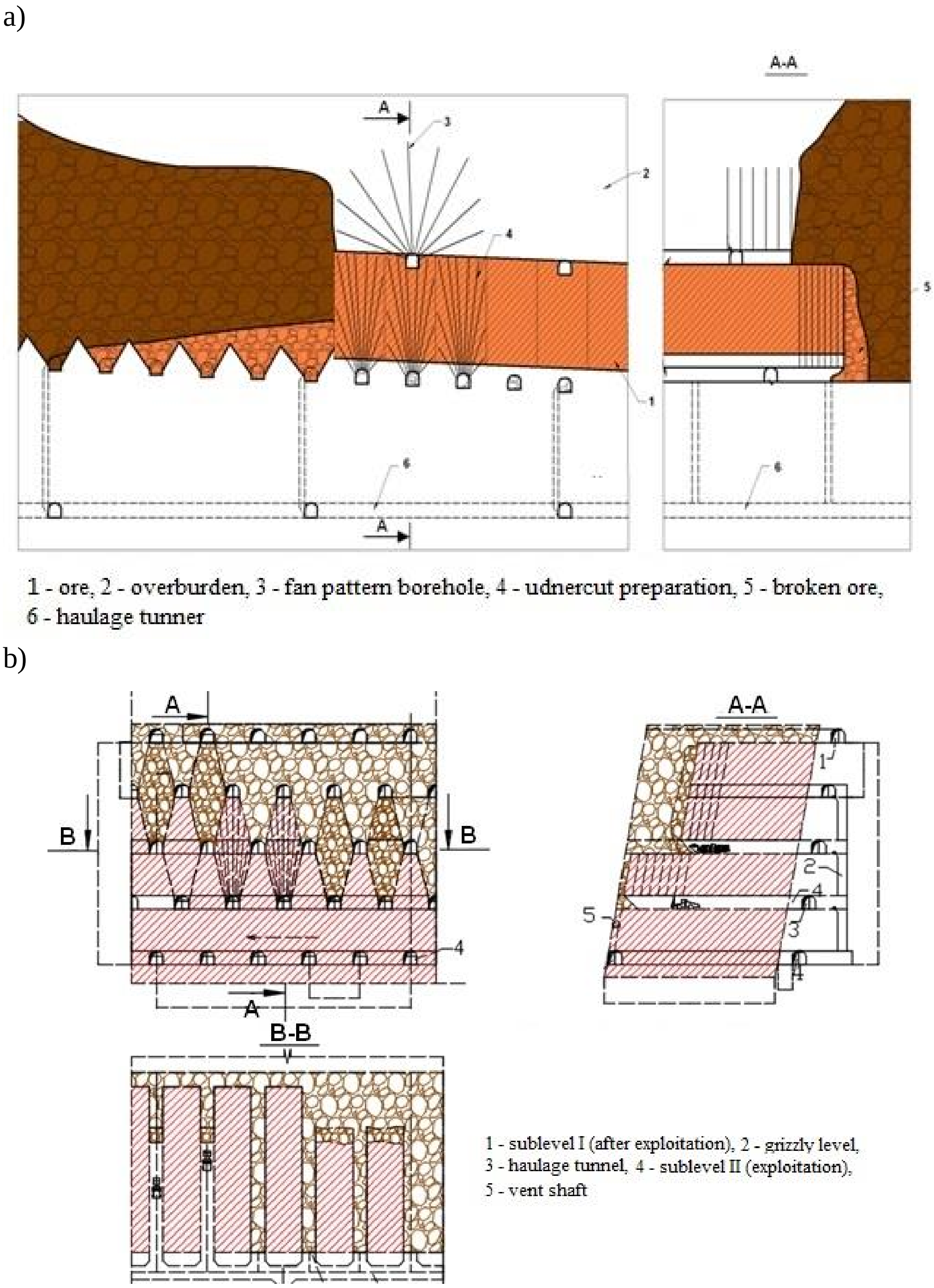


Fig. 1. Sublevel caving in a) Underground mine A, b) Underground mine B.

In Underground Mine A, the ore is extracted by block caving mining method (Fig.1a). The deposit is divided into undercut drifts. The height of the undercut drift depends on the bed thickness and varies from 18 to

50 m. An undercut with haulage is driven under the orebody with drawbells. Long boreholes were drilled in the fan pattern above the undercut. After the blasting, broken rocks were removed from the haulage access by loaders and haul trucks.

In Underground Mine B, the ore is extracted by the sublevel caving mining method (Fig.1b). Sublevel was developed in the ore body at regular vertical spacing. Blastholes around 30 m long were drilled from the access drifts in a ring pattern. The ore in the stope was blasted and collected in the draw-points. Blasting of each sublevel started at the hanging wall and mining then proceeded towards the footwall. As mining progressed downward, each new level was caved into the mine openings, with the ore materials being recovered while the rocks remained behind (Harraz, 2014). Loading was then continued until it was decided that waste dilution was too high (Fig. 1b).

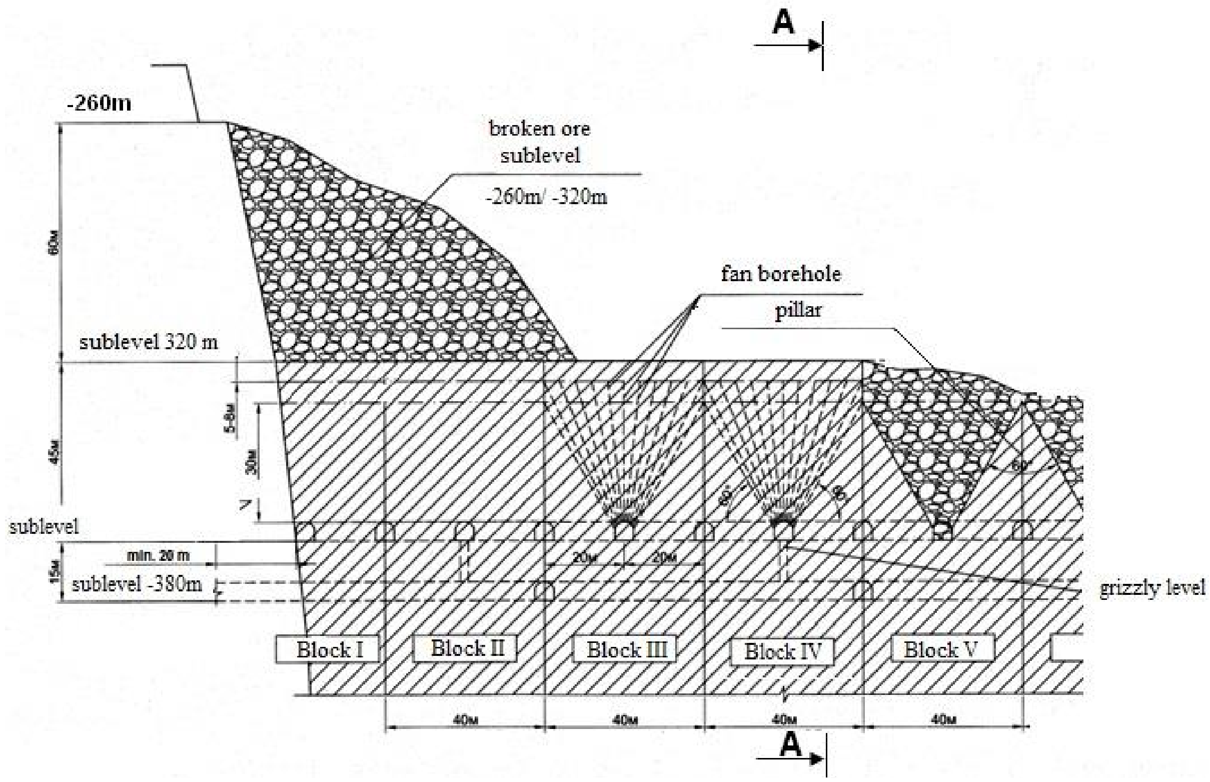


Fig. 2. Sublevel caving in a) Underground mine C.

In Underground Mine C (Fig. 2), diamonds are extracted from two adjacent kimberlite deposits by block caving mining method. Kimberlite deposits are located between limestones of the lower Ordovician. Diamonds are located in the eclogite deposit, which is in contact with peridot-dunite rocks. In 2010 the exploitation system was changed from open pit mining to underground mining.

Methodology

The evaluation of muck pile fragmentation was performed in the *AutoCAD* and *SPLIT Desktop* 2.0 software. The first stage of the analysis consisted of the preparation of photographic documentation in accordance with the recommendations described in (Aler et al., 1996; Batko and Soğys, 2007; Farmarzi et al., 2013; Li et al., 2016). On the muck pile surface, an object with known dimensions was placed. In terms of *Auto CAD* analysis it was the SHSS-T rescue apparatus with a dimension of 111 × 146 × 248 mm, and in terms of *Split Software 2.0*, it was a ball with a diameter of 250 mm.

After the muck pile was scaled, manual delineation was made in *Auto CAD*. The manual delineation was presented in Fig. 3. The final stage of the process was to determine the broken rock diameter in accordance with the scaled object.

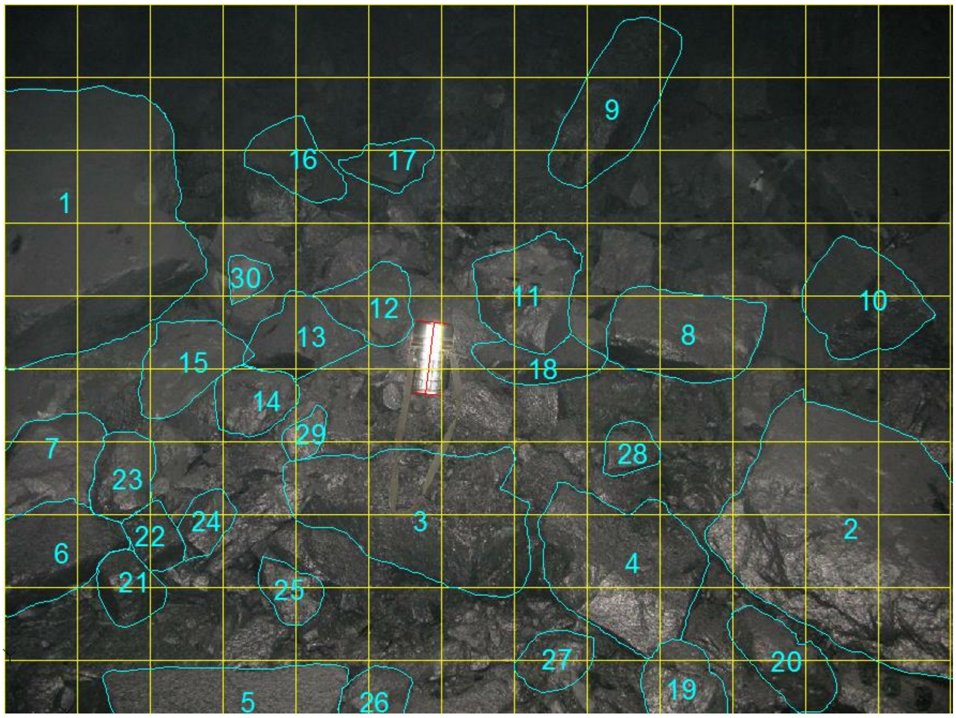
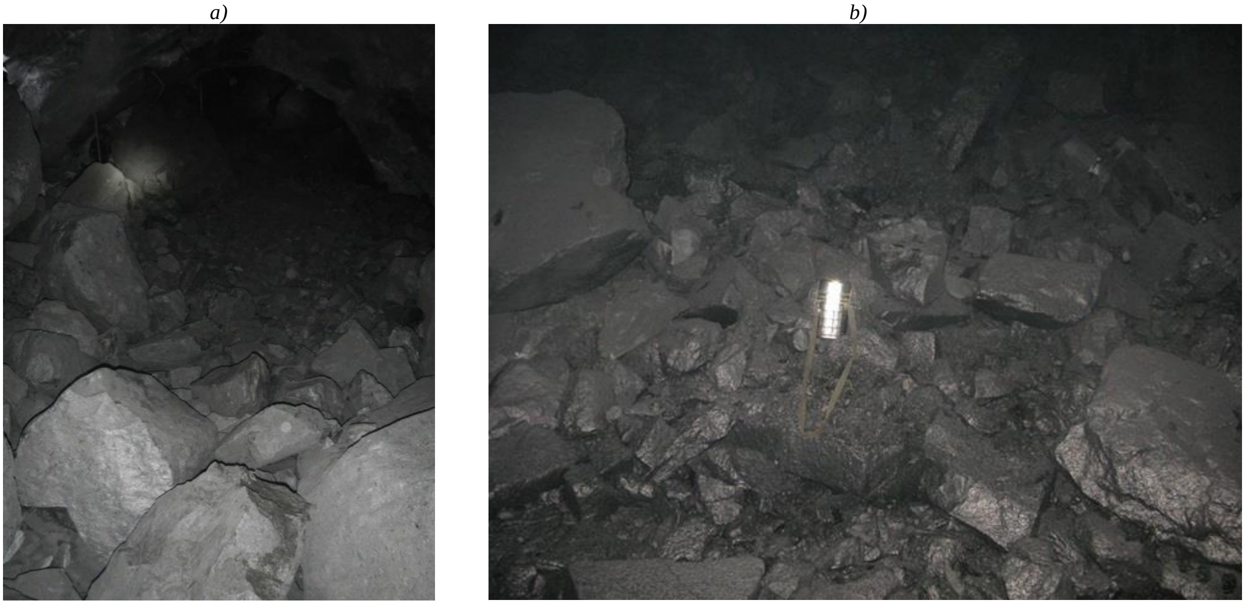


Fig. 3. Delineation in AutoCAD.

In the figure, it can be observed that the results obtained in the *Auto CAD* software were encumbered with a significant error. It was caused by a hinder delineation of the contour of the small rock. Further evaluation of the rock fragmentation is performed by the manual comparison of broken rock dimensions with scaled object dimensions. For the above reason, the exemplary results of the analysis were only presented for Underground Mine B.

The muck pile analysis obtained by the block caving mining method in three Russian underground mines was performed in the *Split Desktop 2.0.* software, in accordance with the methodology described in papers (Biessikirski et al., 2017a; Biessikirski et al., 2017b). An exemplary photos of evaluated muck piles were presented in Fig. 4a-c.

a) *b)*



c)

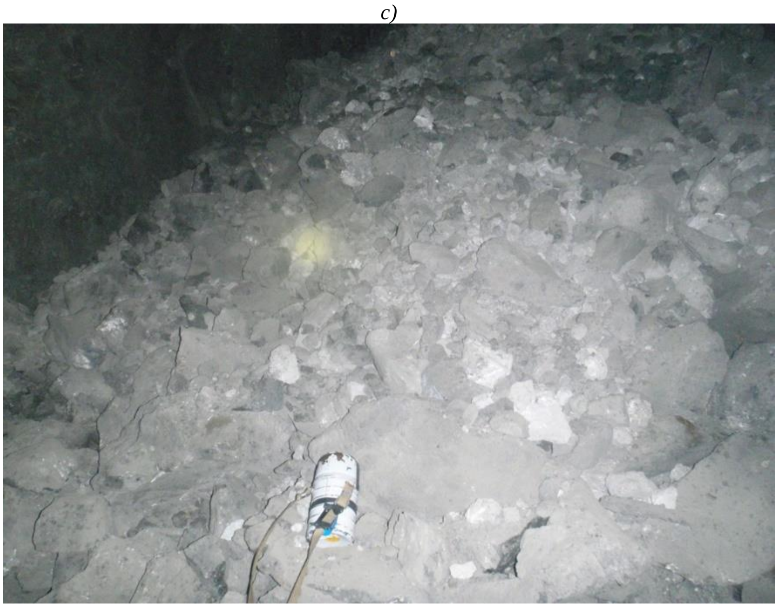


Fig. 4. Analysed muck pile: a) Underground mine A, b) Underground mine B, c) Underground mine C.

After delineation, the evaluation module was initiated by the user. The assessment of the broken rock fragmentation was made based on the Schumann distribution. As a result of the analysis, a logarithmic linear representation of the cumulative size distribution was obtained. An exemplary cumulative size distribution derived from the single muck pile analysis of the broken rock in underground mine A was presented in Fig. 5.

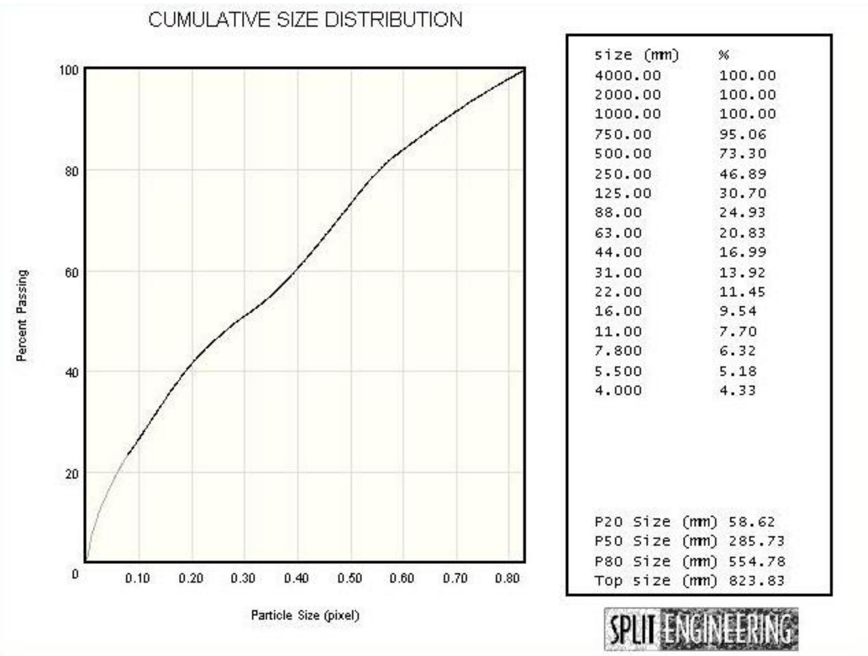


Fig. 5. Cumulative size distribution muck pile no. 1 series 1.

In Fig. 4, it can be observed that the number of oversized rocks (fractions above 700 mm) was ca. 4.94 %. The top size of the grain was ca. 823.83 mm. Furthermore, the fine particles (grains under 4.75 mm) were ca. 4.33 % (PN-B-02480: 1986).

Muck pile fragmentation analysis

An assessment of muck pile fragmentation was made for three different Russian underground mines. Each muck pile was analysed by *AutoCAD* and *Split Desktop 2.0* software, based on the three photographs which were taken from various points. The obtained results were then averaged and presented in Table 1 and Table 2. This resulted in obtaining singular cumulative size distribution, Fig. 6a and Fig. 6b. Detailed results obtained on the basis of *Split Desktop 2.0* software were presented in papers (Biessikirski et al., 2017a; Biessirski et al., 2017b).

Tab. 1. The average percentage of the output particle size based on AutoCAD software.

Grain size, [mm]	**An average amount of broken rock in the analysed muck pile** [%]
[mm]	**Underground Mine B** [%]
< 100	61.25
< 200	65.53
< 400	70.34
< 600	74.70
< 800	87.81
< 1000	100.00

Tab. 2. The average percentage of the output particle size based on Split Desktop 2.0 software.

Grain size [mm]	**An average amount of broken rocks in the analysed muck pile** [%]		
	Underground Mine A	**Underground Mine B**	**Underground Mine C**
< 4.00	2.84	10.30	2.13
< 5.50	3.33	11.14	2.69
< 7.80	4.00	12.07	3.32
< 11.00	4.82	13.25	4.13
< 16.00	5.93	14.80	5.26
< 22.00	7.12	16.42	6.48
< 31.00	8.72	18.57	8.14
< 44. 00	10.81	21.35	10.30
< 63.00	13.60	24.91	13.15
< 88.00	16.97	28.44	16.52
< 125.00	21.25	32.04	21.02
< 250.00	33.67	46.28	33.82
< 500.00	61.48	83.24	58.20
< 750.00	86.95	93.98	75.27
< 1000.00	98.88	97.73	87.55
< 2000.00	100.00	100.00	100.00

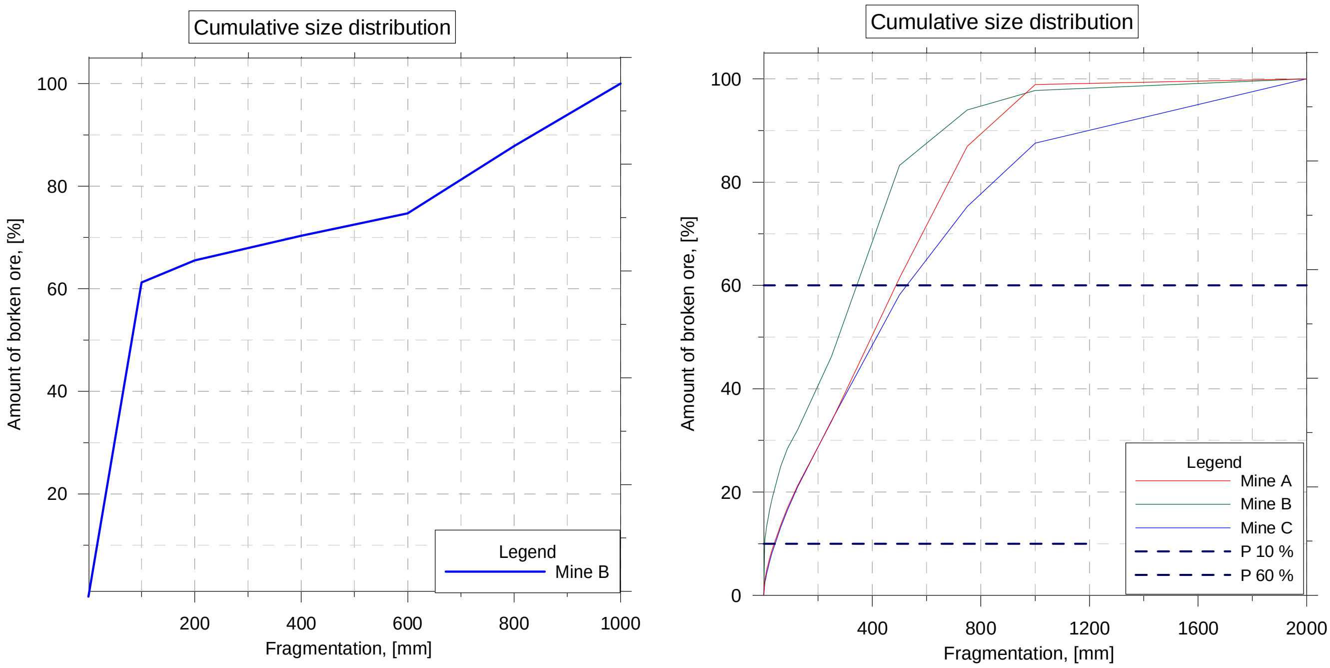


Fig 6. The cumulative size distribution: a) of each stage ï Underground mine A; b) for the whole series ï underground mine A; c) of each stage ï Underground mine B; b) for the whole series ï Underground mine B.

The results of muck pile fragmentation obtained in two different types of software were presented in Fig. 6a and Fig. 6b, and in Table 1 and Table 2. The differences between the results were caused by the limitation of the AutoCAD software during the delineation process. Because of this limitation, results obtained in the *AutoCAD* software should be treated only as cognitive. In further analysis, only the results obtained from *Split desktop 2.0* will be used.

According to the data, Fig. 6b and Table 2, it can be stated that cumulative size distributions obtained after the analysis of three different muck piles in three different underground mines are similar to each other.

Moreover, the average content of fines products was ca. 2.84 % (Underground mine A), 10.30 % (Underground mine B) and 2.13 % (Underground mine C), and the content of oversize rocks was ca. 13.05 % (Underground A), 6.02 % (Underground B) and 24.73 % (Underground C).

In Table 3 the average content of fragmented rock (w) in relation to its average grain size (a) in analysed muck pile was presented.

Tab. 3. The average percentage of the output particle size in relation to the average grain size.

Average grain size (*a*) [mm]	**The average percentage of the output particle size in relation to the average grain size (*w*),** [%]		
	Underground mine A	**Underground mine B**	**Underground mine C**
2.00	1.42	5.15	1.07
4.75	1.91	5.99	1.62
6.65	0.67	0.93	0.63
9.40	0.82	1.17	0.81
13.50	1.11	1.55	1.13
19.00	1.19	1.62	1.22
26.50	1.60	2.15	1.66
37.50	2.09	2.78	2.16
53.50	2.78	3.57	2.85
75.50	3.37	3.53	3.37
106.50	4.29	3.60	4.49
187.50	12.41	14.24	12.8
375.00	27.81	36.96	24.39
625.00	25.48	10.74	17.07
875.00	11.92	3.75	12.28
1500.00	1.12	2.27	12.45

Based on the data presented in Table 3, the weighted average of singular broken rock and Hazenôs index were determined.

Presented in Table 4, weighted averages of broken rock were determined according to Eq. 1.

$$x = \frac{a_1 \cdot w_1 + a_2 \cdot w_2 + \cdots + a_n \cdot w_n}{w_1 + w_2 + \cdots + w_n} \tag{1}$$

where:

x - Weighted average of singular broken rock, Table 4

a_1, a_2, a_n - Average grain size, [mm], Table 3

w_1, w_2, w_n - The average percentage of the output particle size, [%], Table 3

Tab. 4. The average diameter of debris determined for the analysed underground mines.

The weighted average of singular broken rock (x)		
Underground mine A, [mm]	Underground mine B, [mm]	Underground mine C, [mm]
418.41	310.38	527.04

According to Table 4, it can be observed that the weighted average size of broken rocks was between 310.38 ÷ 527.04 mm. Due to the type of exploitation (sublevel caving method), it had been assumed that the optimum size of the material should be within the range of 300 ÷ 400 mm. It is undesirable to obtain a large number of broken rocks over 400 mm due to the possibility of rocks wedging in exhausted draw points. In the case of fines products, a high content thereof would be unfavourable, due to the possibility of excessive densification of broken rock material. According to Table 4 only in the case of Underground mine C the weighted average of broken rock size exceeded the optimum diameter. However, it should be noted that during the gravity movement of broken rock, wedging was not observed.

The detailed assessment of the muck pile was further made according to Hazenôs index (C_u), For example, 2. Hazenôs index is used in geotechnics, and it can be described as a grain size distribution with the permeability for effective diameter P60 and P10, Fig. 6. The high value of Hazenôs index indicates that soil is multi-fraction and the fine fractions will fill the voids between larger grains.

In some Russian underground ore mines, Hazenôs index has been adopted in order to evaluate the effect of blasting works. The assessment by Hazenôs index was performed based on the standard (GOST 25100: 2010), where it is assumed that muck pile with an index greater than 3 should be classified as non-homogeneous.

$$C_u = \frac{d_{60}}{d_{10}} \tag{2}$$

where:

C_u - Hazenôs index

d_{10} - Effective diameter P10, Table 4
d_{60} - Effective diameter P60, Table 4

Tab. 5. Hazenôs index for the analysed mines.

Hazenôs index		
Underground mine A	Underground mine B	Underground mine C
12.42	88.24	12.47

Based on Hazenôs index, Table 5, it can be concluded that all analysed muck piles were non-homogeneous. Expected muck pile ought to be characterised as homogeneous or slightly non-homogenous. In practice, in case of blasting works and sublevel caving mining method, to obtain Hazenôs index equal or close to 3 is impossible. Therefore, the evaluated coefficients were correlated with the average content of fragmented rock in relation to its average grain size (Table 4), and a weighted average of broken rock for an individual underground mine. In Underground mine A, Hazenôs index was evaluated as 12.42 (Table 5), weighted average of broken rock was 418.41 mm (which exceeded the optimum diameter of broken rock: 300.00 ÷ 400.00 mm), and average content of fragmented rock in relation to its average grain size indicated that domain fragmentation was within the range of 125.00 ÷ 1000.00 mm. Similar results were obtained for Underground mine B (C_u = 88.24, x = 310.38 mm, with the dominant fragmentation within the range of 125.00 ÷ 750.00 mm) and Underground mine C (C_u = 12.47, x = 527,07 mm, with the dominant fragmentation within the range of 250.00 ÷ 2000.00 mm). Such a high value of Hazenôs index for Underground mine B can be explained by the significant presence of fines (a = 10.30% for the 4 mm fraction, Table 2) in the analysed muck pile. Moreover, in the case of Underground mine B, the weighted average size of broken rock slightly exceeds the lower value of the optimum range of fragmented material (300.00 ÷ 400.00 mm). The high value of Hazenôs index and calculated the weighted average size of broken rocks indicate the necessity to make a proper correction of blasting work parameters in order to improve broken rock fragmentation.

It can be presumed that if the fractions that are the most dominant in the muck pile were the only ones to be considered (with less dominant fractions like fines or oversize rocks skipped in the analysis), the obtained values of Hazenôs index would be significantly decreased. In such a case, the calculated results for Underground mine A and Underground mine B would probably be close to 3.

Results

The performed analyses indicate the possible application of the photogrammetric technique for underground mining. However, the obtained results should be treated as an approximation due to the fact that photogrammetric techniques only yield information about the fragmentation of broken rock which is located only on the surface of the muck pile. In order to make a precise assessment of the indirect fragmentation evaluation by, i.e. multiple examinations of broken rock fragmentation in various muck pile cross sections should be performed.

The results obtained from the *AutoCAD* and *Split Desktop 2.0* software show differences. These were caused by the limitation of *AutoCAD* during the delineation process. The lack of possibility to make proper delineation of broken rock which had a small diameter and the necessity to make a manual assessment of fragmentation by comparing the dimensions of delineated broken rock with the dimensions of scaling object resulted in a significant error. For these reasons, it is recommended to use software specially designated for fragmentation assessment.

Obtained cumulative size distribution curves for particular underground mines were similar. However, in the case of Underground mine B the high amount of fines products (10.30%, Table 2, Table 3) was observed, and in the case of Underground mine C, the high amount of oversize rock (24.73%, Table 2, Table 3) was noticed.

Calculated weighted average size of broken rocks in Underground mines A and B was within the optimum range (300.00 ÷ 400.00 mm). In the case of Underground mine, C the weighted average size of broken rocks exceeded the optimum values assumed by the mine authorities.

Calculated values of Hazenôs index indicate that analysed muck piles were non-homogeneous. This was caused by the type of exploitation system (blasting works and sublevel caving). Moreover, C_u for Underground mine B was determined as 88.24. Considering the evaluated weighted average size of broken rocks and dominant fragmentation of broken rocks the proper correction of blasting parameters should be made only in the case when similar results are obtained in the subsequent exploitation. In Underground mines A and C, the effect of blasting works was satisfactory. The exceeded value of the optimum broken rock diameter range in Underground mine C did not wedge the draw points during the gravity flow of broken rocks.

It can be presumed that if the most dominant amounts of fragmented rocks in relation to their average grain size were the only ones to be considered, then the determined value of C_u for Underground mines A and C would be close to 3. In terms of Underground mine B, the obtained value would be much lower than 88.24.

Moreover, it can be presumed that fines products would be moving gravitationally throughout the muck pile. However, in order to verify this statement, the abovementioned indirect type of analysis should be performed.

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Design of the forming device with a vertical blow head for the production of synthetic fibers intended for filter process in the mining industry

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It is generally known that the underground water is also extracted into the mining process during the mining of rocks, solids or different liquid substances. These mining processes need to be drained and, given the increased demand for organic processes, water has to be separated from undesirable substances. One of the undesirable substances in the mining process is the oil from the hydraulics mining machines that can contaminate this water. Various technologies and devices are used for filtration, for example, separators. These separators contain filters that can also be made from synthetic fibrous materials. The synthetic fibrous materials for the filters are manufactured by different technologies. The article presents the technology of producing synthetic fiber material by vertical blowing method from thermoplastic raw materials. Namely, the characteristics of the fiber forming device (blow head with an annular converging nozzle) are defined and designed. As a result of the blow head investigation, the values of air rarefaction and air flow in the central hole of the annular nozzle were determined from the process parameters. Also, the result of the blow head study is the determination of the design characteristics, the influence assessment of various design and operational factors on its work, allowing the elementary fibers to be drawn out in a single technological cycle to form a fibrous material for the filters.

Key words: *Filter process, mining industry, blow head, vertical blowing method, synthetic fiber material, annular converging nozzle, rarefaction in the central channel.*

Introduction

In the mining process, besides the semiliquid mixture, mud, slurry, and rocks, there are large volumes of water to remove in order to keep production moving (Qazizada et al., 2018). The authors (Straka et al., 2014) investigated potential reactive materials for the removal of heavy metals from contaminated water. One of the undesirable substances in the mining process is the oil from the hydraulics mining machines that can contaminate this water. Various technologies and devices are used for filtration, for example, separators. These separators contain filters that can also be made from synthetic fibrous materials. Also can be used a highly effective sorbent - a synthetic fibrous material made on TU-2282-001-49396305-99 from the source of raw materials: goods polypropylene, polyethylene terephthalate, waste products of polypropylene and polyethylene terephthalate. The sorbent is a fine-fibred such vata mass ð color from light grey to dark grey (colorless raw materials). The bulk density of the sample fibrous materials is between 160 and 174 kg/m at porosity from 81 % to 81.5%. It is designed for the production of filter materials and other products for cleaning of water, air and soil from pollution with oil products, heavy metals also in the mining industry (Sentyakov et al., 2016; Charvet et al., 2018).

Currently, for most industrial consumers, preference is given to materials that are easy to use, show high efficiency and have a low cost. Therefore, work in the direction of creating a modern technology to reduce production costs while ensuring quality material with high performance is an important task. (Ryauzov et al., 1980).

A well-known traditional technological process of obtaining synthetic fiber materials from primary thermoplastic raw consists in extruding the polymer melt through the spinneret thin holes into the shaft, where the jet is pulled to a predetermined diameter and cooled to a temperature corresponding to the solid state of the thread (Ryauzov et al., 1980). The cured thread is wound on the receiving device and is subjected to stretching and corrugation. At the next stage of the technological process, the tow is cut into elementary fibers (Yankov et al., 2006). The traditional technology of producing staple fibers is a rather complicated and energy-intensive production, involving the use of expensive technological equipment at the stages of the finished fiber production.

In addition, the traditional method is focused on the processing of high-quality industrial raw materials of a certain composition. It is proposed to use household and industrial thermoplastic wastes as raw materials, which are not homogeneous in composition and contain foreign inclusions. As a result, this material has a lower

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viscosity, melting point, as well as low mechanical characteristics, which do not allow winding devices to be used in such conditions. For this reason, it is not possible to obtain from them fibrous nonwoven material according to traditional technology.

From the production point of view, the relevant technology challenge for producing synthetic fibrous materials from secondary thermoplastic raw is the creation of a line that combines the technological operations of elementary threads molding and extrusion, which reduces the number of labor-intensive technological operations and transitions in the processing of fibers and canvas formation, while ensuring the required physical and mechanical properties of the resulting products (Yang, et al., 2017). This circumstance is the main direction of increasing the technical and economic efficiency of the technological process of obtaining fibrous materials from thermoplastic materials (Domnina and Repko, 2017; Shirobokov, 2008).

Of practical interest is the method of obtaining fibrous materials from thermoplastic raw materials melt by vertical blowing method with air, the installation scheme, which is presented in Figure 1 (Sentyakov et al., 2014, Elbakian et al., 2018). The proposed method is fundamentally different from the traditional methods of obtaining elementary fibers and is the most promising in its technical and economic indicators. Such technological process allows to combine the elementary fibers formation and extrusion, reduces the number of labor-intensive technological operations and transitions in the fibers processing and canvas formation, which significantly reduces the production cost (Shirobokov, 2008).

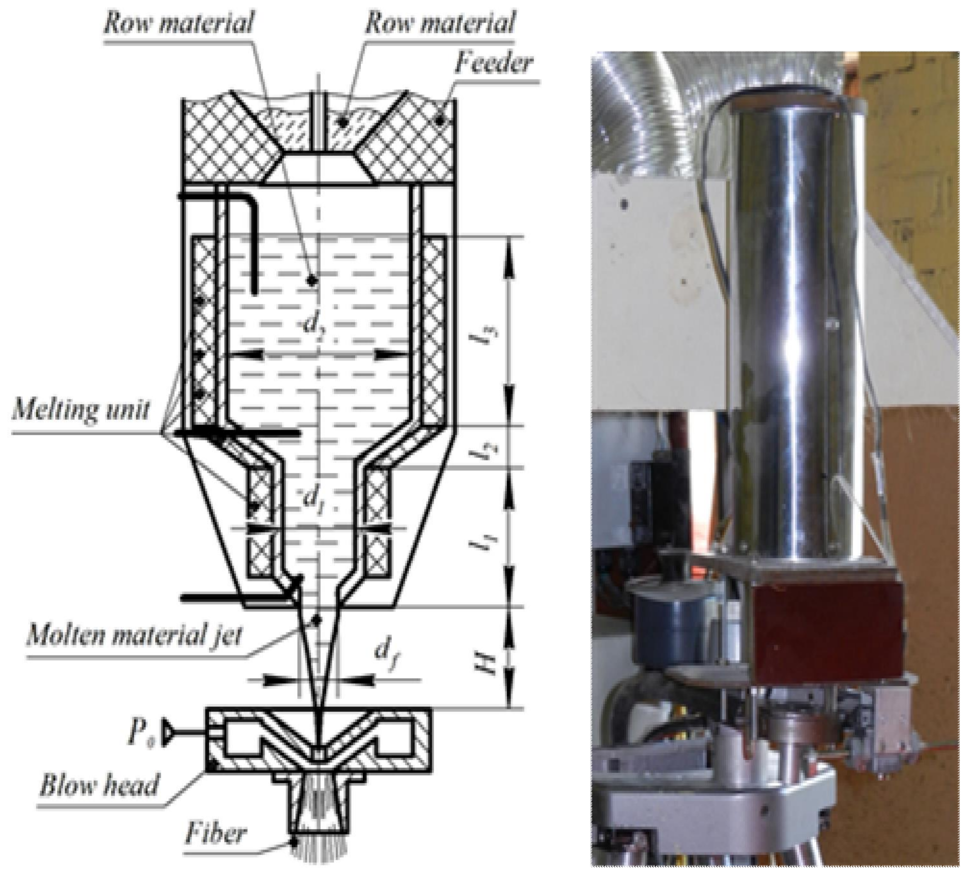

Fig. 1. The experimental setup design for the production of fibrous materials from a thermoplastic melt.

The creation results of new technology for producing fibrous materials from a thermoplastic melt by the method of vertical blowing of the molten material jet, flowing out of the die plate, with air, confirmed its positive qualities, including a significant reduction in the production cost of such fiber compared to traditional technology. In addition, such a technological scheme for obtaining a fibrous material is simple and one-stepped, since all transitions from the raw materials loading to the finished material output are carried out on a single unit. The raw material is a poisonless primary or secondary thermoplastic used for the plastic food dishes manufacture. The finished product is white staple fiber if it is obtained from primary raw materials, and gray, if it is obtained from secondary raw materials - crushed plastic bottles. Such a fiber can be obtained in the form of cotton wool or in the form of canvases, in which the elementary fibers are held together by either natural adhesion or by gluing part of the fibers under the temperature influence. The average elementary fibers diameter can be obtained from 1 to 100 microns, and length - from 1 to 500 mm. The density of cotton wool or canvases is from 10 to 100 kg / m^3. The material has a low hygroscopicity, high strength, and elasticity. It is steady in acids, alkalis, acetone, dichloromethane, is not exposed to microorganisms. The operating temperatures range is from - 60 to 170 ° C. Heat conductivity coefficient - 0.037 ... 0.040 W / (m · K).

The fiber formation from a thermoplastic melt by vertical blowing with air is accompanied by a number of complex and specific phenomena. Therefore, the creation of new advanced technologies, high-performance machines, and units for the production of such materials is impossible without modeling the technological process, which allows to significantly reduce the field tests amount, reduce the cost and development time, as well as select the optimal equipment operation modes.

The research goal was to determine the characteristics, to assess the influence of various design and operational factors on its work, allowing the elementary fibers drawing process to be carried out in a single technological cycle (Hofmann et al., 2018).

Materials and methods of experiment

The effectiveness of such fibrous materials production technology largely depends on the operation of the device, carrying out the process of the molten material jet interaction with the air flow. A distinctive device feature lies in the fact that the elementary staple fibers drawing force is created by an air stream that is specifically directed and determines, in many respects, the productivity and quality of the obtained products. The main factors in the device operation are the air flow parameters, which significantly depend on the device design features (Fabian et al., 2015).

At this stage of the study, the task is to determine the fiber-forming device characteristics, assess the influence of various structural and operational factors on its operation.

The constructive scheme of the blow head with an annular converging nozzle (Sentyakov et al., 2014) is shown in Figure 2.

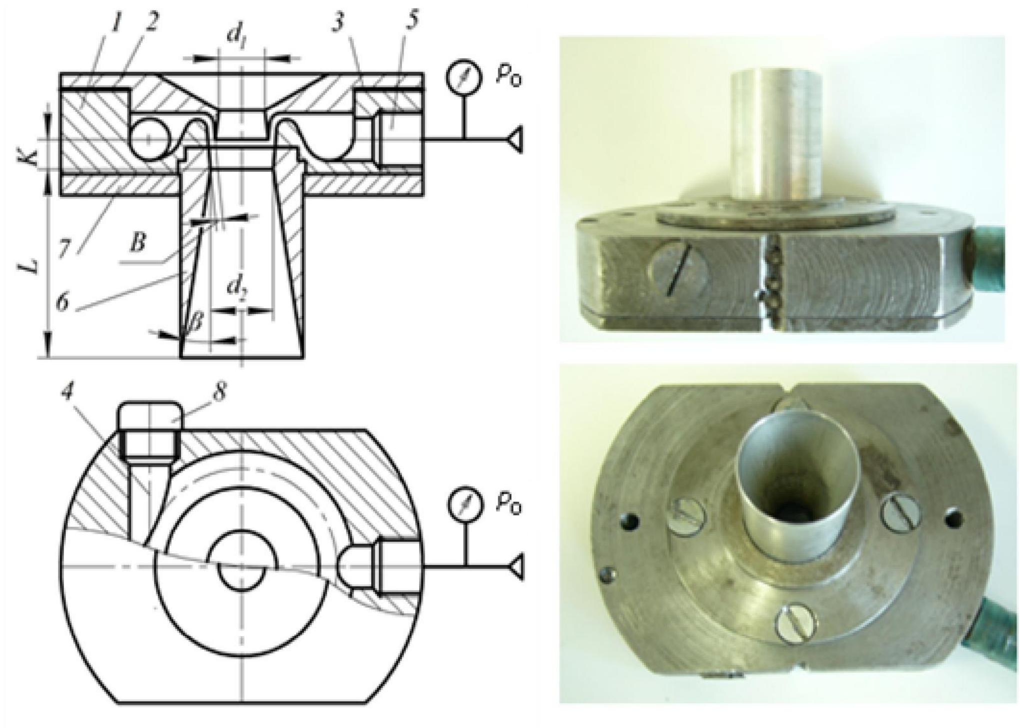

Fig. 2. Blow head design with annular nozzle.

The blow head consists of the lower 1 and upper 2 parts of the body, between which a sealing adjusting washer 3 is installed. The lower 1 and upper 2 parts of the body form an internal annular cavity with channels 4 and 5 for the supply of compressed air and an annular converging gap of width h, through which the compressed air flows into the atmosphere. On the lower part 1 of the body coaxially with an annular nozzle is installed a diffuser 6 of length L, which is fixed by a clamping ring 7. In the upper 2 part of the body, there is a central channel for introducing a molten material jet. One of the supply channels 4 is made tangentially to the inner annular cavity, and the other - channel 5, radially to it. To obtain a swirling air flow at the exit from the blowing head annular nozzle, air is fed with pressure p0 through channel 4, and without swirling - into channel 5. The channel that is not used is closed by a plug 8. A general view of the scheme for determining the rarefaction value in the central channel is shown in Figure 3.

The investigated blow head may be transformed in design by changing the method of supplying compressed air for its power, namely via the radial or tangential channel by connecting or separating the diffuser from the body, placing the steel ball head in the vortex chamber and changing the width of the annular gap B, through which the air flows, by changing the thickness of the sealing adjusting washer 3 between the upper and lower parts of the body. Variants of such blow head structural changes are shown in Table 1.

In the course of blow head experimental studies, the annular gap B width in each of its five versions took values from 0.3 to 0.6 mm, and the rarefaction in the central channel was measured at different supply pressures p_0 - from 10 to 200 kPa.

Tab. 1. Determination of characteristics depending on the blow head design.

A variant of blow head construction	Presence (+) or absence (-) of swirling airflow	Presence (+) or absence (-) of a diffuser	The presence (+) or absence (-) of the ball in the vortex chamber
1	-	-	-
2	-	+	-
3	+	-	-
4	+	+	+
5	+	+	-

Experimental dependencies of rarefaction p_b and the flow rate Q of ejected air in the central channel of the blow head of designs 1 and 2 (according to table 1) on the supply pressure p_0 at different widths of the annular gap B are presented in Figure 3 and 4, which implies that the rarefaction significantly depends not only on the specified parameters p_0 and B, but also on the presence of the diffuser at the blow head outlet. The use of a diffuser allows not only to increase the rarefaction in the central channel, but also to ensure uniform distribution of the airflow parameters downstream of the jet. In the absence of a diffuser, the jet significantly deviates from the blowing head axis due to the non-concentricity of the annular gap (Balog and Mačovský, 2015).

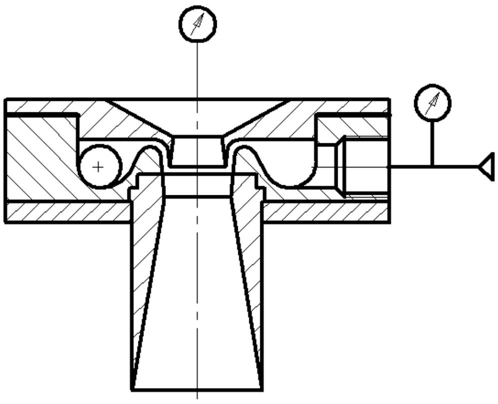

Fig. 3. Scheme for determining the rarefaction value in the blow head central channel.

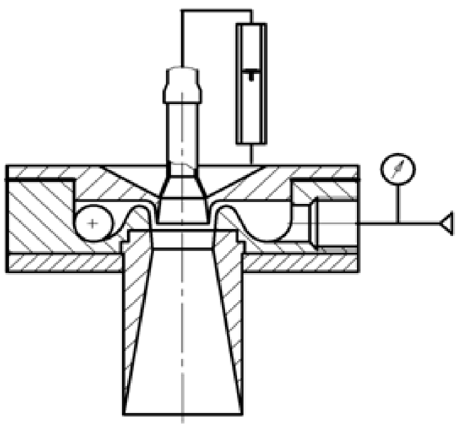

Fig. 4. Scheme for determining the ejected air flow rate through the blow head central hole.

Tab. 2. Levels of factors variation.

Level of factor	Variable factors		
	Supply pressure p_0, [kPa]	**The width of the annular gap** B, [mm]	**Diffuser length** L, [mm]
Upper	200	0.6	40
Lower	50	0.3	0

Tab. 3. The matrix of experiment design.

- experiment	Factors and their interactions								The response function $q = p_{\epsilon}/t\epsilon_{\odot}$
	Z_0	Z_1	Z_2	Z_3	Z_4	Z_5	Z_6	Z_7	
1	+	-	-	-	+	+	+	-	0.015
2	+	+	-	-	-	-	+	+	0.035
3	+	-	+	-	-	+	-	+	0.035
4	+	+	+	-	+	-	-	-	0.1
5	+	-	-	+	+	-	-	+	0.06
6	+	+	-	+	-	+	-	-	0.17
7	+	-	+	+	-	-	+	-	0.095
8	+	+	+	+	+	+	+	-	0.285

Note. The experiment design matrix is constructed in accordance with the recommendations given in the book (Adler and Varygin, 1978). The sign (+) means that the corresponding normalized factor in the corresponding experiment takes the value +1, and the sign (-) - the value -1.

Experiments to determine the rarefaction in the central channel of 1 and 2 blow head versions turned out to be sufficient for processing the results using the theory of experiment design. A matrix of the full factorial experiment was used for three factors varying on two levels. The response function was the dimensionless rarefaction q, determined by the ratio of rarefaction p_b in the central channel of the blow head to the atmospheric pressure p_a. The supply pressure p_0, the width of the annular gap B and the length of the diffuser are taken as independent factors. The values of the variable factors are given in Table. 2. The matrix of experiment design with the results of the experiment is given in Table 3 (Shilyaev et al., 2008).

Presented in Table 3, normalized factors are:

$$Z_1 = 2(p_0 - 125)/150\,;\;\; Z_2 = 2(h - 0,45)/0,3\,;\;\; Z_3 = 2(L - 20)/40$$
$$Z_4 = Z_1 \cdot Z_2\,;\;\; Z_5 = Z_1 \cdot Z_3\,;\;\; Z_6 = Z_1 \cdot Z_3\,;\;\; Z_7 = Z_1 \cdot Z_2 \cdot Z_3$$

Taking the response function in the form of a linear polynomial with the factors interaction and calculating the regression coefficients, we obtain the mathematical dependence of the dimensionless rarefaction q in the central channel of the blowing head without swirling the air flow on the above factors:

$$q = 0,1 + 0,048Z_1 + 0,029Z_2 + 0,053Z_3 + 0,016Z_4 + 0,027Z_5 + 0,008Z_6 - 0,0044Z_7 \quad (1)$$

After conducting experiments on the rarefaction measurement in the central channel of the blow head with intermediate values of the factors p_0, B and L, the adequacy of the obtained formula (1) for calculating the dimensionless rarefaction was checked. The test results are presented in Table 4, where it follows that the discrepancy between the calculated and experimental data in the accepted ranges of factors variation is from 10 to 15 %.

Tab. 4. Check the of the formula (1) adequacy for calculating the dimensionless rarefaction q.

№ experiment	Factors			Experiment result q	Response function $q = p_b/p_0$	Error [%]
	p_0, [kPa]	B, [mm]	L, [mm]			
1	100	0.3	0	0.025	0.0216	13.3
2	125	0.4	0	0.035	0.0391	-11.7
3	150	0.46	0	0.07	0.0598	14.57
4	70	0.6	0	0.05	0.0457	8.6
5	100	0.3	40	0.1	0.0987	1.3
6	150	0.4	40	0.16	0.1627	-1.69
7	70	0.46	40	0.115	0.099	13.9
8	125	0.6	40	0.21	0.19	9.5

Experimental studies of the process of obtaining a fibrous material using a blow head of this design showed that the expansion of intervals indicated in Table 4 is impractical. For example, an increase in the supply pressure p_0 of more than 200 kPa and an annular gap width B of more than 0.6 mm results in a significant increase in airflow and noise level of the blowing head without changing the quality of the produced fiber. Experiments have established that the desire to increase the rarefaction in the central channel of the blow head does not always lead to a positive result. For example, a test of 8 blow head version according to Table. 4, which had the highest rarefaction in the central channel, showed the impossibility of obtaining fibrous material. In this case, there is an intensive cooling of the molten material jet by air flows ejected from the atmosphere and a continuous thread with a diameter of 0.2 ... 0.4 mm is formed. To obtain a fibrous material, the rarefaction in the central channel of the blowing head should be minimal (Farias et al., 2015).

During the blow head investigation, it was revealed that the quality of the obtained fibrous materials is also influenced by the diffuser design parameters, namely, the length L and the cone expansion angle β (Fig. 2).

The diffuser length choice is determined by the need to obtain the maximum degree of the jet stretching and the optimal rarefaction value p_b. A general view of the scheme for determining the rarefaction value in the central channel is shown in Figure 3.

The experiments were carried out with the diffuser constant parameters (Fig. 2): d_1 = 9 mm, β = 12 degrees, d_2 = 11.8 mm and with variable parameters: K = 6.2 ÷ 6.96 mm, B = 0.3 ÷ 0.6 mm, L = 0 ÷ 35 mm. Before each experiment, the annular gap size was established by replacing an adjusting washer, and then the characteristic of the device was taken. Seven diffuser designs with the length L: 5; 10; 15; 20; 25; 30; 35 mm. were investigated (Liu et al., 2016).

Another important diffuser parameter is its shape; therefore, at the second stage of the experiment, the task was to investigate the influence of the diffuser expansion angle β on the rarefaction value p_b in the central channel. The experiments were carried out with the diffuser constant parameters (Fig. 2): d_1 = 9 mm, d_2 = 11.8 mm, L = 20 mm and with variable parameters: K = 6.2 ÷ 6.96 mm, B = 0.3 ÷ 0.6 mm. β = 0 ÷ 14 degrees. Seven diffuser designs with expansion angle β = 0; 3; 5; 8; 10; 12; 14 degrees were investigated;

The pressure p_0 at the inlet to the blow head was measured with an exemplary manometer with a measurement limit of 0.4 MPa accuracy class 0.6. The rarefaction value in the central hole of the device was measured by a TNMP-52 type pressure gauge with a measuring limit of 1250 mm.w.c. Moreover, the flow rate of injected air through the central hole of the device was measured with a PM-04 rotameter with a working measuring range of 0.75-4.3 m^3/h (Domnina, 2017; Volosov and Ped, 1974).

Results and discussions

The experimental study results of the dependence of the rarefaction value in the central hole on the process parameters are shown in Figure 5 and 6. As a result of the dependence analysis, it was found that the rarefaction value is significantly influenced by the value of the annular gap h, and the nature of the swirling airflow through the device's annular cavity, as well as a significant increase in the rarefaction value, is observed with increasing the diffuser length to L = 20 mm. The performed tests allowed us to establish the optimal diffuser length, corresponding to doubled central hole diameter d_1.

Studies have also shown that the rarefaction in the central channel p_b increases with increasing angle of expansion ***ϐ*** = 12 degrees (Fig. 7). A further increase in the angle of expansion ***ϐ*** leads to the airflow separation from the diffuser walls, resulting in the rarefaction value decrease in the central hole.

The results illustrating the dependence of the air flow rate through the annular gap on the pressure in the device are shown in Figure 8.

Studies have shown that air flow rate through the central opening of the device increases with increasing width of the annular gap (Daristotle et al., 2017).

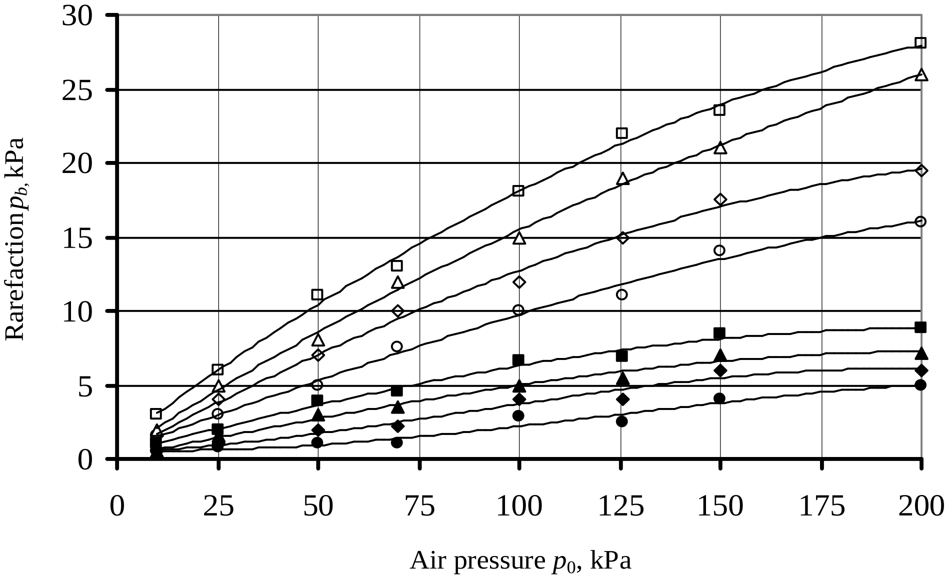

Fig. 5. The dependence of the rarefaction p_b in the central hole on the air pressure in the device p_0: air flow without swirling - Ĭ - at B = 0.6 mm, K = 6.96 mm; φ - at Ɉ = 0.46 mm, ȿ = 6.7 mm; ö - at Ɉ = 0.4 mm, ȿ = 6.4 mm; ȥ - at Ɉ = 0.3 mm, ȿ = 6.2 mm; air flow with a twist - ý - at B = 0.6 mm, K = 6.96 mm; ɀ - at Ɉ = 0.46 mm, ȿ = 6.7 mm; Dž at Ɉ = 0.4 mm, ȿ = 6.4 mm; ǒ - at Ɉ = 0.3 mm, ȿ = 6.2 mm.

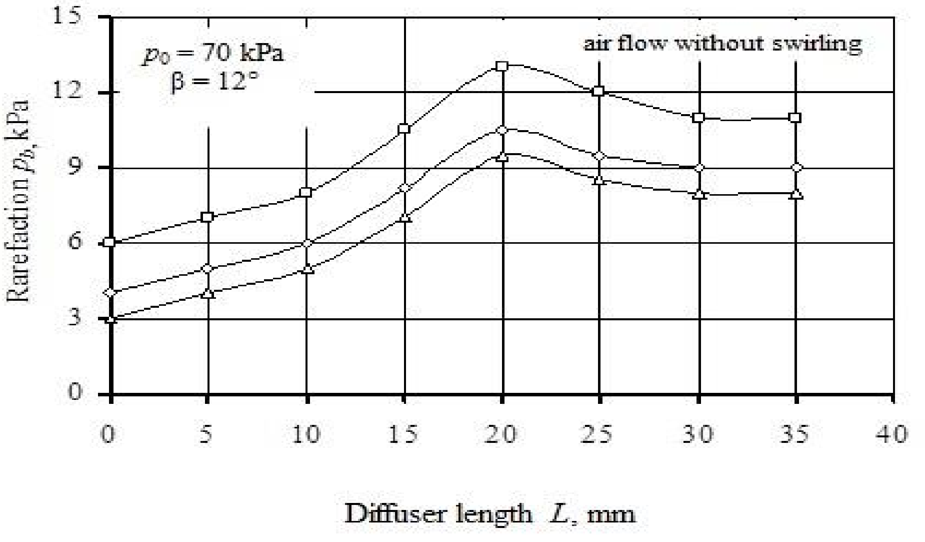

Fig. 6. The dependence of rarefaction in the central hole p_b on the diffuser length L.

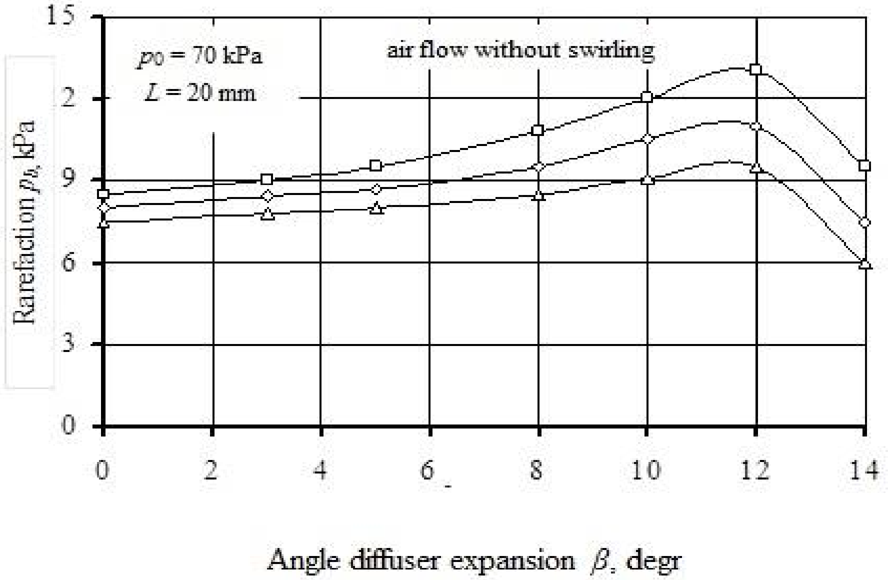

Fig. 7. The dependence of rarefaction in the central hole on the angle of diffuser expansion ***b****.*

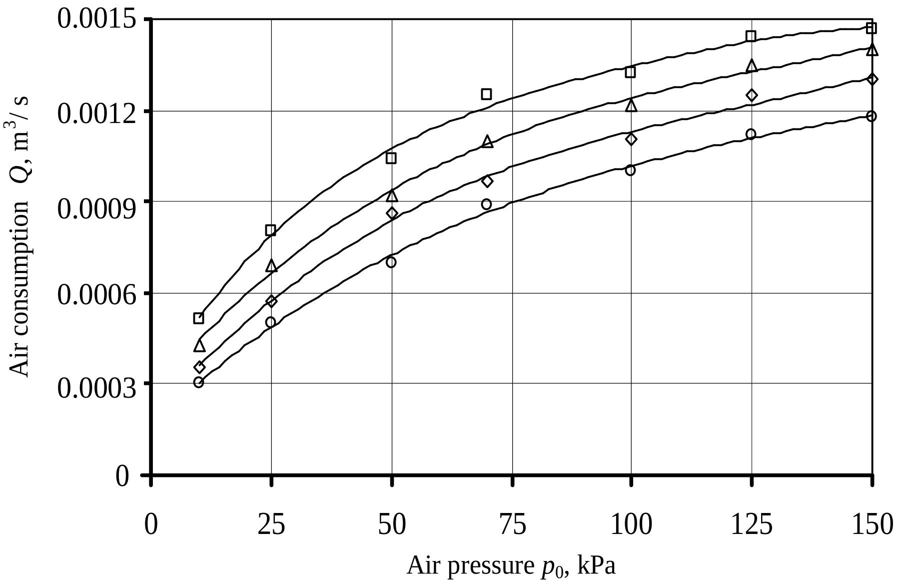

Fig. 8. The dependence of the flow rate Q in the central hole on the air pressure in the device p_0: air flow without swirling - ***Ĭ*** *- with h = 0.6 mm, K = 6.96 mm;* ***φ****- at h = 0.46 mm, K = 6.7 mm;* ***ö*** *- when h = 0.4 mm, K = 6.4 mm;* ***ƺ*** *- at h = 0.3 mm, K = 6.2 mm.*

Summary

Thus, as a result of research carried out on a blow head with an annular converging nozzle to obtain a fibrous material using a vertical blowing of molten material jet with air flow, rational parameters of its flow part, rational pressure range for supplying it with compressed air, and the effect of these parameters on the rarefaction value in the central channel were determined. Quality of the received fiber depends on these parameters (Paschoalin et al., 2017). The obtained characteristics are the initial data for the design of such devices and the development of the technological process and equipment to obtain fibrous materials by the vertical blowing method with the required quality air, which in turn are the basis for the manufacture of construction materials based on polymers. The future potential research will deal with testing of the filtering process with filters from synthetic fibrous materials for the contaminations removal from the underground water that can also be soiled during the mining processes.

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Installation optimization of air-and-water sprinklers at belt conveyor transfer points in the aspect of ventilation air dust reduction efficiency

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Dust content in the air is one of the big hazards experienced in hard coal mines. In the result of mining of seams and driving roadheadings coal and stone dust is generated. Coal dust has explosive properties, and in a certain accumulation and at certain conditions, it can cause a disaster whereas stone dust has a harmful impact on a human respiratory system and may cause pneumoconiosis. Both kinds of dust are present in the mine air simultaneously but in different proportions and concentrations. Design solutions and a principle of operation of Bryza-1200 transfer point sprinklers are described in the paper. An application of these sprinklers in coal mines, in different configurations and development of this solution over a period of a recent few years, is presented. Test results are given and an analysis of dust reduction in the aspect of positioning the components of Bryza-1200 sprinkler, in relation to the conveyor transfer point, is described. A shape of the transfer point, its speed, the direction of ventilation air flow, dimensions of the working and a position of the conveyor transfer point in relation to the working axis have an impact on the installation.

Keywords: *dust, spraying system, prediction, optimization, neural networks*

Introduction

Apart from basic sources of dust generation in mines, there are many secondary sources, among others dust is generated by a movement of personnel as well as transfer points of belt and scraper conveyors. Dust control equipment or spraying devices are used for a reduction of dust. Dust control equipment (ProstaŒski and Jedziniak, 2013) stores sucked in dust in a wet or dry form. Spraying devices neutralize harmful and dangerous dust combining dust particles with water drops, eliminating volatility of dust (KwiecieŒ and Szponder, 1989; Shirey, 1985; ProstaŒski, 2013).

The efficiency of a spraying installation, which can vary from a dozen or so up to nearly 100% depends on a selection of design parameters (location, type, and a number of nozzles) and a selection of supply parameters.

An improvement of dust reduction efficiency is usually achieved by increasing supply pressure which is connected with an increased water flow intensity (KwiecieŒ and Szponder, 1989; Shirey 1985).

A reduction of nozzle outlet hole diameters causes a decrease in water flow intensity; however, then nozzles are subject to stacking and a loss of permeability. Another way, preferred by the KOMAG Institute (Bağaga and Jaszczuk, 2016; Karowiec, 1984; ProstaŒski, 2013, Fabian, 2015) and more and more commonly used, are air-and-water spraying installations in which compressed air is applied for an improvement of the stream spraying quality. These installations, in general, have a better dust reduction efficiency, a significantly smaller water flow intensity and they generate small droplets of median from a dozen or so to tens micrometers.

The KOMAG Institute, as a leader in developing and implementing air-and-water spraying devices, has contributed significantly to their dissemination in mines, which always led to an improvement of safety and work comfort (ProstaŒski, 2018, Ļernecký et al., 2015).

One of basic solutions of air-and-water spraying installation is Bryza-1200 sprinkler which has been applied in nearly a hundred systems, reducing dust concentration in the area of belt and scraper conveyors transfer points (ProstaŒski, 2013).

Bryza-1200 sprinkler

Bryza-1200 sprinkler (Fig. 1) is a device supplied with water and compressed air, which may be delivered from standard mine media, i.e., from the fire extinguishing pipeline and from the compressed air pipeline (ProstaŒski, 2013). Supply pressure of air and water in the installation varies from 0.3 to 0.5 MPa, at the water flow intensity - from 0.5 to 2 dm^3/min. The amount of the water flow, irrespective of the pressure, is established with flow controllers, thus eliminating a need of using pressure controllers. The whole installation usually consists of five nozzles, and it is sufficient for a dust reduction on the transfer point even by 90%. Standard construction of Bryza-1200 sprinklers includes a frame with installed nozzles and chains for its suspension as

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well as a box of media preparation, in which there are cut-off valves, filters, and flow controllers. The components are connected with flexible hoses. In the case of water pressure exceeding 1 MPa, a reduction valve is applied. Bryza-1200 sprinkler is additionally equipped with drain grid, to which the wet dust adheres. Such a simple construction of the installation ensures its failure-free operation and easy maintenance. During an observation of the sprinkler operation and in the result of suggestions made by its users, a few alternative solutions of Bryza-1200 sprinkler and its additional equipment, have been developed.

To automatize an operation of Bryza-1200 sprinkler, a few additional solutions have been offered. One of them consists in an installation of feed height sensor, located over the conveyor belt, indicating a presence of coal on the belt and causing a start of spraying nozzles through an electrovalve and a return valve controlled with pressure, coupled with it. Another solution, enabling to automatize an operation of Bryza-1200 sprinkler is an application of non-electric roller water divider, designed at the KOMAG Institute. A divider roller is set in rotary motion with a belt, it opens a water valve, causing a flow of water stream which supplies the nozzles with water and compressed air. Its flow is possible due to the use of a return valve controlled by air. Another solution, which has not been implemented yet, is a combination of Bryza-1200 sprinkler application with the dust sensor located over the conveyor. After exceeding the given value of dust concentration in the air, the dust sensor causes a start of the electrovalve opening a flow of water and indirectly a flow of compressed air. An element improving a spraying efficiency is a feeder of dust wetting agent. This agent, batched to the water, reduces its surface tension, facilitating adhesion of dust to water drops. The feeder, designed at KOMAG is a non-electric device, supplied with a stream of spraying water. The spraying installation is also equipped with return valves controlled with pressure. Their series of types have been designed for operational pressures of spraying installations. All these solutions have been designed to suit the best operation of Bryza-1200 sprinkler to local userǿs conditions, taking into consideration the accessibility of supply media and dust concentration.

Fig. 1. Bryza-1200 transfer point sprinkler.

Measurement results of dust reduction efficiency

Each time the first implementations of Bryza-1200 sprinkler have been connected with instructions for users within optimal use of the spraying installation and measurements of dust concentration reduction efficiency. In the majority of cases tests of total dust and respirable dust have been carried out (ProstaŒski, 2013). An assessment of desired positions of spraying systems in relation to the stream of running coal and a position of drain grid has been impeded by existing installation conditions, a shape of transfer point, a position of conveyors (angular, parallel) and a transportation direction of material in relation to the direction of the air stream. In the result of tests different results of dust reduction efficiency have been obtained, but usually, they exceeded 70%. It has also been observed that apart from certain exceptions, air-and-water spraying devices reach a higher efficiency of dust reduction for respirable dust than for total dust (Fig. 2). Probably it is caused by a degree of stream spraying to the diameter size of drops similar to the size of dust particles.

The taken measurements have shown that if the sprinkler is carried away, the dust concentration and reduced dust concentration as well as dust reduction efficiency decrease (Fig. 2).

Another reason for an impact on the results of dust concentration reduction is an installation of drain grid, which improves a sprinkler operational efficiency (Fig. 3). This phenomenon can be seen, in particular, for respirable dust (ProstaŒski, 2018; ProstaŒski, 2013). In general an efficiency of a spraying installation increases when the water flow intensity increases in the scope from 0.5 to 2.0 dm^3/min (in a few cases 10 dm^3/min). It can be concluded that if the air velocity increases, the spraying water flow intensity should be increased, keeping the right relationship between water flow intensity and air velocity. In accordance with the run-of-mine transportation direction with the direction of ventilation does not have a significant impact on dust reduction while using Bryza-1200 sprinkler. Other factors, such as geometric relationships of the spraying installation, i.e., the height of sprinkler attachment and drain grid over the conveyor and a difference of height of cooperating belts affect the dust reduction efficiency. Obtaining an optimal spraying efficiency is not a simple issue, and each time it requires an individual approach. Type of coal and use of agent reducing water surface tension and a location of the conveyor in relation to the working axis also have an impact on the dust reduction efficiency. However, these last parameters have not been recorded.

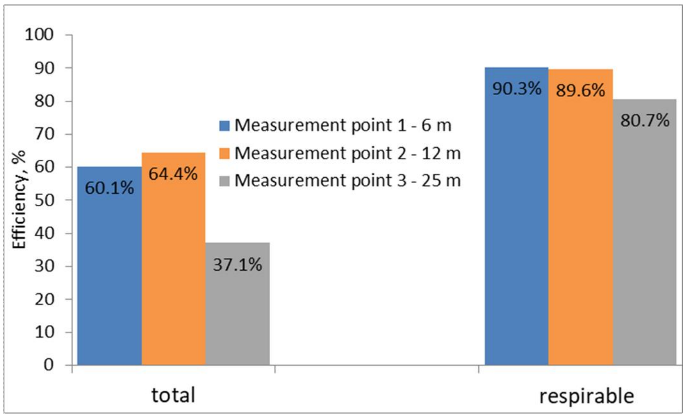

Fig. 2. Dust reduction efficiency in different measurement points.

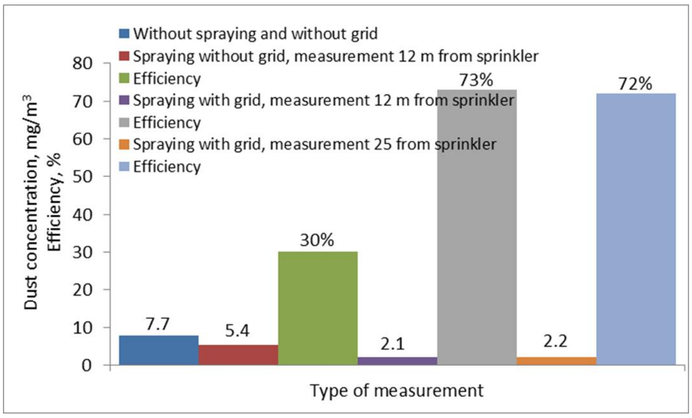

Fig. 3. The efficiency of total dust reduction: measurement without grid 12 m from the sprinkler; with grid 12 m from the sprinkler; with grid 25 m from the sprinkler.

Analysis of test results of Bryza-1200 sprinklers used in coal mines

As can be seen from the conducted analysis (Fig. 4 and 5), it is difficult to select individual parameters of optimal positioning of air-and-water spraying installation unmistakably.

The conducted measurements indicate an increase in dust reduction efficiency explicitly while using Bryza-1200 sprinklers when a concentration of respirable dust increases. Similar, although less apparent trends can be observed in the case of total dust reduction efficiency. From Fig. 4a it can be concluded that at smaller distances of drain grid from sprinkler a reduction of respirable dust efficiency can be seen. This trend is less observable in the case of total dust (Fig. 4b). It is probably connected with smaller volatility of bigger dust grains. Lack of drain grid also affects a reduction of sprinkler efficiency. In the paper, for a presentation of results, in the case when it lacks the drain grid, it has been assumed that it is installed in the distance of 30 m from the sprinkler, in other words outside the furthest measurement point. As can be seen, when it lacks the drain grid one of the lowest results of dust reduction efficiency are experienced (Fig. 4).

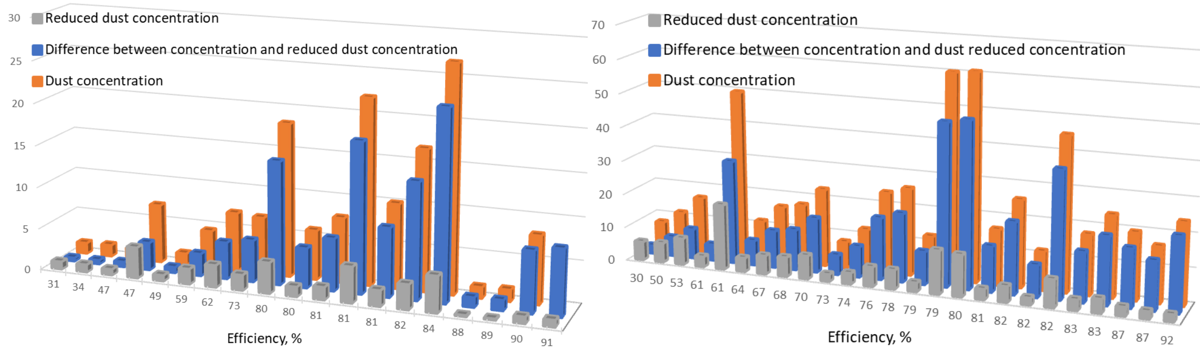

Fig. 4. Impact of dust concentration level on the efficiency of its reduction: a) for respirable dust, b) for total dust

A presentation of results, as regards total dust, is not unmistakable. It is affected by a repeatedly bigger weight of dust and significantly smaller distance of the settlement place from the dust raising place. However, only for about 4 % of results a dust reduction efficiency below 50 % and for about 30 % of results the efficiency lower than 70 % have been achieved. However, 40 % of results have had the efficiency exceeding 80 %. Such a high dust concentration reduction of respirable dust has been obtained for 58 % of measurements.

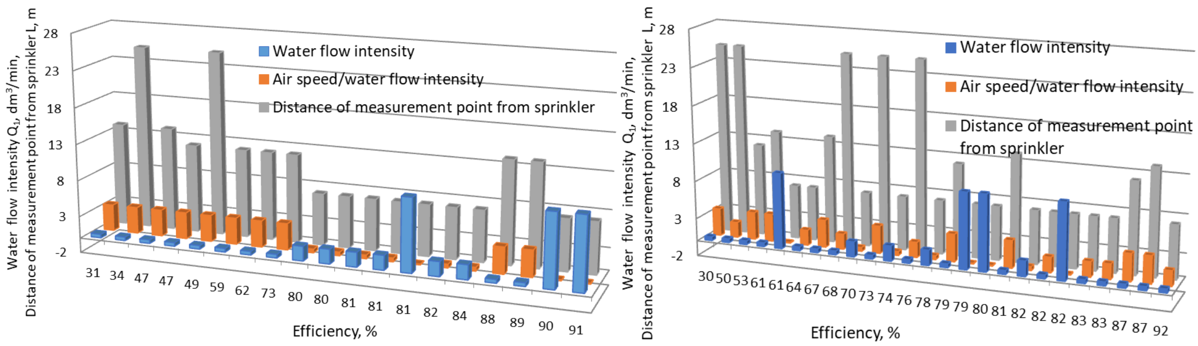

Fig. 5. Impact of selected parameters on dust reduction efficiency: a) for respirable dust, b) for total dust.

It should be borne in mind that the tests have been conducted in different configurations of the transfer point position, at the differentiated operational intensity and different types of coal, whose parameters have not been taken into consideration in this analysis. In the result of such an approach the obtained relationships should be treated as a generalization of the issue, and the results of the analysis show approximate values.

However, the conducted measurements show the usability of Bryza-1200 air-and-water sprinkler for a reduction of dust on conveyor transfer points. The high efficiency of dust reduction is achieved at a small water flow intensity, which has no negative impact on work conditions of the staff operating transfer points and it does not cause softening of the floor in the area of a transfer point.

Possibilities of selecting parameters of spraying installation

The tested solutions have shown unmistakably that not only mining conditions in the working but also design features and supply parameters of spraying equipment have an impact on dust reduction efficiency. It is mainly seen in the case of reducing respirable dust. A correct selection of the type and configuration of the spraying device, adjusted to the transfer point in the roadway working, is not simple and explicit.

A selection of spraying devices is made, basing on the experience of producers and users, and often it may not be optimal.

A difficulty in assessing the impact of individual parameters on a dust reduction efficiency causes a need for searching tools for general and criterial optimization of spraying installation application method in given conditions.

It should also assume satisfactory thresholds of achieved efficiency which will enable convenient use of sprinklers in given application conditions, without a necessity of meeting all the given parameters, resulting from an optimization process.

An application of conventional models (Branny, 2008; Colinet et al., 2010; Changchi, Zhizong, and Dewen, 1996; Karowiec, 1984; Konduri, McPherson and Topuz, 1997; KwiecieŒand Szponder, 1989; Lange, 1996) of coal dust displacement and settlement and its combination with water drops, using mathematical function for a description, enables an exact description of the phenomenon but it is not practical to a big extent for a selection of spraying installation parameters. Statistical models (for example, multi-dimensional linear) (Colinet et al., 2010), which give a possibility of imaging a dust reduction process and a selection of application parameters of spraying installations to given operational conditions, are more practical. However, these models are complicated as regards their generation, and they require some appropriate theoretical knowledge (Jasiulek, Stankiewicz and WoszczyŒski, 2016; Latos and Stankiewicz, 2015; Stankiewicz, Jasiulek, Rogala-Rojek and Bartoszek, 2013).

Use of artificial neural networks seems to be a simple method of constructing a model describing a phenomenon of dust reduction using a spraying installation. This method is used successfully in many branches of science and industry for imaging processes and their control (Shirey, 1985; ProstaŒski, 2018; ProstaŒski, 2013).

Use of neural network consists of its training on a certain data set to foresee output data with it later on. It is essential that the network gets an ability to foresee exclusively on the base of delivered data. However, it is not supplied with any formulae or relationships between the input data and foreseen results. A right selection of the network type and structure is also of importance. An advantage of neural networks is a possibility of searching models for processes or phenomena, whose structure or operational principles have not to be recognized and described yet and only their quantities, which affect the phenomenon under testing and its result, are known. Thus this tool can be used successfully for modeling a dust reduction efficiency while using air-and-water spraying system.

Construction of a neural model requires a determination of network input parameters and its reply. Input and output quantities are measurement data and parameters obtained from conducted measurements on real objects.

It is best to apply classical classification models, for example, multi-layer perceptron trained with the use of a reverse propagation of errors (Jonak, ProstaŒski and Szkudlarek, 2003) for a description of this kind of phenomena. While constructing a model, it should also be determined which of the variables will be the network replies. The network reply enables a correct selection of spraying installation parameters in the aspect of dust reduction efficiency.

The tests of spraying equipment, conducted in mines, indicated an impact of the environment on spraying efficiency. Such parameters are among others:

- air flow velocity (m/s),
- the quantity of feed on the conveyor (Mg/h),
- the quantity of dust generated by the transfer point and its environment,
- coal moisture content (%),
- accordance of air flow direction with the conveyor movement direction (+/-),
- location of transfer point in relation to the working axis,
- the height of transfer point (mm),
- the difference in conveyor belts levels on transfer point (mm),
- run-of-mine transportation velocity (m/s),
- total dust concentration in the mine air (mg/m^3),
- the concentration of respirable dust in the mine air (mg/m^3).

Achieved efficiencies will also be input data of the network:

- reduction efficiency of total dust in the mine air (%),
- reduction efficiency of respirable dust in the mine air (%).

Above given quantities may be input parameters for a neural model. Other parameters, describing a specificity of a spraying system installation, are among others:

- spraying water flow intensity (dm^3/min),
- air pressure (MPa),
- water pressure (MPa),
- number of spraying nozzles (pieces),
- the height of sprinkler suspension (mm),
- type of nozzles.

These and other parameters can be network replies, in other words, quantities which can be affected. In relation to the number of measurements and degree of individual parametersô impact on a reply, a suitable number of layers in a neural model should be accepted, sufficient for getting a reply with the proper correlation coefficient, where a model will react properly on changes of these parameters. Another important feature of a neural model is an acceptable time of learning and getting a reply.

Reliability of a correct model selection and of getting replies close to these, which are obtained in real conditions, gives a testing set, on which a quality of the network adaptation to independent sets of measurement data is checked. To obtain reliability of a correct model operation, the network is checked on the third, independent validating set. A correlation coefficient which equals 1 proves a stiff adaptation of the network to the relationships under analysis, and it will render impossible to get correct network replies, for example, out of the scope of the data under consideration (ProstaŒski, 2002).

A model with a number of neurons in the input layer equal to the number of input data and the number of neurons in the output layer, equal to the desired number of network replies, seems to be most obvious. Intermediate layers, aimed at improving the efficiency of the network operation, should be included in the structure of such a network. Obviously, in the case of a smaller experience of a creator or lack of recognition of a phenomenon under description, intermediate layers should be selected experimentally, obtaining in each following modeling, a network of acceptable learning speed and a satisfactory quality of reply. However, assuming that this phenomenon is tested, using this method for the first time, it is correct to assume one reply of a network, i.e., one output neuron. In this way, it is possible to construct a few networks, each time expecting a different reply. Some of the output data, which can be affected in a smaller degree, can be placed in an input layer, for example, water flow intensity or water and compressed air pressure.

Effects of modeling using an artificial neural network

Initial tests of measurement data sets with use of artificial neural networks have shown that a set, concerning respirable dust, is much more foreseeable than the total dust. A possibility of conducting an introductory analysis easily and quickly can give at once a reply as regards correctness of conducted considerations, confirming or not suppositions in the scope of an impact of individual factors on dust reduction efficiency.

An assessment analysis of the impact of a drain grid on a dust reduction efficiency has been conducted as the first one. An automatic network architect has generated 10 network models, where the best model is shown in Fig. 6. Basing on this model an average error 3.6, a standard deviation 2.6 and a correlation coefficient 0.13 have been determined. These quantities disqualify the network as regards its correctness and cannot be accepted at a selection of a drain grid location.

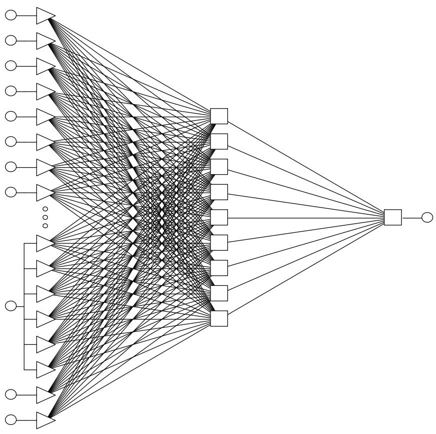

Fig. 6. Model of a neural network for searching an optimal position of drain grid in the case of total dust measurements.

Basing on collected data, it has been possible to present an impact of drain grid position (*d*) on dust reduction efficiency (*s*) in relation to dust concentration, a difference between dust concentration and reduced dust concentration (**φs**) and a distance of the sprinkler from the measurement place (l) ï Fig. 7. From the conducted analysis it can be concluded that the bigger distance of drain grid (*d*) and the smaller dust concentration (*a*), the bigger dust reduction efficiency (Fig. 7a). The situation looks different in the case when the difference between total dust concentration and reduced dust concentration decreases, then an increase in dust reduction efficiency is achieved at a smaller distance of the drain grid from the sprinkler (Fig. 7b). An increase of dust reduction efficiency is achieved at the biggest distance of the grid and possibly small distance of taking measurements of dust concentration from the sprinkler (Fig. 7c). It should be borne in mind that the presented relationships show the relationships between individual parameters in the context of relations determined by the neural network.

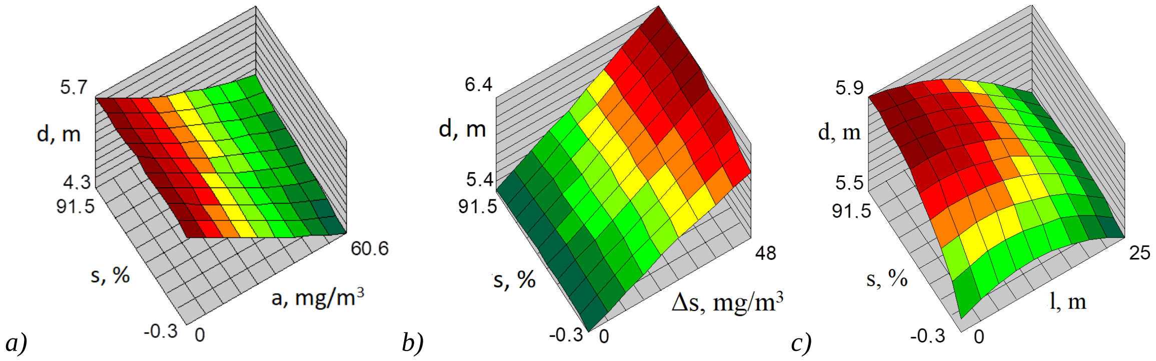

Fig. 7. Impact of drain grid distance from the sprinkler on reduction efficiency of total dust in relation to a) dust concentration (a); b) difference between dust concentration (φs); c) distance from sprinkler to concentration measurement place (1).

A significantly better quality of the network has been achieved for an efficiency assessment in the context of total dust measurement concentration. The best structure, among 10 suggested networks, is presented in Fig. 8. In the case of this network, an average learning error has been about 2, at an average standard deviation of about 5 and a correlation coefficient of about 0.7. Obviously, these parameters also disqualify this model. However, it is much more reliable than the previous one.

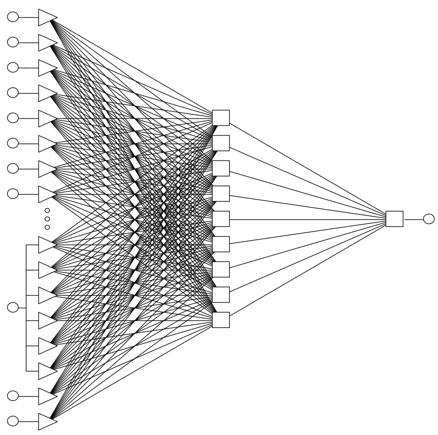

Fig. 8. Model of the neural network at searching for optimal placement of total dust concentration measurement.

Testing an impact of measurement place (1) of dust concentration (*a*) on dust reduction efficiency (*s*) in relation to dust concentration (*s*), a difference in dust concentration and reduced dust concentration (**φs**) and the distance of drain grid suspension from a sprinkler is presented in Fig. 9. In this case the bigger distance of the measurement place from the sprinkler (1), the higher efficiency of decreasing total dust quantities (Fig. 9a). A similar trend can also be observed at a decreasing difference between dust concentration and reduced dust concentration (Fig. 9b). Different relationships occur in the case of an impact of distances between measurement places and a sprinkler on dust concentration in d/v_1, which have an insignificant impact on dust reduction efficiency (Fig. 9c).

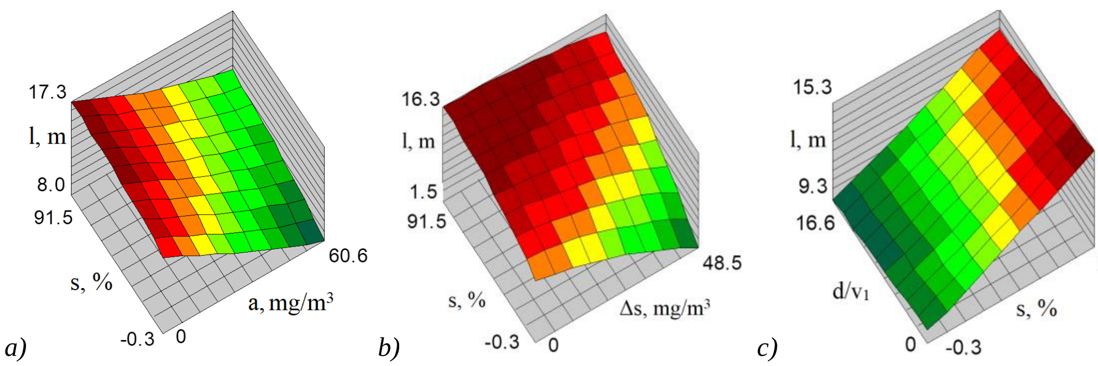

Fig. 9. Impact of measurement distance from the sprinkler on a reduction of total dust concentration in relation to a) dust concentration (a); b) difference between concentration and reduced dust concentration (φs); c) air velocity and the distance of drain grid from the sprinkler (d/v1).

A better quality network has been obtained at an assessment efficiency of respirable dust reduction in relation to the distance of drain grid suspension. The best network, among 10 selected ones, is shown in Fig. 10. In the case of this network, an average learning error has been about 2 and a standard deviation ï about 3 and a correlation coefficient has been about 0.99. These results of network learning are significantly better. It also concerns imaging relationships accompanying a reduction efficiency assessment of respirable dust, although also, in this case, they may be regarded as insufficient ones.

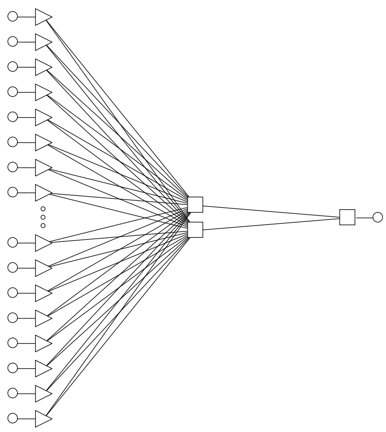

Fig. 10. Model of the neural network at searching the optimal suspension place of drain grid for respirable dust fraction.

The distance of drain grid suspension in relation to the parameters *s*, ***φ****s*, *a*, *d* is shown in Fig. 11. In all the three cases, these relationships are shown by inclined planes. When the distance of the grid suspension from the sprinkler increases, the reduction efficiency of respirable dust increases and it also increases when the dust concentration decreases (Fig. 11a). The efficiency of dust reduction also increases proportionally to an increase of the distance of the drain grid from the sprinkler and an increase of the difference between dust concentration and reduced dust concentration (Fig. 11 b). An increase of reduction efficiency of respirable dust is also observed at the simultaneous increase of the drain grid suspension distance and the distance of measurement place (Fig. 11 c).

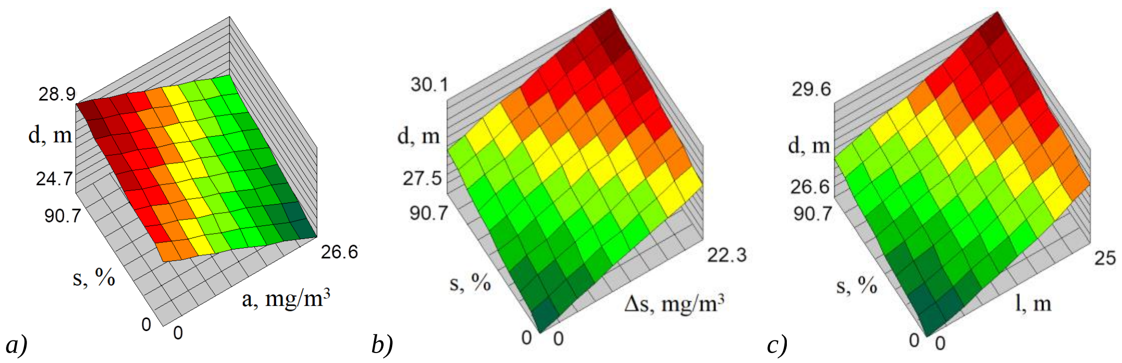

Fig. 11. Impact of the distance between drain grid from the sprinkler on reduction efficiency of respirable dust in relation to a) dust concentration (s); b) difference between dust concentration and reduced dust concentration (φs); c) distance of taking concentration measurement.

Even better results of learning have been obtained for a reduction efficiency assessment in the case of respirable dust in relation to the distance of dust concentration measurements (Fig. 12). An average error has been 0.6, standard deviation - 2.7 and correlation coefficient - 0.98.

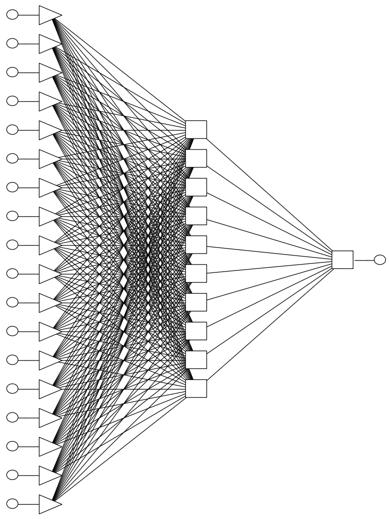

Fig. 12. Model of the neural network of searching optimal measurement place of respirable dust concentration.

In the case of testing a distance impact of concentration measurement on reduction efficiency of respirable dust in relation to *a*, ***φs***, d/v_1, bent, regular surfaces (Fig. 13) of these relationships have been obtained. A location of measurement place has a small impact (Fig. 13a) on an efficiency assessment of dust reduction in relation to respirable dust concentration. This relationship looks different in the case of different ***φs*** quantities, where the biggest efficiency is achieved for the furthest measurement places and the biggest ***φs*** (Fig. 13 b). There is no essential impact of 1 and d/v_1 on dust reduction efficiency. Then an efficiency increase depends exclusively on an increase of d/v_1 (Fig. 13 c).

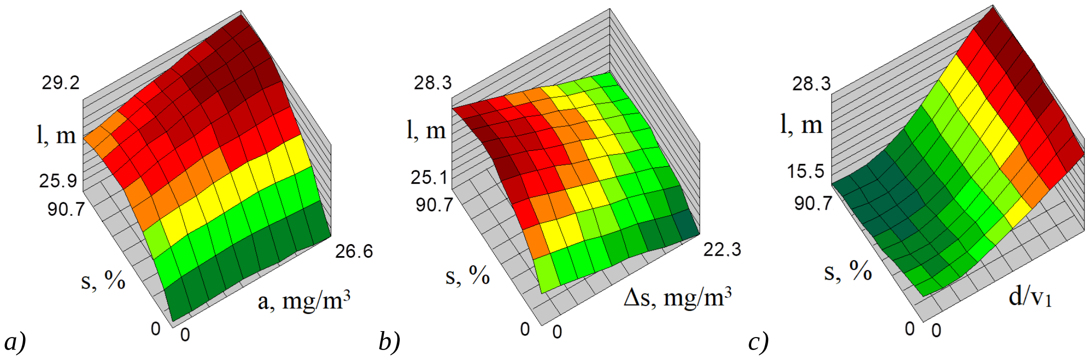

*Fig. 13. Impact of measurement place on reduction efficiency of respirable dust in relation to a) dust concentration (s); b) difference between dust concentration and reduced dust concentration (**φs**); c) air velocity and the distance of drain grid from sprinkler (d/v1).*

Summary

The tests of reduction efficiency of air-borne dust, using air-and-water Bryza-1200 sprinkler, conducted in mines, have shown a high efficiency of this installation, however basing on these tests it is difficult to assess a real impact of individual parameters of a spraying installation on the efficiency. It is also difficult to assess the impact of the weight of these parameters at different mining conditions. The presented relationships of an impact of individual parameters on dust reduction efficiency reveal many diversities, in particular in the case of total dust. Having at disposal, a relatively small number of measurements and incomplete data as regards conducted measurements, an application of a simple and effective tool, such as neural networks for presenting an impact of individual parameters on dust reduction efficiency and trials of imaging relationships among all the available quantities, is suggested. Due to this fact, the use of neural network seems to the most appropriate and the simplest approach to the phenomenon. Basing on the obtained test results, preliminary models of neural network have enabled to image a phenomenon of air-borne dust with different, not always satisfactory quality. The main objective of this experiment has not been to get the best reply, but to show that getting it is possible in a simple way, without a deep understanding of relationships among individual parameters. Initial tests of spraying installations with the use of neural networks, conducted at the KOMAG Institute have indicated the right direction of research work despite a relatively small population of learning and validating sets. In the learning process, as regards an efficiency assessment of respirable dust, a correct image of spraying parameters has been achieved. Within a simulation framework a network of multi-layer perceptron type, consisting of a dozen or so neurons in the input layer, ten neurons in the hidden layer and one output neuron, imaging an optimal measurement place of dust concentration, has been suggested. This network has been selected as the best one from 720 tested models of the network. It has been approved by the Statistica-Neural Networks Software as effective and correct, and its average learning error has been 0.6, standard deviation - 2.7 and correlation coefficient - 0.98.

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Terminological definition of the terms ĂPingeñ(Binge)

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The paper comprehensively analyses the montane concept of pinge (ping). It points to the origin of the word pinge and its primary meaning in historical - medieval mining. The main part of the paper is concerned with the evaluation of the conceptual terminological inaccuracy of the word pinge and analyses the individual historical montane shapes in detail. These shapes are often terminologically interchanged with pingen and also inaccurately considered pingen. These are surface concave montane shapes of relief in general, i.e., verhau, shurf, sinks, depressions, etc. These shapes have fundamentally different morphological and morphometrical properties and parameters, considering the different genesis of their origin. Nevertheless, these basic properties, which are not interchangeable were not considered in different scientific disciplines and were regarded as pingen. This part of the paper also evaluates the basic properties of pingen based on their relation to the relief and landscape, which must be studied from anthropogenic geomorphology and mining engineering. Final part of this paper offers the results of the research based on the analysis of medieval and modern archive sources, the oldest montane papers, the latest scientific studies and field research, which unequivocally establish the meaning of pinge in mining engineering (montane anthropogenic geomorphology), which is acceptable for all scientific disciplines concerned with this unique shape. A part of the final discussion is the exact definition of the term pinge and its meticulous delimitation.

Keywords: *pinge, concept genesis, conceptual terminological inaccuracy, morphological and morphometrical properties, new definition*

Introduction

Termination of mining activity in many mining areas during the 20th century brought a slow but important change in the outlook on the remains of mining activity. In many rich medieval mining sites significant terrain changes took place and after mining and processing of raw material the landscape was altered significantly, often degraded severely and recultivated only minimally. To this day there are many administration buildings, ore sorting machines, crusher houses, smelting houses, adit collars, drainage adits and shafts in the vicinity of mining and ore processing sites. However, it can be said that the cultural and historical value of these buildings was realised in the second half of the 20th century, followed by the creation of mining museums and exhibitions. With the development of tourism and its various forms during the 20th century a very specific type of tourism ï mining tourism - came into being, gradually hierarchically incorporated into geotourism. Abandoned ghost towns came back to life and depopulated mining areas gradually became centres of attention once again. Various ways of understanding of customary, initially mining expressions have appeared across scientific fields with the development of research methods and evaluations of various manifestations of mining activities, but also in their methodological classification and determination. An example can be the inconsistent view on the meaning of the Old German word ñpinge,ò or ñpinging,ò which is understood differently in mining disciplines and for example in post-war European geomorphology significantly complicates classification of montane anthropogenic forms of relief. The following study looks for the primary meaning of this term, using the original expert mining literature and archive map sources.

The origin of the word pinge comes from the old (early Medieval) German, where the word Bingen denoted pit or depression, probably tunnelled anthropogenically. Medieval German mining then gradually adopted this term. The first prospectors began to name randing and opening pits, which they tunnelled at the exit of ore veins or ore pillar on the surface. Primarily these forms at the beginning of early Medieval mining occurred at first during searching for deposits of industrial minerals (randing = pinging), which could gradually become surface or shallow subsurface mining. The significance of this type of ñsurveyò grew upon searching for the directional sequencing of the ore veins in the surface projection and upon determining their incline.

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The technique of exploring the deposits and subsequent surface and subsurface mining using mining pits ï pingen was not discovered during the Middle Age. Logical principle of this method had already been used by prehistoric and ancient miners.

Theoretical framework and background

Beginning and history of using pingen (pinging) as a mining technique

The first anthropogenic mining depressions were produced with a gradual increase in demand for quality raw materials in the Stone Age (Paleolithic period, three million ï 8 300 BC). The actual surface mining of inorganic minerals ï stone began in the later Stone Age (Neolithic ï about 6 000 BC), leading to the formation of mining pits, which can be identified as pingen. The demand for quality stone with specific properties (fissility with sharp edges, hardness, resistance, and later drillability) exhausted not only surface sites of its collection but also those sites, where the stone was extracted on the surface from the weathered or non-weathered surrounding rock. As a primaeval miner gradually penetrated deeper for quality stone, the first shallow mining pits were created on the plains and slopes. Upon removing overburden, that is the soil and overburden rocks, there was a waste, which was spilt near the shallow pits, forming ring-shaped mounds (waste dumps) around the ñmining pit.ò Extremely corrugated relief emerged at the mining sites. During the mining process, the man could proceed steadily and dug pits after pits; in this case, the excavation material was often deposited in the previous pit. The active mining pits have gradually acquired a characteristic funnel shape, which is now called pingen. Primary surface mining in pingen resulted either in mining pits or open quarries (if mining spread to the sides, only then to depth), or mining shafts (if mining progressed primarily to the depth and only then to the sides) (Fig. 1).

The process of genesis of surface forms nowadays referred to as pingen, can also be described chronologically. By gradual monitoring the quality of extracted material and its selective extraction in the direction to the middle of the Earth, the pingen (mining pits) gradually deepened and shafts 15 to 20 meters waste dumps around deep occurred, depending on the height of groundwater and difficulties it caused. Their diameter was originally close to their depth. After sinking of a shaft, their bottom was gradually expanded, and the bell-chamber with a circular plan was formed around the perimeter. Simple ladders made from tree trunks and branches were used to enter the shaft. If the bell chamber was larger, the dump was heaped directly into its space (backfill). The next step towards the formation of broken mines was the star-shaped formation of the underground tunnels from the bell chamber or the bottom of the shaft (thill). Respective corridors followed the occurrence of extracted material, or the extracted mineral deposits and created vast underground spaces ï the first mines. The height of corridors reached only the proportions necessary for human movement, up to 1.2 m, their length could be more than 20 m. Corridors ran from the entrance shaft in a star shape. With sufficient occurrence of quality stone, the tunnels were expanded to the sides, and first mine chambers were created underground (Borkowski, 1995).

This mining technique was used in the following millennia; it was the main mining technique of surface and subsurface mining even in the Middle Ages.

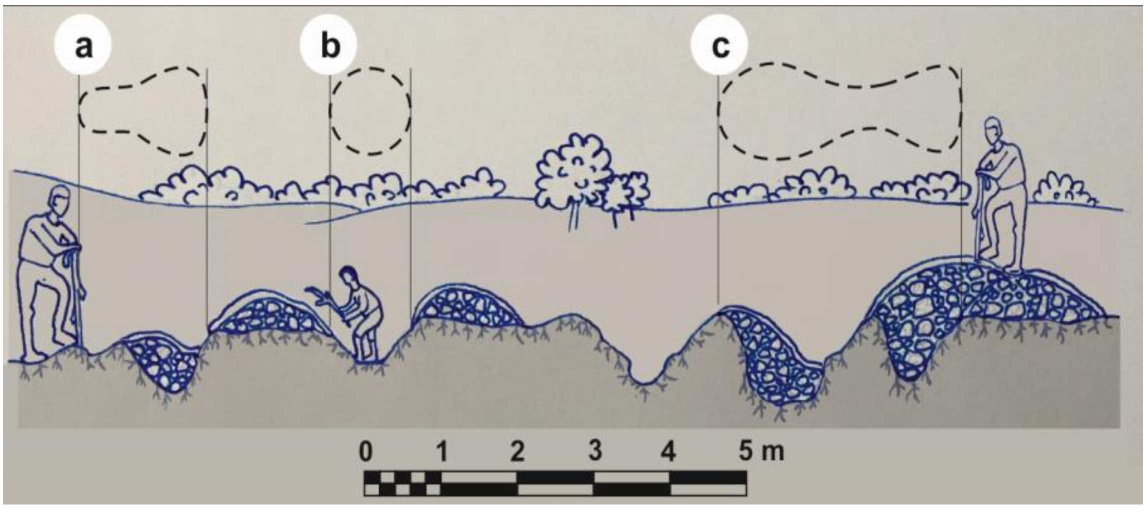

Fig. 1. Stone mining using surface mining pits ï pingen. Legend: a) combined pinge, b) pinge with a regular circular plan, c) pinge with an irregular plan. Upon progressing mining, exhausted pingen were often covered with heaps of dump.

In the Middle Ages, mining rich ores on the surface in oxidation zones was gradually followed by subsurface and deep mining of minerals. The deep part during the Middle Ages is thought to begin from the 12th or 13th century. Pingen as a mining method became more and more obsolete, and in modern times they have exclusively been used as exploring and searching method. In Jan Möhling's textbook on mining geometry, the key shows both mining and exploring pingen (Fig. 2), including waste dumps around them (Möhling, 1793).

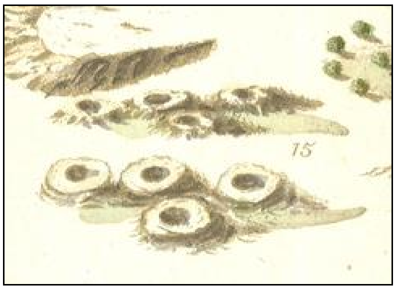

Fig. 2. The map key of Johann Möhling published in 1793 (Möhling, 1793).

General occurrence of pingen created while searching for deposits (geological survey) rises until the 16th and 17th century. The second type of surface active mining pingen, created by a drop of hanging wall into the mine basement began to appear later (18th and 19th century). While the first type did not reach large proportions (often only 1-2 m deep), the second type reached a depth of several meters, even tens of meters (depending on the size of undermined space, the thickness of hanging wall, wall stability, etc.). This form of relief and its sudden emergence is problematic especially in inhabited areas and sometimes is periodically filled with water. Unlike natural forms, pinge is an artificial object of anthropogenic origin with mining or searching (exploring) function. Regarding shape, and partially regarding their origin, they resemble natural karst forms of relief ï karst pits, sink-holes, which however lack heap or ring-shaped mound.

Inconsistency in the meaning of the term pinge

Currently, in mining or anthropogenic geomorphology, but also in other sciences, where it is the object of study (thus also among the general public), the understanding of the word pinge is broad, inconsistent in meaning, even fundamentally different. This problem can be observed not only in Central European science (Poland, Slovakia, Czech Republic, Hungary), including German-speaking countries but also in European or world science, written in English.

Nowadays, the term pinge (die Pinge ï in German) does not have an exactly defined object of naming in English, therefore does not name an actual anthropogenically created montane relief (as it is in other languages), but there is also no definite fixed nor word equivalent or phrase. Sporadically, in scientific papers written in English, there is a phonetic form ï ña pingeò (plural ï pingen), taken over from the German language, however, more often there are other terms without specific meaning. For example: ñopen mining pit,ò ñmining hole,ò ñmining sink-holeò or ñanthropogenic sink-hole.ò Descriptive expressions such as ñopencast mining,ò ñmine slumpò or ñglory holeò are also used.

It is clear that the term ñpingeò (in German ï Binge, Pinge, in Czech ï pinka, in Slovak and Polish ï pinga) has not been exactly defined by any scientific discipline yet, be it mining, anthropogenic geomorphology, economic geology, geography, geoecology or history, which are concerned with research and analysis of these relief relicts. This term, although actively used in mining terminology, remains without the exact definition of form (shape), process (activity) by which it came into being, or conditions, which it must meet. Terminological consolidation of this term is largely outside the fundamental interest of the disciplines in question. It can be said that they are in a terminological ñshadow,ò despite being very common, widespread and visually interesting and specific relics of old mining.

The term pinge or its close equivalent refers to many genetically and morphologically different surface concave forms of montane relief. This ambiguity is caused by taking over the term by geographers in the 20th century aits using without realising its exact meaning in mining and its application to a wide range of visually similar shapes. This inconsistency then leads to numerous misunderstandings and misinterpretations of the term. This way, all surface mines in the shape of the pit with various plans up to outstanding line shapes (excavations, abatis, mining excavations), often of great proportions are called pingen, regardless of the process, in which they originated, the function they performed and properties they have. Even greater disunity arose after the term ñpingenò was used to name brakes or debris-filled mines of concave shape, without a heap or ring-shaped mound. This problem was pointed out by geologists (Kuģvart & Böhmer, 1972), montane archaeologists in the Czech Republic (Nováļek, 1993; Hrubý et al., 2016), but also anthropogenic geomorphologists in Slovakia

(Hronļek, Rybár & Weis, 2011; Hronļek& Weis, 2010a, 2010b). This problem also prevails today in German mining, montane archaeology, and related sciences. As K. Nováļek writes, the partial thematic solution was the work of a German geographer D. Düsterloh (Düsterloh, 1967), who, however, did not find any followers, who would solve this problem in a complex way.

To this day, this problem has not been solved, and there are no general classification systems, which would explicitly define and divide concave montane relief shapes acceptable to all scientific disciplines which study these shapes.

Until now, the Czechoslovak geomorphological school or specialists in anthropogenic geomorphology have confirmed with German view on the issue of pingen. Contemporary German mining literature considers pingen shapes created directly by surface mining activity, by digging, deepening or disintegration (Grabung), regardless of the purpose of this activity (mining, exploration, etc.), but also shapes created by indirect mining activity, in breaking or collapsing (Einsturz) of hanging wall. This approach was also adopted by the Czech geomorphologist L. Zapletal (1968, 1969), whose work was also based on German school. He introduced the term ñpinkaò into Czech science as an equivalent of the German ñpinge.ò German literature, however, uses the term ñpingeò to denote shapes, created by inbreaking or collapsing of hanging wall (overburden), almost exclusively in cases where this secondary form continues to be an active mining space. This fact, however, is not defined as conditional for using the term ñpingeò.

As it was already mentioned, the process of formation of these forms of relief, their function in mining and their features were not taken into account geographically. This approach (ambiguity and inconsistency) was taken over by other experts in the Czech Republic, for example, Kirchner, Smolová (2010). In Slovakia, this view was taken over by the doyen of Slovak montane archaeology J. Mazúrek (1965, 1987, 1989) as well as by younger experts, for example, V. Ļech with J. Krokusová (2013), and Krokusová with Ļech (2014). The situation in Poland was similar, as was the case with older authors (Klimaszewski, 1961, 1978; Hornig, 1968), which continues with current generation (Pogrórski, 1999, 2001; Wójcik 1996).

The problem of inconsistency of the concept of ñpingeò on all levels was pointed out by numerous current experts in the Czech Republic, for example, K. Nováļek (1993) and K. Hrubý with a team of co-authors (Hrubý et al., 2016). These authors have tried partially unify this problem. In Slovakia, the attention to this problem was paid by P. Hronļek and K. Weis (Hronļek & Weis 2010a; Hronļek, Rybár& Weis, 2011), who attempted to define the geometry and necessary properties of this form of relief.

In general, the term ñpingeò can be defined by monolingual dictionaries and diverse scientific literature as a funnel or sphenic depression, created or related to mining activity. Its origin is directly related to the surface mining of minerals or with collapsing of shallow mines.

It can be said that this definition is general, vague and ambiguous. It is also confusing since the term ñpingeò is also used to refer to sink-holes and decline, which again suggests the absence of emphasis on Genesis, function and typical properties of this form.

Division of ñpingen,ò more precisely the concave surface shapes of montane relief from their relevant properties ï genesis, shape and original function

Nowadays almost all concave montane depressions without more specific identification of genesis, shape, proportions or original functions are called ñpingenò in anthropogenic geomorphology. This approach, however, does not pass the basic sorting of montane shapes, which can be divided into:

- those, which were created by a direct activity of anthropogenic geomorphological processes during mining activities, i.e., as a result of mining activity and using of mining techniques ï the so-called primary mining relics
- secondary mining relics, which are a reaction to a mining activity or they are created for mining. They were not created by anthropogenic geomorphological processes (mining activity), but the activity of natural geomorphological factors (Zapletal, 1968, 1969, 1978; Hronļek, Rybár & Weis 2011; Ļech & Krokusová 2013).

These processes or their combination are then reflected also to their morphological and morphometric parameters, which at first glance create similar or identical shapes from individual concave anthropogenic montane shapes of relief. This fact leads to their mistaken association and identification as ñpingen.ò This error is pointed out by historical context, which stems from medieval mining and geognosy.

Despite the fact that there is no unified relevant scientific classification of surface concave montane shapes of relief and exact terminological determination of the word ñpingeò, a basic division can be established, based on the genesis of their origin in accordance with the current understanding of a wide spectrum of scientific disciplines worldwide to: dug-out (excavated), crushed and debris-filled.

Dug-out (deepened, sunken) pingen are created by a direct exploring or mining activity, based on which they can be divided into two subtypes: exploring pingen and mining pingen. They are created by digging or disintegration of hard rocks of the mantel. This process is called pinging. Once the pits are abandoned, they change their basic shape by the influence of natural geomorphological factors. The duration and intensity of these factors determine the stage of development of shape from almost ideal inverted hollow cone to a reversed shallow elliptical paraboloid.

This shape then can also be defined as a funnel-like terrain depression, which was primarily created while surface mining of minerals (pinging), or while exploring. Pingen were excavated on assigned small-scale mines with a mandated randing right.

The method was widely applied in the initial survey of a deposit, when it was necessary to map out exits, width and the course of veins on the surface, as well as while looking for the continuation of the vein structures just below the surface, or when determining their incline. This is why the line arrangement of pingen rows or disorganised texture of pinge fields can be often encountered (Mazúrek, 1987). Such an understanding of the term ñpingeò is more akin to the original old mining term, which referred to shallow randing surface pits, which were used to either monitor the course of ore veins on the terrain surface or to carry out selective mining. The funnel-like shape of ñpingenò is strikingly similar to naturally created karst sink-holes. However, they cannot be confused.

According to Polák (1952), Slotta (1991), Hronļek, Weis (2010a) pingen are exploratory or mining pits. A similar view is shared by K. Nováļek, who names them ring-shaped mounds. However, this terminology is not accurate, since ring-shaped mound, as a specific type of heap is only convex, above the ground part of pinge (Nováļek, 1993). The inconsistency in defining mining or exploratory pingen is also reflected in the fact that some authors consider ingenuine mining pits, or trenches, which they denote as line ranging or mining ditches (Weiss, 2005).

Fig. 3. Digging of exploration pits in the first half of the 16th century by G. Agricola (Agricola, 1556). Line grouping of pingen to pingen rows (a) and sheet layout of pingen into the pinge field (b).

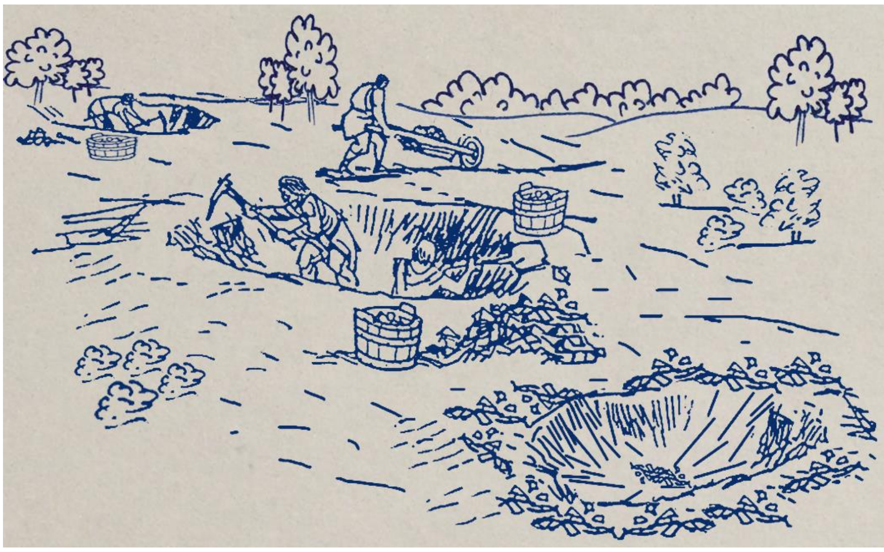

Fig. 4. Reconstruction of the digging of exploration and mining pingen inline form after the exit of ore vein to the surface ï pingenrow.

The correctness of the definition of pingen of this origin is pointed out by one of the oldest expert montane works of G. Agricola (Agricola, 1556), where the author mentions concave shapes of similar character (Fig. 3). When defining and identifying explorative or mining pinge (Fig. 4, 5), its second integral part in the shape of the convex part, made of the heap cannot be forgotten. Regarding pingen, the basic shape of the heap can be distinguished, which was created especially by upthrowing of disintegrated material evenly around its perimeter. These heaps are created on flat relief and are called ring-shaped mounds. On the slopes with moderate incline, the heaps have a plan in the shape of the moon (reminiscent of letter C), which are similar to desert sand barchan dunes. Typology of montane heaps is a concern of Zapletal (1968, 1969), Hronček a Weis (Hronček & Weis, 2010b, 2014). The slope heaps were created in the steepest slopes since throwing disintegrated rock only at the lowest point of the edge of the pinge required the least amount of energy.

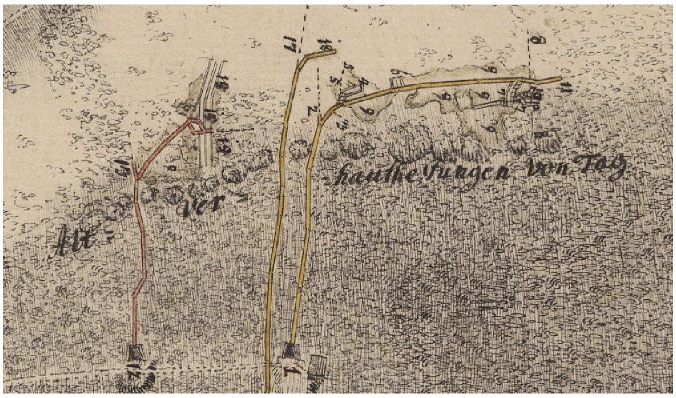

Fig. 5. Old mining pingen in the transitional stage to overhaul (HKG VI. Inv. Nr. 00095).

Mining pings are known for example near the town Suhl in Thuringian Forest (Thüringer Wald) in Germany or the historic ore ward Siegerländer in western Germany, on the hills Mědník, Díže in the Ore Mountains in the Czech Republic, pingenon the vein Schweizer in Jáchymov in the Czech Republic, Lomnické pingen to the north from Sokolov in the Czech Republic, the pingen from the vein Terézia in Banská Štiavnica (Fig. 6) and extensive pinge fields of vein Rabenstein from early Middle Ages, as well as veins Východná, Štefan, Leopold, Brenner, Moltra etc. in Hodruša-Hámre in Slovakia (Kaňa et al., 2016) and others (Fig. 7, 8, 9).

Fig. 6. Abandoned mining pinge on the Terézia ore vein exit in Banská Štiavnica.


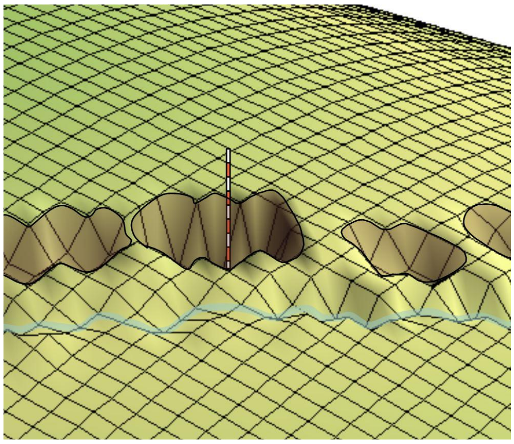

Fig. 7, 8. An example of the exploration pinge (on the left) and its 3D model created by detailed field research (on the right). Pinge is in the pingen row on the ore vein exit (east direction from Margecany) in the Čierna hora Mts. (eastern Slovakia).

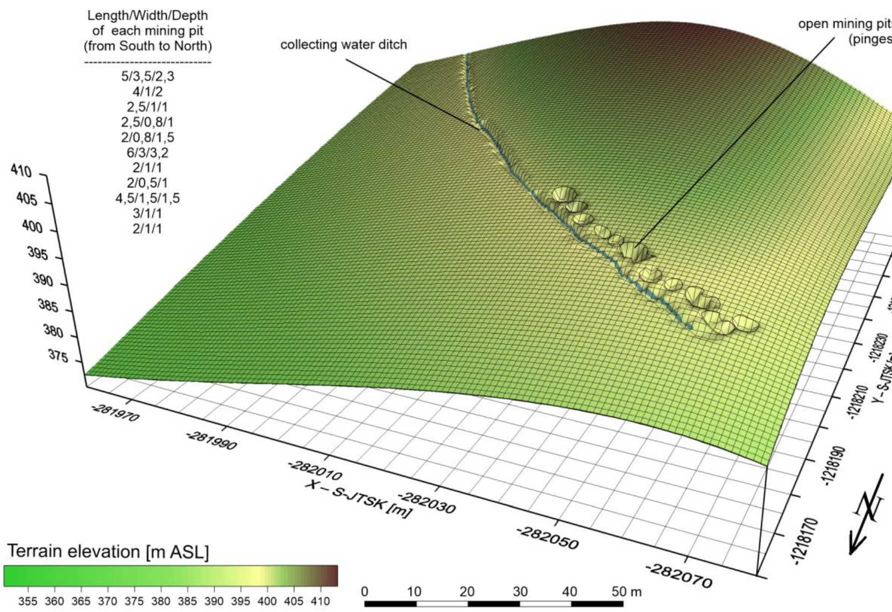

Fig. 9. 3D model of exploratory pingen row dug on the ore vein exit (east direction from Margecany) in the Čierna hora Mts. (eastern Slovakia).

Collapse pingen (Fig. 10, 11) did not occur directly through the extraction of mineral raw materials, but only as a secondary consequence of natural geomorphological processes. However, we propose to consider as pingen only those who were still part of an active mining area even after the collapse of their overburden, where the mining continued partly using surface mining (in the sense of the German approach).

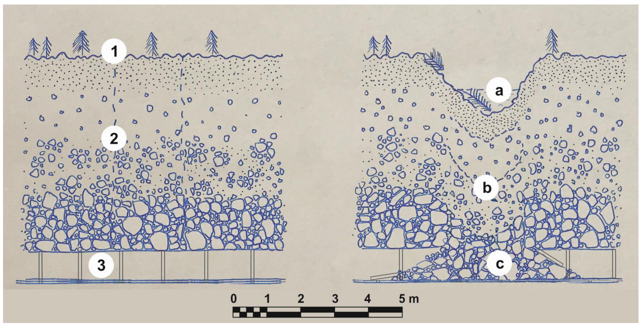

Fig. 10. Pingen created by a sudden (catastrophic) collapse of the overburden. Legend: 1. natural relief, 2. overburden layers, 3. mining underground, a. Collapse pingen, b. Collapsed overburden,c. mining collapse.

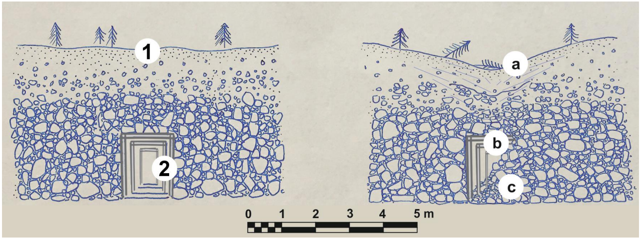

Fig. 11. Pingen created by a sudden (catastrophic) collapse of the overburden. Legend: 1. natural relief, 2. mining corridor, a. Collapse pingen, b.mining corridor, c. mining collapse.

However, if the overburden collapsed into abandoned and inactive galleries or chambers, we recommend using only the term sink-hole. The forms of relief called pingen until now can be divided according to their genesis into two subtypes ï sink-holes and depressions.

These shapes are formed when the degree of the undermining of the surface reaches the stage when the forces supporting the overburden reach their critical values, and the destruction of the ceiling (overburden) into the extracted spaces takes place. Collapses can occur over big chambers, above the intersections of the mining corridors (these places usually coincide with ore veins crossings), and in shallow depths also over exploratory galleries. These situations may occur suddenly forming a catastrophic situation of a rapid collapse of the overburden in the mining underground. An example of a typical collapse can be seen at the locality Ġturec near Kremnica (Slovakia) with a length of 700 m, a width of 250 m and a depth of 170 m (Fig. 12, 13).


Fig. 12, 13. The sink-hole Ġturec, an aerial view, and photography (SMM Archive, Banská Ġtiavnica)

The sink-hole walls are usually very steep, often perpendicular. Different rock formations reminiscent of rocky towns can be found in their space. From the 17^{th} century, vast sink-holes - called pingen occurred when the whole mine collapsed. Well-known examples include Altenberger Binge, Geyerischen Binge, Seiffen Pinge in Germany, a few pingen in the surroundings of HŚebeļné (Schneppôs pinge, Wildbahn pinge) in the Kruġné Hory Mts. in the Czech Republic, Ġturec in Kremnica in Slovakia and many others.

If the sinking process of the terrain above mining works is very slow and long-term, or if it occurs in several stages, then we are clearly talking about sinks caused by collapses at the mining works level when the sinking of the overburden is limited by the volume of underground space. Walls, or slopes of sinks, are not steep but slightly inclined, corresponding formally to the shapes of trough depressions. This is absolutely typical for coal basins where the method of mining in a relatively plastic and unstable overburden causes sinks in relief, sometimes with huge horizontal dimensions and variable height amplitudes (Fig. 14, 15).


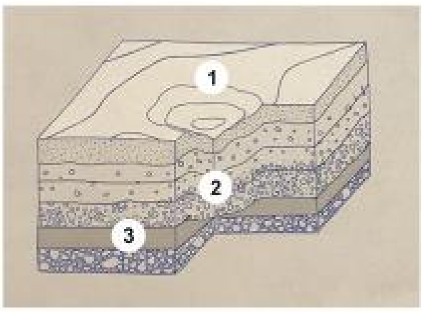

Fig. 14, 15. A depression "pinge"(or sink-hole) in Lehota pod Vtáļnikom. Legend: 1. sink-hole, 2. collapsed overburden, 3. mining underground.

Depressions are caused by relatively slow downward movements of overburden over excavated spaces, which occur gradually rather than momentarily. The main factors influencing the intensity of depressions and sinking of the overburden are the depth of the excavated spaces, thickness of excavated layers, area of excavated spaces, shape and length of the mining surface, layers slope angle and the mining technology (blasting or excavators, etc. cave-in with a partial back-fill, with a full back-fill, blasting, excavators, etc.). Time is an independent factor.

A large number of sinkholes - collapse pingen originated in the Middle Ages together with the beginnings of the subsurface (underground) mining ï in German der Deckelbaum, shallow, subsurface mining. It was an excavation of a dense network of shallow blind shafts connected by side by side leading corridors or narrow profile gallery passages. Several chambers were gradually connected to this network. All of these underground mines were following ore bands. Using this method led to such a disruption of the rock stability that the overburden usually collapsed into underground spaces (Nováļek, 1993).

A typical feature of all collapse pingen is that they consist only of the concave part. There is no heap near them because the heap of dirt from the originally excavated underground is located at a variable distance from the collapse pinge, at the site of the original exit of the mining works on the surface.

Screed pingen (Fig. 16), or more precisely scree-collapse depressions, are created by a screening of the mouth of vertical or nearly perpendicular mines on the Earth's surface, commonly including shafts, staple (small-profile shafts) or chimneys. The transport, extraction or ventilation function of an abandoned, secured shaft or an old ventilation chimney is terminated, i.e., degraded, usually by screening or flooding. After the mining process termination, a gradual rotting of the timbering and the subsequent weathering of mouths of these shapes takes places, which leads to the destruction of the immediate surroundings of the mouths, chimneys and blind shafts created by geomorphologically less resistant rocks. Applying gravitational forces over a longer period causes the rock material to collapse into the interior of the mining bodies, which are gradually filled up or clogged. This process creates a typical funnel shape of the pinge, but we do not consider these shapes pingen. It is necessary to preserve their original definitions together with their attributes - screed, collapsed, flooded, extinct, etc. (Fig. 17, 18, 19)

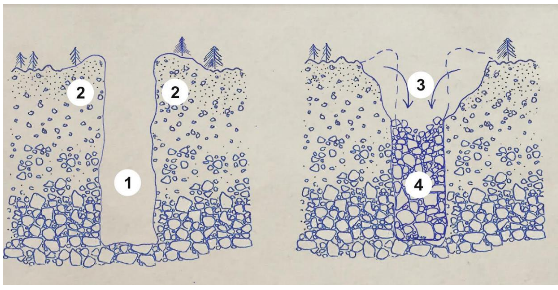

Fig. 16. Procedure for the formation of screed pingen in the openings of vertical mining works. Legend: 1. Vertical work in operation (shaft or small shaft) 2. vertical walls before destruction 3. crater depression similar to pinge (screed pinge), 4. collapsed vertical mining area.

Fig. 17. Screed pinge on Bärenleuten ore vein, chimney discharge to the surface in Hodruģa-Hámre.

Fig. 18. Screed pinge on the old nameless shaft on Všechsvätých ore vein in Hodruša-Hámre.

Fig. 19. An example of a flooded screed opening of an abandoned shaft Ján-Jozef near Banská Štiavnica.

Heaps in their surroundings are not determinant for this category of shapes, because these can be accompanied by bring-shaped mounds especially in the mouths of shafts resp. On the slope - slope heaps, as is the case with the dug (exploratory and mining) pingen.

On the contrary, heaps are not present in the vicinity of shafts which have been traditionally long-term and intensively used for transportation or ventilation, but smaller terraces for manual capstans are preserved in the nearby surroundings of the shaft opening (Agricola, 1556), as well as larger terraces as the relics of horse-whims.

These forms have also been and are often understood as pingen, but this interpretation is incorrect and leads to inaccurate and often false scientific conclusions.

Basic properties of pingen based on their relationship to relief and landscape

Basic characteristics necessary for scientific analysis can be derived from many concave surface montane relief shapes - currently inaccurately referred to as pingen - according to their basic shape, position or specific properties in the terrain and the landscape (Hronček, Rybár& Weis, 2011).

According to the **genesis of origin**, as detailed above, pings have been defined as dug, collapsed and screed, by the current inconsistent scientific literature. As we point out in Results and discussion, we do not approve such classification.

According to the **original function**, we distinguish pingen for exploration and extraction. As many exploratory pingen have continuously taken over the mining function a definite determination of their function is complicated. Similarly to subsurface mining, the function may change several times. This is determined by external conditions such as the yield of the used raw material processing technology, fluctuations in raw material prices, and so on. The function (exploratory or mining) is the distinguishing criterion of the new definition of the term pinge. Only using this criterion is it possible to avoid erroneous classification of sink-holes, depressions and collapses of shafts and chimneys (screening) as pingen!

From a **morphological point of view**, the pingen are defined as concave - hollow (clamped) shapes about the natural relief, with their entire volume under the natural relief. Exploratory and mining pingen have convex shapes in the form of heaps that protrude above the natural relief.

According to **the technology of origin**, pingen can be divided into hand-raked or partially digged (the historically oldest form, only primitive tools and their fragments like antlers are present, with preserved fragments of wooden sticks in exceptional cases), digged using typical mining tools (hammer, chisels, various types of hoes, shovels ï worn out parts can be found in the adjacent heap or the mould volume) and combined digged and blasted where the penetration into cohesive rocks was achieved by means of black rifle dust (from 1627 to the end of the 19th century) or by other explosives (the 20th century).

Depending on the **occurrence and field arrangement** ï classification in the sense of pingen lines and pingen fields. Linear arrangement same as or parallel to the vein exit or the tracked structure is typical for pingen lines. Pingen fields are characterised by a seemingly chaotic arrangement without an apparent implicit condition. Like the pingen lines, both arrangements can have either exploratory or mining (digging) functions.

Based on **location in the terrain (on a slope), i.e. according to the local morphological situation**, we recognize alluvial pingen (on the floodplains), foothill pingen (created at the foot of slopes), slope pingen (on slopes), plain pingen (on plain reliefs) and peak pingen (lying on peak platforms and ridges).

Depending on **the angle of slope -** the basic shape properties of mining and exploratory pingen also change with the slope angle. (this is a diagram)

Depending on the size, we can recognise macro-shapes, meso-shapes, micro-shapes, and nano-shapes. Pingen can reach the size of macros-shapes only exceptionally. Macro-shapes are the largest; they must measure more than one hundred meters or up to several hundred meters in several directions (for example, width and length). This includes the largest sink-holes caused by undermining, but only under the condition of continuing mining (in the underground, or on the surface of the sinkhole and in its volume).

Meso-shapes are shapes reaching average sizes, i.e., several tens of meters, and in one direction they can exceptionally exceed hundreds of meters. Mainly sink-holes ï collapse pingen, for example, long linear excavations can be included, for example, the Rabenstein site in Hodruġa-Hámre, or Terézia, Bieber and Spitaler veins in Banská Ġtiavnica (Slovakia), Wolfspinge near the village of PotŢļky in Kruġné Hory Mts in West Bohemia.

Exploratory and mining pingen are typical micro-shapes with nano-shape elements. For micro-shapes, the individual basic dimensions (width, length, depth) are within ten meters (in units of meters), with one of the dimensions exceeding that limit. Nano-shapes have the smallest size, reaching a size of one meter, and it is difficult to identify them in the terrain because of their age and progressive re-naturalisation.

Depending on **the ground plan -** circular, elliptical, irregular, linear, etc. The shape is usually dependent on function, and the total depth reached in a certain type of rock.

According to the presence of water - dry, watered and flooded pingen can be distinguished. When it comes to watering, it can be permanent, or it may be formed by periodical rainfall or precipitation water. Pingen can be permanently flooded with free water if there is a high content of clay in the screen, or if impervious rocks at the bottom of the pinge are present.

About **the natural relief**, the surface forms of the montane anthropogenic relief can be divided into concave (i.e., hollow) and convex (i.e., heaped) shapes (Zapletal 1968, 1969). The concave shapes of the montane relief are completely, or only by a substantial part, located below the level of the original natural relief (when it is not part of a mullock tip). The convex shapes of the montane relief are overlapped above the original relief (heaps and moulds). Exploratory and mining pingen usually have preserved heaps, so we classify them as a combined ï concave-convex shapes.

According to **re-naturalisation or age** ï in scientific literature pingen characteristics referring to the periodisation of history, resp. Mining is used. We can talk about prehistoric pingen, ancient, medieval, modern time pingen, or pingen created lately. It is also common to use the time scale according to centuries. An accurate determination of the age is problematic because it can only be determined by historical sources. Only the most significant pingen have the exact year of origin written in archive documents.

The relative expression of individual concave shapes age is used in research, for example:

a) alive **-** shapes that are in the stage of development, i.e., they are at the stage of youth,
b) mature **-** shapes that are already developed, i.e., they are at the stage of maturity, with mostly sharp edges,
c) vanishing ï shapes that gradually begin to be subjected to the natural geomorphological process, signs of "re-naturalisation" are visible, and we can talk about the senescence of these forms. Their edges are mostly rounded, and the concave part is partially screed,
d) the extinct shapes may disappear by a natural way, which is incomparably longer than anthropogenically-determined extinction through plantation or degradation; these shapes can be documented only by montane archaeology.

Related to the **vegetation,** we can talk about "bare" pingen, which are not covered by vegetation either because of their age or inappropriate ecological properties of the site. Pingen can be only exceptionally found without a vegetation cover due to the degree of landscape devastation, pingen age, and progressive succession. The presence of resistant pioneer tree species and herbs is common. The degree of plant coverage, resp. the coverage of pingen and its surroundings by individual categories of secondary landscape structure is used to be expressed in the percentage scale.

Results and discussion

The primary objective of the presented work is to clarify the term "pinge" from a semantic point of view and to determine it according to its characteristic features. The aim is to clarify terms related to particular concave montane relief forms which are inconsistently and inexactly explained by the contemporary scientific literature and are often considered as pingen. The term pinge is often used as a synonym for different relief forms or misplaced as equivalent.

From history, Montanism and anthropogenic geomorphology only shape directly created by exploration or extraction of mineral resources can be considered as pingen. Then we can only talk about two types of pingen – exploratory and mining. Their inseparable parts are heaps, whether different types of slope heaps or moulds.

Pinge can be defined as a depressed relief form, mostly of a funnel-like shape created directly by surface exploration (digging) or surface mining – so-called "pinging" (selective surface mining). It is exclusively an anthropogenically formed landscape form. The typical feature of a pinge is the presence of a mould (on a plain) or a sloping heap (on a slope). Two types of spatial layout, linear (pinging) and irregular (randomly spaced) planar (pingen field) layout are characteristic of pingen. However, the occurrence of pingen is always in groups. Pingen does not occur as a random depressed form of relief in montane (mining) landscape.

Based on our analysis, it is clear that other shapes of the concave montane relief did not necessarily need to have a typical funnel shape. They have a different genesis, an absent heap, large dimensions and have their specific systematic apparatus. Despite these facts, they have been called pingen. The most common confusions occur with the following montane forms:

Schurf (schürfe – testing and mining ditches and trenches) is a very old name for exploratory and mining linear shapes (von Charpentier, 1778; Karsten & Dechen, 1835). The shape is a shallow pitch, usually no more than 2 m deep, which is tens of meters long. A schurf has linear heaps in the form of mounds on both sides along its length if the angle of the slope is favourable. These shapes are often referred to as pingen or Surfping (Steen, 2013) in the literature. The historical name schurf can be identified with the term (exploratory, mining) excavation pit (trench), which is a linear exploratory surface mining work, more than 1.5 m wide, and its length must be bigger compared to its depth, while the inclination of the walls depends on rock coherence.

Mining forms referred to as **verhau** (ditch) represent an interstage between surface and subsurface mining. These are narrow linear excavations with very steep or perpendicular and usually rocky walls. They reach considerable depths, sometimes even a few tens of meters (Kratochvíl, 1952). Essentially they represent embossed "negatives" of extracted ore vein structures. These terms are sometimes used as a synonym for old mining (Lueger, 1910). These shapes are inaccurately referred to as pingen, or literally Verhaupingen (Steen, 2013).

Interesting verhaus are for example: verhau at Badenweiler and Münstertal in Schwarzwald in Germany, Himmelsehre near Salzburg in Austria, Vlčí jámy, Červená jáma near Horní Blatná in the Krušné Hory Mts. in the Czech Republic, Bludná at Sněžná hůrka in the Krušné Hory Mts., Malá Čertova Zeď in North Bohemia, Rabenstein in Hodruša Hámre in Slovakia (Fig. 20, 21).

Fig. 20, 21. Verhau on Rabenstein ore vein in Hodruša-Hámre (photo L. Lužina).

Mining (exploratory) ditches are long line shapes excavated **-** dug off the hillside approximately following the contour. Their bottom is horizontal and does not fall below the lower edge of the ditch. A linear slope heap shaped like an earth mould is at its lower edge along the entire length. **Mining cuts** have a similar genesis (Hrubý et al., 2016). These are small dugout spaces, usually with an elliptical ground plan and an adjacent slope heap. These shapes are morphologically almost identical to the collapsed shaft openings with the character of bowl-like depressions.

Mining pits - soil ditches and surface quarries of different sizes are created exclusively by mechanisms or by combined blasting and breaking (stone) and are related neither to exploratory nor selective surface mining. These shapes are often unique. Therefore their group occurrence on a relatively small area is improbable.

Creep is a natural or a controlled burial of underground spaces due to the destruction of the ceiling or walls into the excavated underground mining area. The direction of movement is predominantly vertical, and its result is the filling of the originally empty mining spaces with overburden rocks. This process does not create pingen, but sink-holes or depressions. Sink-hole is a more frequent shape, which arises from a sudden catastrophic collapse of the overburden into the excavated spaces. The dimensions of sink-holes can reach several hundreds of meters in all directions. Depression is a part of the earth's surface of a bowl-like or funnel-like shape, in larger dimensions of a pan-like shape, caused by the slow downward (dropping) of the overlying layers into the excavated spaces. By large depressions, we can talk about ñdrop valleys.ò The basic parameters of depressions are derived from their edge, slope, bottom, depth, and area.

Screed openings and the exits of vertical mining works to the surface cannot be regarded as pingen, even though in many cases they retain a heap that originated during the excavation of the original vertical mining work. These concave shapes can be defined either as collapses, backfills (natural but also anthropogenic) of the mouths of vertical mines on the earth's surface. They acquire the characteristic conical shape gradually through natural erosion of the side walls. They arise primarily by screening or filling of the blind shafts or shafts. A blind shaft is an exploratory or mining work mined vertically into the Earth's crust, with a circular or square floor plan with an area of up to 3.75 m^2 and a depth of approx. 20 m, max. To 40 m. The shaft is a vertical or almost vertical underground mining work with a circular, square or rectangular ground plan, which mouths to the surface. With a floor plan of more than 3.75 m^2 and a depth of several tens of meters, it served for various mining purposes (for example, transport, mining, ventilation, drainage, etc.).

Creating new Slovak or national equivalents for individual shapes of concave montane relief is not a very good solution in the current globalising world. These forms are more like a shape-like relief form for which faulty synonyms are used, but the actual term concepts are different. It is, therefore, preferable to use original (often German) mining terms with their original meaning, being precisely defined and denoting the same montane shapes in each language. A partial solution has so far been a sporadic use of a phonetically almost identical term a pinge in English, or pinka in Czech and Polish, or pinga in Slovak. The adherence to their meaningful content as suggested by this study is, however, more important. The unambiguous definition of the term pinge is given together with the determination of the most important determinant, i.e., the original function because the criterion (the genesis of form) used so far has not been unambiguous. Definitions of individual concave montane shapes are derived from the historical early medieval mining, which was formed mainly in the Northern Alps, in the mountains north of the Alps and in the Carpathian region. It has a well-established logic and a steady meaning, which can also be applied in the current scientific research. A similar situation occurs with the one-word terms schurf or verhau. It is best to use the international term "quarry" for excavation pitches that can be identified with historical quarries.

Conclusion

The unambiguity and punctuality of the term pinge are indispensable for research in all disciplines dealing with surface concave relief forms such as Montanism, anthropogenic geomorphology, bearing geology, geography, landscape ecology and planning, and history.

The above-presented unambiguous classification is also necessary for practice and secondary use in montane tourism. It is tourism oriented towards those areas of the Earth that are of exceptional value from a geological point of view. These usually represent key witnesses of the evolution of a certain part of the Earth's surface and made a significant contribution to the development of human society.

From the montane tourism point of view, two possibilities of pingen visits can be offered. Pingen documented near mining towns such as Banská Ġiavnica can be observed from existing educational trails. Similarly in the Pukanec Forest, where they are a part of the Pukanec educational trail. Some pingen can hide a buried shaft underneath. The material clogging the mouth of the shaft can occasionally collapse to a depth of several tens of meters, says the official web site of Pukanec.

Area of Staré Hory offers the possibility of observing the anthropogenic form of relief of montane origin - pingen. Staré Hory is well-known for the mining of copper ore in the past.

All of the mentioned areas have one thing in common. It is easy to combine montane tourism with traditional tourism, because of existing touristic infrastructure - accommodation, meals, urban and natural attractions. This could greatly facilitate the promotion of montane tourism. Reciprocally, the montane aspect of tourism could enhance local tourism. The holiday area Harmonia in the surroundings of Modra may serve as a good example.

The second option is to create a separate product which includes an attribute of adventure. The idea is to show pingen in nature, discovered by montane tourists themselves without touristic and educational trails. A beautiful example of exploration as well as the specific route is given by Adrián Harničár, Daniela Mlynarčíková and Peter Roth in their article The Extinct Mining Works on the Upper Hornád River, in which they identify and document individual pingen located in the Vernár village surroundings. These can be found and explored, and it was interesting to include them in the montane tourism concept within the area above. There are many possibilities.

Pingen and then their involvement in montane tourism abroad can be seen, for example, in Altenberg and Geyer in Germany, Czechia – Krušné Hory Mts., England - prehistoric mining.

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