Artur Dyczko

METODOLOGY FOR RUN-OF-MINE QUALITY MANAGEMENT IN A HARD COAL MINE

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Methodology for Run-of-Mine Quality Management in a Hard Coal Mine

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1. Introduction

The term sustainable development, for the first time defined in 1987 in the report ‘Our Common Future’ prepared by the UN World Commission on Environment and Development, gained great popularity in scientific circles - but also on a broader scale - in the awareness of the general public. In this report the sustainable development was defined as a process aimed at satisfying development aspirations of the present generation in a way enabling a realisation of the same aspirations by the next generations. At the same time, because of a multitude and diversity of factors that can affect this phenomenon, three main areas were distinguished, on which the planning of an effective strategy for achieving sustainable development should be focused. These are:

– an environmental protection and a rational management of natural resources,
– an economic growth and a fair division of resulting benefits,
– a social development.

Looking at the areas, distinguished above from the mining sector perspective, one may say that the sustainable development in the mining industry consists in such management of mineral resources that in the final effect the carried out activity would be economically efficient, environment friendly, and socially acceptable. The definition formulated in such a way gives rise to a few basic questions in the context of our national mining sector, i.e.:

– Whether the management of resources carried out now by mining entrepreneurs, having a significant impact on the future of Polish mining sector, is carried out according to sustainable development principles?
– Whether the conviction, functioning in the general awareness of Poles, that national hard coal resources are sufficient, seems to be longer-term justified?
– Whether a high level of mined minerals’ dilution in the underground mining should be tolerated, at its negative impact on the efficiency of carried out production processes?

Poland belongs to a relatively small group of countries, in which a generation of electricity and heat is based almost entirely on solid mineral fuels: hard and brown coal. In 2016 approx. 79% of electricity was generated from those two fuels. In 2015 in the electricity production structure a decreasing trend of electricity generation from solid fuels maintained (this share went down by 12% from 2005) with an increasing share of generation from RES. This situation results from our country richness in these fuels and the lack of significant amounts

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of other primary energy carriers. Hard coal in Poland, because of the size of possessed resource base as well as the hitherto and planned raw material and power policy, plays and will continue playing in the future a role of energy security and energy independence guarantor for at least several years to come.

In the post-war period twenty two big hard coal mines were constructed, five big brown coal mines, eighteen iron ore mines, six copper ore mines, four zinc and lead ore mines, as well as a number of other opencast and underground mines of chemical, rock, road, and building raw materials. A permanent growth of the hard coal output maintained during 33 years till 1979, when the peak annual output of 201 million tonnes was achieved. An average annual increase in the hard coal output in that period was 4.5 million Mg/y, i.e. 15,000 Mg/d. Such an increase would not have been possible without a construction of new mines. By 1980 the mines of total production capacities of 253,000 Mg/d were constructed in Poland, which was 40% of the national production potential of hard coal mines operating at that time. In terms of new mining plants construction in the 1980s Poland ranked second in Europe, after the USSR. Till 1990 the production potential of mines increased by another 82,000 Mg/d, so that at the beginning of the 1990s the overall production potential of the Polish mining sector was 335,000 Mg/d. In parallel to a construction of new mines the existing mines were subject to a reconstruction and modernisation. At the beginning of 1960s the process of smaller mines merge into bigger ones was started, with a modernisation of basic technical and organisational lines at the same time. During the next twenty years more than twenty small mines were merged into bigger units. In 1980 the total number of hard coal mines was 66 (instead of 81).

Social and political-economic transformations, which started in 1989, forced the hard coal mining sector to adapt to the market economy. This applies in particular to a new approach to a determination of the Polish economy demand for hard coal, a profitability of operating mines, with special emphasis on the pace of closing down permanently unprofitable mines and on the related legal, social, financial, and environmental protection issues.

The past thirty years of mining plants operation under new economic conditions, based on market rules, forced a thorough review of possessed deposits opening and of mining plans, adequately to the situation on the fuel market, depending on the customers demand. It is necessary to admit, that this period caused a significant deterioration of the sector situation, against a background of it only one entity was capable of meeting the requirements of new economic conditions and changing its operational strategy, to dominate the market. This refers to the Lublin Bogdanka mine, which now operates in the Lublin Coal Basin (LCB) as a single mine within a joint stock company named LW Bogdanka SA.

The plan of the Lublin Coal Basin (LCB) construction in the 1970s, as well as a general atmosphere accompanying the investment, were typical for all the great constructions of that time. The construction of this ‘pilot’ mine, the first one in the entire basin, was started in 1975. Numerous mistakes, made during the mine construction, resulted both from a poor recognition of geological and mining conditions and from the haste, caused by shortening the construction deadlines on the occasion of the party leaders’ visits, and consecutive Polish United Workers' Party conventions, and sometimes also just from wrong decisions. The last ones were the reason of major delays in the mine construction schedule, huge costs, as well as lost human efforts. Because of great adversities in the initial period of mine construction, such as passing the layers of Albian (quicksand) by the shafts being sunk, the convergence of newly driven horizontal workings, a reconstruction of damaged shaft insets, the first longwall, referred to as a test one, was carrying out solitary mining operations of approx. 800 Mg/d until April 1986, that is for more than 5 years.

The commissioning of shaft S.1.4 in Nadrybie, which took place in November 1985, allowed to increase the output to approx. 2,500 Mg/d. The second longwall 2/1 in the Bogdanka panel was started in April 1986, and the first longwall in the Nadrybie panel (2N) was started in September 1987. The breakthrough in the coal mining in the ‘Bogdanka’ mine occurred in November 1988, when after 11 years of construction the second hoisting skip shaft S.1.3 equipped with a 30 Mg skip and capacity of 18,000 Mg/d was commissioned.

In the years 1988-92 the number of operating longwalls was increasing relatively rapidly (maximum 5) and the coal output in the mine was also growing rapidly. December 1992 witnessed another important moment - when the Coal Preparation Plant was commissioned, enabling coal preparation and its quality improvement. The last important event, after 24 years from the government decision concerning the K-2 mine construction stoppage, was commissioning of the shaft S.2.2 in Stefanów together with the infrastructure. In the present realities ‘Stefanów’ is the third prospective mining panel of LW Bogdanka SA, which enabled to double the mine capacity.

The period of past 40 years of coal mining in Bogdanka is the time of laborious struggle with the nature, with technical adversities, but also - and perhaps first of all - of applying scientific bases of the plant modernisation and reconstruction, the mine construction and expansion cycle optimisation under geological-mining conditions as well as social and economic conditions of the

LCB. This is a brand new approach to develop and extract resources, deposited deeper and under more complicated geological-mining conditions, as well as a continuous development and improvement of the methods comprised by the designing theory of mines. This is a thorough restruction and a creation of foundations to increase the labour productivity and to cut the costs, to achieve the leading position in the hard coal sector (the highest quality, the biggest concentration of mining operations, the lowest costs).

A drastic drop in coal prices in the years 2014-2016 and difficulties with selling the coal from the Bogdanka mine forced a development of a new production strategy of the mine, whose foundation consisted in controlling the mining production in terms of increased yield of saleable coal, the production parameters stabilisation, and an improvement in the run-of-mine (ROM) coal quality (a reduction of ROM dilution).

The issues, characterised above, provided a premise to start a few years ago, the study and research work on the assessment of the hard coal dilution impact on the efficiency of production processes in the LW Bogdanka SA and on a development of an appropriate system for the mining production management support with the use of IT solutions and monitoring of production processes directed towards the ROM quality stabilisation and improvement (Dyczko 2018).

The research work carried out by the author, as regards those issues, is described in this Monograph in the following arrangement:

– In Chapter 2 the Author presents the analysis of the state of the issue, to which he devoted his research. It starts from a description of resources classifications, used in the country, taking into account the quality of minerals and characteristics of commercial nature of resources, emphasising a recollection of the binding definitions of mineral losses and dilution, crucial for the analysis of the dilution impact on the efficiency of carried out mining operations. The scope of a further analysis, both in terms of the subject-matter and volume, required limiting of the selected methods of mathematical modelling within the theory of mines designing, and basically of the analytical methods, methods of variants, or combined methods applying computer techniques – the methods using an economic criterion in the field of issues related to the mine size, the mine model, the sequence of deposit mining within the mine field of the coal mine, the size of mining panels and mechanised longwall parameters, and finally the methods of qualitative parameters estimation developed mainly for the needs of coal mining sector.

- In Chapter 3 the Author analyses the reasons for the ROM dilution origination in the process of hard coal mining, starting the considerations from the analysis of such elements as the structure of deposit opening, the amount of dirt, geological-mining conditions of beds deposition, and the technique and technology of coal deposits mining impact on the origin of dilution.
- In Chapter 4 the Author performs an assessment of the ROM dilution impact on the economic efficiency of an underground mining plant production process. It is based on the appropriate data, which originated from three longwalls situated in a thin seam of the LW Bogdanka SA mine, being in various phases of the mining process. The empirical data were subject to the statistical analysis directed towards developing a mathematical model of longwall advance depending on the quality structure and amount of the ROM originating from the analysed plough faces. With use of the Monte Carlo simulation and also of the cluster analysis as well as the scenario analysis a mathematical model was developed, being the basis for an analysis of the economic efficiency of the mining process in the scenarios assuming an improvement in the ROM quality.
- In Chapter 5 the Author describes his own concept of the production process automation and monitoring, which he implemented in the LW Bogdanka SA, consisting of a few interconnected components. It results from the experience gathered during an implementation of big research projects for the KGHM Polska Miedź S.A., JSW S.A., and Tauron Wydobycie S.A. The Integration Platform is a central element of the presented solution, being a significant tool for the information integration, allowing a standardised exchange of data among systems built in various technologies, or using various communication protocols.
- In the summary (Chapter 6) the Author presents the most important observations and conclusions resulting from the carried out research work.

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2. Studies and Analyses Preceding a Development of Methodology for Run-of-Mine Quality Management in a Hard Coal Mine

A division of the deposit and its resources because of the mineral quality is important, when a diversification of the mineral mineralogical and petrographic features or a diversification of useful or harmful components content or other parameters, characterising its quality, is reflected in its various ways of use or causes a necessity of applying various parameters of its preparation, refinement or dressing. In the case of hard coal the division is based on a set of physical properties, deciding about the direction of the coal use and its quality (division into coal grades), depending on the ash and sulphur content. The division criteria must always be clearly defined, i.e. limit values of this feature or a set of features provided, based on which an appropriate grade or mineral type is separated (Nieć et al. 2015).

If a few distinguished mineral grades exist in the deposit, their distribution must be presented on maps and cross-sections, and their resources calculated separately. So in the case of coal deposits the resources of their individual types are calculated separately as well as the resources of deposit parts, in which coal differs in the ash content. Depending on the mineral qualitative features and on its resources, as well as on the deposit structure and conditions of its stratification, the deposit may present a different economic value. This may apply to the entire deposit and also to its separated parts, differing in the structure, mineral quality, deposition conditions, or even the exploration degree. The following factors decide about the economic value of the deposit (or its separate parts):

- a deposit attractiveness for management, determined by the conditions of deposit stratification, depth of deposition, its structure, and the mineral properties,
- a degree of deposit development for management,
- a suggested method of deposit management,
- a risk level related to an assessment of technical and economic conditions of mining, preparation, and sales of raw materials produced from the mineral.

The aforementioned factors are closely interrelated, but two groups may be distinguished among them:

- those related to natural properties of the deposit,
- those related to actions aimed at its management.

The criteria of resources commercial nature are determined at the stage of designing the deposits mining system. A design of the deposit management together with a technical and economic analysis of the plant construction (expansion) conditions are the basis for a determination of commercial resources. This analysis should specify the optimum variant of resources use in relation to geological conditions of the deposit stratification and current technical and economic possibilities. The technical and economic analysis should also clarify and justify basic elements, deciding about the management of resources, in particular:

- a choice of the deposit opening method and its mining system, including of determination of anticipated losses as regards geological resources,
- a determination of basic mining-geological parameters, specifying the criteria of the resources classification to commercial resources,
- a determination of mining and preparation costs for minerals in the deposit,
- a determination of economic factors, decisive for the deposit or its specified parts mining efficiency.

The aforementioned elements should be determined for all the analysed options of the deposit use in the case when the geological and mining conditions of the deposit stratification provide possibilities for various solutions of its mining method and size. For the optimum variant of resources use the limiting parameters should also be determined. They should be met by the mineral as well as by the conditions of the deposit (or its part) stratification to classify balance resources as commercial resources (Nieć 1982).

Recoverable reserves are separated from commercial resources, i.e. commercial resources less losses. The planning process of a mining plant construction or expansion, preparation of technical and economic assumptions, and a technical design of the mine may be carried out after a previous deposit exploration and an approval of the documentation containing a balance of geological resources of the mineral deposit and the commercial resources approval (Nieć 1982). Fig. 2.1 presents a classification of resources due to their economic importance.

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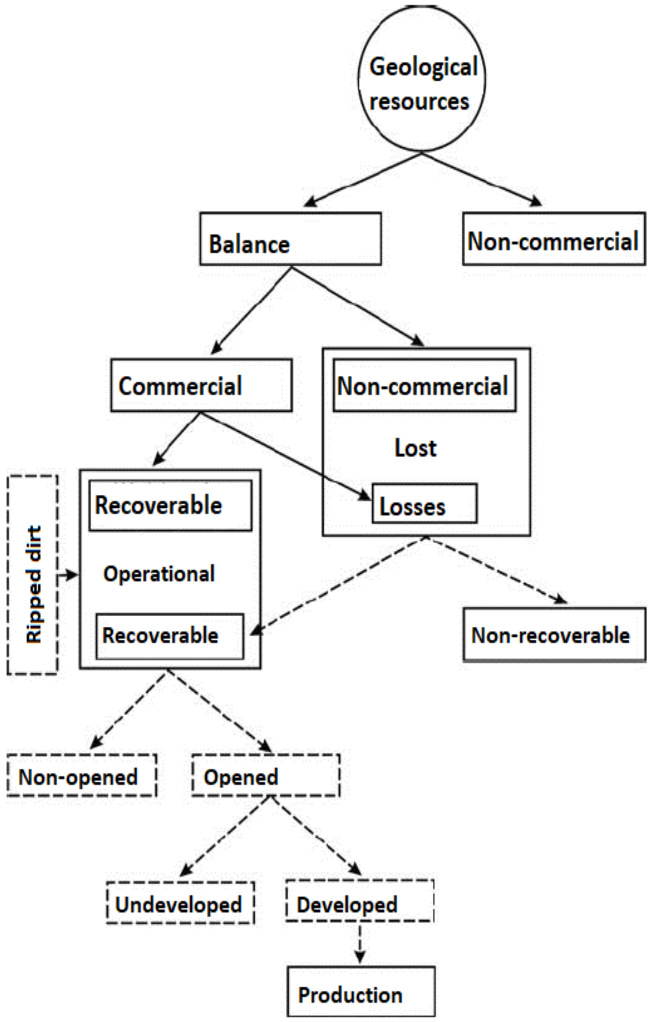

Fig. 2.1. Classification of resources due to their economic importance *(Nieć 1982)*

The capacity of a plant mining a mineral deposit and its life are determined during the mine designing process by reducing the commercial resources by the losses anticipated in the adopted deposit mining system and in the adopted mineral preparation processes.

The approved balance geological resources are divided into commercial and non-commercial resources in the design of deposit management, taking into account a complex use of the main mineral and accompanying minerals as well as waste and the projects ensuring environmental protection.

The resources trapped in pillars (protective, safety, support) are one of the main factors resulting in their classification to non-commercial ones. Also the deposit parts are considered non-commercial, because their mining could create hazards to the life of working personnel and to the plant operational continuity. They mainly include (Nieć 1982):

- seams underworked by previous mining operations,
- zones of water, gas, fire etc. hazards occurrence, which cannot be controlled by the available technical measures,
- zones highly disturbed tectonically, where the rock mass pressure is difficult to be controlled.

Underworking of seams is a frequent phenomenon in multi-seam deposits (e.g. of coal seams), especially those mined in the caving system. Depending on the mined seam thickness and its distance from the overlying seam, the resources of underworked seam, if it is not mined in advance, may be considered as non-commercial ones or as the resource losses.

Mining losses (quantitative losses) substantially reduce the deposit mining efficiency, resulting *inter alia* in increased expenditure on exploration and development operations. Instead, the dilution (qualitative losses) is a direct reason for an increase of transport and preparation costs, and it also reduces the value of the product obtained in the preparation process. Table 2.1 presents sources of mineral dilution and prevention methods.

The reason of the resources classification as non-commercial ones may be caused by their stratification in separate lenses, lobes, or apophyses, difficult to open and mine due to the necessity of changing a regular line of the mining front or a change of the system and related additional opening costs. Fig. 2.2. presents examples of mineral dilution (Nieć 1982).

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Sources of mineral dilution and prevention methods *(Nieć 1982)*

Table 2.1.

Reason of dilution	Way of dilution origin	Possible prevention methods
Roof and floor rocks ripping	too small deposit thickness as against required working dimensions	selective mining
	varying position of mineralised parts in the profile, diversified deposit roof and floor morphology, minor folding, faults	generally unavoidable; in exceptional cases precise exploration and selective mining
Deposit discontinuity, dirt bands	lack of information, large-size dirt bands and parts	precise exploration and selective mining
	thin dirt bands occurrence, lack of selective mining possibility	dilution cannot be avoided
Dilution with dirt	roof rock falls or caving	appropriate mining technology, leaving a safety berm
	incomplete overburden removal in opencast mines	control of overburden removal cleanness
	mixing the ROM from ‘work in stone’ or backfill	control of the ROM transport; appropriate backfilling technology
Non-clean mining	leaving richer roof or floor parts of the deposit	control of mining cleanness
ROM scatter and segregation	scatter and segregation at mining and transport	control of mining and transport cleanness; cannot be entirely avoided
Weathering	long-term deposit exposure, long-term outdoor storage of the ROM	appropriate organisation of mining operations, transport and preparation

A high variability of thickness, forcing to rip the dirt to obtain the required cutting height and also a varying but large number of dirt bands can result in a significant dilution or force to separate the dirt from the ROM. It increases the production costs, especially the costs of preparation operations.

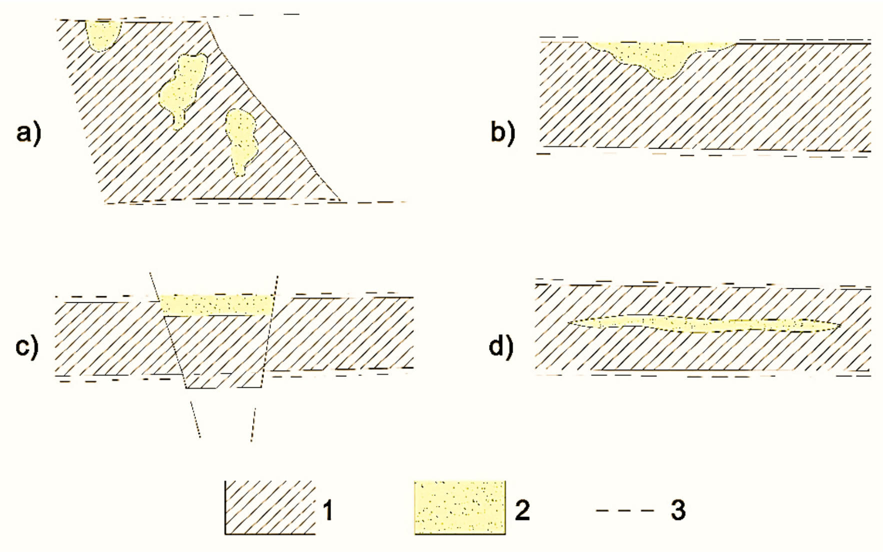

Fig. 2.2. Examples of mineral dilution *(Nieć 1982)*

a - Karst pockets, b - washout, c - small faults, d - thin dirt bands that cannot be separated during mining. 1 - deposit, 2 - diluting rocks, 3 - working contour

A prevention of an excessive dilution of minerals, in the result of roof and floor rocks ripping, primarily consists in an appropriate choice of mining systems and machinery sizes to the balanced thickness of the mined deposit. Also selective mining of useful mineral and stone is used, with a possibility of a partial or total stone placement in goafs.

It is difficult to avoid a dilution resulting from unpredicted deposit thickness changes or changes of its location as well as of small dirt bands. It may be decreased by condensing the grid of exploration points and using specially chosen deposit mining systems.

The dilution caused by the dirt from roadways can occur in the case of combining coal and stone transport in transfer nodes into one transport system. In this case it is necessary to separate shifts for stone transport in the daily time balance.

A useful mineral, mined by the mining entrepreneur, usually has worse properties than those, which were found during the deposit sampling. It is said then that the phenomenon of the ROM pollution in the case of coal mining and its dilution in the case of ore deposits mining are experienced.

Originally the difference between the dilution and pollution resulted from the fact, that in the case of ore deposits the reduction of useful component in the mined mineral as compared with its content in commercial reserves did not result

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from mixing of dirt and non-commercial mineral with the commercial one, as it happens in the case of hard coal, but was also related to the loss of richly mineralised small ROM fractions in the processes of mineral mining, transport and storage. Until quite recently in the world literature the *dilution* occurring especially in the ores mining processes, is defined as dilution, and in the case of hard coal mining as so-called *out of seam dilution* (OSD). The use of *dilution* term also for an analysis of pollution experienced at hard coal mining is now increasingly popular, therefore the Author of this work - also based on the results of his own studies - considered the use of dilution term also in the case of hard coal justified, like many authors worldwide, and hereinafter uses both terms: pollution and dilution as equivalent.

The phenomenon of mined mineral pollution is closely related to the progressing process of mining industry mechanisation and to an increase of so-called productivity of a mining plant. The pursuit of economic efficiency improvement in the case of the carried out mining operations is indispensably related to the will of increasing the mining concentration and using economies of scale, consisting in achieving increasingly higher output at a defined level of fixed costs, which result in a deterioration of the mined mineral quality.

The phenomenon of pollution, understood in this way, is one of the basic factors, to a big extent deciding about an economic efficiency of acquisition and use of minerals mined by the entrepreneur, resulting in nothing else but a qualitative loss of the acquired raw material.

The fact of deposits depletion is another factor, having an equally significant impact on the pollution of the mined mineral. Over time mines transfer production to poorer quality deposits, because better quality ones have already been depleted.

Generalising the term of mineral dilution, the following definition may be given for any phase of the mining process: ‘ROM dilution consists in a reduction of the useful component content in the mineral in a specified phase of the mining process, resulting from technical-mining properties characterising this phase’.

For the term of useful mineral dilution defined in such a way three basic dilution categories may be distinguished, i.e.:

– resulting from non-commercial mineral mixing with the commercial one - this category of pollution results in an increased amount of the mined mineral. The content of useful components in the mined mineral goes down, while their amount may grow proportionally to the content of useful components in non-commercial resources;

- resulting from dirt mixing with commercial mineral - this category of pollution also results in an increased amount of the mined mineral. The content of useful components in the mined mineral goes down, while their amount remains unchanged proportionally to the content of useful components in non-commercial resources;
- resulting from the loss of fine fractions of the ROM rich in the useful component, small in terms of quantity (so-called qualitative losses) - the third category of minerals pollution has a nature of qualitative losses. The ROM weight slightly decreases, while the percentage content and amount of useful minerals in the mineral goes down (this category of pollution characterises mainly the ore deposits mining).

When analysing the issue of ore deposits dilution, Wright (1983), Ingler (1984), and Knissel (1995) drew attention to the cause of the phenomenon, breaking it down to:

- a mining dilution resulting from the carried out mining operations. In the result of numerous studies and observations it was noticed that during mining of a deposit, where the dirt situated close to the mined mineral crumbles into pieces of the same size as the mined mineral or bigger, the dilution is usually minimal. While in a situation, where the dirt crumbles into the pieces smaller than the mined mineral, the effect of dilution is definitely greater;
- a structural dilution is related to the nature of the mineral deposit occurrence and originates due to the presence of individual spoil formations within the deposit in such a way, that the selective mining is not possible or so-called deposit washing occurs, caused by the groundwater penetrating the deposits (occurs in the case of certain copper and uranium ore deposits).

However, Noppe (2003) suggested a classification of the ROM mined in mines, most popular in the world literature, stating that the pollution of the mined ROM with dirt, in the case of longwall mining in the case of an extraction at the full seam height is unavoidable, and its sources may be divided into three main groups (Fig. 2.3):

- the primary dilution originating from ripping of the mined seam roof or floor (accidental or intentional), made by a longwall shearer or a heading machine,
- the secondary dilution related to ripping of the floor uplift or cutting rocks from the roof fall during cutting, resulting in mixing this material with coal,

– the tertiary dilution - considering the coal pollution during the cleaning of powered roof support units or mixing the dirt from driving of drifts with the coal ROM from longwalls.

As it is estimated in the literature (Saeedi et al. 2008) in underground mines so-called *off-seam dilution* (OSD) ranges from 15% to 30% depending both on mining-geological conditions and on technical parameters of the mining system. It happens mainly via a deterioration of the ROM quality and it generates additional costs related to milling, transporting and transferring the dirt polluting the coal, especially at transport along big distances.

The literature analysis state of the issue, carried out by the Author, showed that both in the world and national literature the effect of the *off-seam dilution* - OSD, despite its unquestionable impact, on the mining efficiency, has not been fully and comprehensively analysed so far. In particular, this applies to a description of the issue nature together with understanding factors that affect its origin, and there are not many papers available which describe this phenomenon in the case of coal longwall mining and they are not representative.

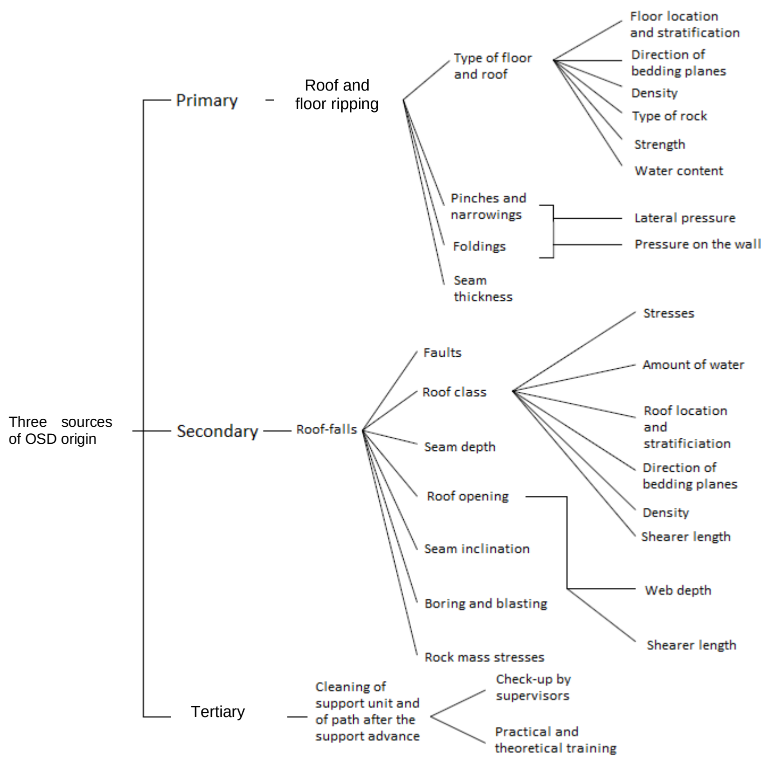

Fig. 2.3. Pollution sources in a longwall together with pollution reasons (*Noppe 2003*)

The first papers on the analysis of pollution understood as linking of useful mineral content with the mined rock, the useful mineral content in the solid, and the useful mineral content in the dirt were prepared by Popov in 1971 (Popov 1971). Michail Ivanowich Agoszkow (1974) was the main advocate of this idea, who determined the relationship among the mentioned parameters and the pollution. Agoszkow has stated that the coal pollution to a large extent depends on the coal content in the rocks. Therefore, high pollution values do not always mean a low productivity of mining operations. Similar studies were presented by Nazarchik (1972) as regards mining of thin ore deposits. The Nazarchik's study focused on the analysis of chambers width, average vein width, unit weight of the parent rock, unit weight of the mined rock in the vein, metal content in the mined rock in the vein and in parent rocks (Nazarchik 1972). The third attitude to dilution, based on the description of cases from the Ruttan mine was suggested by Pakalnis (1986). This method is not used in the industrial practice, but it was one of the initial attempts aimed at a quantitative assessment of the dilution. Clark developed a new method of dilution estimation based on the format of modified stability graph and expressed the chamber stability as the dilution estimator (Clark 1997, 1998). Other studies were based on the data gathered in the Stillwater mine, Montana (Annels 1996). In this case the dilution was estimated by the application of an exponential curve based on studies concerning the current dilution related to various thicknesses of the mined ore deposit. According to the studies carried out by Annels (1996), the dilution grows when the seam thickness decreases.

In their papers Soviet scientists devoted a lot of attention to the phenomenon of dilution - especially in the ores mining industry - from the point of view of rational deposit utilisation. In this field the papers by Kaplunow, Mielnikow, Agoszkow, and Bajkow (Agoszkow 1974); (Agoszkow, Panfilow 1973); (Agoszkow, Rysow 1967); (Bajkow 1973, 1978); (Bajkow, Kuczko 1974); (Kaplunow 1938, 1948); (Mielnikow 1973, 1974) are most important, dealing with the methodology of the amount of losses and ore dilution calculation depending on the mining system, analyses of losses and dilution origination, and an economic evaluation of mining operations both for underground and opencast mines. The method of ore loss amount and dilution calculations, provided by Kaplunow (1938), was binding in the Soviet Union. Soviet papers on losses and dilution drew attention to their role in the economic assessment of the deposit management. A number of methods were developed in this field, which may be divided into three groups. Group one may include those methods, in which the basis of assessment consists of value and natural indicators (value of production, labour productivity, capital intensity, and others) considering a negative impact

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of mining losses and ore dilution (Kaplunow 1948). Group two comprises profitability of ore mining and processing as well as the profit obtained from one tonne of mined ore, considering the loss originating from ore mining and dilution (Gamow 1973); (Rzhewski 1964). In group three there are methods, based on conventional profits obtained from one tonne of balance resources (Agoszkow, Panfilow 1973); (Abigmow, Owobienko 1971); (Bajkow, Kuczko 1974). Methods of this group most effectively present the amounts of ore mining and dilution losses in the economic assessment of the deposit management.

Those studies were used to develop similar methods for the hard coal mining sector. Noppe (2003) carried out studies on the measurement and control of dilution during underground coal mining, and Chugh et al (2002, 2003, 2004, 2005) analysed out-of-seam dilution in an underground coal mine in the USA, to understand its impact on the coal mining and preparation as well as on waste disposal.

As Noppe noticed in his study, an overall level of dilution for the mine can be calculated, comparing backwards the quality of coal ROM with the quality of coal anticipated under in situ conditions, without any dilution. This method does not allow to determine current types of pollution, to identify areas of the mine responsible for the dilution origination nor its causes. However, this information is indispensable to reduce the dilution to acceptable levels.

Saeedi et al (2008) in their papers determined quantitatively the level of dilution for longwall mining of hard coal in the Tabas mine in India. They managed to determine an increase in the out-of-seam dilution in the analysed mine by 1%, which affected the net profit, reducing it by 0.75%, and the capacity of the coal preparation plant was reduced due to the necessity of preparing a larger weight of the ROM which resulted in a unit cost of production increase by 1.17 USD/Mg. The authors extended this study with numerical modelling of the out-of-seam dilution (OSD) in the retreat longwall mining (Saeedi, Shahriar, Rezai, Karpuz 2010). There were numerous papers on the ROM pollution in mining operations, with its measurable results and indirect and direct effects on production costs, e.g.: (Wright 1983); (Pakalnis 1986); (Elbrond 1994); (Pakalnis, Poulin, and Hadjigeorgiou 1995); (Villaescusa 1996, 1998); (Bock 1996); (Bock, Jagger, and Robinson 1998); (Revey 1998); (Scoble and Moss 1994), and (Brewis 1995).

The pollution of the mined ROM has been and continues to be the subject of many numerical modelling analyses and techniques, carried out to study parameters affecting both the size and the nature of dilution depending on the used mining system. Suorineni et al (1999) studied the relationship between dilution occurrence and faults existence. In the result of their study they proved a pretty obvious premise, that the fault existence in the neighbourhood of an opened mining chamber can increase the size of roof fall hazard, resulting in the dilution growth. Wang (2004) obtained similar results of his study and developed a method of numerical modelling based on the assessment of the roof rock stress and geometry impact on the stability of open chambers and on dilution. He showed that with an increasing coefficient of radius and stress ratio the dilution goes up. Henning and Mitri (2007) suggested another approach to the use of numerical methods in studying the effect of dilution. They developed a series of 3D numerical models to analyse the influence of mining depth, in situ stresses, as well as the chamber geometry and direction of deposit arrangement on the amount of dilution.

The process of mined ROM pollution in the Polish mining sector was analysed in an entirely different way. In the national literature on the subject, historically the term dilution was always related to metal ore resources and did not apply to hard coal deposit resources. The very issues of economic assessment of the ore dilution impact on all the stages of mining operations were described in numerous papers, e.g.: (Krasiczyński 1974); (Krasiczyński, Majchrowicz, Wilczyński 1974a, 1974b); (Dziura, Zapotocki 1976), (Praca Zbiorowa Instytutu Górnictwa Politechniki Wrocławskiej 1972); (Wanielista 1979); (Cieszkowski et al 1984); (Dyczko 1998, 2002, 2004). The authors dealt with selected issues related to ore losses and dilution, and also defined terms and methodology of calculations, relationships between losses and ore depletion, an economic analysis of ore losses and dilution, as well as they studied the relationships among losses and dilution as well as selected parameters of mining systems.

The biggest attention was paid to dilution, and in particular to its adverse impact on the efficiency of the carried out production processes in the Polish copper mines (KGHM Polska Miedź S.A.), although it took a really measurable character after 1989, together with going away from a centrally controlled economy in favour of a free market.

Wawrzyniak (1976) and Krasiczyński (1974) showed the interrelationship between ore mining losses and dilution. Konstantynowicz (1971) presented the dependence of metal mining losses on ore losses and dilution in his paper. He stated at the same time that ore mining losses are interdependent on the ore dilution. Wawrzyniak (1976) also drew attention to the interrelationship between mining losses and ore dilution, emphasising that they should be considered comprehensively for individual mine areas. It should be stated that the considerations, presented in Krasiczyński (1974) and Wawrzyniak's (1976)

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papers, on interrelationships between ore mining losses and dilution, were not supported by practical studies, therefore it was not found, to what extent this interrelationship exists in reality. The papers of Wanielista (1979) and Krasiczyński (1972) indicated the need of an economic analysis of mining losses and ore dilution size.

Wanielista's (1979) paper presented a method of an economic assessment of mining losses and copper ore dilution. This method allowed to determine costs, which may be incurred at the choice of a mining system and of the appropriate technical undertakings to reduce the mining losses and ore dilution. Instead, Krasiczyński in his PhD thesis (Krasiczyński 1972) presented an efficiency calculation of copper ore deposit mining in the Legnica-Głogów Copper Basin (LGCB), taking into account the amount of mining losses and ore dilution. Kot (1972) also applied the criterion of mining systems efficiency in his research, taking a discounted profit on panels mining as an efficiency measure, expressed by the current efficiency index. In this method it is debatable to assume a relationship between the efficiency index value and the area of the mined panel. In practice such an approach can result in improper interpretation of mining systems efficiency at mining small area panels. Partial analyses of mining losses and ore dilution amounts, originating at the LGCB and reasons for them, are provided in the papers prepared at the Institute of Deposit Mining Technology, Silesian University of Technology (Dziura, Zapotocki 1976) and at the Institute of Mining, Wrocław University of Technology (Praca Zbiorowa Instytutu Górnictwa Politechniki Wrocławskiej 1972). Dziura and Zapotocki (1976) analysed the development of mining losses and ore dilution in the 'Lubin' and 'Polkowice' mines over the years 1969-1973. They showed that the used mining systems, variability of the deposit thickness as well as a mineralisation and tectonic disturbances have a decisive impact on the amount of losses and dilution. They did not manage to determine the degree of mentioned factors influence on the amount of mining losses and dilution. Based on changes of their values in individual periods, they concluded that there was a relationship between mining losses and ore dilution. They analysed the recovery issue of deposit trapped in the support and protective pillars, and they also analysed mining losses and ore dilution in the 'Lubin' and 'Polkowice' mines in 1971, depending on the existing mining-geological conditions. They indicated a relationship between losses and dilution as well as between the used varieties of chamber-and-pillar mining systems and the mined panels tectonics.

Some authors studied the relationship of losses and dilution at selected parameters of mining systems. Krasiczyński and co-authors (Krasiczyński, Majchrowicz, Wilczyński 1974) discussed the advisability of copper ore deposit mining with use of chamber-and-pillar systems within a thickness range of 2.2÷2.7 m (parts of the deposit thickness smaller than the lower limit of those systems applicability, which is directly related to the ore dilution amount). The authors adopted a discounted profit on the panel under mining as the criterion for the chamber-and-pillar systems applicability. In conclusions of their paper they stated that the application of chamber-and-pillar systems in a deposit of 2.2÷2.7 m thickness may be justified in the case of a high content of useful component in diluting rocks. Otherwise it is advisable to apply a longwall system (Krasiczyński, Majchrowicz, Wilczyński 1974a).

Wanielista drew attention to the fact that parameters characterising the minerals quality (mineralisation, pollution etc.) are specified by technical standards, contracts with customers, and technological and economic conditions related to the anticipated direction of mineral use (Wanielista 1986). Jureczko, Buczek, and Drewniak (1986) noticed that the data describing conditions of minerals existence, apart from the technical aspect, derive from both natural conditions of mineral deposition, and the achieved technology level, enabling mining under conditions ensuring both economic profitability as well as the required work safety and comfort. Due to their studies they also confirmed an obvious thesis developed by foreign authors that the necessity of providing the staff with indispensable work safety objects and ensuring safe conditions (ventilation, cooling, etc.) by means of available technical means, a specific cost barrier is encountered frequently. It determines the mining method, and hence to a large extent also the qualitative parameters of the mined mineral. This balance provides the basis for - as they stated – a determination of extreme parameters enabling effective mining of resources (Jureczko, Buczek, Drewniak 1986).

The mining dilution of hard coal in the Polish mining sector has always been perceived as an element of resource losses (Drewniak, Rosielski 1980), for which determination rules are related to the design of mining operations together with an accompanying analysis of maximum resources use possibility carried out on the basis of an identification of geological and mining conditions. This approach used the linking of a mining plant operation with a so-called rational resource management by an introduction of the systematic and comprehensive control of proper resources utilisation. As a result, a necessity occurred to determine formal and legal solutions enabling the supervision authority to approve the acceptable level of losses (including dilution), whose mining is technically not possible both due to the shortage of technical means and the necessity to achieve an appropriate output volume.

As Lisowski (1981) noticed a few decades of mining practice showed that such a way of loss indicator settlement resulted and still frequently continues to result in the fact that the losses, assumed by mines, were generally high, and the resource utilisation indices were low, which in the social perception (high losses and low deposit utilisation) caused a negative assessment of the carried out mining operations. Therefore a natural reaction of mines consisted in pursuing ‘moderation’ of such assessments. The path from this reaction to resources reclassification, to their reduction and other similar operations was already short. Moreover, such operations frequently resulted in the loss of resources left in the deposit for a so-called oblivion. Their size was frequently larger than the amount of mined resources. The process of the so-called resources oblivion occurred both at the stage of planning and designing of mining operations, and also during a registration of their effects. Therefore, as noticed by Lisowski, the issue of correctness of formal and legal solutions, regulating mining management and utilisation of mineral raw material resources, should be considered primarily from the point of view of correctness of objectives, which they serve, and effectiveness of their role fulfilment. As the balance nature criteria, commercial resources determination criteria and categories of useful mineral losses are the basic instruments, by means of which, the resource utilisation is regulated. They should be considered one by one, focusing attention mainly on them. When assessing the criteria for balance nature as well as commercial and recoverable usefulness of resources from their effectiveness point of view one can say that these criteria fulfil their role (not always in the best way) only at specific points of activity aimed at the deposit utilisation. Beyond those points they lose effectiveness. Additional information starts to decide then about the way of deposit utilisation, the more detailed, the later is the phase of management and mining, while formal parameters, determining the balance nature and commercial usefulness of the deposit, become less important (Lisowski 1981).

Noticing these issues in the 1980s many authors (Wanielista, Kicki, Nieć, Darski, and Sobczyk) started to look for solutions aimed at resolving the imperfection issue of calculations in the field of resource losses and values of basic measures of their utilisation in the mining industry. A discussion started on the direction, in which the future changes should go.

According to Lisowski the starting point of carried out discussions consisted of the assumption that each generation - understood as a generation of miners, having a specific technology available - has the right to mine those deposits (or their parts), which ensure economic efficiency (profitability) to the business. However, no generation had the right, and still continues that, to destroy the deposit, and what it leaves for future generations should be handed over to them with the maximum amount of information facilitating the exploitation, if any.

Also the contemporary criteria, according to which geological resources are determined, distinguishing a mineral from dirt or separating a category of 'commercial geological reserves', are only a current interpretation of this principle. Thinking realistically - is it possible to imagine the use of another principle? Which generation would agree to mine deposits economically unprofitable for the benefit of future generations, which will be probably wiser and richer than the previous ones. Despite the fact that the sense of discussed principle seems obvious, practical methods of economic profitability determination for mining of these and not other parts of the deposit today are much more difficult than in the past. The carried out discussion resulted in establishing that a certain decision on the issue of mining or leaving a specific part of the deposit for future generations must be based not only on the balance nature criteria, but on a comprehensive economic analysis considering local natural and technical conditions (e.g. mutual arrangement of seams, available equipment). A significant element of this analysis must consist in an assessment method for the degree of underworked deposit parts damage (destruction), which would enable both a correct assessment of resource losses, and also a determination of the state of resources left for the future use. Moreover, in the process of use advisability assessment and mining losses assessment of the deposit, the balance (and commercial) criteria for resources usefulness should be replaced with an assessment of their economic efficiency, taking into account the value of all destroyed or damaged geological resources (Lisowski 1981).

It is known now that there is no contradiction between the pursuit of deposit protection and the requirement of maximum efficiency of the carried out business activity. Contrary to appearances, the economical (not wasteful) management of resources is highly profitable.

In his papers the author many times emphasised (Dyczko 2013, 2014, 2016) that the basic criterion for a classification, at a given time, whether to mine these and not other deposits or their parts (seams) must consist in a criterion of maximum economic efficiency observing a condition that the calculation is carried out in relation to the entire geological resources, in a long-term approach (30÷50 years), considering economic effects caused by resource losses, their underworking (and thereby increasing the costs of future mining) and a depreciation of the value of resources left in residual deposit parts. Worldwide since the 1970s, and in Poland since the end of 1990s, the economic efficiency calculation for the carried out process of resources depletion has been performed using computerised planning systems in decision-making processes. Computerisation seems nowadays a more effective enforcer of rational deposit management than formal and legal regulations used in the past, frequently not standing up to the confrontation with the destructive pressure of quickly changing needs of the economy.

The issue of quantitative and qualitative parameters rationalisation in designing the development and mining operations (of a hard coal deposit) has been so far a subject of many papers. Fixed or averaged qualitative parameters of the deposit were taken in the definite majority of papers from this field. Analytical or statistical methods were applied in classical papers, e.g. by: (Goszcz 1971); (Hurysz, Sikora 1963); (Magda 1990a); (Rabsztyn 1970a); (Sitko 1973); (Sitko, Chmiela, Kozyra 1973); (Walisko 1962), and economic or technical criteria for the solution assessment and selection, e.g. in papers by: (Kamionka, Zbyradowski 1967); (Kamionka 1971); (Karbownik, Pogonowski, Piwko, Łyczbiński 1978); (Karbownik, Pogonowski, Sznurawa 1979); (Kindla, Kozdrój, Sitko 1968), (Konopko, Hryniak, Kula, Kurzeja 1979), (Magda 1990b); (Magda, Domański 1991); (Magda, Franik, Domański 1992); (Parysiewicz, Wolski 1962), and (Soja 1964). Due to computational difficulties certain simplifying assumptions were used for a hard coal deposit of a great variability of qualitative parameters, to represent it reasonably precisely.

In his papers Magda (1982; 1983; 1984) formulated a concept of a space-time representation of coal seams mining, being theoretical premises expanded for the total production process within a real mine, mining a seam type deposit by the longwall system – a representation based on the idea of merging (integrating) smaller structural elements into bigger ones, up to obtaining a production process model in the real mine. This representation of the production process, whose element consists in an optimisation of dilution level within the mine and its individual structural elements, is comprised by mathematical modelling and it is an extensive issue, comprising a huge number of papers and studies carried out in the past and now by numerous specialists, studying this issue from the point of view of defined decision criteria.

A broad application of computer techniques in the designing practice of mines enabled to develop methods based on mathematical modelling (numerical methods), where the computational possibilities were practically reduced to minimum. Due to computer technique application it was possible to achieve high-precision effective solutions quickly. In addition, the computer technique enabled an automation of designing processes and a selection of a rational or optimal solution out of practically any number of design variants.

In the global mining IT tools, used for designing and planning the mining production, appeared on the market more or less at the same time, substantially affecting the quality improvement of the carried out mining-preparation process (Kaiser et al 2002). The first packages of software, supporting the deposits mining, appeared as early as in the 1970s, and the pressure of gold producers to search for effective tools, minimising losses related to exploring, documenting, and mining overly diluted raw material (average gold price on global stock exchanges at that time did not exceed USD 50/oz) are considered the main catalyst of their development. Created tools were developed both by mining corporations and research centres, whose staff overnight were becoming creators of innovative products, which on a competitive market quickly changed into commercial software, frequently more functional than that one developed within mining companies (Kapageridis 2005).

Finally at the beginning of the 1980s a majority of the world corporations abandoned their own research projects aimed at building IT tools supporting the process of mining production planning and scheduling in favour of a quickly developing commercial market.

In recent decades an incessant development of mining production planning systems is witnessed and one can risk a statement that it is not possible now to find a company, involved in mining of minerals, which would not use engineering software in the scheduling and planning process.

A particular progress in the field of deposit modelling and use of IT tools in the process of mining is visible in deep mines of noble metal ores, in which numerous companies obtained the first direct benefit from using those tools by reprocessing the data and using the information, which previously was considered useless.

The design supporting packages have a countless number of algorithms, from the simplest to the most complicated ones. Since the moment of geological operations until the ROM transport, an integrated algorithm system ensures a possibility of a continuous calculation of deposit resources, of controlling the workings state, and numerous others, at a permanent accessibility of a friendly graphical environment. The presence of algorithms and their complexity allow to control the time necessary to achieve the intended modelling objectives (Kapageridis 2005).

A generation of mining plans is a process proceeding linearly. The information from boreholes and the geodetic data are gathered to determine the structure and to analyse the deposit quality. The gathered data are then used to

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develop a model of the deposit and to determine its resources, based on the mining limitations.

The mining planning utilises the information from modelling to design 3D blocks of size enabling the production scheduling. The amount and quality of mineral in individual blocks are estimated and then - taking mining limitations into account - a production schedule is developed.

The optimisation of prepared schedule usually requires a creation of a few alternative production scenarios. If the production process is effective, integrated, and linear then the amount of time necessary to prepare a single scenario is relatively small. In addition, this process should be repeated, if new geological information becomes available. This is aimed at ensuring that the assumptions of the created plan will reflect the current conditions existing in the deposit. The inaccuracy of models can result in an origination of errors in the next steps of mining planning, which in turn can end up with an unexpected increase in the cost and decreased revenue on the mining activity (Wilkinson 2010).

The software for designing mining operations gets increasingly more dynamic, both in terms of the modelling itself and visualisation. The simulation of, say, movement of cutting machines, during which a series of optimised technological schemes is followed, is no more something extraordinary. Such schemes, developed in the 3D environment and visualised with appropriate time synchronisation, provide an almost real image of the minerals mining process (Kapageridis 2005).

Recently the geological-mining software, taking advantage of a dynamic development of IT technologies, is subject to changes typical for the entire sector, which in relation to the applications in the mining industry indicates the following trends (Jurdziak, Kawalec 2011):

- a potential concentration of companies delivering software to the mining industry,
- an expansion of the scope of actions covered by computer assistance – a development of new algorithms for selected processes of modelling and optimisation, an implementation of new IT technologies, a creation of new IT tools,
- an integration of proposed solutions, replacing the tools serving ‘island’ processes with comprehensive solutions to support documentation, to analyse and optimise the entire chain of value creation in a mining enterprise,

- an avalanche type growth of diverse data, gathered and processed with use of dedicated specialised software,
- a construction of IT environments to manage the process of specialised data processing to ensure a continuity and digital security of computer support on the scale of the mining corporation.

The 3D modelling is a flywheel of the progress witnessed nowadays, which recently became an extremely important tool opening new development directions. Mines of metal ore deposits in the Republic of South Africa, by means of special exploration techniques, based on broadly understood mining geophysics, for quite a long time have carried out practical planning and management of mining production (Campbell 1994); (De Wet et al 1994). Domestic methods for designing elements of underground coal mines, developed with the use of computer techniques, were presented e.g. in the following papers: (Karbownik, Pogonowski, Piwko, Łyczbiński 1978); (Karbownik, Pogonowski, Sznurawa 1975); (Magda, Franik 1989); (Magda 1994); (Stokes 1994); (Janik, Kuś 1992); (Kicki, Dyczko, Timler 2009); (Dyczko 2009); (Kicki, Dyczko 2008, 2009, 2010,2012); (Dyczko, Kicki, Stopkowicz 2007); (Dyczko 2013); (Dyczko, Galica, Sypniowski, Szot 2013); (Dyczko, Kowalczyk, Mól 2016); (Kowalczyk, Galica, Dyczko, Kołomański, Mól 2016); (Dyczko, Kołomański, Kowalczyk 2016); and (Dyczko 2016).

The scope of this publication, both in terms of the subject-matter and volume, required a limitation of selected methods of mathematical modelling in the field of mines designing theory, and basically to analytical methods, methods of variants, or combined methods applying computer techniques - methods using an economic criterion in the field of issues related to the mine size, mine model, the sequence of deposit mining within the mine field of the coal mine, the size of mining panels and mechanised longwall parameters, and finally methods of qualitative parameters estimation developed mainly for the needs of the coal mining sector.

Methods for a determination of optimum mine size parameters may be found in the following papers: (Ajdukiewicz 1957); (Bromowicz 1963); (Chudek, Dąbrowski 1979); (Chudek, Paździora, Dąbrowski 1979); (Jawień 1965); (Krupiński 1963); (Lama 1977); (Sharp 1974); (Siska, Vitek 1967); and (Wolski, Pogonowski 1968a, 1968b). Analytical methods deserve special attention among them: Ajdukiewicz (1957) and Bromowicz - Jawień (1963); Jawień (1965); and Krupiński (1963). Also the issues of determining the size of panels and a related choice of mechanised longwall parameters are connected with the issues of deposit opening structure on the mining level. The following papers may be

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mentioned in the field of optimum panels size: (Gazda 1971); (Pogonowski, Karbownik 1978); (Saginow, Kwon 1972); (Saginow, Kwon, Adiłow 1974); (Stachowicz 1975); (Suchan 1974); and (Wyra 1981), among which the paper by Suchan (1974), albeit prepared for the needs of ore mining, contains a big cognitive material in the field of methodology for the optimum panel size determination, with special emphasis on dilution as qualitative losses. In the field of mechanised longwall parameters a determination of the following papers should be mentioned: (Bindels 1964); (Claes et al 1975); (Denk 1977); (Horak 1966); (Kamionka, Wasilewski 1967); (Kamionka, Zbyradowski 1969); (Kamionka, Zemła 1972); (Karbownik, Pogonowski, Sznurawa 1975); (Łokszin, Korobki 1968); (Nowak, Chmiel 1981); (Parysiewicz, Wolski 1962); (Rabsztyn 1970b); (Rabsztyn, Kozdrój 1967); and (Soja 1964), the majority of them present analytical methods.

The main parameters of mine size discussed now, such as the area of mine field, commercial resources, resource losses, a period of mine life, and an output determine so-called operational dilution, which is a process planned by the engineer designing the mining plant operations, when the mining plan is being developed. The dilution can also appear unexpectedly and then it is called an unplanned dilution. The following papers may be mentioned from the field of aforementioned mining economy assessment and the calculation of mining investments efficiency: (Biezrukowa and Malkin 1979); (Dorstewitz 1967); (Jawień, Hajdasiński 1973); (Karbownik 1982a); (Rokita, Strzoda 1970, 1976); (Sitko 1976); (Sitko Chmiela, Kozyra 1973); (Węgierski, Wolski 1964); (Wolski, Pogonowski 1968a), and the (Fiszel 1969) paper deserves special attention in the field of investment efficiency theory. As regards underground mine construction cycle the following papers may be mentioned: (Cyrnek 1974); (Cyrnek, Soliński 1977); (Karbownik 1981, 1982b); (Pogonowski 1971); (Soliński, Cyrnek 1979); and (Dyczko et al. 2013).

The management of the hard coal deposit in the Lublin Coal Basin encountered in practice new, unrepeatable in other Polish coal basins, geological and mining conditions and related new research problems. Difficulties with mastering specific conditions of the deposit location became an inspiration for the origination of many papers, published and unpublished, related to designing of the deposit mining management. Among the published papers, in particular on a development and designing of mining operations, the following papers may be mentioned: (Gałąź, Stachowicz 1994); (Gałąź, Stachowicz 1995); (Gawroński, Kozek, Stachowicz 1988); (Głuch, Kosiński, Limburski, Stachowicz 1987); (Stachowicz, Krukowski 1994); and (Stachowicz, Kosonowski, Kozek 1995), and among those unpublished, e.g.: (Lachman 1978); (Soliński 1977); (Jaworski, Kozek 1987); (Gałąź 1994); (Dyczko 2006, 2007); and (Dyczko et al 2011, 2012, 2013, 2014, 2015).

The necessity for coal producers to adapt to customer requirements under free market economy conditions requires developing of new methods, which would facilitate a selection of rational solutions in a qualitatively new situation, which has not been considered in development and mining operations designing methods elaborated so far. The analysis of the issue showed the lack of such a designing method for a deposit of high variability of qualitative parameters. This work is an attempt to fill the gap in this research area and at the same time it is oriented towards practical aspects.

The topic of this publication is related to a development of assessment methodology to investigate the impact of ROM pollution on the efficiency of hard coal production process measured by means of economic indicators. Geological in situ observations, carried out by the Author, of roadheadings, mining longwalls, as well as headgates and tailgates allowed to develop some methods of current monitoring of the deposit mining cleanness in the Lublin Coal Basin. A very important aspect of this work consisted in an assessment of possibilities to use the knowledge on the scale of ROM pollution forecast in the process of mining production scheduling. The computerisation progress in the field of mining production planning allows for an efficient modelling of the deposit structure and its qualitative parameters, and it also enables a development of an appropriate mathematical model to forecast the amount of ROM pollution in a mining plant together with a economic assessment of the production process efficiency.

The long-term research work on the undertaken issues allows stating that the coal pollution is an unfavourable process, having a negative impact on the ROM extraction efficiency, however, there are possibilities to manage (control) the amount of pollution, both through the use of available technological and technical solutions, changing the process of deposit mining. The scale of the pollution impact on the efficiency of ROM extraction may be controlled and evaluated with support of appropriate IT solutions.

To determine the impact of pollution - as qualitative losses - on the efficiency of coal extraction in the conditions of an underground mining plant, using the example of LW Bogdanka SA, and to develop a model for a management and monitoring of the extracted ROM pollution, a research methodology was adopted, based on experimental studies and in situ measurements carried out during the mining operation of a specific coal seam part with use of the longwall system in LW Bogdanka SA. A broad scope of data and measurement results were analysed using quantitative and qualitative methods. Numerous methods, techniques, and tools were used in the detailed studies within the undertaken issues, including:

- the expert assessment,
- the statistical analysis (descriptive statistics, correlation analysis, regression analysis, variance analysis),
- the probabilistic modelling (Monte Carlo simulation, distribution sampling, statistical distribution parameters determination, selection of probability distributions, random numbers generation),
- the cluster analysis,
- the mathematical modelling,
- the assessment of economic efficiency (using the measure of Net Present Value - NPV).

The choice of those methods on one hand resulted from the features of empirical tests (quantity, quality, specific nature and scope of data) and on the other hand - it was conditioned by an indication of achievable economic effects of the mining process rationalisation, consisting in mining less dirt, where it is justified from the geological and technical point of view.

3. Analysis of ROM Pollution Origination in the Process of Hard Coal Mining

The hard coal mining is inseparably related to an extraction of dirt. The coal ROM from underground mines worldwide contains 10÷60% of non-flammable mineral substances, that is waste (Palarski 2009). The process of coal separation from waste takes place during mining processes via selective mining or in preparation processes. The share of dirt is unfortunately unavoidable and it is related to the location of coal seams in the environment of other rock formations. These are sedimentary rocks, most often sandstone, shale and claystone. These rocks get to the coal ROM during:

- an operation of opening workings,
- an operation of development activities,
- an operation of mining activities, including:
  - ripping of the working roof,
  - ripping of the working floor,
  - roof rock falls.

In addition, the dirt exists in the form of bands in the coal seam. It is assessed that each mined tonne of hard coal is accompanied by 250÷300 kg of coal waste. During roadways driving in seams of small thickness, the stone weight share on the belt transporting the ROM from the face, can reach even 70%. In general, because of varying quality of mined seams the net output does not change. The record one was achieved in 2004 and it amounted to 5.5 million Mg at the yield of 78.26% (Kicki et al 2016).

A large amount of dirt originates from development operations carried out primarily in the seams of small thickness. Under the LW Bogdanka conditions such a seam is the seam 385/2 mined by means of a shearer from 1997, and from 2010 by a plough. Large amounts of dirt get to the ROM during driving of headgates and tailgates, both in the case of the plough and shearer mining technologies. From the working cross-section it is easy to conclude, where it is precisely visible, that only a small part of the working cross-section is covered by the coal seam. The case complicates even more, when to satisfy the needs of plough mining technology the headgates and tailgates must have bigger dimensions at a smaller seam thickness. The factors affecting the amount of obtained waste rock in the mine, assuming the ROM mining as ‘clean’ as possible, may be divided into four groups (Kicki et al 2016):

- the structure of the mine, i.e. the way of deposit opening and development as well as of the main development workings arrangement in relation to

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each other and to the shafts. The amount of stone which may be reduced due to the deposit structure at the deposit opening, where primarily the amount of waste originating from the mine goes down;

- geological and mining conditions of the deposit location, in particular a description of basic parameters characterising the deposit, i.e. thickness of seams, conditions of their deposition, faults, type of roof and floor rocks, as well as thickness of dirt bands. In general, there is no possibility to have a major impact on these factors;
- technical conditions of mining operation determining the use of mining systems, the width of roof opening and the scope of necessary ripping. In this case it is possible to limit the ROM pollution with dirt by an application, on a broader scale, of the longwall retreat mining (after driving headgates and tailgates and identifying tectonic disturbances, if any);
- the technology of coal mining, which affects appropriate ROM crushing, which in the case of lack of fines preparation has a significant impact on the amount of dirt obtained in the preparation process. A development of mechanisation of mining and loading has an adverse impact on the degree of the ROM pollution (it is difficult now even to separate coarse stone from the ROM, while there was no problem with that when conveyors were not used).

3.1. Structure of deposit opening versus cleanness of deposit mining and amount of dirt

To be capable of analysing the structure of deposit opening, it is necessary to understand the original model of the mine, that is the spatial division of the deposit proposed by the designer - within the mine field - by vertical and horizontal planes and auxiliary levels, wings, sub-panels, panels, hence the way of opening workings (shafts, cross-cuts, drifts) arrangement in stone as well as of development ones (roadways, inclines) in the deposit and also in relation to each other. In practice the mine model depends primarily on the mine field shape, then on natural conditions of deposit location, disturbances, richness, the number and mutual arrangement of seams, and their inclination. So it is possible to state that the mine model depends on very many interdependent factors, causing the need to design a proper structure of the deposit opening on a given level. Two basic types of deposit structure opening (development) may be distinguished with many varieties, i.e.:

- the structure with seam development of the deposit,
- the structure with stone (geometrical) development of the deposit.

In practice both mentioned structures can exist together, creating so-called complex models, frequently very complicated. During the mine life only a part of the workings from the general mine layout is driven and operated. Other workings are either liquidated as already unnecessary, or have not been driven yet. So in every phase of the mine life only a part of designed mine model is under operation. The situation of mine model changes over time and in space, vertically at opening and closing the old levels, and in the vertical and inclined planes at the development and mining of panels and mining a specific level.

The structure with seam development of the deposit, used in the coal mining, is frequently referred to as a coal model and features a smaller number and length of stone workings as against the structure with stone development of the deposit, enabling to get the smaller CAPEX and also a shorter time of level (mine) construction.

The structure with stone development of the deposit consists in driving the majority of the deposit development workings in the rock, which is frequently related to much higher CAPEX at the beginning of mine commissioning, but later on it allows to maintain lower costs of development workings maintenance (driven in stone they longer preserve their function without the necessity of a reconstruction). The stone structure has a doubtless advantage of possibility to create a larger number of independent areas and panels in the seam, an easier ventilation isolation of individual panels and a simple organisation of the main transport in a level, with a possibility of full automation.

The mine structure has a substantial influence on the amount of dirt. The application of deposit development structure allows to reduce the amount of dirt originating from the mine underground operations, however its use is possible only in favourable geological-mining conditions, in the case of seams deposited in a shallow and regular way, and now in Poland there are not many deposits of this type.

3.2. Geological-mining conditions of deposit location versus cleanness of deposit mining

The amount of hard coal losses and pollution is mainly affected by the following mining and geological factors:

- the deposit tectonics,
- the type and nature of roof and floor layers,
- the seam thickness,
- the variability of angle of dip and strike,

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– the hazards (gas, fire, water),
– the other factors, which can include: cost of mining operations and coal sales prices, partly satisfied demand for winning machines and powered roof supports adapted to varying thicknesses and angles of seam dip.

The deposit tectonics is a significant factor affecting the amount of coal losses in the mining process. Faults of throw bigger than the thickness of mined seam result in the necessity of carrying out new fringe drifts and because of that leaving fault pillars. Water-bearing faults, due to the necessity of leaving safety pillars, to a significant extent affect the amount of losses. Faults of high throws are usually accompanied by a number of small faults, forming so-called fault zones, in which the mining operations encounter huge technical difficulties and which are limited most frequently due to economic reasons. Statistical studies showed a clear impact of the type and nature of roof and floor layers on the amount of losses and pollution (Jureczko et al 1986). This applies in particular to mining seams of very weak roofs, easily falling, cracked, whose maintenance requires to leave a coal layer in the roof. Both very weak roofs and swelling floor layers in each case cause specific, technical, mining difficulties, contributing to increased amounts of losses.

The seam thickness is one of the most important factors affecting the pollution amount. Because of limited height of supports in the case of high-thickness seams, it is frequently necessary to leave a layer of coal in the roof or floor. In the seams of varying thickness, because of difficulties in adapting appropriate winning machines, it is necessary to mine the seam with the predetermined height, resulting in leaving non-extracted coal layers in the seam roof or floor.

Also the angle variability of seam dip and strike has an adverse impact on clean mining of the deposit. This is related to an application of full mining mechanisation and production high concentration at a high speed of mining fronts advance.

Gas, fire, and water hazards usually cause difficulties in the case of mining seams or their parts. They frequently result in leaving safety pillars or in the necessity of driving new longwall fringe drifts, leaving a non-extracted part of the longwall panel.

As it was determined in the result of studies (Jureczko et al 1986), the losses, caused by complicated tectonics, seam thinning and termination, are most frequent reasons in the group of losses inside panels due to geological reasons. Mining reasons result almost only from the support pillars for the main transport roadways and also from the protective and support pillars for operating mine levels. The situation is similar in the group of losses outside panels due to geological-mining reasons, with the difference that the share of geological reasons prevails, determining to a large extent the size and shape of the panel.

3.3. Technology and engineering of coal deposits mining operations versus cleanness of deposit mining

A choice of the mining system is the basic task of each mine. It should be made considering fully the progress in the mining knowledge in the field of theory and practical mining experience related to mining of the given deposit or a deposit located in similar geological conditions. A properly chosen system must ensure maximum work safety and the highest production efficiency at the lowest mining losses.

The technical and economic criteria for a selection of the deposit mining technology impose the need to resolve a number of partial issues, such as:

- the method of deposit mining,
- the method of roof management (caving, packing, retarded caving),
- the geometry of workings and pillars,
- the type of support,
- the machinery and equipment,
- the quantitative and qualitative losses of useful mineral.

In underground mines, in which the effect of dilution occurs, a selection of proper mining system plays a significant role, and in particular an adaptation of the mining height to changing parameters of the deposit, mainly its thickness. This requirement is closely related to a possibility of using appropriate mining machines. Also the natural hazards have an important impact on the system choice. This implies a necessity of a current modification or a creation of new mining system solutions, which take into account state-of-the-art achievements in the field of technical progress and methods for controlling those hazards.

An application of the proper mining system limits the amount of losses and pollution, however, it is related to the necessity of using more expensive mining systems and technological operations, allowing to extract the deposit in the ‘clean’ way (e.g. the use of consolidated stowing is effective, nevertheless it is always related to a significant cost increase). On the other hand the amount of losses and dilution as well as the type of mining system are already determined during a creation of the deposit management concept, because the amount of mining losses directly results in a reduction of the possessed resources of the mined mineral, which can have a big impact on increasing mining costs in the

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future. All the aforementioned factors resulted in the fact that over the years of the Polish underground mine operations many mining systems were used, which may be divided *inter alia* by the type of mining working, the way of roof management, the deposit thickness etc. Fig. 3.1 presents a simplified classification of coal seam mining techniques.

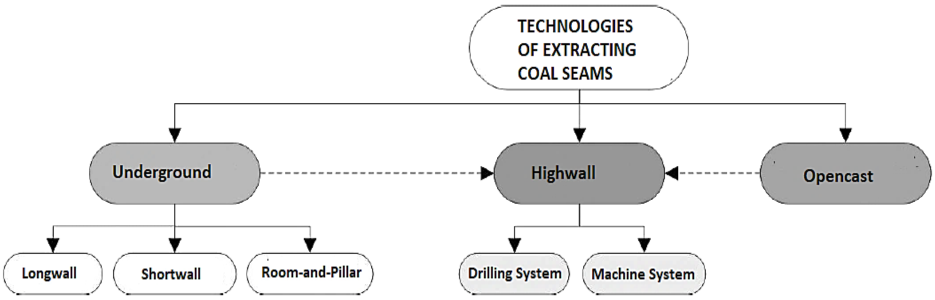

Fig. 3.1. Simplified classification of coal seam mining techniques

The simplified classification of coal seam mining techniques presented above, comprises mining methods used both in the past as well as at present, in which three basic methods of coal mining, comprising underground, highwall, and opencast techniques can be distinquished. In the case of underground mining the mining systems can be divided by the type of working, the way of roof management, the deposit thickness etc. The mining system must be rationally well-suited to the existing conditions first of all, that is it must meet the following basic criteria:

– an economic profitability,
– a maximum deposit utilisation, i.e. minimisation of useful mineral losses,
– to provide the working staff with proper occupational health and safety conditions,
– to ensure proper ecological conditions, i.e. a minimisation of mining damage and environmental pollution with solid, liquid, gaseous, and radioactive (if such occurs) waste.

For many years the longwall system has been the basic mining system in the Polish hard coal mining sector. It consists in mining a part of the seam, most frequently rectangular in shape, limited by roadways, by means of one long face. The mining voids (goafs), formed after the seam mining by the longwall technique, are liquidated. This part of the mining process can be realised by full or partial caving of roof rocks or filling with the stowing material. The caving process of roof rocks is the prevailing method of goafs liquidation in the Polish mining sector in the last decade. In 2016, the Katowicki Holding Węglowy

(Katowice Coal Holding), as the last one in Poland used hydraulic filling in its coal mines, made the decision to abandon this way of goafs liquidation. A high versatility and favourable economic conditions cause that the longwall system is one of the most widespread technologies of underground mining worldwide. Fig. 3.2 presents a spatial diagram of the longwall system.

Proven variants of this system are used for seams of different dip and thickness. Basically also the type of roof rocks does not limit the use of such systems. Only strong disturbances in seams deposition, such as folding and a dense faults system cause problems in the longwall mining. Mining by long faces (250÷300 m) allows to obtain favourable economic and technical indices, in particular in the conditions of long panels, which due to numerous tectonic disturbances are more and more difficult to achieve under the conditions of the Polish mines. Basic advantages of longwall systems include:

– a small amount of development activities,
– low mining losses,
– a high concentration of mining operations,
– an easy roof management,
– a possibility of full work mechanisation.

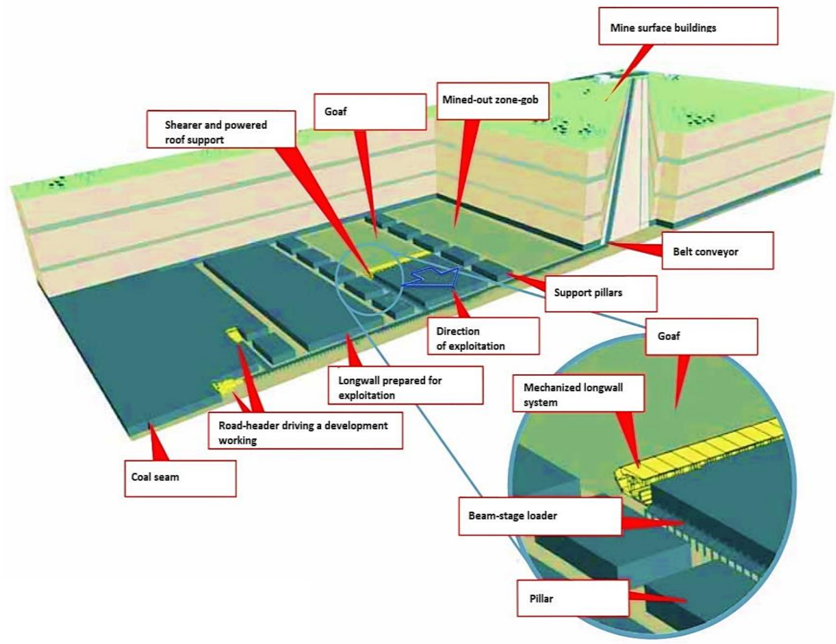

Fig. 3.2. Spatial diagram of the longwall system (*The Coal Resource 2008*)

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Because of the direction of the developed deposit part, the mining activities along the strike, cross-working, and diagonal face mining systems are distinguished. The direction of mining against the deposit strike is the distinguishing feature in this division. In the case of mining in accordance with the strike, the longwall systems belong to the group of mining along the strike systems (Fig. 3.3), while in the case of mining perpendicularly to the strike - to the group of cross-working systems. If the direction of mining is askew against the strike, such systems belong to the group of diagonal face mining. The mining operations along the strike are the prevailing mining system in the Polish mining industry.

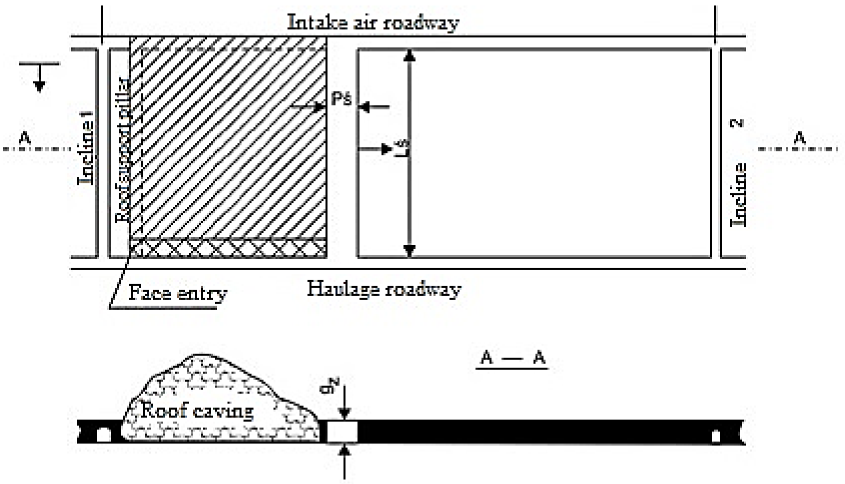

Fig. 3.3. Mining along the strike system with roof caving (*Sikora et al 1995*)

In recent years effective mining of coal from thin seams, below 1.5 m, became a breakthrough. In low shearer-equipped longwalls there is - both natural and forced by geometrical dimensions of the longwall equipment – a trend to increase the longwall height, which is most frequently realised by ripping the seam floor and roof. This approach results in a deterioration of the ROM quality. Considering rational and efficient management of coal resources, deposited in small-thickness seams, mining operations with the use of plough technique has been undertaken. An effective return of the plough technique to the Polish mining sector after many years was conducted by the JSW SA and LW Bogdanka SA, where this technology was treated as the project of strategic importance for a development of both companies.

3.3.1. Bogdanka experience in limiting the dirt in the case of various mining techniques

In the LW Bogdanka SA the mined rock originates mainly from driving mining workings in coal seams that have dirt bands and from the layers of dirt ripped in the working roof and floor. The seam thickness is smaller than the height of the mining working (e.g. seam 385/2 is 1.5÷1.9 m thick, including the dirt band of 10÷20 cm, and the height of development working is approx. 4.5 m). The cross-section of the roadway driven in seam 385/2 comprises approx. 10 m^2 of coal and approx. 18 m^2 of dirt.

In the Bogdanka mine 4.5÷4.7 million Mg of mining waste are obtained now. More than a half of them are recovered and are used economically. The rest goes to the mining waste disposal facility in Bogdanka. The largest amount of generated waste - more than 84% - originates from the coal preparation processes (Table 3.1).

Balance of mining waste generated in the years 2005÷2019 (*LW Bogdanka 2020*)

Table 3.1.

Year	Total generated waste (Mg)	Coal preparation waste (Mg)	Underground stone (Mg)
2005	2,619,039	2,168,835	450,205
2006	3,074,492	2,610,271	464,221
2007	3,337,444	2,926,537	410,907
2008	3,047,323	2,568,598	478,725
2009	3,788,150	3,200,198	587,952
2010	3,288,948	2,547,739	741,209
2011	4,050,085	3,362,872	687,212
2012	4,742,458	4,096,022	646,436
2013	5,064,500	4,426,158	638,342
2014	5,624,451	5,013,323	611,128
2015	4,770,255	5,058,631	288,376
2016	5,639,735	5,928,398	288,663
2017	5,116,205	5,304,256	188,051
2018	6,051,640	6,216,257	164,617
2019	5,617,362	5,943,950	326,588

The main places of rock origination comprise at present and probably will comprise in the future as well such mining operations as:

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– driving horizontal and inclined roadways in the seam 385/2, in panel VI, in Nadrybie and in panel VI and VIII, in Stefanów, and on the level 990 in Stefanów,
– mining the longwalls,
– reconstructions and floor ripping of the workings operated in Stefanów and Nadrybie panels, including floor ripping of the headgates and tailgates of plough longwalls (the ripping is carried out in rock layers).

Fig. 3.4 presents amounts of mining waste generated by the LW Bogdanka against the output background. As the presented specification shows each tonne of coal mined in the years 2005÷2019 generated on average 0.61 Mg of waste.

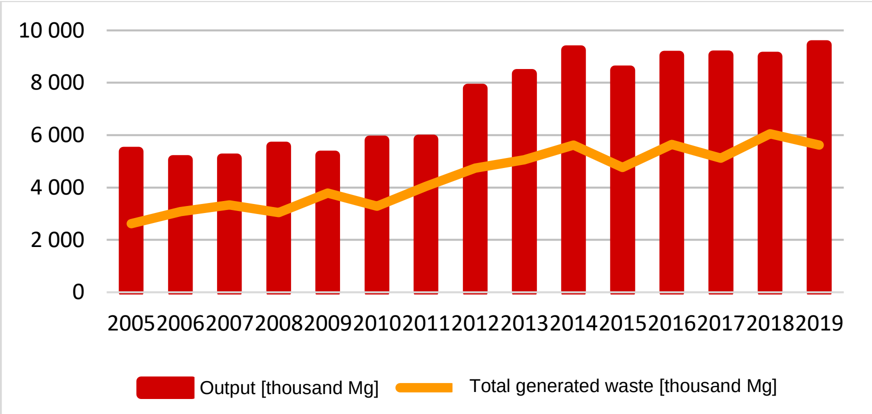

Fig. 3.4. Coal output and amounts of waste generated in LW Bogdanka in the years 2005÷2019 (*Own study based on MEERI PAS studies*)

An estimated forecast of the amount of extracted stone consists of the following rock sources:

– approx. 3,500 t/d - roadways driven in Stefanów and Nadrybie panels;
– 5,900 ÷ 7,800 t/d - roadways and longwalls mined in Stefanów and Nadrybie panels;
– 6,750 ÷ 8,650 t/d - roadways and longwalls mined in Stefanów and Nadrybie panels together with extracted stone delivered in cars from mining operations carried out in the Bogdanka panel (option).

Table 3.2 provides the forecast of dirt amount by 2034 according to the current deposit model and production schedules. This forecast was made by the Author based on geological and mining data available at the LW Bogdanka and his own original production schedule, which was expanded till the end of mining operations in the current mine field of the coal mine, i.e. by 2034.

LW Bogdanka production forecast together with the dirt amount
(*own study acc. to LW Bogdanka figures*)

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Table 3.2.

No	Years	Run-of-mine output [Mg]	Net coal yield [%]	Forecast of dirt [Mg]
1	2020	13 076 923	65.00%	4 576 923
2	2021	13 076 923	65.00%	4 576 923
3	2022	13 076 923	65.00%	4 576 923
4	2023	13 076 923	65.00%	4 576 923
5	2024	13 076 923	65.00%	4 576 923
6	2025	13 076 923	65.00%	4 576 923
7	2026	13 076 923	65.00%	4 576 923
8	2027	13 076 923	65.00%	4 576 923
9	2028	13 076 923	65.00%	4 576 923
10	2029	13 076 923	65,00%	4 576 923
11	2030	13 076 923	65.00%	4 576 923
12	2031	13 076 923	65.00%	4 576 923
13	2032	13 076 923	65.00%	4 576 923
14	2033	13 076 923	65.00%	4 576 923
15	2034	13 076 923	65.00%	4 576 923

The amount of dirt provided in the table derives from the output planned in the mine and from so-called coal yield, being a percentage of the coal weight in the total weight.

Many years of studies on mining waste properties in the LW Bogdanka SA confirm the stability of physicochemical parameters. In terms of petrography it is a mixture of mainly claystone and mudstone, i.e.:

- dark grey clayey shale of clear cleavage, laminated with an organic matter (approx. 10%),
- grey claystone (approx. 30%),
- grey mudstone without stratification (approx. 35%),
- grey-brown clay ironstone (approx. 15%),
- light-grey fine-grained sandstone (approx. 10%).

Having in mind the location of the Bogdanka mine in the region of precious natural protected areas, in the era of growing environmental awareness of the society, the aspect of natural environment care becomes increasingly important. The amount of waste generated in the production process is one of bigger issues related to the mine environmental impact.

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3.3.2. Geological and mining conditions of coal mining against the background of mining strategy at the LCB

The Lublin Coal Basin features a simple structure. The existing Lublin and Westphalian A-B beds form the Lublin formation, which is richest in coal. This formation is most frequently built of mudstone-claystone series, containing approx. 42% of mudstone and 34% of claystone, often with sphaerosiderite and inferior with sandstones (approx. 14%). The geological structure of the discussed area comprises Quaternary, Tertiary, Cretaceous, Jurassic, and Carboniferous formations. Table 3.3. presents a lithological and stratigraphic profile of the LCB deposit.

Lithological and stratigraphic profile of the LCB deposit (*Porzycki 1972*)

Table 3.3.

<table>
<tr><th colspan="5">Stratigraphy</th><th>Deposit thickness [m]</th><th>Lithology</th></tr>
<tr><td>Quaternary</td><td colspan="2">Holocene
Pleistocene</td><td colspan="2"></td><td>2.00÷80.00</td><td>sands, peats, all-in-aggregates, gravels, clays</td></tr>
<tr><td>Tertiary</td><td colspan="2">Pliocene</td><td colspan="2"></td><td>3.90 – 30.00</td><td>clays</td></tr>
<tr><td rowspan="2">Cretaceous</td><td colspan="2">upper</td><td colspan="2" rowspan="2"></td><td>499.90÷616.20</td><td>marls, chalkstone, chalk-like limestones, limestones with flint inserts and nodules, sanded up limestones</td></tr>
<tr><td colspan="2">lower (Albian)</td><td>0.60÷8.60</td><td>sands and quartz sands with glauconiteand phosphatic nodules, sandy marls</td></tr>
<tr><td>Jurassic</td><td colspan="2">upper medium</td><td colspan="2"></td><td>96.00÷154.85</td><td>pelitic, oolitic, organodedritic, detritic, sandy, crystalline limestones, calacareous and dolomitic sandstones, dolomites, claystones, mudstones, sandstones</td></tr>
<tr><td rowspan="6">Carboniferous</td><td rowspan="4">upper</td><td rowspan="4">Westphalian + Namurian</td><td>Lublin beds</td><td>Lublin formation</td><td>45.30÷408.10</td><td>claystones, mudstones, sandstones, coal seams</td></tr>
<tr><td>Kumów beds</td><td rowspan="2">Dęblin formation</td><td>169.80÷242.60</td><td>sandstones, mudstones, claystones, thin coal seams, few inserts of carbonate rocks</td></tr>
<tr><td>Bug layers</td><td>93.40÷123.90</td><td>claystones, mudstones, sandstones, thin coal seams, inserts of carbonate rocks</td></tr>
<tr><td>Komaró w beds</td><td>Terebina formation</td><td>176.15÷236.50</td><td>claystones, mudstones, sandstones, limestones, thin coal inserts</td></tr>
<tr><td rowspan="2">lower</td><td colspan="2">upper Visean</td><td rowspan="2">Huczwa formation</td><td>94.25÷108.70</td><td>limestones, marls, sandstones, mudstones, thin coal inserts</td></tr>
<tr><td colspan="2">variegated series</td><td>8.00÷37.80</td><td>sandstones, claystones, mudstones, limestones of variegated colouration</td></tr>
<tr><td>Devonian</td><td>upper</td><td colspan="3"></td><td>drilled roof part</td><td>sandstones with inserts of claystones and also limestones and dolomites</td></tr>
</table>

The majority of the LCB seams feature a complex internal structure with sedimentation and tectonic disturbances, although in the Bodganka deposit they are less numerous than in other coal basins. Up to even five dirt bands occur in the seams, most frequently from 5÷8 cm to more than 20 cm thick, causing the seams splitting. The share of dirt bands in the seam is variable from 3÷5% (at one dirt band) to more than 15% (at the occurrence of three or four dirt bands, e.g. in the seam 382). The existing faults, which were originally found during the deposit exploration by means of seismic methods, after a thorough analysis turned out to be rock mass disturbances in the zone of small flexure-type seam deviations.

Mining and geological conditions of deposits location as well as technical and organisational factors decide about the development direction of underground hard coal mining technology. The following parameters are decisive as regards mining and geological conditions:

- the depth of deposit location,
- the deposit thickness,
- the deposit inclination,
- the physical and mechanical properties of rocks,
- the geological disturbances,
- the natural hazards occurrence.

The following factors are decisive for a selection of the mining technique:

- the conditions of seam deposition, in particular the deposit thickness,
- the longwall parameters, in particular its height,
- the coal properties expressed by coal cuttability and side-crumble angle (breaking characteristics),
- the required daily output.

In the case of the coal seam No 385/2 in the ‘Bogdanka’ panel in the LW Bogdanka SA the following characteristics of mining and geological conditions were found:

- the seam thickness: approx. 1 m, where the seam thickness in the first longwall in panel one (No 1/I/385/2), which LW Bogdanka SA intends to mine as the first, is 0.93÷1.3 m, while in the entire seam, consisting of a few longwalls, in certain places the thickness reaches 1.7 m,
- the primary temperature of the rock: up to 32°C,
- the methane hazard: category I, workings with degree ‘c’,
- the rock-bumps hazard: does not occur,
- the compression strength of coal seam: 8÷19 MPa,

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– the type of roof rock/compression strength - claystones and medium-compact mudstones/4.5÷38 MPa,
– the type of floor rock/compression strength: claystones and poorly-compact mudstones/2.5÷20 MPa,
– the seam breaking characteristics (acc. to Protodiakonow): 0.75÷1.20.

Figs. 3.5÷3.10 present the exploration state of seams planned for mining in terms of thickness and dirt band thickness:

Fig. 3.5. Map of the seam 385/2 thickness (*own study*)

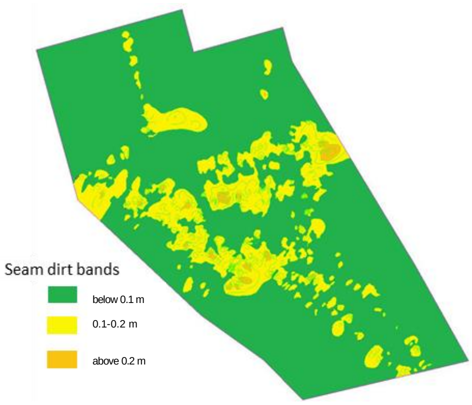

Fig. 3.6. Map of the seam 385/2 dirt bands (*own study*)

Fig. 3.7. Map of the seam 389 thickness (*own study*)

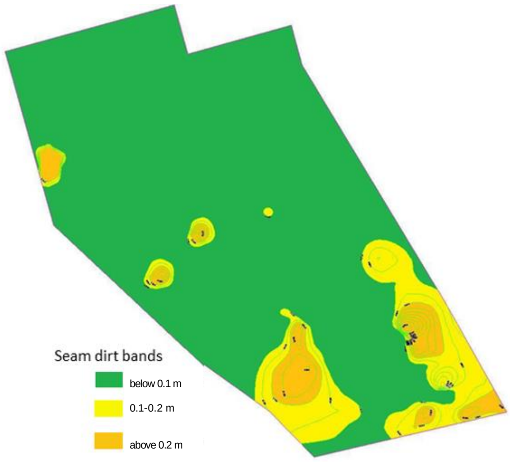

Fig. 3.8. Map of the seam 389 dirt bands (*own study*)

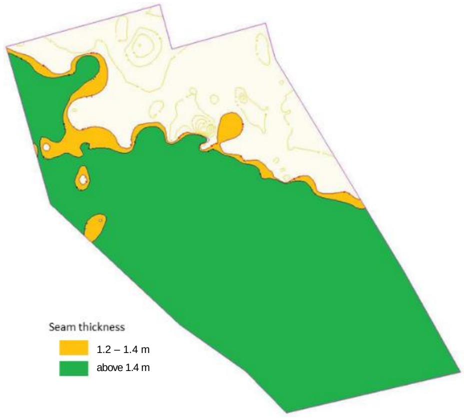

Fig. 3.9. Map of the seam 391 thickness (*own study*)

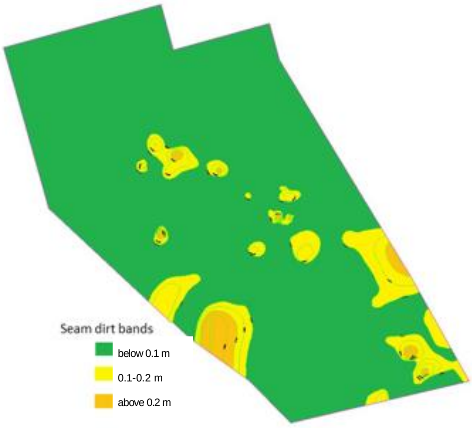

Fig. 3.10. Map of the seam 391 dirt bands (*own study*)

The presented figures show that the greatest amounts of dirt bands occur in the seam 389 (yellow corresponds to the dirt band thickness from 0.1 m to 0.2 m, while orange - to thickness exceeding 0.2 m), where in the south-eastern part the dirt bands thickness reaches 0.55 m. It is worth emphasising the fact that the pollution of the seam 391 is small, where in the area of coal thicker than 1.2 m, the dirt bands thicker than 0.1 m constitute approx. 18% of the area. For a comparison, this index for the area 389 is 52%.

The seam 385/2 is one of the richest and most regular LCB hard coal seams, mined mainly within the Bogdanka panel (the deposition depths approx. 950 m). The seam 385/2 is constituted of humic micro-banded coal with prevailing glance coal, with occasionally occuring sapropelic coals. The dirt bands existing in this seam feature varying thicknesses, ranging from 0.10 m to 0.16 m. In the seam roof the alternating layers of claystone, mudstone, and sandstone, which are classified as roof class II, III, and IV occur. Tables 3.4 and 3.5 present general and qualitative parameters of the seam 385/2.

Selected parameters of the seam 385/2
(*own study based on LW Bogdanka SA figures*)

Table 3.4.

Parameter	Type/value
thickness	from 1.3 m to 1.6 m
dip	from 0° to 2° towards the west
dirt bands (composition)	claystones and coal shale
flooding	generally dry seam; local flooding
roof-floor conditions	good
coal compression strength	from 8 MPa to 19 MPa

Qualitative parameters of the seam 385/2
(own study based on LW Bogdanka SA figures)

Table 3.5.

Qualitative parameter	Type/value
Coal type	32 - gas-flame coal, 33 - gas-flame coal, 34 - gas-coking coal
Ash content	from 3.59% to 35.19%; on average 8.37%
Calorific value	from 19 878 kJ/kg do 30 226 kJ/kg; on average 25 972 kJ/kg
Total sulphur	from 0.52% to 2.2%; on average 1.11%

3.3.3. Experience and efficiency of longwall systems use in the LW Bogdanka SA in the case of various mining technologies

The underground mining operations, carried out for more than 30 years in the Lublin Coal Basin, have been based only on the shearer mining technique. This technique requires a permanent presence of shearer operators at the longwall and the shearer operator to follow the operated machine. This fact and also the shearer and the armoured face conveyor dimensions decide that the shearer technique can be efficiently used in longwalls higher than 1.6 m. The Bogdanka mine in its mining activities so far, has refined the technology of longwall mining with use of the shearer longwall cutting to a satisfactory level. Because of that it reached a high concentration of mining operations, high productivity, and favourable economic results when mining the seams thicker than 1.6 m. This technique, in the case of the LW 'Bogdanka' SA, in the seams 2.0÷2.5 thick allows to achieve a daily output from one longwall up to 20,000 Mg/d of the coal ROM, and in longwalls 1.6÷2.0 m high - up to 15,000 Mg/d.

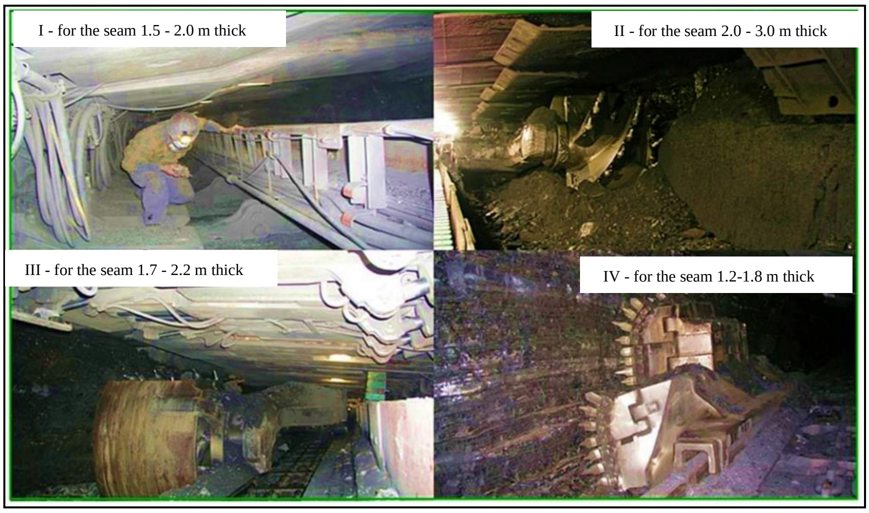

Fig. 3.11. Coal face systems in ‘Bogdanka’ mine (*Dyczko et al 2011*)

As it has already been emphasised, in low shearer-equipped longwalls there is - both natural and forced by geometrical dimensions of longwall equipment - a trend to increase the longwall height, which is most frequently realised by ripping the mined seam floor and roof. This results in pollution, and at the same time it causes a deterioration of the ROM quality. Having in mind a rational and effective management of coal resources deposited in the seams of small thickness, in 2010 the LW Bogdanka SA started mining with the use of plough technique, based on the solutions proved in the German mining sector (Dyczko et al 2011).

An effective implementation of the plough technique in the LW Bogdanka SA was considered to be the project of strategic importance for the Bogdanka mine development and it has been consistently carried out since 2010, when the first plough (test) longwall was started, equipped with a new, fully automated plough set Bogdanka-1.

An implementation of the plough technique was to ensure a simultaneous accomplishment of two objectives:

- an economic mining of the seams thinner than 1.6 m, which allowed to increase the base of coal commercial resources in the seams 1.2 m to 1.6 m thick,
- an improvement of the ROM quality by reducing the amount of ripped dirt from the roof and floor, due to ‘clean’ mining.

In 2008 the plough technique implementation in the LW Bogdanka SA was preceded by a preparation by the MEERI PAS of the report: ‘Analysis of plough technique situation worldwide and of possibility to implement it in the LW Bogdanka SA, in which it was shown that:

- in the areas covered by the licence granted to the mine, thin seams 1.2÷1.5 m in thickness, and even thinner (0.93÷1.3 m thick), constitute a large amount, and their mining will cause an increase in the resource base of commercial resources, profitable to mine, extending their period of mining and the mine life,
- profitable mining is possible with the use of state-of-the-art solutions in the plough technique, which was abandoned in Poland at the end of the past century,
- it is necessary to consider incurring additional expenditures related to a development of the mine infrastructure (an increase in the roadway cross-sections, a change of workings supports),
- an efficient mining is possible, which was confirmed by the carried out economic analyses,
- an implementation of plough technique will require learning new skills by the mine staff, and as a result an increase in the staff training expenditure (as of 2008 situation).

Numerous scientific papers, both in the country and worldwide, show that it is advisable to start mining 1 m thick deposits with use of plough sets. A high productivity, lower mining costs, and a limited amount of dirt in the mineral decide about that. The rock ripping and the roof rock falls are the main factors resulting in the ROM quality deterioration. These effects have a negative impact on the cost of all the processes related to the carried out operations of the mining plant. The originating production costs are primarily related to:

- a significant increase in the energy utilised for cutting,
- a maintenance and wear of cutting equipment,
- a shortened service life of belt conveyors,
- an energy consumption, related to waste transport,
- a higher cost of preparation,
- a mining waste dumping,
- an environment reclamation.

As it has already been mentioned, fighting the pollution (rock ripping) to cut the production costs may be carried out by the plough technique application. In the Author’s opinion it is necessary to take any possible steps, which will improve the ROM quality and reduce the negative environmental impact. A one-percent reduction in the ROM pollution with stone resulted frequently in a two-digit improvement in the production profitability, which was shown using an example of American mines (Kryj et al 2011).

For quite a long time the LW Bogdanka SA mine has been contending with a high coal pollution, focusing on cutting the production costs and improving the mining efficiency.

An improvement in the ROM pollution degree is confirmed by the comparison of production results of two longwalls in the seam 385. The first longwall, 7/V/385, mined with use of a shearer system obtained 68% of coal in the ROM, hence 32% was the dirt. The second longwall, 1/VI/385, which was a test one for the new plough system, achieved a yield of 75% coal in the ROM and 25% of rock.

The hitherto experience of the Bogdanka mine in the plough technique implementation covers the following scope:

– a development of assumptions and a purchase of two plough sets in the years 2008-2010,
– a performance of panel development, a purchase of the first plough set (Bogdanka-1), equipping and mining the first plough longwall 1/VI/385 in the seam 385/2 in Nadrybie in 2010,
– a performance of the panel development and moving the first plough set to the longwall 7/VII/385 in the seam 385/2 in Stefanów and its operation in the period: October 2011 - March 2013,
– a purchase of the second plough set Bogdanka-2 for the longwall 2/VI/385 (longwall started mining in November 2012),
– a performance of a development and starting the third plough longwall in the panel VI (longwall 3/VI/385) in November 2014 (equipped with the Bogdanka-3 plough set),
– a purchase of the Bogdanka-4 plough set and its assembly in the longwall 1/I/385 in the Bogdanka panel.

The first plough set, purchased in 2009, enabled mining of longwall 1/VI/385 in seam 385/2 in 2010, achieving the planned output of approx. 10,000 Mg/d in the case of the seam approx. 1.55 m thick. Previously a full panel length of the longwall 1/VI/385, equal to 1,750 m, was achieved in the March-November 2010 period. The parameters obtained by this longwall between April and October were as follows:

– the average daily advance - 10.0 m,
– the average daily gross output - 8,200 Mg,
– the average monthly advance - 247.4 m,
– the average daily time of plough operation - 5 h 23 min.,
– the total longwall output - 1,382,000 Mg.

The next serious challenge in the field of the plough technique implementation at the LW Bogdanka SA consisted in mining the longwall 7/VII/385, 305 m long and with the panel approx. 5 km long in the seam of 1.25÷1.60 m thickness, which was equipped with the set from the longwall 1/VI/385. This longwall was started in October 2011 in the Stefanów panel and it was conditioned by commissioning of a new skip in the shaft 2.1.

The first plough set Bogdanka-1, moved from the longwall 1/VI/385 to the longwall 7/VII/385, obtained a record daily output of approx. 25,000 Mg/d, at the longwall advance of 27 m/d. The longwall 7/VII/385 operated in the Stefanów panel obtained an average daily output of 11,500 Mg/d of the coal gross ROM, at an average advance of 12 m/d. At the longwall 7/VII/385 new solutions were introduced in the field of self-loading of the coal thrown by the plough to the headgate and the tailgate and in supporting and driving roadway stable-holes. Some changes were also introduced in the technology of its operation. These changes are the subject of a detailed assessment and will be implemented successively.

The next longwalls, equipped both with the Bogdanka-1 set and the Bogdanka-2 set, confirmed great possibilities in the field of obtained productivity, available to plough sets. The next two plough sets were purchased in 2014, where the fourth one - Bogdanka-4 - was a result of an unsuccessful attempt to purchase a shearer set for low seams. The production results for the longwalls mined with plough sets and the obtained productivity are presented in Table 3.6.

Output from longwalls equipped with plough sets
(*own study acc. to LW Bogdanka figures*)

Table 3.6.

Plough set Bogdanka-1					
No	**Longwall**	**Period of longwall operation**	**Panel length**	**Total gross output**	**Daily gross output**
			[m]	**[t]**	**[t/d]**
1	**1/VI/385**	Mar-Nov 2010	1,750.0	1,381,938	7,942
2	**7/VII/385**	Oct 2011-Mar 2013	5,022.0	4,837,385	11,329
3	**1/VIII/385**	Jul 2013-Jan 2015	5,015.0	4,753,877	10,201
Total			**11,787.0**	**10,973,200**	**10,284**
Plough set Bogdanka-2					
1	**2/VI/385**	Oct 2012-Jul 2013	2,310.0	2,059,178	9,713
2	**6/VII/385**	Nov 2013- ctd.	3,448.0	3,370,436	9,336
Total			**5,758.0**	**5,429,613**	**9,476**
Plough set Bogdanka-3					
1	**3/VI/385**	Nov 2014- ctd.	410.0	361,023	6,447
Total			**410.0**	**361,023**	**6,447**

The stock of four plough sets secures the mine needs in the field of mining seams 1.2÷1.6 m thick for at least a few next years.

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The mine intends to maintain and develop both techniques of mining longwalls, i.e. shearer face systems and their plough equivalents. The target mining model assumes a performance of the total output obtained from 4 simultaneously operating longwalls, having at least six machinery sets available, including e.g. 3 plough sets and 3 shearer ones, or 4 plough sets and 2 shearer ones.

Summarising, it is necessary to emphasise that the scope of individual techniques applicability is affected by many factors. In general, they may be divided into geological and technical factors (Table 3.7).

Most important factors affecting the scope of individual techniques application (*own study*)

Table 3.7.

<table>
<tr><th colspan="3">Factor</th><th>Plough</th><th>Shearer</th></tr>
<tr><td rowspan="6">Geological</td><td colspan="2">Seam thickness</td><td>from 0.6 m to 2.3 m</td><td>from 1.5 m to 6.0 m</td></tr>
<tr><td colspan="2">Coal hardness</td><td colspan="2">comparable properties</td></tr>
<tr><td rowspan="2">Inclination</td><td>longitudinal</td><td>up to 45 deg</td><td>up to 20 deg</td></tr>
<tr><td>transverse</td><td>gradient 45 deg, dip 20 deg</td><td>gradient up to 20 deg, dip 20 deg</td></tr>
<tr><td colspan="2">Passing of faults</td><td>average</td><td>good</td></tr>
<tr><td colspan="2">Seam corrugation</td><td colspan="2">a plough passes in an easier way due to a smaller length</td></tr>
<tr><td rowspan="5">Technical</td><td colspan="2">Roof conditions</td><td>a smaller web depth facilitates a roof control</td><td>frequent downfalls</td></tr>
<tr><td colspan="2">Floor conditions</td><td colspan="2">comparable properties (bases of powered roof support units adapted to difficult conditions)</td></tr>
<tr><td colspan="2">Granulation of run-of-mine</td><td>a bigger amount of thick coal sizing</td><td>strongly disintegrated run-of-mine</td></tr>
<tr><td colspan="2">Overall dimensions of workings</td><td>requires wider workings</td><td>most often the return drive situated in the longwall face</td></tr>
<tr><td colspan="2">Automation</td><td>partly automated operation</td><td>a significantly smaller degree of automation</td></tr>
</table>

Fully automated plough sets allow mining very thin seams, which in the case of shearers is much more difficult. This results mainly from the fact that operators must follow the moving shearer. The 1.5 m longwall height is the limit, at which the work of the staff in the mining face is difficult (Fig. 3.12).

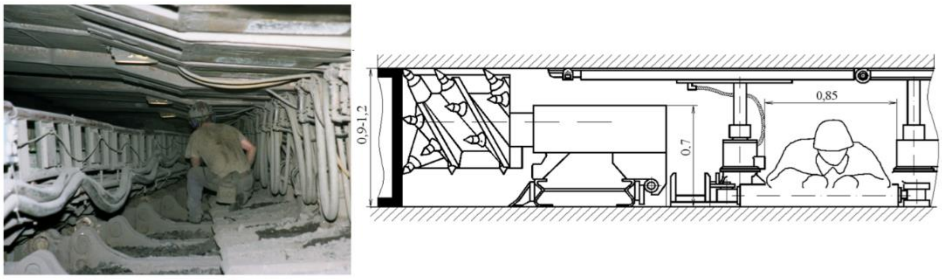

Fig. 3.12. Difficult working position in a low shearer longwall
(*Bondarenko et al 2003*)

The hitherto experience in the field of thin hard coal mining in the LW Bogdanka SA allows a flexible strategy of mining activities and a choice of equipment guaranteeing efficient mining operations.

4. Assessment of Dilution Impact on the Economic Efficiency of an Underground Mining Plant Production Process

A realization of economically efficient mining operations requires a control a few key geological and mining variables, related to the deposit, the coal deposition conditions in the seams, or the used mining technology.

The papers by e.g.: (Karbownik 1987); (Lisowski 2001); (Magda et al 2002); (Lorenz et al 2002); (Przybyła, Chmiela 2007); (Rajwa 2007); (Dyczko, Kopacz 2008a, 2008b); (Lubosik 2009); (Sobczyk 2009); (Brzychczy 2012); (Wodarski, Bijańska 2014); (Grudziński 2009, 2012); (Turek 2013); and (Kopacz 2015a, 2015b) show in various ways the interrelationships between geological and mining parameters, as well as, their impact on the mine model, mining technology, process costs, quality of commercial products as well as economic effectiveness of mines. Those papers emphasise in particular the relationship between the seam thickness and the mining technology, the assessment of technical possibilities of longwall mining, the impact of mineral (coal) yield on the advance of faces, and finally on the costs in the places of their origination. A selection of technology under the conditions of deteriorating seam thicknesses enables to support possibilities of gainful (economically profitable) mining activities. So if there is a relationship between the thickness of mined seams and the used mining systems or the work organisation system and process costs, thereby there should be an economic justification for an introduction of changes that can result in mining smaller amounts of rock in underground workings.

In the Author's opinion such a relationship exists and an attempt to improve the quality of the mining process ultimately consisting in mining less dirt should improve the economic efficiency of mining operations. Obviously, the final effect of the scale of economic results should be related to the rationality of their achievement under specific deposit conditions and the applied technology.

4.1. Assessment of geological-mining parameters relationships in relation to the mining faces costs

Within a preliminary assessment of the economic efficiency of the longwall mining process, the basic information on the selected longwalls mined with use of the shearer and plough technology in the LW Bogdanka SA was compared. Against this background the presence (or lack) of basic relationships between geological features of seams and longwalls characteristics as well as the costs aggregated on them were shown. These relationships were determined based on the correlation analysis. In particular, the relationships between the following properties were included in the scope of assessments:

- the thickness, net coal yield and output,
- the advance and average daily net and gross output,
- the costs of mining the faces and the total net and gross output from longwalls,
- the costs of mining the faces and the net yield of coal.

Table 4.1 specifies basic technical data characterising shearer-equipped longwalls in the period from starting the mining operations until liquidation, while Table 4.2 presents basic technical data related to plough-equipped longwalls in terms of the average monthly figures in the years 2012÷2019. Figure 4.1 presents the results of correlation analysis of the above variables for the shearer-equipped longwalls in the years 2011÷2019, while Fig. 4.2 presents the matrix of mutual correlations for the range of variables presented in Table 4.2 in the case of plough-equipped longwalls.

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Specification of shearer-equipped longwall parameters in the years 2011-2019 (own study)

Table 4.1.

Specification	Period of face mining	Panel length	Length	Average seam thickness in coal	Average height of mined face	% of ripping	Average daily progress	Average daily net output	Average daily gross output	Total coal output	Total ROM output	Average net coal yield
13/II (p.382)	12	2 564	310	2.25	2.53	12%	8.84	9 489.00	12 155.00	2 751 872.00	3 525 037.00	78.10%
3/II (p.382)	8.2	2 160	289	2.4	2.92	22%	10.3	10 975.20	15 938.00	2 304 846.00	3 347 046.00	68.90%
6/IV/385	7.2	1 363	297	1.78	2.17	22%	7.7	6 321.00	9 284.00	1 112 440.00	1 634 006.00	68.10%
9/IV/385	4.0	1 077	296	1.9	2.28	20%	10.6	8 972.40	12 743.00	915 163.00	1 299 758.00	70.40%
4/IV/385	16.0	3 096	294.5	1.8	2.16	20%	7.9	6 372.00	9 691.00	2 510 499.00	3 818 404.00	65.70%
7/IV/385	6	1 110	296	1.7	2.14	27%	7.2	6 522.24	9 467.88	887 024.70	1 287 631.90	68.89%
8/IV/385	6	1 070	296	1.65	2.09	29%	5.8	4 766.39	7 895.97	800 753.32	1 326 522.69	60.36%
2/I/385	11	1 640	313	1.4	2.01	39%	5.9	4 444.33	8 274.06	1 177 747.90	2 192 625.10	53.71%
3/I/385	10	1 601	313	1.43	1.97	35%	6.0	4 488.78	8 356.80	1 151 489.02	1 995 964.68	57.94%
8N	7	1 053	322	3.01	3.35	34%	6.0	7 626.15	12 870.99	1 441 342.15	2 432 617.77	59.25%
4/IV/389	9	2 240	296	2.07	2.52	18%	8.8	9 922.14	12 441.74	2 411 080.40	3 023 342.70	79.75%
3/IV/389	9	2 411	296	2.07	2.49	19%	9.0	9 822.92	12 317.32	2 551 587.47	3 401 372.27	75.10%
1/V/391	12	2450	310.5	2.42	2.79	18%	8.7	11 098.67	14 022.19	2 974 443.10	3 757 948.14	79.15%
2/V/391	11	2 450	310.5	2.37	2.63	17%	8.8	10 550.68	13 292.41	2 911 986.40	3 668 706.10	79.37%
3/V/391	13	2 472	310.5	2.36	2.5	23%	7.1	8 716.96	11 504.42	2 902 746.70	3 830 970.40	75.77%
4/V/391	9	1 827	310.5	2.37	2.68	20%	7.5	9 152.80	11 849.55	2 170 112.27	2 919 918.58	74.42%

Table 4.2.

Specification of plough-equipped longwall parameters in the years 2012-2019 (own study)

Specification	Period of face mining	Panel length	Length	Average seam thickness in coal	Average height of mined face	% of ripping	Average daily progress	Average daily net output	Average daily gross output	Total coal output	Total ROM output	Total ROM output
1/VI/385	8	1 744	250	1.54	1.77	20%	7.5	6 016.31	7 988.08	1 040 820.83	1 381 937.80	75.32%
7/VII/385	18	5 022	300	1.43	1.68	25%	11.5	7 807.36	11 326.37	3 333 741.80	4 836 360.30	68.93%
1/VIII/385	19	5 024	300	1.4	1.7	26%	10.3	6 901.72	10 075.42	3 264 511.60	4 765 674.20	68.50%
2/VIII/385	19	5 021	300	1.36	1.72	30%	9.8	6 443.57	10 139.24	3 335 178.20	4 910 071.80	67.93%
2/VI/385	10	2 310	250	1.54	1.93	28%	10.2	6 392.90	9 592.00	1 374 473.00	2 062 275.80	66.65%
6/VII/385	20	4 850	300	1.47	1.78	26%	10.2	6 937.23	10 096.19	3 315 996.30	4 825 979.20	68.71%
5/VII/385	16	4 685	300	1.58	1.9	26%	11.1	7 939.58	11 683.62	3 335 178.20	4 910 071.80	67.93%
3/VIII/385	15	3 460	300	1.39	1.73	30%	8.9	5 936.90	9 364.57	2 293 709.51	3 621 062.78	63.34%
3/VI/385	11	2 295	250	1.48	1.97	34%	10.5	6 051.96	9 552.95	1 270 912.34	2 006 120.52	63.35%
4/VI/385	9	2 320	220	1.44	1.77	31%	10.6	5 033.71	8 011.47	1 082 248.70	1 722 465.10	62.83%
5/VI/385	8	1 844	300	1.32	1.73	33%	8.5	6 629.01	10 369.71	1 186 592.98	1 856 178.41	63.93%
6/VI/385	9	1 600	300	1.35	1.68	30%	8.5	6 297.56	9 851.23	1 023 683.70	1 635 488.47	62.59%
1/I/385	14	2 043	318	1.36	1.57	32%	6.4	4 454.80	6 975.00	1 505 716.47	2 357 556.57	63.87%
2/II/385	10	1 640	318	1.35	1.72	30%	7.6	5 143.88	8 087.50	1 075 071.90	1 690 287.70	63.60%
3/II/385	9	1 640	318	1.17	1.55	34%	6.0	4 783.81	7 521.38	925 928.40	1 605 895.03	57.66%

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Correlations (Sheet 1)
Marked correlation coefficients are essential with p < .05000
N = 5 (Lack of data removed by cases)

Variable	Face exploitation time (months)	Panel length [m]	Length [m]	Average seam thickness [m]	Average cutting height [m]	% of ripping [%]	Average daily advance [m]	Average daily coal output [Mg]	Average daily ROM output [Mg]	Total ROM output [Mg]	Total coal output [Mg]	Average net coal yield [Mg]	Costs in total [thousand PLN]	Materials [thousand PLN]	Outside services [thousand PLN]	Energy [thousand PLN]	Repairs [thousand PLN]	Payroll [thousand PLN]	Depreciation [thousand PLN]	Other costs [thousand PLN]
Face mining time [months]	1.00	0.96	0.22	-0.03	-0.12	-0.36	-0.61	-0.32	-0.35	0.82	0.86	-0.05	0.81	0.40	-0.71	0.76	0.93	0.91	0.43	0.89
Panel length [m]	0.96	1.00	0.16	0.22	0.14	-0.38	-0.39	-0.06	-0.08	0.93	0.96	0.03	0.88	0.59	-0.65	0.82	0.95	0.88	0.58	0.84
Length [m]	0.22	0.16	1.00	0.07	-0.18	-0.92	-0.28	-0.03	-0.29	0.28	0.11	0.85	0.50	0.14	-0.25	0.49	0.45	0.57	0.51	0.59
Average seam thickness in coal [m]	-0.03	0.22	0.07	1.00	0.97	-0.33	0.54	0.92	0.86	0.55	0.47	0.53	0.49	0.88	-0.09	0.52	0.25	0.04	0.62	0.11
Average height of mined face [m]	-0.12	0.14	-0.18	0.97	1.00	-0.08	0.60	0.91	0.92	0.44	0.39	0.31	0.33	0.82	-0.03	0.38	0.10	-0.14	0.67	-0.06
% of ripping [%]	-0.36	-0.38	-0.92	-0.33	-0.05	1.00	0.07	-0.23	0.02	-0.54	-0.38	-0.89	-0.69	-0.38	0.21	-0.64	-0.61	-0.69	-0.72	-0.65
Average daily advance [m]	-0.61	-0.39	-0.28	0.54	0.60	0.07	1.00	0.83	0.86	-0.19	-0.22	0.20	-0.30	0.10	0.78	-0.36	-0.48	-0.49	0.07	-0.66
Average net daily output [Mg]	-0.32	-0.06	-0.03	0.92	0.91	-0.23	0.83	1.00	0.96	0.26	0.18	0.49	0.18	0.62	0.32	0.18	-0.06	-0.20	0.59	-0.23
Average gross daily output [Mg]	-0.35	-0.08	-0.29	0.86	0.92	0.02	0.86	0.96	1.00	0.19	0.16	0.24	0.06	0.56	0.36	0.05	-0.17	-0.33	0.44	-0.37
Total coal output [Mg]	0.82	0.93	0.28	0.55	0.44	-0.54	-0.19	0.26	0.19	1.00	0.98	0.31	0.97	0.82	-0.62	0.93	0.93	0.81	0.84	0.81
Total ROM output [Mg]	0.86	0.96	0.11	0.47	0.39	-0.38	-0.22	0.18	0.16	0.98	1.00	0.12	0.91	0.77	-0.63	0.67	0.91	0.78	0.73	0.77
Average net coal yield [Mg]	-0.05	0.03	0.85	0.53	0.31	-0.89	0.20	0.49	0.24	0.31	0.12	1.00	0.47	0.41	-0.03	0.47	0.29	0.32	0.71	0.34
Cost in total [thousand PLN]	0.81	0.88	0.50	0.49	0.33	-0.69	-0.30	0.18	0.06	0.97	0.91	0.47	1.00	0.78	-0.68	0.98	0.96	0.87	0.87	0.91
Materials [thousand PLN]	0.40	0.59	0.14	0.88	0.82	-0.38	0.10	0.62	0.56	0.82	0.77	0.41	0.78	1.00	-0.53	0.83	0.62	0.38	0.91	0.51
Outside services [thousand PLN]	-0.71	-0.65	-0.25	-0.09	-0.03	0.21	0.78	0.32	0.36	-0.62	-0.63	-0.03	-0.68	-0.53	1.00	-0.78	-0.73	-0.57	-0.46	-0.82
Energy [thousand PLN]	0.76	0.82	0.49	0.52	0.38	-0.64	-0.36	0.18	0.05	0.93	0.87	0.47	0.98	0.83	-0.78	1.00	0.93	0.79	0.89	0.90
Repairs [thousand PLN]	0.93	0.95	0.45	0.25	0.10	-0.61	-0.48	-0.06	-0.17	0.93	0.91	0.29	0.96	0.62	-0.73	0.93	1.00	0.95	0.71	0.96
Payroll [thousand PLN]	0.91	0.88	0.57	0.04	-0.14	-0.69	-0.49	-0.20	-0.33	0.81	0.78	0.32	0.87	0.38	-0.57	0.79	0.95	1.00	0.56	0.93
Depreciation [thousand PLN]	0.43	0.58	0.51	0.82	0.67	-0.72	0.07	0.59	0.44	0.84	0.73	0.71	0.87	0.91	-0.46	0.89	0.71	0.56	1.00	0.64
Other costs [thousand PLN]	0.89	0.84	0.59	0.11	-0.06	-0.65	-0.66	-0.23	-0.37	0.81	0.77	0.34	0.91	0.51	-0.82	0.90	0.96	0.93	0.64	1.00

Fig. 4.1. Correlation matrix for variables determining shearer-equipped longwalls (*own study*)

Variable	Correlations (Sheet 1) Market correlation coefficients are essential with p < .05000 N=5 (Lack of data removed by cases)								
	Advance	**Seam height**	**Partings**	**Roof fall**	**Opening height**	**Net output**	**Gross output**	**Coal yield**	**Operational costs**
Advance	1.0	- 0.68	- 0.60	- 0.66	- 0.74	0.91	0.93	0.57	0.82
Seam height	- 0.68	1.0	0.46	0.75	0.96	- 0.77	- 0.77	- 0.79	- 0.89
Partings	- 0.60	0.46	1.0	0.52	0.39	- 0.68	- 0.66	- 0.41	- 0.63
Roof downfall	- 0.66	0.75	0.52	1.0	0.87	- 0.90	- 0.88	- 0.98	- 0.94
Opening height	- 0.74	0.96	0.39	0.87	1.0	- 0.86	- 0.86	- 0.90	- 0.94
Net output	0.91	- 0.77	- 0.68	- 0. 90	- 0.86	1.0	1.0	0.83	0.97
Gross output	0.93	- 0.77	- 0.66	- 0.88	- 0.86	1.0	1.0	0.82	0.96
Coal yield	0.57	- 0.79	- 0.41	- 0.98	- 0.90	0.83	0.82	1.0	0.92
Operational costs	0.82	- 0.89	- 0.63	- 0.94	- 0.94	0.97	0.96	0.92	1.0

Fig. 4.2. Correlation matrix for variables determining plough longwalls (*own study*)

The correlation analysis of the faces mined with shearers showed strong and in numerous cases statistically significant correlation relationships between:

– the period of longwall operation as well as the total coal and ROM output from the longwall, total operating costs, including direct costs of materials and energy consumption, repairs, payroll, and other costs,
– the seam thickness and daily net output,
– the level of ripping and the coal yield,
– the daily advance and average daily net and gross outputs,
– the daily gross output and seam thickness as well as the cutting height,
– the total coal and ROM outputs from the longwall and the total operating costs, including direct costs of materials and energy consumption, repairs, payroll, and other costs.

In the case of longwalls, where the plough technique was applied, strong and in numerous cases statistically significant correlation relationships were obtained between:

– the advance, net and gross outputs and operating costs,
– the seam thickness, roof fall, mining height, net and gross outputs and operating costs,
– the net coal yield, seam height, roof fall, and operating costs.

The performed correlation analysis:

1. Emphasised the existence of relationships between the face advance, seam thickness, daily coal and total output, and to a larger degree - dirt and operating costs:
   – in particular relationships between the costs, which in a definite majority may be classified as variable costs are visible.
2. Showed that in the case of shearer-equipped longwalls there is a medium-strong negative correlation relationship between the advance and operating costs at the face:
   – in the case of plough faces this relationship is stronger, albeit the correlation is positive.
3. Did not confirm the existence of correlation relationships between average net coal yield and the mining costs (the lack of relationship can result indirectly from the estimation error of the average coal yield as a quotient of the net and gross output for the entire longwall):
   – there is a correlation relationship between the average yield and average daily gross outputs in the case of thin longwalls.

Finally the obtained assessment results confirm the legitimacy of further work directed towards showing economic effects, resulting from the rationalisation of mining through a change of targets and task parameters from quantitative to qualitative or a mutual optimisation of both of them to reduce the amount of originating spoil.

It is necessary to emphasise the fact that due to a small number of analysed faces the carried out correlation analysis does not decide about the current relationships between the analysed variables. A numerical assessment of the statistical significance features an error related to the quantity, where it is not possible to determine the direction of this error impact and its ultimate value. Nevertheless, in accordance with the intuition and knowledge in this field, the obtained results properly show the directions and the strength of the relationships existing in reality.

The faces equipped with a shearer as a cutting machine, featuring now a higher economic efficiency, become a model for plough longwalls.

4.2. Methodology for the assessment of ROM pollution impact on the economic efficiency of coal mining operations

An assessment of the ROM pollution impact on the economic efficiency of coal mining operations was carried out taking into account the life cycle of a mine section – a longwall face. The life cycle of a typical longwall face comprises basically the phase of the panel development and access to the longwall, driving the development roadways making the longwall contour, the equipment installation in the face, the operational phase of mining, the equipment dismantling and liquidation (Magda et al 2002); (Kustra, Sierpińska 2013).

Because of a specific nature of the possessed data, which were described in detail further on, the entire analysis was carried out based on averaged data for the longwalls 1/VIII/385, 6/VII/385, and 3/VI/385, generated for a large population of 10,000 variables, covering thereby a full life cycle of one longwall face. Fig. 4.3 presents the assessment process.

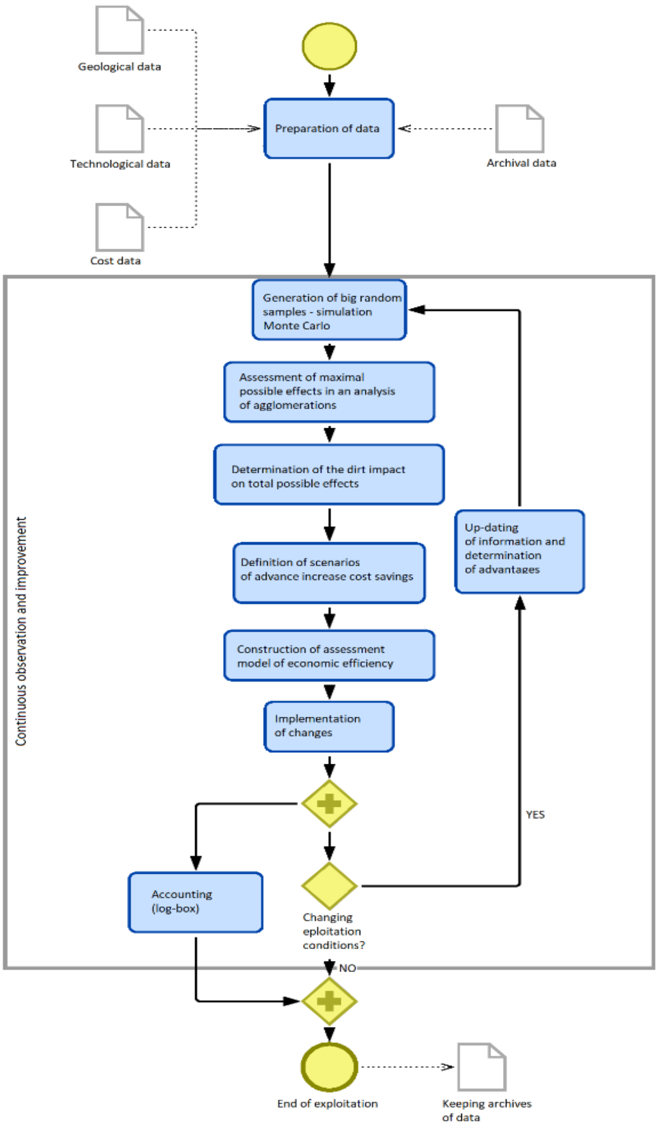

Fig. 4.3. Procedure and analytical assessment process of coal extraction economic efficiency (*own study*)

It comprises a few key components (stages):

- a preparation and statistical processing of data related to plough faces,
- a generation of large random samples,
- a development of economic efficiency assessment model, including:
  - a development of a mathematical model,
  - results and conclusions.

In particular the analysis, the research problem assessment and conclusions assume the following actions:

1. Gathering geological data (qualitative data concerning the ROM from the SYSKON system (ash content, calorific value, moisture content, ROM weight).
2. A preparation of other technological data (advances, failures at faces statistics from the SIK system, and linking them with the SYSKON system data).
3. A preparation of cost data (costs of analysed mining faces, costs of accompanying processes, in which potential savings are sought due to the rationalisation of the mining process and a reduction of rock mining).
4. Implications concerning the model of historical data on the work and results of longwall faces with similar technical parameters, in similar geological conditions, so-called experience use.
5. Data analysis (statistical analysis of data originating from plough faces: descriptive statistics, assessment of graphs - graphical visualisation of data, regression and variance analysis, identification of correlation relations).
6. A selection of mixes distribution and a generation of large random samples (development of a simulation model: selection of statistical or empirical distributions best fitted to the data from the sample, identification of correlation relations between variables, generation of large data sets).
7. A development of the economic efficiency assessment model (development of a discount model in the NPV method), including:

7.1 A development of a mathematical model (linking ROM qualitative characteristics with technical parameters and mining costs, in particular – a parametrisation of the ash content and the ROM weight impact on the advance of longwall faces and a preparation of identification mechanisms for object features (qualitative parameters and costs) with the ash content, ROM weight, and advance):

- cluster analysis (division of data from the population to groups (clusters): grouping k-means by minimisation of the Euclidean distance of the key parameter (longwall advance) and generation of the vector of object features: qualitative parameters of the ROM, cost level, savings level, failure rate level for individual clusters) – a calculation of the maximum effect related to the advance increase.

7.2 A verification of total effects obtained within the cluster analysis based on the carried out questionnaire survey, including:

– an assessment of achievable benefits potential (resulting from the mining process optimisation, i.e. a reduction of the dirt amount).

7.3 A selection of other variables of the discount model (output level, number of longwalls in thin seams, determination of revenue, mining costs, selection of the discount rate).

7.4 A generation of assessment results in the form of scenarios (a preparation of baseline, pessimistic, optimistic, full optimisation effects scenario), including:

- a recalculation of the model for individual scenarios.

Analysing the above items, it is possible to notice that the assessment process focused significantly on linking qualitative features of the mined mineral with the features of the mining system in the conditions of given longwalls as well as mining costs.

An introduction of the Monte Carlo method to the analysis was aimed at an improvement of the modelling quality and a generation of large data sets and the willingness to present the results based on a model plough face in the entire life cycle.

The cluster analysis was carried out with use of the data obtained in the Monte Carlo method, taking the advance as the differentiating parameter. In the next stage, based on the questionnaire survey of mine employees, adjustments of maximum and theoretical effects of optimisation were estimated, resulting from an assessment of the impact of geological-mining, technical, and organisational conditions on the process of mining in thin seams. Because of the willingness to obtain transparency as high as possible, the results of analyses were presented in the form of scenarios. Taking into consideration the scale of obtained results, the following scenarios were ultimately defined:

- baseline scenario,
- optimistic scenario,
- pessimistic scenario,
- scenario of full optimisation effects.

The results of economic efficiency assessment for individual scenarios were referred to the current situation, which was marked in the work as the scenario (current situation). A wide range of achievable results of economic-financial modelling was presented on the scale of results spread between the pessimistic scenario and the full optimisation effect scenario.

4.2.1. Scope of analyses and selection of faces

The assessment of ROM pollution (dirt) impact on the economic efficiency of mining was performed based on the data related to three plough faces operated in thin seams. The scope of analysis comprised all the longwalls, in which geological profiling was carried out. The profiling, referred to, was related to the detailed data describing inter alia the geological structure of mined faces and - what is especially important in the performed analyses - the information about the avoidable amount of dirt due to an improvement in the plough operation. The scope of analysis comprised finally the period from October 2014 to mid-April 2015. In this interval the daily statistics were available for:

- the amount of mined ROM,
- the feed quality (calorific value, ash content),
- the raw ROM moisture content,
- the feed weight,
- the failure rate (failure period, failures division: mechanical, electrical, mining),
- the avoidable amount of dirt as a result of mining process rationalisation,
- the mining costs.

Such a limitation of the analysis horizon results from the process of completing data specifications from various LW Bogdanka SA IT systems and an elimination of defective information during the SYSKON system equipment calibration. The characteristics of analysed longwalls are presented below (Table 4.3).

Characteristics of analysed plough longwalls
(own study based on LW Bogdanka SA figures)

Table 4.3.

Specification	3/VI/385	6/VII/385	1/VIII/385
District name	G-6	G-4	G-1
Set name	Plough set 3	Plough set 2	Plough set 1
Plough	Bucyrus 3	Bucyrus 2	Bucyrus 1
Number of suport units	142 x 1.75 m	172 x 1.75 m	172 x 1.75 m
Total panel length	2300 m	4850 m	5024 m
Face length	250	303.8	303.8
Analysis of data from panel length	410 m – 1500 m	2700 m – 4400 m	4531 m -5024 m
Analysis of data in the period	12/01/2015 - 15/04/2015	13/10/2014 - 15/04/2015	24/11/2014 - 27/01/2015
Average seam thickness	1.45	1.42	1.31
Average roof fall	0.55	0.5	0.4
Average floor ripping	0.33	0.35	0.43

The faces in question were in different life cycle phases during the time, when it was possible to gather the information from technical systems and when the geological profiling of workings was carried out. During the considered period the longwall 3/VI/385, mined in the district G-6, was in the start-up phase. It started operation in January 2015. At that time it featured a smaller advance than at the moment of production stabilisation. In addition, in district G-6 apart from the longwall start-up also the plough set 3 was in the start-up phase. This fact and the stoppages of face mining, occurring in the next months and related to difficulties in coal selling, reduced the statistical reliability of the acquired data. Ultimately a decision was made to limit the data series for this face only to a few dozen representative observations. Figure 4.4 presents the longwall progress in the consecutive days of mining. During the initial days of mining daily advances are much smaller than those recorded after the period of output stabilisation.

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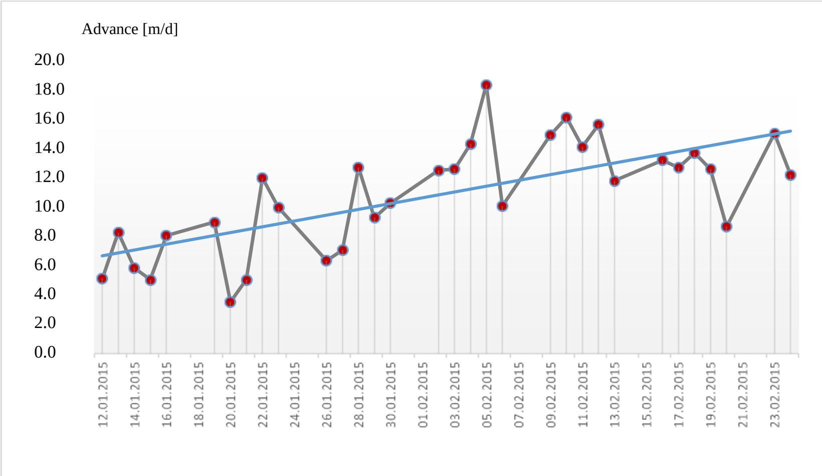

Fig. 4.4. Advance of the longwall 3/VI/385 in the analysed period (*own study*)

For the longwall 6/VII/385 mined in the district G-4 the period of analysis comprises the section between 2,700 m and 4,400 m of the panel length (the total panel length was 4,850 m). The data set comprises almost five calendar months and 1,700 m of the panel length.

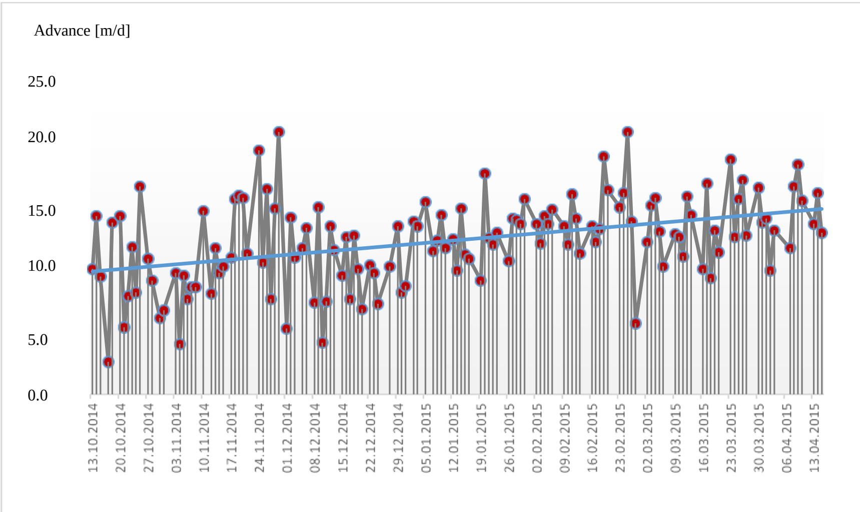

Fig. 4.5. Advance of the longwall 6/VII/385 in the selected period (*own study*)

Fig. 4.5 presents changes in the daily advance of the analysed longwall. It features a relatively stabilised range of fluctuations, which results in the improvement of the statistical modelling quality (distributions choice).

The longwall 1/VIII/385, mined in the district G-1, was finishing its life. The mining operations were completed in January 2015. In the final phase of mining it featured smaller advances and a higher ash content. That was related to the rock mass characteristic at that time and to the technological conditions of mining. Fig. 4.6 presents a graph of advance for this longwall.

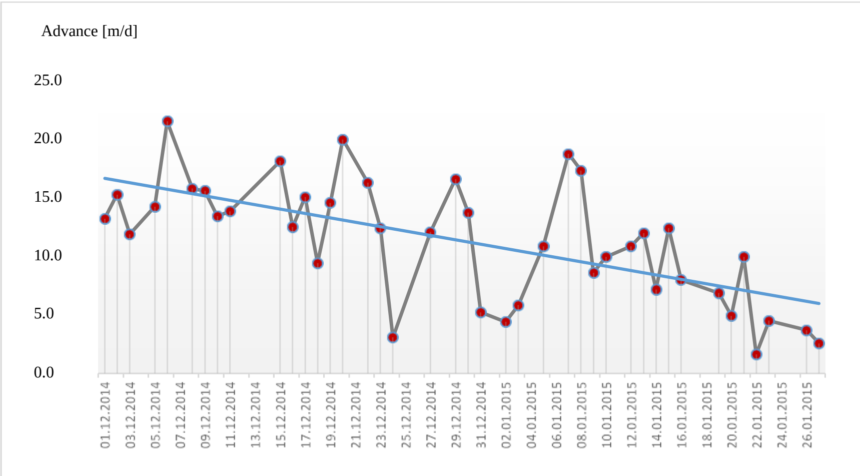

Fig. 4.6. Advance of the longwall 1/VIII/385 in the selected period (*own study*)

At the analysis of the aforementioned advance of longwalls, it is possible to notice that fluctuations in the daily advance are relatively large. Moreover, the advance characteristic of the longwalls 3/VI and 6/VII is relatively similar (in the longwall 6/VII, despite the fact that this is a phase of operations, a minor upward trend of the daily advance and the lack of clear output stabilization are visible). Thereby it makes difficult to separate clearly the start-up phase and the phase of operations at full mining capacity.

Summarising, it is necessary to state that because of the longwalls operation in various phases of life cycle, the performance of analyses based only on one of the faces would be related to:

– a disturbance to the assessment quality (non-standardised number of observations),
– a loss of a significant part of the statistical reliability and a limitation of the scope of conclusions,
– a limitation of achievable effects concerning a reduction of rock amount and economic effects only to one phase of a mining face life cycle.

The above observations became a contribution to an attempt to create a model (typical) plough face, whose assessment will be made in the entire life cycle, combining the start-up, operation, and liquidation phases. A development of the stochastic model was a solution for the above situation (for the needs of the Monte Carlo simulation), based on which large sets of representative data were generated, characterising the life cycle of a model plough face. The related details are described in chapter 4.5.

4.2.1.1. Monte Carlo simulation

The Monte Carlo simulation consists in substituting statistical distributions for uncertain variables in spreadsheets, and then sampling them many times (Jermakow 1976). In this way thousands of random variables executions and the same number of forecast value (e.g. NPV) measurements are obtained. In such models it is also possible to correlate variables. The determined correlation relationships affect generation procedures of individual values. The model becomes more complete and consistent with reality.

The Monte Carlo simulation was used in the work in the following scope:

- an analysis of choice of statistical distributions known under a mathematical form and a statistical quality assessment of theoretical distribution matching the empirical data. This work was carried out for all the three plough faces in the field of the following data:
  - the qualitative data of raw ROM: calorific value, ash content,
  - the daily advance,
  - the ROM (feed) weight,
  - the avoidable amount of dirt (as a result of mining process rationalisation),
- a preparation of a mix of distributions in the entire life cycle of a ‘typical’ plough face:
  - a generation of the distributions mix for:
- calorific value, ash content, daily advance, ROM weight, avoidable amount of dirt,
  - an introduction of correlation relations,
  - a selection of sampling algorithm,
  - a determination of the number of recalculations (iterations),
  - a simulation,
  - reporting and interpretation of results.

4.2.1.2. Cluster analysis

The cluster analysis was performed to aggregate the obtained observations. The data were divided into smaller subsets - clusters (variables grouping the daily advance), Fig. 4.7. As a result of statistical analyses it was found that for various variable features the optimum number of clusters (groups) would range from 8 to 12. Ultimately this number was fixed as 11. This resulted from the variance minimisation inside the clusters, maintaining a division of variables and an effectiveness of the computational model operation as broad as possible.

Label of lines	Average advance	Variance of advance	Label of lines	Average advance	Variance of advance	Label of lines	Average advance	Variance of advance	Label of lines	Average advance	Variance of advance
cluster_5	4,73	1,93	cluster_3	4,44	1,74	cluster_2	4,14	1,55	cluster_5	3,99	1,47
cluster_1	8,30	0,68	cluster_4	7,81	0,62	cluster_3	7,37	0,60	cluster_2	7,02	0,52
cluster_0	10,61	0,31	cluster_0	10,07	0,31	cluster_0	9,60	0,29	cluster_10	9,13	0,27
cluster_2	12,29	0,20	cluster_1	11,73	0,18	cluster_5	11,21	0,17	cluster_1	10,70	0,16
cluster_3	13,75	0,18	cluster_5	13,09	0,14	cluster_4	12,52	0,13	cluster_9	11,95	0,11
cluster_7	15,24	0,22	cluster_2	14,37	0,15	cluster_9	13,70	0,12	cluster_0	13,05	0,10
cluster_6	17,06	0,41	cluster_7	15,78	0,21	cluster_1	14,89	0,13	cluster_4	14,13	0,10
cluster_4	19,93	1,89	cluster_6	17,59	0,42	cluster_8	16,25	0,20	cluster_6	15,28	0,13
Sum	**13,34**	**8,52**	cluster_8	20,47	1,80	cluster_7	18,02	0,39	cluster_7	16,62	0,19
			Sum	**13,34**	**8,52**	cluster_6	20,86	1,72	cluster_8	18,37	0,39
						Sum	**13,34**	**8,52**	cluster_3	21,22	1,63
									Sum	**13,34**	**8,52**

Fig. 4.7. Average advance and variance for a variable number of clusters (*own study*)

The cluster analysis was performed in the RapidMiner Studio 5 software. The cluster analysis grouped samples according to a minimisation algorithm for the Euclidean distance between samples. Observations were grouped in the clusters in consecutive iterations. In the final phase the observations, combined in clusters, were replaced with their representative having the coordinates consistent with the gravity centre of those observations. In this way bigger and bigger groups were formed, to which next 'similar' clusters were assigned.

The cluster analysis with use of k-means method was applied in the work in the following scope:

- a performance of preliminary grouping (in 2,500 clusters),
- a performance of cluster analysis for preliminary grouped data related to the daily advance, weight, average calorific value and ash content, and grouping into 11 clusters,
- a generation of descriptive statistics of the above parameters for each cluster separately,
- a calculation of the new ash content in each group, linked with the avoidable amount of dirt,
- linking the adjusted ash content with the advance in a new cluster via a minimisation of the Euclidean distance between the ash content in the sample and the ash content in individual clusters.

4.2.1.3. NPV method

According to the traditional formula the net present value (NPV) is a sum of current (net present) annual cash flows, less the initial capital expenditure. The NPV reflects the project value at a given discount rate and at a number of assumptions, related to cash flows. So the NPV is a measure of the investment value. A general equation, allowing to calculate the NPV, is as follows:

$$NPV = \left[\sum_{t=1}^{n} \frac{CF_t}{(1+d)^t} \right] - I_0$$

where:

CF_t – cash flow in year t [PLN],

I_0 – initial investment [PLN],

d - discount rate [%],

n – total number of periods required for the project implementation.

NPV has numerous advantages as a tool for an economic efficiency assessment. It takes into consideration the changes of money value over time and provides the evaluation of a single project. In general, one can say that the higher NPV, the bigger benefits result from the project implementation.

4.2.2. Data sources

The data for the technical-economic model originate from a few sources (IT systems):

- the monitoring system of the ROM quality from longwalls - SYSKON,
- the system for LW Bogdanka S.A. Management Information (SIK),
- the developed block model, based on the analysed longwalls observations,
- the LW Bogdanka SA financial-accounting system.

The SysKonSystem is used in the LW Bogdanka SA for a continuous measurement of qualitative parameters (ash content, moisture content, calorific value) of hard coal transported by the belt conveyor for the entire width of the coal stream (Fig. 4.8).

Fig. 4.8. SysKon400, the system for a continuous control of coal qualitative parameters in the LW Bogdanka SA (*Dyczko et al 2015*)

The data, used in the performed analysis, originate from the SysKon400 type devices, which ensure a continuous measurement of qualitative parameters and weight of hard coal transported by the belt conveyor. The data, aggregated in one day, are used in the analysis. The moisture content is measured based on the principle of phase shift and microwaves attenuation in a significant volume of the measured stream. The coal weight is determined by means of a technological mass installed in the device. The measurements continuously monitor the quality of the ROM transported by the belt conveyor (Fig. 4.9).

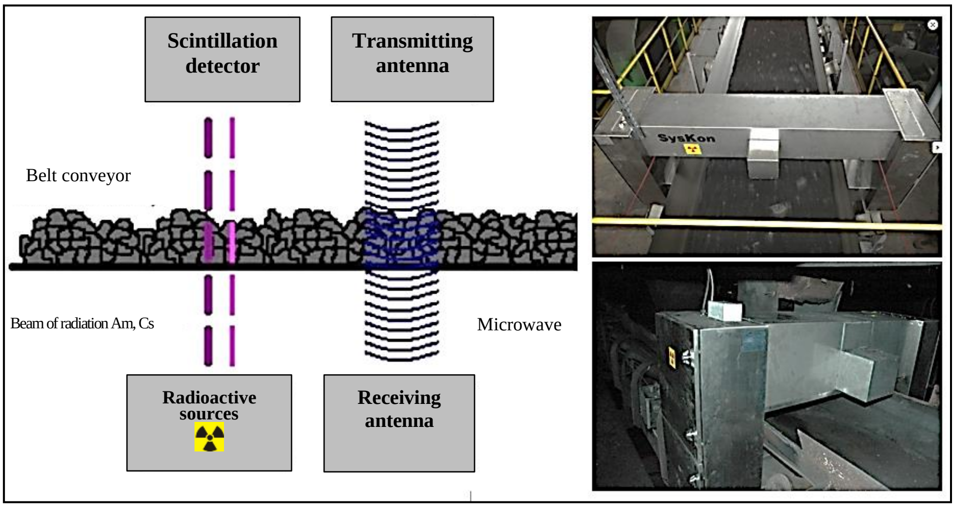

Fig. 4.9. Operational diagram of ROM quality analyser (*Dyczko et al 2015*)

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The measuring devices, in accordance with the manufacturer's specification, measure the ash content with accuracy of 1% to 2%, and the moisture content with accuracy of 0.5% to 1%. The devices are equipped with reporting software, which allows determining quickly the share of individual components, such as coal, dirt bands in the coal seam, roof caving, and floor ripping. Because of minute usefulness of the data and their small daily variability (scatter from 5.5% to 8%) detailed analyses of the moisture content impact on other variables of the mining process (e.g. advance) were given up in the research work. Table 4.4 presents an example of the source data used in the analysis.

The Management Information System, developed by the IT Department operating at Bogdanka, was the second system, from which the data for the analysis originated. The system contains information on key indices related to the operational activities. The information on the advance of a given longwall and on failures was used in the model. Table 4.5 presents an example of input data fragment used in the analysis.

Characteristic of analysed plough longwalls (*own study*)

Table 4.4.

Shift	Ash content [%]	Moisture content [%]	Calorific value [kJ/kg]	ROM weight [Mg]
17/11/2014, Sh. I	46.2	6.3	14,030	2,258
17/11/2014, Sh. II	32.0	6.4	19,265	1,668
18/11/2014, Sh. III	47.8	7.2	13,102	2,338
18/11/2014, Sh. I	47.7	7.3	13,088	2,345
18/11/2014, Sh. II	46.8	7.1	13,546	3,827
19/11/2014, Sh. III	46.7	7.9	13,254	2,236
19/11/2014, Sh. I	37.5	7.7	16,731	2,840
19/11/2014, Sh. II	20.8	7.2	23,157	3,441
20/11/2014, Sh. III	25.7	7.1	21,392	2,173
20/11/2014, Sh. I	44.8	7.4	14,166	3,396
20/11/2014, Sh. II	45.7	7.5	13,777	1,658
21/11/2014, Sh. III	45.8	7.2	13,831	1,850
21/11/2014, Sh. I	33.9	6.6	18,518	2,480
21/11/2014, Sh. II	17.2	6.7	24,706	1,734
22/11/2014, Sh. III	19.2	6.7	23,946	1,241

Characteristic of analysed plough longwalls (own study)

Table 4.5.

Date	Progress	Failures
13/10/2014	9.7	[el.] 10:05 -11:35 No communication at the face - damaged fibre-optic cable [mech.] 12:40 -13:10 Damaged pressure T-connection for support units 159 and 160 supply [mech.] 21:05 -2130 Main drive of plough, no cooling
14/10/2014	13.8	[el.] 20:15 -21:30 No communication, main drive of plough
15/10/2014	9.1	[mech.] 7:15 -8:25 Replacement of plough chain tray tensioning actuator [el.] 21:05 -0:20 damaged cable controlling plough parameters (MD)
17/10/2014	2.5	[mining] 1:00 -03:10
18/10/2014	13.3	[mining] 10:40 -14:20 Face conveyor, pan raised in s:4

The block model, based on observations from profiling of the analysed longwall faces, was the third crucial source of data. A stratigraphic model of the Bogdanka deposit was developed on the grounds of boreholes from the surface (Fig. 4.10) and underground test boreholes.

It was developed due to an interpolation and superposition of coordinates of stratigraphic strata and surfaces findings. It covered 13 documented coal seams, for which qualitative models of the deposit were generated, describing the calorific value, ash content, sulphur content, and coal density.

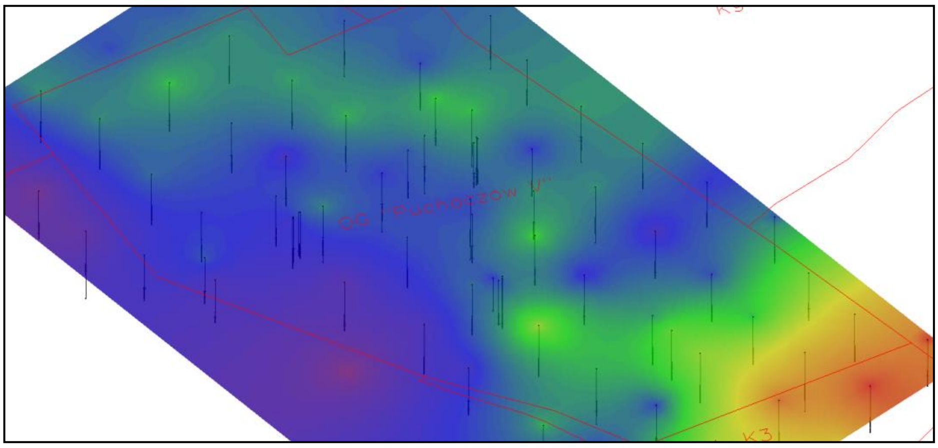

Fig. 4.10. Model of the ground surface and location of surface test boreholes (*Dyczko et al 2015*)

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A model of roof rock fall was developed based on the stratigraphic model prepared in this way. Before taking the decision about the deposit block model use to forecast the roof rock fall under the Bogdanka mine conditions, the Author together with a team of geologists carried out at the LW Bogdanka SA, during 8 months at the turn of 2014 and 2015, a series of underground observations and measurements aimed at studying the origin of roof rockfalls, a development of algorithms of their occurrence forecasting, as well as technical possibilities of a prevention. In total the profiling was made in 68 cases (Fig. 4.11).

Fig. 4.11. Geological measurements and observations at the LW Bogdanka SA (*Dyczko et al 2015*)

During the measurements special attention was paid to observations of plough longwalls and of roadways. The geological profiles of longwalls were made every 25÷40 metres, depending on the conditions existing in the working (sometimes the mining height was so small, that it was not possible to make profiles at equal spacings).

The measurements, carried out in the longwall faces less than 1.5 m high, were made using a laser rangefinder, by means of which the seam thicknesses, lithological separations thickness (indicating the roof fall), and longwall face levelling were measured. The measurements contained also the information on the coal seam floor position against the chain conveyor (Fig. 4.12).

Fig. 4.12. Measuring instruments used during geological observations (*Dyczko et al 2015*)

Detailed notes with geological sketches were made in the case of finding an occurrence of any geological disturbance in the face. Geological cross-sections of the longwall were prepared based on all the available data, which were then analysed together with the mining supervisory personnel (Fig. 4.13 and Fig. 4.14).

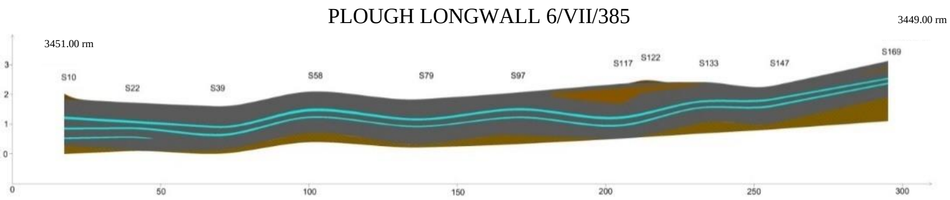

Fig. 4.13. An example of geological cross-section (elevated) in a thin coal seam without any major disturbances (*Dyczko et al 2015*)

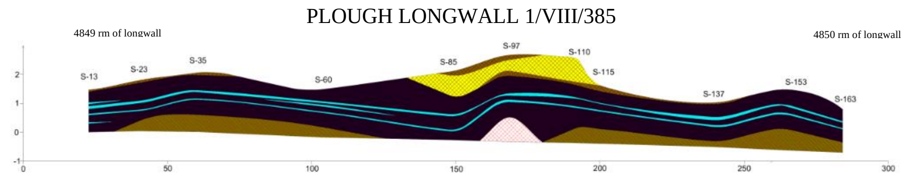

Fig. 4.14. Profile of the plough longwall 1/VIII/385 of 30/12/2014 with a visible roof fall (yellow background, *Dyczko et al 2015*)

The measurements in the headgate and in the tailgate were taken every 20 metres with an observation of the roof, concentrating the profiling activity in the case of any geological disturbances occurrence. The mapping of both longwalls as well as of the headgate and of the tailgate was then used for a creation of

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geological profiles. The information, obtained during observations, was fully used during a development of the block model forecasting roof rock fall, which was made in the MineScape software of the ABB Company (Fig. 4.15).

The created model was aimed at presenting a full lithology of the overburden, of the coal seam and of the dirt bands in the seam. Checking a possibility of presenting and analysing the roof rock fall was the most important element of the developed model.

The presented block model, was based on the previously developed stratigraphic model of the Bogdanka deposit, created by an interpolation and superposition of coordinates of the stratigraphic strata and surfaces findings. The modelling process consisted in downloading the information from the Geological Database, which was exported in the form of lithological data within the deposit. The information on the lithology, used for the model development, originated from profiling the roadways and longwalls described above (Dyczko i in. 2015).

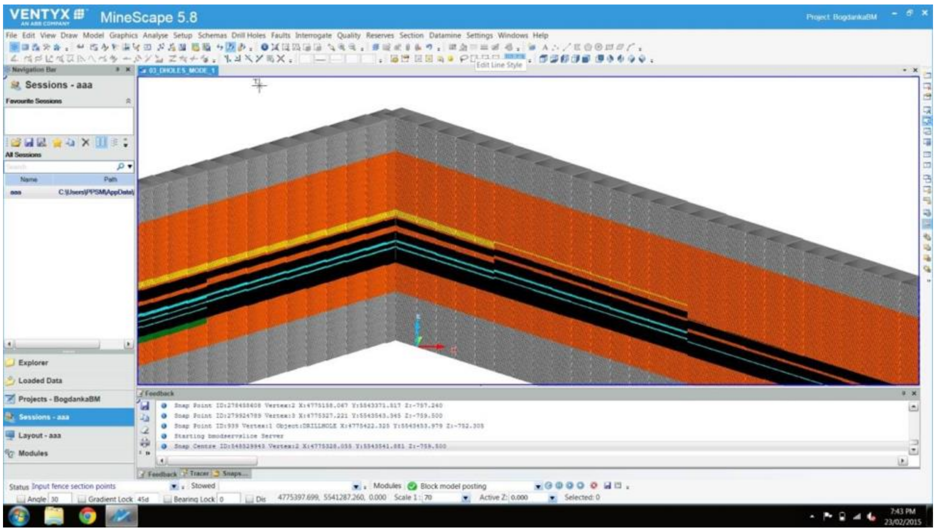

Fig. 4.15. Block model of the deposit in the analysed longwall 6/VII/385 (*Dyczko et al 2015*)

It should be emphasised that the data from underground boreholes were not used in the model development, because they were drilled in the roadway roof, due to which they were outside the analysed seam, that is outside the area of the main interest. The range of the developed model was determined as 4 m above the seam and 3 m below the seam, where the most important zone, from the analysed roof fall point of view, is situated 2 m above the seam. The model was created inside the body, which limits its area. The developed model allowed to present the course of the roof fall and to calculate the weight of the roof fall or the selected lithology at the given density for individual rock types.

The data contained the information about the studied seam thickness, lithology of out-of-seam rocks, and the dirt, which could be avoided in the case of mining in accordance with the geological service guidelines.

The amount of avoidable dirt was determined based on an assessment of possibilities to rationalise the mining operations at minimal roof ripping, observing technological requirements, i.e. a possibility to change the angle between the support units consistent with the technological specification and the mining height enabling a safe operation. Fig. 4.16 presents a fragment of the source data from the profiling model.

The financial-accounting system of the Company was the last source of data, with the costs aggregated to the areas of analysed longwalls and haulage processes, horizontal and vertical transport, as well as a preparation and the costs of other supporting processes, such as ventilation, drainage, and OHS.

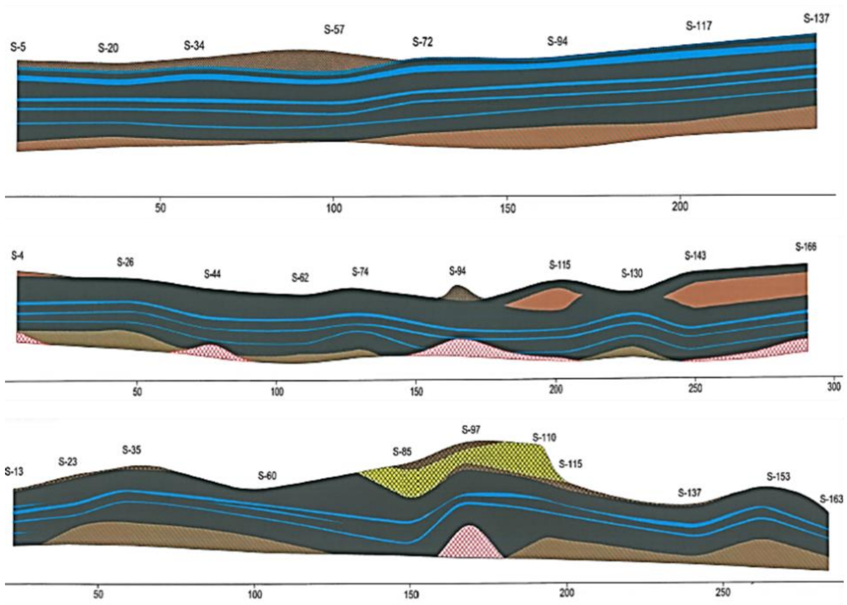

Fig. 4.16. Selected profiles of plough longwalls (*Dyczko et al 2015*)

https://doi.org/10.32056/KOMAG/Monograph2022.1

4.2.3. Statistical assessment of empirical data

The first step of data analysis consisted in an expert assessment – a graphical visualisation of the data obtained from the test samples. It was made in the form of Excel graphs and presented in Figs. 4.17÷4.19.

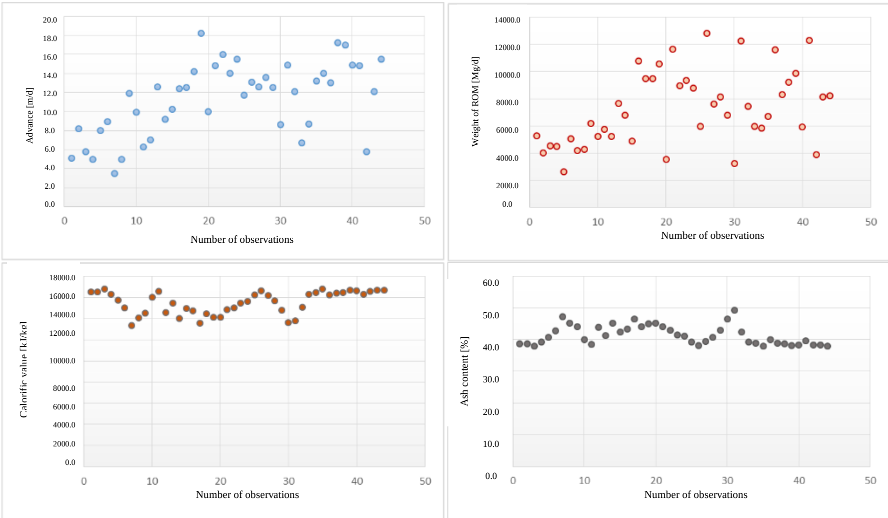

Fig. 4.17. Graphs of advance, ROM output (weight), calorific value, and percentage ash content in the longwall 3/VI (*own study*)

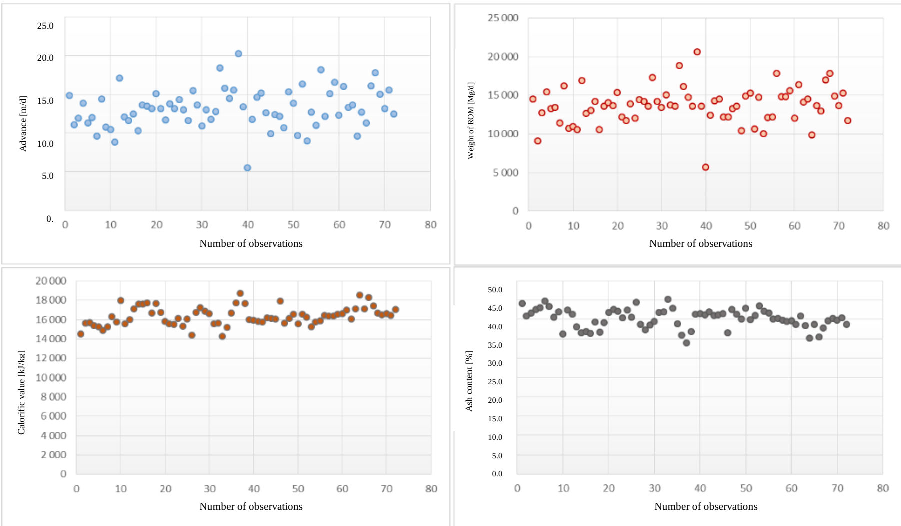

Fig. 4.18. Graphs of advance, ROM output (weight), calorific value, and percentage ash content in the longwall 6/VII (*own study*)

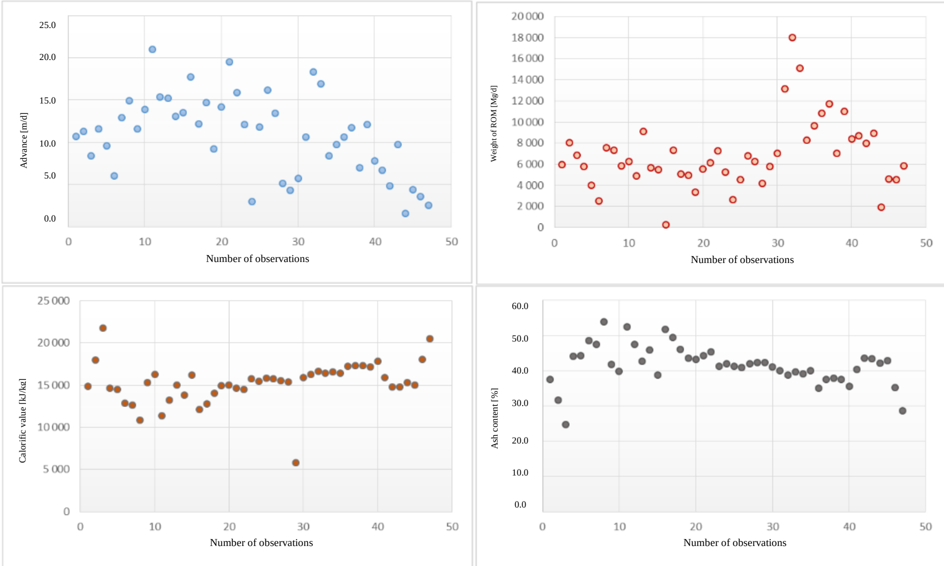

Fig. 4.19. Graphs of advance, ROM output (weight), calorific value, and percentage ash content in the longwall 1/VIII (*own study*)

The variability of information in the samples was the highest in the case of the longwalls 1/VIII and 3/VI. The longwall 6/VII, mined in January 2015 in the district G-4 for more than a year, and whose liquidation was planned for July 2015, featured a lower variability and a relatively higher forecasting capability. Basic descriptive statistics of analysed samples are presented below (Tables 4.6÷4.8).

Descriptive statistics of the longwall 3/VI (G6) (*own study*)

Table 4.6.

Specification	**Advance**	**Weight**	**Calorific value**	**Ash content**	**Failure time**
	[m/d]	[kg]	[kJ/kg]	[%]	[hours]
Average	11.3	7,246.9	15,523.6	41.4	02:11:10
Standard error	0.6	409.1	160.9	0.5	00:28:59
Median	12.3	6,800.9	15,717.9	40.7	01:15:00
Standard deviation	3.8	2,713.7	1,067.4	3.0	02:02:57
Variance of sample	14.50	7,364,432.43	1,139,317.61	9.26	00:10:30
Kurtosis	-0.88	-0.80	-1.13	-0.53	-0.40
Skewness	-0.3	0.4	-0.5	0.7	1.0
Range	14.7	10,155.5	3,471.8	11.4	05:50:00:
Minimum	3.5	2,631.8	13,322.0	37.9	00:25:00
Maximum	18.20	12,787.30	16,793.81	49.27	06:15:00
Meter	44	44	44	44	18

Descriptive statistics of the longwall 6/VII (G4) (*own study*)

Table 4.7.

Specification	Advance	Weight	Calorific value	Ash content	Failure time
	[m/d]	[kg]	[kJ/kg]	[%]	[hours]
Average	13.2	13,678.5	16,362.8	40.2	02:20:38
Standard error	0.3	278.1	112.0	0.3	00:21:54
Median	13.1	13,658.3	16,272.2	40.5	01:00:00
Standard deviation	2.5	2,360.0	950.2	2.4	02:53:52
Variance of sample	6.28	5,569,609.81	902,839.51	5.94	00:21:00
Kurtosis	0.88	1.71	-0.03	-0.20	3.96
Skewness	0.1	-0.1	0.3	-0.4	2.07
Range	14.8	14,978.6	4,486.9	11.4	12:40:00
Minimum	5.5	5,677.9	14,249.7	33.9	00:10:00
Maximum	20.30	20,656.48	18,736.66	45.28	12:50:00
Meter	72	72	72	72	63

Descriptive statistics of the longwall 1/VIII (G1) (*own study*)

Table 4.8.

Specification	Advance	Weight	Calorific value	Ash content	Failure time
	[m/d]	[kg]	[kJ/kg]	[%]	[hours]
Average	10.9	6,884.3	15,274.3	41.7	02:54:03
Standard error	0.7	475.2	360.4	0.8	00:39:12
Median	11.6	6,269.8	15,359.1	42.1	01:30:00
Standard deviation	4.7	3,258.0	2,470.8	5.5	02:59:37
Variance of sample	22.3	10,614,357.8	6,104,788.0	30.8	00:22:24
Kurtosis	-0.59	2.54	4.49	1.58	2.51
Skewness	-0.07	1.14	-0.88	-0.45	1.71
Range	19.4	17,698.0	15,916.2	29.2	10:40:00
Minimum	1.6	298.5	5,811.7	24.8	00:20:00
Maximum	21.0	17,996.5	21,727.9	53.9	11:00:00
Meter	47	47	47	47	21

4.2.3.1. Assessment of data samples forecasting capability - regression, variance, and correlation analysis

The choice of proper mathematical (statistical) model is the key in forecasting future trends (consecutive observations) based on the empirical data. In the study the regression and variance analysis were the basis of forecasting capability assessment for the advance data at selecting a linear, non-linear or another model. A preliminary assessment of the data for individual longwalls (Figs. 4.20÷4.22) shows a small forecasting capability of samples and difficulty in a statistical model choice.

A trial to select a linear model for all the longwalls was conducted in the first stage. It was determined that the advance will be the forecast variable, while the ROM weight from the longwall, calorific value, and percentage ash content will be independent variables.

The next tables present the results of regression and variance analysis of empirical data (Tables 4.9÷4.11). These analyses were carried out by means of the Statistica software.

Regression, variance, and correlation analysis in the set of the longwall 3/VI (G6) data (own study)

Table 4.9.

N=44	Summary of dependent changeable regression: Advance of longwall (3/VI), R= .77633436 R2= .60269504 corrected g. R^2= .57289717 F(3.40)=20.226p < .00000 Estimation error 2.4883					
	BETA	**Stat. Error BETA**	**B**	**Stat. Error BETA**	**t(40)**	**Level p**
W. free			67.77806	66.51361	1.01901	0.314322
Weight – longwall (3/VI)	0.807646	0.107124	0.00113	0.00015	7.53939	0.000000
Calorific value – longwall (3/VI)	-0.581153	0.626321	-0.00207	0.00223	-0.92788	0.359034
Ash content – longwall (3/VI)	-0.628806	0.626951	-0.78519	0.78288	-1.00296	0.321910

Effect	**Variance analysis: DV: Advance of longwall (3/VI)**				
	Sum of squares	**df**	**Average of squares**	**F**	**Level p**
Regression	375.6943	3	125.2314	20.22611	0.000000
Reminders	247.6629	40	6.1916		
Total	623.3573				

	Analysis of Durbin-Watson and series correlation of regression	
	Durbin-Watson	**Series correlation**
Estimation	1.713998	0.096849

Specification: correlations	**Advance**	**Weight**	**Calorific value**	**Ash content**
Advance	1.00	0.81	0.06	-0.04
Weight	0.81	1.00	0.00	0.02
Calorific value	0.06	0.00	1.00	-0.99
Ash content	-0.04	0.02	-0.99	1.00

Variable	**In the equation there are variables: DV: longwall (3/VI) advance (Analysis of regression)**						
	Beta	**Par. Corr.**	**Sem. Corr.**	**Tolerance**	**R-sq**	**t(40)**	**Level p**
Weight – longwall (3/VI)	0.807646	0.766132	0.751395	0.865553	0.134447	7.53939	0.000000
Calorific value – longwall (3/VI)	-0.581153	-0.145157	-0.092475	0.025320	0.974680	-0.92788	0.359034
Ash content – longwall (3/VI)	-0.628806	-0.156625	-0.099957	0.025270	0.974730	-1.00296	0.321910

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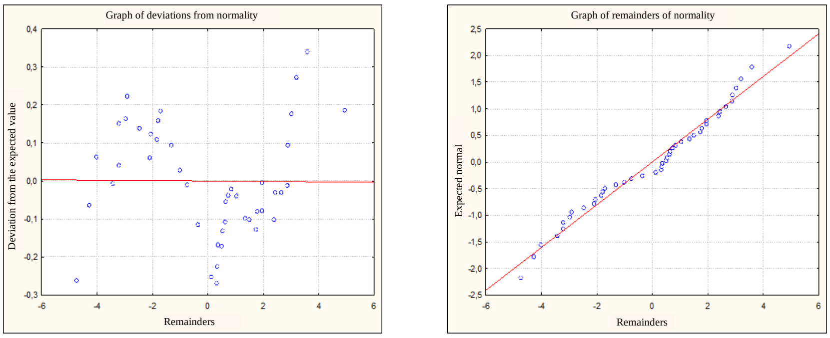

Fig. 4.20. Graphs of deviations from normality and normality of remainders in the set of longwall 3/VI data (own study)

At the analysis of results of the linear regression model for the data from the longwall 3/VI, it is possible to notice that the level of the advance variability explanation, measured by the determination coefficient R^2, is approx. 57%. This is a too low value of the determination coefficient to be able to state that such a model features an appropriate forecasting capability and a proper description of the advance by means of the indicated independent variables (the only statistically significant parameter is the ROM weight from the longwall, the t-Student test level of significance is 0.0000). Also the correlation matrix indicates a strong relationship of the weight with the advance. In this case the value of the Pearson linear correlation coefficient r is 0.81. Also the calorific value and percentage ash content are very highly correlated with each other (-0.99). The other correlation relationships are weak and statistically insignificant. The Durbin-Watson statistics developed on the level lower than 2, which can indicate an existence of the autocorrelation relationship (for delays equal to and higher than 1). The lack of random nature and normality of residual components, illustrated by Fig. 4.21, reduces the forecasting capability of the model, too. Table 4.10 presents the results of the regression, variance, and correlation analysis in the set of the longwall 6/VII data.

At the analysis of results of the linear regression model for the data from the longwall 6/VII, it is possible to notice that the level of the advance variability explanation, measured by the determination coefficient R^2, is nearly 83%. This is a relatively high value of the determination coefficient, at statistical significance positively verified by the Fisher test F(3.68) (default significance 0.05% vs critical significance (p-value) close to 0.000). Also the contribution of the explained advance variability is contained in the regression component (sum of squares 372).

Regression, variance, and correlation analysis in the set of the longwall 6/VII (G4) data (own study)

Table 4.10.

N=72	Summary of dependent changeable regression: Advance of longwall (6/VII), R= .91382329 R2= .83507301 corrected g. R^2= .82779682 F(3.68)=114.77p < 0.0000 Estimation error 1.0398					
	BETA	**Stat. Error BETA**	**B**	**Stat. Error BETA**	**t(68)**	**Level p**
W. free			11.86497	25.64344	0.46269	0.645064
Weight – longwall (6/VII)	0.908563	0.049581	0.00096	0.00005	18.32476	0.000000
Calorific value – longwall (6/VII)	-0.125477	0.303604	-0.00033	0.00080	-0.41329	0.680692
Ash content – longwall (6/VII)	-0.156523	0.303927	-0.16084	0.31232	-0.51500	0.608222

Effect	**Variance analysis: DV: Advance of longwall (6/VII)**				
	Sum of squares	**df**	**Average of squares**	**F**	**Level p**
Regression	372.2420	3	124.0807	114.7679	0.000000
Remainders	73.5178	68	1.0811		
Total	445.7599				

	Analysis of Durbin-Watson and series correlation of regression	
	Durbin-Watson	**Series correlation**
Estimation	1.496539	0.240758

Specification: correlations	**Advance**	**Weight**	**Calorific value**	**Ash content**
Advance	1.00	0.89	0.14	-0.14
Weight	0.89	1.00	0.12	-0.12
Calorific value	0.14	0.12	1.00	-0.98
Ash content	-0.14	-0.12	-0.98	1.00

Variable	**In the equation there are variables: DV: longwall (6/VII) advance (Analysis of regression)**						
	Beta	**Par. Corr.**	**Sem. Corr.**	**Tolerance**	**R-sq**	**t(68)**	**Level p**
Weight – longwall (6/VII)	0.908563	0.911920	0.902464	0.986620	0.013380	18.32476	0.000000
Calorific value – longwall (6/VII)	-0.125477	-0.050056	-0.020354	0.026313	0.973687	-0.41329	0.680692
Ash content – longwall (6/VII)	-0.156523	-0.062332	-0.025363	0.026257	0.973743	-0.51500	0.608222

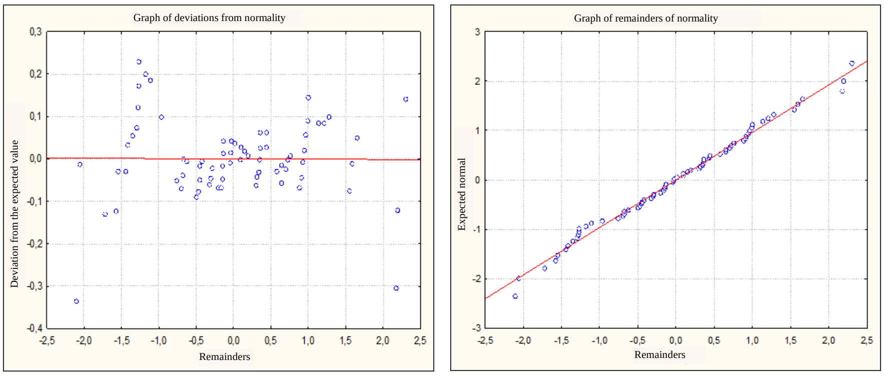

Fig. 4.21. Graphs of deviations from normality and normality of remainders in the set of the longwall 6/VII data (*own study*)

In this model only the ROM weight is a statistically significant parameter (level of t-Student test significance is 0.0000), which also confirms a strong correlation relationship between the weight and the advance (0.89). Also the calorific value and the percentage ash content are very highly correlated (-0.98). The other correlation relationships are weak and statistically insignificant.

The Durbin-Watson statistics, developed on the level lower than 2, can indicate the existence of autocorrelation relationship (for delays equal to and higher than 1). The assessment of random nature and normality of remainders, illustrated by Fig. 4.22, is better than in the case of the longwall 3/VI.

Table 4.11, by means of next specifications and graphs, presents the results of the regression, variance, and correlation analysis in the set of the longwall 1/VIII (G1) data.

Regression, variance, and correlation analysis in the set of longwall 1/VII (G1) data (own study)

Table 4.11.

N=47	Summary of dependent changeable regression: Advance of longwall (I/VIII), R= .58222370 R2= .33898444 corrected g. R^2= .29286708 F(3.43)= 7.3505 p < .00044 Estimation error 3.9737					
	BETA	**Stat. Error BETA**	**B**	**Stat. Error BETA**	**t(43)**	**Level p**
W. free			-24.9163	13.24233	-1.88156	0.066676
Weight – longwall (I/VIII)	0.408431	0.127173	0.0006	0.00018	3.21162	0.002500
Calorific value – longwall (I/VIII)	0.257358	0.214051	0.0005	0.00041	1.20232	0.235818
Ash content – longwall (I/VIII)	0.681374	0.213981	0.5804	0.18228	3.18427	0.002699

Effect	**Variance analysis: DV: Advance of longwall (I/VIII)**				
	Sum of squares	**df**	**Average of squares**	**F**	**Level p**
Regression	348.195	3	116.0650	7.350473	0.000440
Remainders	678.976	43	15.7901		
Total	1027.171				

	Analysis of Durbin-Watson and series correlation of regression	
	Durbin-Watson	**Series correlation**
Estimation	0.973343	0.506906

Specification: correlations	**Advance**	**Weight**	**Calorific value**	**Ash content**
Advance	1.00	0.23	-0.29	0.33
Weight	0.23	1.00	0.34	-0.32
Calorific value	-0.29	0.34	1.00	-0.93
Ash content	0.33	-0.32	-0.93	1.00

Variable	**In the equation there are variables: DV: longwall (I/VIII) advance (Analysis of regression)**						
	Beta	**Par. Corr.**	**Sem. Corr.**	**Tolerance**	**R-sq**	**t(43)**	**Level p**
Weight – longwall (I/VIII)	0.408431	0.439846	0.398195	0.950501	0.049499	3.211616	0.002500
Calorific value – longwall (I/VIII)	0.257358	0.180346	0.149071	0.335512	0.664488	1.202321	0.235818
Ash content – longwall (I/VIII)	0.681374	0.436819	0.394804	0.335732	0.664268	3.184272	0.002699

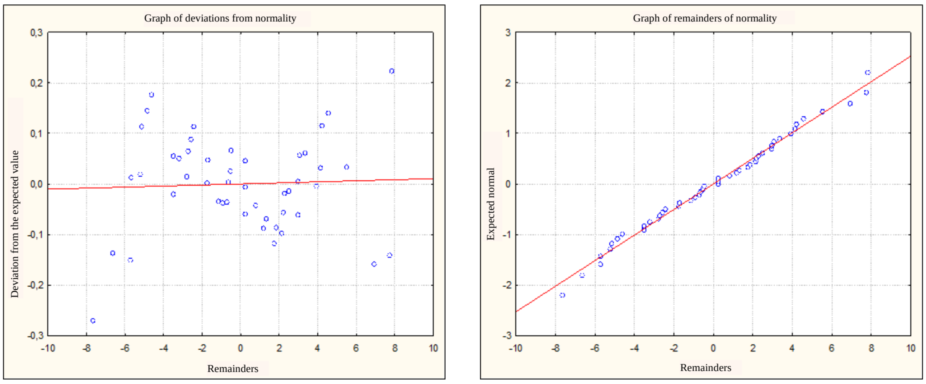

Fig. 4.22. Graphs of deviations from normality and normality of remainders in the set of the longwall 1/VIII data (*own study*)

At the analysis of results of the linear regression model for the longwall 1/VIII, it is possible to notice that the level of the advance variability explanation, measured by the determination coefficient R^2, is only 30%. This is a too low value of the determination coefficient to be able to state that such a model features an appropriate forecasting capability and the proper advance description by means of indicated independent variables. Two statistically significant parameters exist in the model, i.e. the ROM weight and the percentage ash content (level of t-Student significance test is 0.0025 and 0.0027, respectively).

Based on mutual correlation matrices, it is possible to state that the calorific value and the ash content feature the highest mutual correlation (-0.93). The correlation relationship between the advance and the ash content is (0.33), while between the advance and the weight - only 0.23.

The Durbin-Watson statistics, developed on the level lower than 2, can indicate the existence of autocorrelation relationship (for delays equal to and higher than 1). The lack of random nature and normality of residual components, illustrated by Fig. 4.22, reduces the forecasting capability of the model, too. The graph of remainders normality is relatively good (there is a small number of outlier observations), where in the graph of deviations from normality arranged concentrations are visible, which may be related to an autocorrelation process.

Conclusions from the statistical analysis

1. There is a strong dependence of the advance on the ROM weight. It is visible in all three models, also despite a low correlation relationship in the liquidated longwall model (1/VIII). So it seems that in the mine this relationship results from the focus on the task – the quantitative parameter (output maximisation) at the cost of the qualitative parameter, i.e. maximisation of mining cleanness.

In practice this can mean a maximisation of the amount of produced ROM, however, it does not have to imply a significant increase in the amount of produced saleable coal due to a poor feed quality.

2. The model for the longwall 6/VII (G4) was chosen in the best way, in which its statistical significance and a relatively high forecasting capability were shown.
3. The statistical model for the longwall 1/VIII, albeit of the low forecasting capability, includes the percentage ash content in the scope of statistically significant parameters. However, it should be mentioned that the correlation relationship between the advance and the ash content differs and it is relatively low in all the analysed sets.

On this basis it was found that:

1. The mine favours the quantitative rather than the qualitative criterion in its production targets. The maximisation of output tonnage is strongly visible in growing advances and in decreasing calorific values of the ROM mined from these longwalls.
2. The obtained model results justify a possibility of the model development for a model plough longwall face, whose life cycle will consist of the start-up phase (3 weeks), the operational phase of mining - approx. 34 weeks, and the liquidation phase lasting 4 weeks. The period of the model face existence (based on the data from the mine on an average plough longwall duration existence period) was calculated as 41 weeks. However, in practice the length of individual phases differs depending on the working conditions.
3. The operation of the model face construction, on which large data sets were generated, was carried out in the Monte Carlo simulation method.
4. Drawing far-reaching conclusions on the basis of the linear model may be erroneous (in particular for a population comprising a few data sets in the entire life cycle).
5. In the further work a decision was made to replace the conclusions based on linear models with the conclusions based on cluster analysis. It will be the basis for an identification of sets of observations as similar as possible and a determination of descriptive statistics, for them, i.e. technical and economic relationships.

4.2.3.2. Preparation of large data samples for a plough face - Monte Carlo simulation

The statistical analysis of data showed a possibility of the model development for a model plough face, whose life cycle will consist of the start-up phase (3 weeks), the mining phase (34 weeks), and the liquidation phase (4 weeks). The design of a theoretical model for the model face was based on a conception of merging statistical distributions into larger sets- so-called distribution mixes (Kopacz 2008). The process of the model development from a mix comprises a few phases:

- a selection of statistical distributions for the empirical data from the longwalls 3/VI, 6/VII, and 1/VIII.
  - a verification of goodness of fit by Anderson-Darling (A-D), Kolmogorov-Smirnov (K-S), and χ^2 statistical tests,
- a generation of random variables from the selected statistical distributions,
- data merging and repeated choice of the mix distribution,
  - an assessment of the mix distribution selection quality,
- a generation of large random samples from the mix distribution.

Table 4.15 presents 3 best fitted statistical distributions and an assessment of their goodness of fit to the empirical data from the longwalls 3/VI, 6/VII, and 1/VIII samples. As mentioned before, the theoretical distributions were selected to empirical data by means of the Crystal Ball software and the Monte Carlo method.

Ultimately the A-D test (sensitive to outliers and so-called fat tails; the smaller the A-D statistics, the better) was chosen as the criterion deciding about the best fitting distribution, accepting all the limitations and drawbacks of parametric tests, and the distributions were ranked on its basis. The p-value (critical value) statistics higher and significantly higher than 0.05 (default value in the software) were added and placed in Tables 4.12 through 4.14.

https://doi.org/10.32056/KOMAG/Monograph2022.1

Table 4.12.

Selection of distributions for the sample from the longwall 3/VI (own study)

	Distribution	A-D	A-D P-Value	K-S	K-S P-Value	χ^2	χ^2 P-Value	Parameters
Advance	Triangular	0.32	---	0.08	---	6.91	0.14	Minimum=1.55256, Likeliest=13.2, Maximum=19.59818
	Min Extreme	0.38	0.42	0.08	0.70	4.73	0.45	Likeliest=13.09552, Scale=3.25198
	Normal	0.65	0.08	0.14	0.05	9.09	0.11	Mean=11.27727, Std. Dev.=3.80745
Weight	Distribution	A-D	A-D P-Value	K-S	K-S P-Value	χ^2	χ^2 P-Value	Parameters
	Triangular	0.22	---	0.08	---	3.27	0.51	Minimum=1705.2016, Likeliest=5817.88, Maximum=14157.77861
	Gamma	0.27	0.69	0.09	0.61	4.00	0.41	Location=1188.22117, Scale=1293.02962, Shape=4.68565
	Normal	0.48	0.23	0.11	0.19	5.09	0.41	Mean=7246.90568, Std. Dev.=2713.74878
Calorific value	Distribution	A-D	A-D P-Value	K-S	K-S P-Value	χ^2	χ^2 P-Value	Parameters
	Triangular	1.32	---	0.18	---	8.73	0.07	Minimum=12790.08845, Likeliest=16793.80855, Maximum=16832.81778
	Weibull	1.39	0.10	0.18	0.03	15.64	0.00	Location=9184.88529, Scale=6762.424, Shape=6.96627
	Normal	1.60	0.00	0.20	0.00	33.45	0.00	Mean=15523.63391, Std. Dev.=1067.38822
Ash content	Distribution	A-D	A-D P-Value	K-S	K-S P-Value	χ^2	χ^2 P-Value	Parameters
	Beta	0.44	---	0.08	---	5.09	0.17	Minimum=37.67643, Maximum=49.43954, Alpha=0.74246, Beta=1.58148
	Pareto	0.63	---	0.14	---	10.18	0.07	Location=37.82254, Shape=11.28139
	Normal	1.37	0.00	0.16	0.00	29.82	0.00	Mean=41.43453, Std. Dev.=3.0432

Table 4.13.

Selection of distributions for the sample from the longwall 6/VII (own study)

	Distribution	A-D	A-D P-Value	K-S	K-S P-Value	χ^2	χ^2 P-Value	Parameters
Advance	Logistic	0.17	0.93	0.05	0.83	5.22	0.63	Mean=13.12416, Scale=1.38648
	Student's t	0.19	---	0.07	---	2.44	0.88	Midpoint=13.17361, Scale=2.31233, Deg. Freedom=13.48103
	Normal	0.26	0.71	0.07	0.52	4.67	0.70	Mean=13.17361, Std. Dev.=2.50566
Weight	Distribution	A-D	A-D P-Value	K-S	K-S P-Value	χ^2	χ^2 P-Value	Parameters
	Logistic	0.27	0.62	0.07	0.30	8.00	0.33	Mean=13687.76533, Scale=1268.69602
	Student's t	0.32	---	0.08	---	8.56	0.20	Midpoint=13678.5075, Scale=2057.51253, Deg. Freedom=8.33617
	Normal	0.56	0.15	0.08	0.28	11.61	0.11	Mean=13678.5075, Std. Dev.=2360.00208
Calorific value	Distribution	A-D	A-D P-Value	K-S	K-S P-Value	χ^2	χ^2 P-Value	Parameters
	Lognormal	0.27	0.60	0.06	0.73	11.06	0.09	Mean=16363.54626, Std. Dev.=950.80037, Location=7638.44705
	Gamma	0.27	0.59	0.06	0.79	11.06	0.09	Location=10441.46538, Scale=150.43219, Shape=39.3623
	Normal	0.44	0.29	0.07	0.46	12.72	0.08	Mean=16362.8218, Std. Dev.=950.17867
Ash content	Distribution	A-D	A-D P-Value	K-S	K-S P-Value	χ^2	χ^2 P-Value	Parameters
	Weibull	0.22	0.69	0.06	0.81	6.06	0.42	Location=27.35292, Scale=13.87257, Shape=6.16091
	Beta	0.28	---	0.05	---	4.11	0.53	Minimum=29.29248, Maximum=45.75418, Alpha=6.18829, Beta=3.11567
	Normal	0.55	0.16	0.08	0.40	8.28	0.31	Mean=40.24155, Std. Dev.=2.43727

Table 4.14.

Selection of distributions for the sample from the longwall 1/VIII (own study)

	Distribution	A-D	A-D P-Value	K-S	K-S P-Value	χ^2	χ^2 P-Value	Parameters
Advance	Weibull	0.17	0.88	0.07	0.84	3.21	0.52	Location=-5.56711, Scale=18.15363, Shape=3.89111
	BetaPERT	0.18	---	0.07	---	1.85	0.76	Minimum=-2.22252, Likeliest=11.17709, Maximum=22.68438
	Normal	0.23	0.79	0.07	0.78	3.21	0.67	Mean=10.8617, Std. Dev.=4.72544
	Distribution	**A-D**	**A-D P-Value**	**K-S**	**K-S P-Value**	χ^2	χ^2 **P-Value**	**Parameters**
Weight	Lognormal	0.48	0.12	0.09	0.35	3.55	0.47	Mean=6888.11671, Std. Dev.=3180.17649, Location=-6359.12024
	Logistic	0.51	0.14	0.08	0.49	2.53	0.77	Mean=6618.42599, Scale=1706.21958
	Normal	1.05	0.00	0.13	0.07	7.64	0.18	Mean=6884.27532, Std. Dev.=3257.96836
	Distribution	**A-D**	**A-D P-Value**	**K-S**	**K-S P-Value**	χ^2	χ^2 **P-Value**	**Parameters**
Calorific value	Logistic	0.51	0.14	0.12	0.05	6.96	0.22	Mean=15372.08795, Scale=1238.99768
	Student's t	0.64	---	0.14	---	5.94	0.20	Midpoint=15274.28023, Scale=1986.94834, Deg. Freedom=5.6609
	Normal	1.19	0.00	0.16	0.00	14.11	0.02	Mean=15274.28023, Std. Dev.=2470.78692
	Distribution	**A-D**	**A-D P-Value**	**K-S**	**K-S P-Value**	χ^2	χ^2 **P-Value**	**Parameters**
Ash content	Logistic	0.29	0.56	0.09	0.41	3.21	0.67	Mean=41.77158, Scale=2.94241
	Student's t	0.41	---	0.09	---	2.19	0.70	Midpoint=41.65922, Scale=4.9218, Deg. Freedom=9.38148
	Normal	0.63	0.09	0.11	0.23	3.55	0.62	Mean=41.65922, Std. Dev.=5.54865

The performed assessment of goodness of theoretical distributions fit to empirical data, carried out by Anderson-Darling (A-D), Kolmogorov-Smirnov (K-S), and χ^2 statistical tests allows stating that for each longwall face and all the analysed variables, it is possible to select the proper statistical distribution. The highest goodness of fit was obtained for the longwall 6/VII. The p-value statistics are high and sufficient to verify the correctness of the theoretical distribution selection in relation to the empirical data. The lowest selection quality was obtained in the case of the longwall 3/VI, which was in the start-up phase. The values of p-value tests are higher than 0.05, and there is an area of statistically relative certainty in relation to the performed distributions selection. Finally, it is possible to state that for all the data sets and all the variables there are no grounds to reject a zero hypothesis about the consistency between theoretical distributions and their empirical distributions. Theoretical distributions may be used to generate large random samples.

Estimation results for a mix of distributions for large data sets

To ensure an appropriate number of data for a cluster analysis and further research work, large data sets (approx. 2,500 observations) were finally generated in the Monte Carlo method for the following variables:

- the advance,
- the ROM weight,
- the calorific value,
- the ash content,
- the avoidable amount of dirt.

The Latin hypercube technique was used for sampling. It enables uniform sampling in the entire scope of the distribution domain. The number of iterations (calculations) in the simulation was 2500. For the above variables the mixes were created from the following distributions, for variables, as appropriate:

- **the advance**: triangular distribution (1.6; 13.2; 19.6) (longwall 3/VI) (7% contribution); logistic distribution (13.1; 1.39) (longwall 6/VII) (84% contribution); Weibull distribution (5.6; 18.2; 3.9) (longwall 1/VIII) (9% contribution),
- **the ROM weight**: triangular distribution (1,705; 5,818; 14,158) (longwall 3/VI); logistic distribution (13,688; 1,269) (longwall 6/VII); lognormal distribution (6,888; 3,180; -6,359) (longwall 1/VIII) (*),
- **the calorific value**: triangular distribution (12,790; 16,794; 16,833) (longwall 3/VI) (7% contribution); lognormal distribution (16,364; 951; 7,638)

(longwall 6/VII) (84% contribution); logistic distribution (15,372; 1,239) (longwall 1/VIII) (9% contribution) (*),

- **the ash content**: beta distribution (37.7; 49.4; 0.74; 1.58) (longwall 3/VI) (7% contribution); Weibull distribution (27.35; 13.87; 6.16) (longwall 6/VII) (84% contribution); logistic distribution (41.77; 2.94) (longwall 1/VIII) (9% contribution) (*),
- **the avoidable amount of dirt**: maximum extreme values distribution (416.8; 266.6).

explanation: (*) - *Percentage contribution of distribution should be understood as a percentage share of empirical observations generated from the given distribution in the mix.*

Moreover, the correlation relationships of variables (distributions) were introduced to the simulation, observed in empirical samples. In this way, a high degree of generated samples consistency with empirical data was ensured. The level of consistency in this case was verified by the assessment of correlation relationships between individual variables in empirical samples and secondary correlations - in random samples.

The distribution of avoidable dirt was designed based on an expert assessment of possibility of leaving in the plough longwall solid a specified amount of stone, in accordance with geological guidelines. It was found for this distribution that the range of observations can vary between 0 Mg and 1300 Mg per day. Finally, the distribution of maximum extreme values was adopted with the expected value of 571 Mg/d (median was 515 Mg/d) (Fig. 4.23). This distribution was related by a strong correlation relationship with the distribution of percentage ash content (ρ=0.85).

A graphical illustration of the created distribution mixes for analysed variables is presented below. They represent variables for a model plough face in the entire life cycle.

The distribution mixes are not expected to be consistent with a single theoretical model (green line on graphs; in special cases such consistence may occur), hence a verification of the goodness of fit, attached in the tables describing the distribution, is only illustrative information.

The distribution, obtained for the ROM weight, seems to be the most atypical example of the mix model. It features a relatively high number of observations in the left tail of the distribution (small advances and small output for the start-up and liquidation phases) and more than average concentration of information in the area of the mean value.

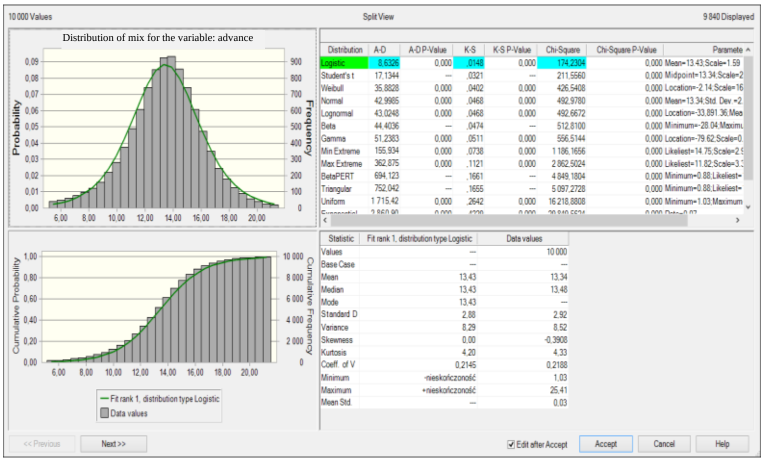

Fig. 4.23. Distribution of mix for the variable: advance (*own study*)

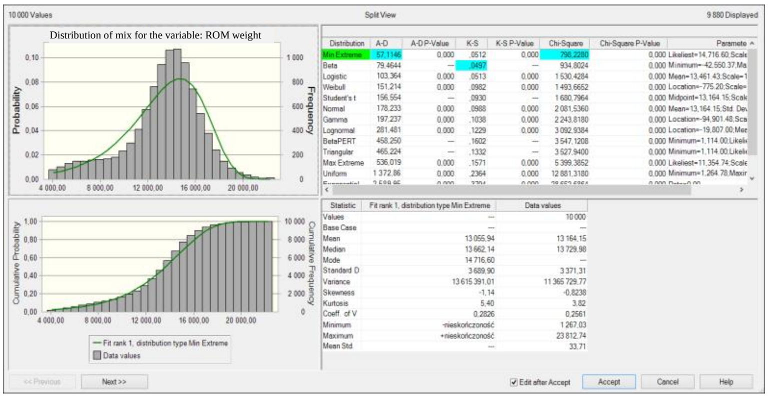

Fig. 4.24. Distribution of mix for the variable: ROM weight (*own study*)

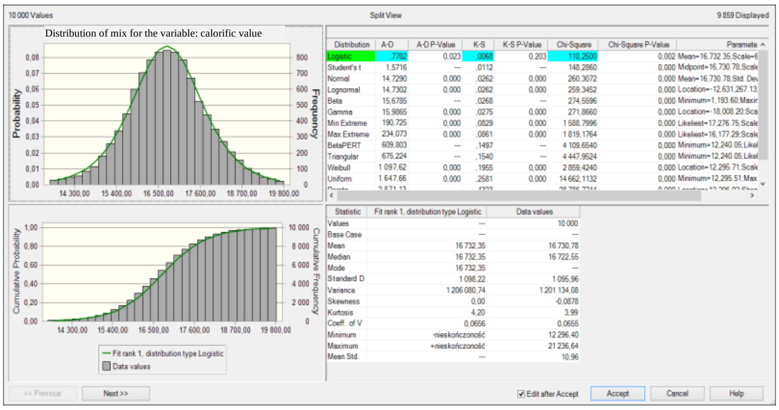

Fig. 4.25. Distribution of mix for the variable: calorific value (*own study*)

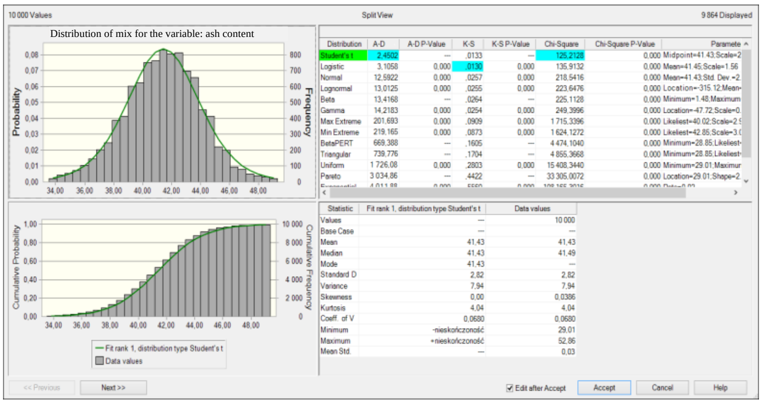

Fig. 4.26. Distribution of mix for the variable: ash content (*own study*)

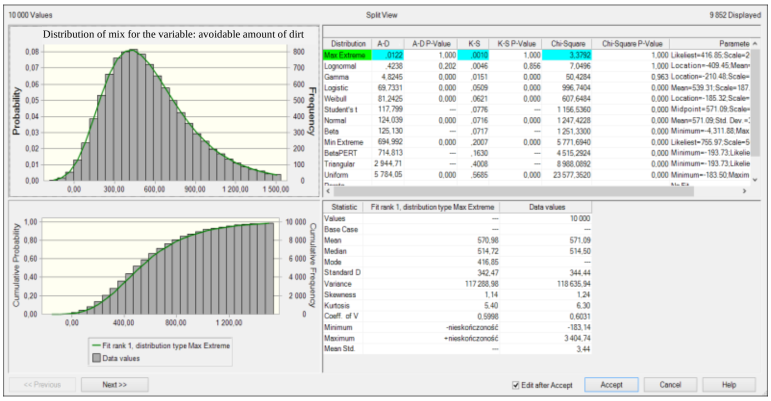

Fig. 4.27. Distribution of mix for the variable: avoidable amount of dirt (*own study*)

Tables 4.15 and 4.16 also present the results of correlation analysis in the developed ‘test’ model, based on empirical samples and models of mixes. Differences between cross-correlations in both tables are small and fully acceptable.

Table of primary correlations (correlations in samples for three analysed plough faces) (own study)

Table 4.15.

Specification	Calorific value	ROM weight	Ash content	Advance
Calorific value	1.00			
ROM weight	0.15	1.00		
Ash content	-0.95	-0.15	1.00	
Advance	0.30	0.70	-0.40	1.00

Table of secondary correlations (estimated correlations based on mix models (own study)

Table 4.16.

Specification	Calorific value	ROM weight	Ash content	Advance
Calorific value	1.00			
ROM weight	0.15	1.00		
Ash content	-0.94	-0.15	1.00	
Advance	0.30	0.69	-0.41	1.00

Finally, by means of the Monte Carlo method the model of the model plough face was reconstructed. It was consistent with the primary specific nature of the data. The large data sets were generated as well. The phase of analytical material assessment and preparation is ended at this stage and the phase of building mathematical model assumptions is started. It is integrated with the model of economic efficiency assessment.

4.3. Model of economic efficiency assessment

The analysed data, both from empirical samples and generated by means of the Monte Carlo method, were grouped in 11 clusters in the RapidMiner Studio 5 software implemented for the model needs.

4.3.1. Mathematical model and cluster analysis

Finally, the group of 11 ‘clusters’ determined within a detailed analysis was estimated as the optimum between the number of generated sets, analysis time, and the obtained results during successive approximations. The advance was the grouping parameter (Fig. 4.28). Each of the created clusters featured a defined vector of analysed parameters.

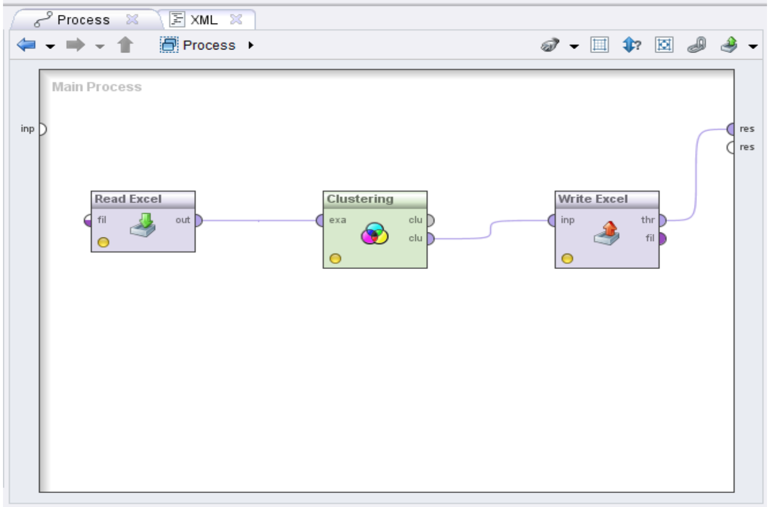

Fig. 4.28. Cluster analysis model in the RapidMiner software (*own study*)

Fig. 4.29 presents a graph of mean ash content vs advance for a given cluster in the mathematical model.

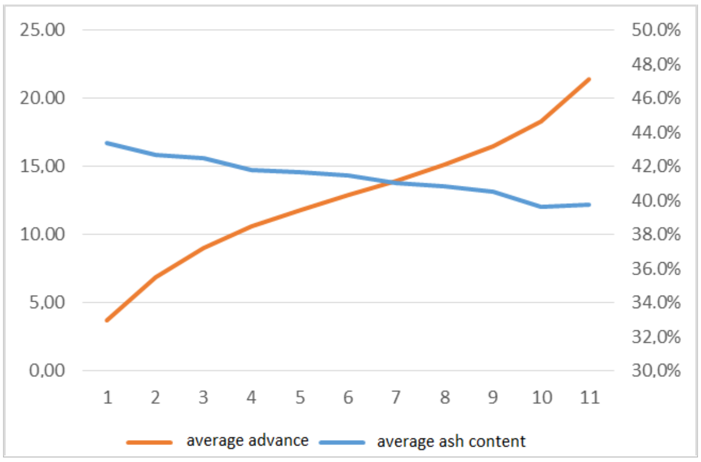

Fig. 4.29. Dependence of average advance on the ash content (*own study*)

The presented graph illustrates a situation, where the dirt is the dominating factor conditioning the advance, which is avoidable as a result of appropriate management of the mining process. In practice, instead of one negative factor, the mining process is conditioned by many factors (geological factors - practically unmanageable, technological and organisational factors - manageable only partly). The model allocates individual observations to clusters of defined advance value, and the other features of the given observations are then inherited to the cluster. In accordance with the above, the reduction of the dirt amount in the longwall results in a reduction of the ash content, followed by a selection of a new cluster with a higher advance, used to calculate maximum (theoretical) effects related to an increased longwall advance. The impact recalculation of the tonnage of spoil ROM left in the longwall on the reduction of the feed ash content level was significant in the analysis. The formula applied in the calculation is based on the principle of mass and energy conservation. An assumption was made that the separable rock has a zero calorific value.

It was found that the initial mass of the ash in the ROM must be equal to the separable dirt mass and the initial mass of the ROM reduced by the separable rock mass (of certain unknown - new ash content), which is presented by the following formula:

$$m_p \times p_p = sk_{du} \times 100\% + (m_p - sk_{ds}) \times p_n$$

where:

- m_p – the total ROM weight [Mg],
- p_p – the original ash content consistent with the ash content meter readings [%],
- sk_{ds} – the avoidable amount of dirt [Mg],
- p_n – the new ash content in the changed structure of the longwall ROM [%].

After transformations the formula for the new ash content estimation was obtained:

$$p_n = \frac{m_p \times p_p - sk_{du} \times 100\%}{m_p - sk_{du}} \ [\%]$$

The determination of a target advance was the next stage of calculations. By means of the Euclidean distance function a cluster featuring the highest similarity of ash content in relation to the original data from the sample was assigned. Based on that an average advance characteristic of a given cluster was assigned. The total effect of efficiency increase was calculated by means of the formula:

$$E = \frac{\sum n_p}{\sum p_p} - 1 \ [\%]$$

where:

- E – the efficiency increase [%],
- p_p – the sample advance [m],
- n_p – the new sample advance [m].

https://doi.org/10.32056/KOMAG/Monograph2022.1

After a performance of the cluster analysis the ordered set of data was presented in Tables 4.17 and 4.18.

Set of calculation data (own study)

Table 4.17.

Advance	Calorific value of feed [MJ/kg]	ROM weight [Mg]	Ash content in feed [%]	Avoidable rock (theor.)	Amount of energy [GJ]	Amount of energy calculated per unit of advance [GJ/m]
15.7	16,842.0	13,670.0	42.0%	582.4	230,228.7	14,682.4
13.0	16,554.0	14,921.0	42.0%	484.1	247,006.8	19,045.7
12.0	17,591.0	13,016.0	39.0%	175.4	228,959.0	19,106.4
4.1	17,883.0	3,731.0	39.0%	256.1	66,722.7	16,374.6
10.7	16,325.0	12,687.0	43.0%	695.4	207,108.9	19,407.7
12.1	17,037.0	16,691.0	39.0%	183.4	284,365.0	23,510.3
10.7	15,965.0	11,414.0	44.0%	642.3	182,217.3	16,987.8
10.8	16,533.0	6,646.0	41.0%	522.4	109,882.2	10,210.4
9.2	15,908.0	12,174.0	44.0%	367.9	193,664.4	21,011.0
11.8	16,321.0	12,866.0	42.0%	564.9	209,993.3	17,835.9
11.7	16,578.0	12,820.0	41.0%	516.5	212,534.6	18,183.9
12.9	18,030.0	13,921.0	39.0%	294.9	250,994.8	19,408.2
10.5	17,597.0	11,972.0	39.0%	265.4	210,667.7	20,100.9
15.6	15,292.0	16,090.0	45.0%	776.3	246,051.7	15,745.8
17.8	16,475.0	17,469.0	42.0%	704.1	287,805.0	16,162.3
13.2	16,904.0	12,683.0	41.0%	607.3	214,387.4	16,253.4
16.5	16,547.0	19,672.0	43.0%	702.4	325,510.8	19,726.1
10.9	16,851.0	12,556.0	40.0%	288.9	211,580.8	19,435.0
12.3	17,831.0	14,385.0	38.0%	433.8	256,488.3	20,820.7
14.1	16,183.0	14,873.0	43.0%	747.5	240,691.2	17,136.8
11.7	17,949.0	3,318.0	40.0%	222.0	59,557.6	5,104.0
12.4	16,368.0	12,330.0	41.0%	480.1	201,812.5	16,337.9
14.3	15,430.0	16,006.0	45.0%	884.6	246,977.5	17,231.8
14.0	16,424.0	10,625.0	42.0%	547.0	174,504.0	12,444.2
11.0	16,200.0	13,263.0	42.0%	974.7	214,864.1	19,474.0
13.3	16,076.0	15,281.0	43.0%	705.0	245,654.4	18,513.6
11.1	16,613.0	14,143.0	0.41	372.7	234948.8	21247.4
14.6	16834.0	15,483.0	0.42	756.3	260642.9	17859.1
11.9	17361.0	14,686.0	0.38	382.1	254963.0	21519.5
14.7	17079.0	13,465.0	0.42	559.2	229956.2	15635.3
12.9	16859.0	14,047.0	0.41	490.9	236821.6	18380.3
8.2	18464.0	8,861.0	0.37	327.9	163612.8	20046.7
18.2	16176.0	20,066.0	0.42	663.7	324585.8	17804.9
10.4	18117.0	14,182.0	0.36	46.3	256932.5	24748.1
15.8	15590.0	16,731.0	0.44	769.3	260833.6	16464.9

Set of calculation data, cont. (own study)

Table 4.18.

New ash content [%]	Advance cluster	Average advance in cluster [m/day]	Shortest distance	New cluster	Average advance in cluster [m/day]	New advance [m/d]	Target cluster
39.0%	9	16.2	0.01	11	21.4	21.4	11
40.0%	6	12.5	0.00	10	18.3	18.3	10
38.0%	6	12.5	0.02	11	21.4	21.4	11
34.0%	1	3.5	0.05	11	21.4	21.4	11
39.0%	4	10.2	0.00	11	21.4	21.4	11
38.0%	6	12.5	0.01	11	21.4	21.4	11
41.0%	4	10.2	0.00	8	15.1	15.1	8
36.0%	4	10.2	0.03	11	21.4	21.4	11
42.0%	3	8.6	0.00	5	11.8	11.8	5
40.0%	5	11.4	0.00	10	18.3	18.3	10
39.0%	5	11.4	0.01	11	21.4	21.4	11
38.0%	6	12.5	0.02	11	21.4	21.4	11
38.0%	4	10.2	0.02	11	21.4	21.4	11
42.0%	9	16.2	0.00	5	11.8	16.2	9
39.0%	10	18.1	0.00	11	21.4	21.4	11
38.0%	7	13.6	0.02	11	21.4	21.4	11
40.0%	9	16.2	0.00	9	16.5	16.5	9
39.0%	5	11.4	0.01	11	21.4	21.4	11
36.0%	6	12.5	0.04	11	21.4	21.4	11
40.0%	7	13.6	0.00	10	18.3	18.3	10
36.0%	5	11.4	0.04	11	21.4	21.4	11
39.0%	6	12.5	0.01	11	21.4	21.4	11
41.0%	8	14.8	0.00	7	14.0	14.8	8
39.0%	7	13.6	0.01	11	21.4	21.4	11
37.0%	5	11.4	0.02	11	21.4	21.4	11
40.0%	7	13.6	0.00	10	18.3	18.3	10
39.0%	5	11.4	0.00	11	21.4	21.4	11
39.0%	8	14.8	0.01	11	21.4	21.4	11
36.0%	5	11.4	0.04	11	21.4	21.4	11
39.0%	8	14.8	0.01	11	21.4	21.4	11
39.0%	6	12.5	0.00	11	21.4	21.4	11
35.0%	3	8.6	0.05	11	21.4	21.4	11
40.0%	10	18.1	0.00	10	18.3	18.3	10
36.0%	4	10.2	0.03	11	21.4	21.4	11
41.0%	9	16.2	0.00	8	15.1	16.2	9

https://doi.org/10.32056/KOMAG/Monograph2022.1

For the next clusters at a given advance the average expected values of the other qualitative and quantitative parameters were estimated. Table 4.19 presents values of these parameters in individual clusters.

Average values of the ROM parameters for each cluster (own study)

Table 4.19.

Cluster	Average ash content [%]	Average advance [m/day]	Average calorific value [kJ/kg]	Average weight [Mg]	Average amount of energy [GJ]
1	43.4%	3.68	15,941.17	5,361	85,316
2	42.7%	6.86	16,156.33	7,658	123,408
3	42.5%	9.03	16,285.65	9,396	152,724
4	41.8%	10.60	16,568.28	10,984	181,774
5	41.7%	11.82	16,625.05	12,229	203,174
6	41.5%	12.91	16,703.36	13,485	225,115
7	41.0%	13.97	16,832.77	14,549	244,902
8	40.9%	15.12	16,974.43	15,400	261,237
9	40.5%	16.48	17,080.94	16,346	279,092
10	39.6%	18.29	17,433.36	17,681	308,308
11	39.8%	21.39	17,327.64	18,985	329,147

Figures 4.30 to 4.34 were prepared to present the whole input sample and variability of the ROM parameters. Fig. 4.30 presents the distribution of consecutive clusters vs the advance. It is visible that the extreme clusters feature the widest spread of data. The closer is the graph centre, the data range in individual groups is smaller and smaller.

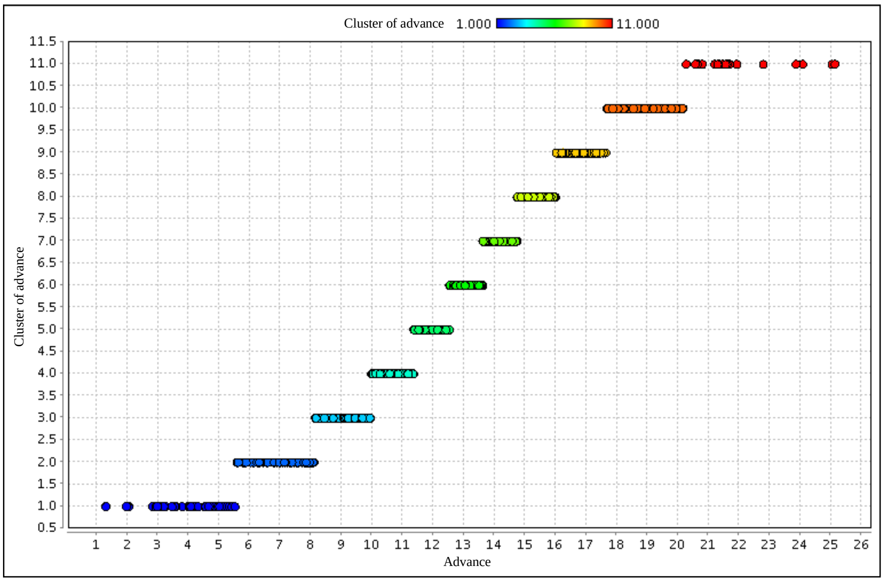

Fig. 4.30. Relationship between clusters and advance. Advance in m/d (*own study*)

Fig. 4.31 presents the relationship between the weight and the advance in individual clusters.

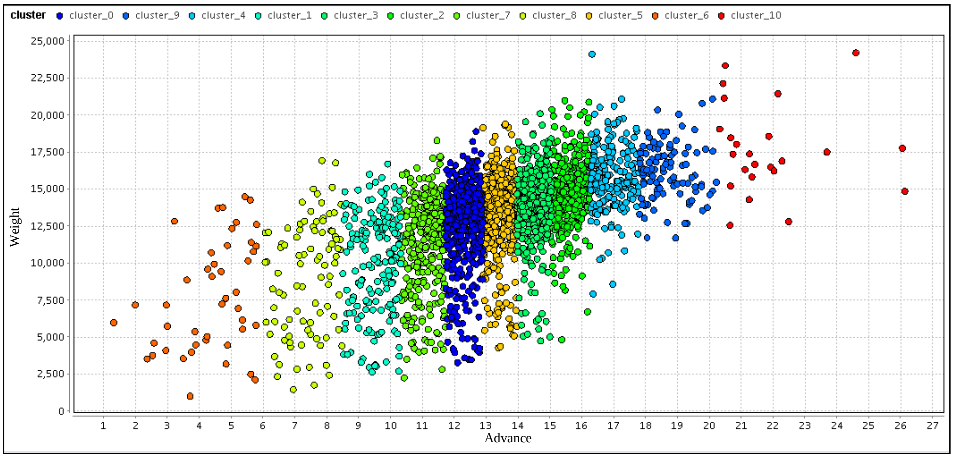

Fig. 4.31. Variability of the weight vs the advance in clusters. Advance in m/d, the ROM weight in the feed [Mg] (*own study*)

Fig. 4.32 illustrates the change of the ash content vs the advance, broken down into clusters. The decreasing ash content is visible with the increasing advance.

https://doi.org/10.32056/KOMAG/Monograph2022.1

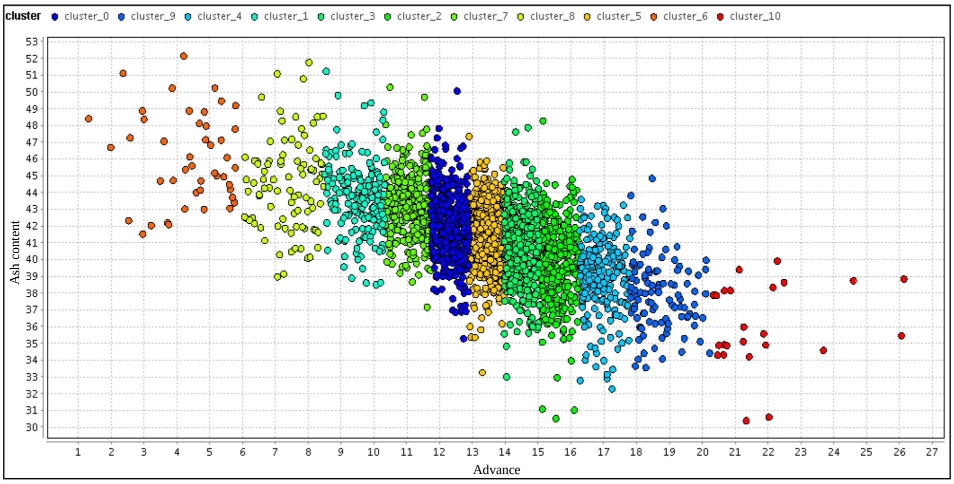

Fig. 4.32. Variability of the ash content vs the advance in clusters. Ash content in %, advance in m/d (*own study*)

Fig. 4.33 illustrates the impact of avoidable rock on the advance. The greatest effect is visible at small advance values, accompanied by a high ash content. The smallest effect occurs for a high advance, implying a low ash content in accordance with Fig. 4.30.

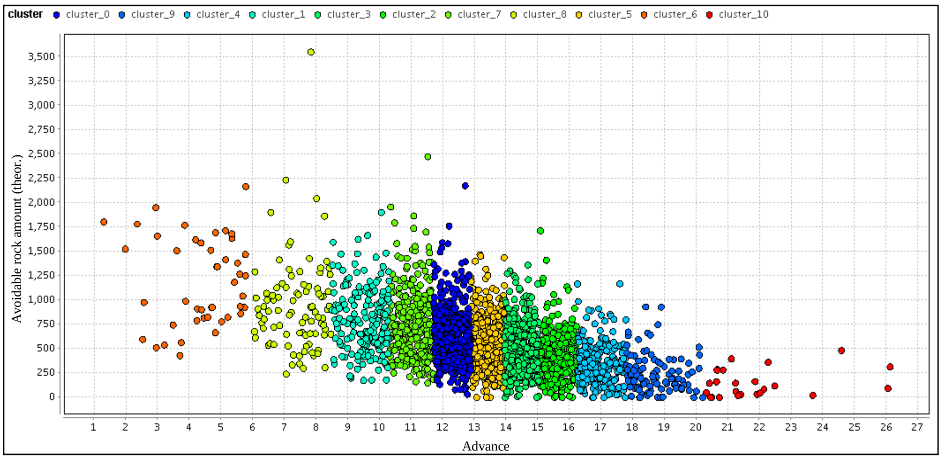

Fig. 4.33. Variability of the avoidable rock amount vs the advance in clusters. Avoidable ROM weight in Mg, advance in m/d (*own study*)

Fig. 4.34 presents the relationship between the weight (z axis) and the ash content (y axis) and the advance (x axis) for consecutive clusters.

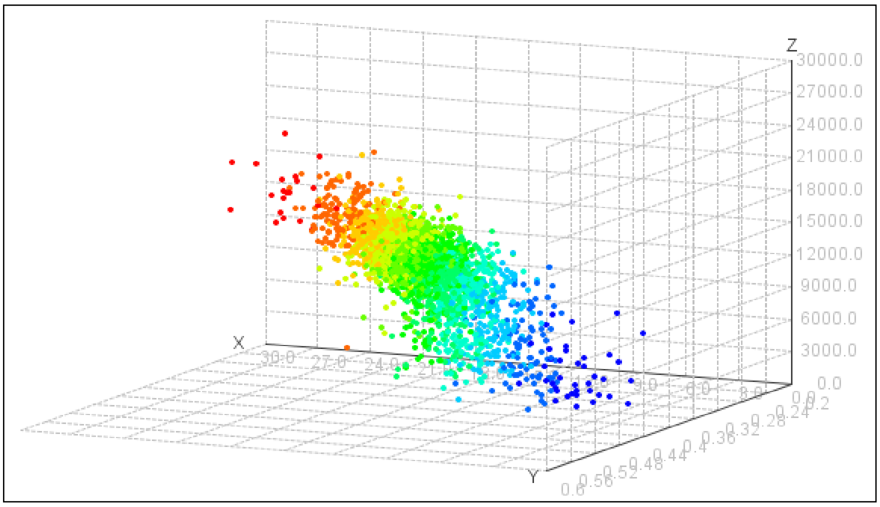

Fig. 4.34. The ROM weight in Mg vs the advance in m/d and the ash content in % in clusters (*own study*)

Summarising the above, it should be stated that based on the carried out cluster analysis, assuming under theoretical conditions an elimination of unfavourable factors impact on the mining process, it was calculated that the maximum achievable increase in the advance vs the average values observed in the LW Bogdanka SA is 8 m during a day. In reality this effect does not decompose into one, but into a dozen or so factors limiting the advance of longwalls. Because of that a questionnaire survey was carried out. Its results allowed to estimate the impact of geological and mining, technical and organisational factors on the advance. These factors were divided into controllable (i.e. such, which decision makers can actually manage) and remaining out of control, the impact on which is entirely excluded or limited very significantly. The prepared pessimistic scenario means a situation, in which the majority of effects, resulting from an introduction of actions aimed at improving the cleanness of mined ROM, is not achievable due to geological-mining, technical, and organisational conditions.

The optimistic scenario assumes the existence of rock mass, mining technology, and work organisation related factors, which are mostly controllable. In particular, this applies to technical and organisational issues.

4.4. Impact of geological-mining, technical, and organisational factors on plough longwalls advance - questionnaire survey

During this work preparation a questionnaire form was developed, addressed to the higher and medium-level supervisory personnel, and to the face staff employees. On its basis an achievable effect of advance increase was decomposed to the factors affecting the cleanness of mining.

The main objective of the carried out questionnaire survey was to obtain an authoritative assessment opinions of the LW Bogdanka SA employees on the factors affecting the amount of output from the plough-equipped longwalls and the ROM pollution, as well as on the level of the staff knowledge and awareness about the costs of mining operation. The collected answers reflected the realities of work in the plough-equipped longwalls and show paths to optimise the mining process with the use of plough technique.

4.4.1. Questionnaire and applied survey methods

The survey data were gathered by the questionnaire technique using a survey form, consisting of a description (four questions) and five subject-matter questions. Questions were specified in a few forms:

– a determination of the staff impact scale (from 1 - very small, to 5 - very big) on a given factor,

– a percentage determination of the share of factors affecting the output amount and cleanness,

– a determination of the coefficient of work at the plough-equipped longwalls arduousness (from 1 - very small, to 5 - very big).

Each question contained an instruction clarifying the way of answers provision. The questions asked in the survey were related to:

– the social description, concerning the district, position, length of service and department, in which the respondent was employed (4 questions),
– the knowledge about the impact of three factors (level of output, cleanness of mining, organisation of work) on the costs of coal mining in the plough-equipped longwalls (1 question),
– the assessment of geological, technical, and organisational factors impact on the amount and cleanness of output from the plough-equipped longwalls (2 questions with three sub-questions),
– the determination of the staff and supervisory personnel influence on a reduction of negative effects of the negative factors existence in the plough-equipped longwalls (1 question),
– the assessment of the arduousness of specific actions performance in the plough-equipped longwalls (1 question).

The data were introduced to the database created in the Microsoft Excel 2010 and analysed there.

4.4.2. Surveyed groups of respondents

440 questionnaires were distributed for the survey, of which:

– 110 pcs were filled in the Nadrybie panel,

– 110 pcs were filled in the Bogdanka panel,

– 220 pcs were filled in the Stefanów panel.

Before the survey started the respondents were informed about the survey objective and its anonymity. Questionnaires were transferred to the panel managers and distributed among the employees. Overall, 198 questionnaires were collected, 165 of which were analysed; the other ones were incomplete or filled in incorrectly.

4.4.3. Descriptive analysis of survey results

The number of collected questionnaires in all the mining areas was 165, of which 57% were related to the Stefanów panel, 33% - to the Bogdanka panel, and 10% - to the Nadrybie panel. The attendance was 45%, of which 41.6% were analysed (Fig. 4.35).

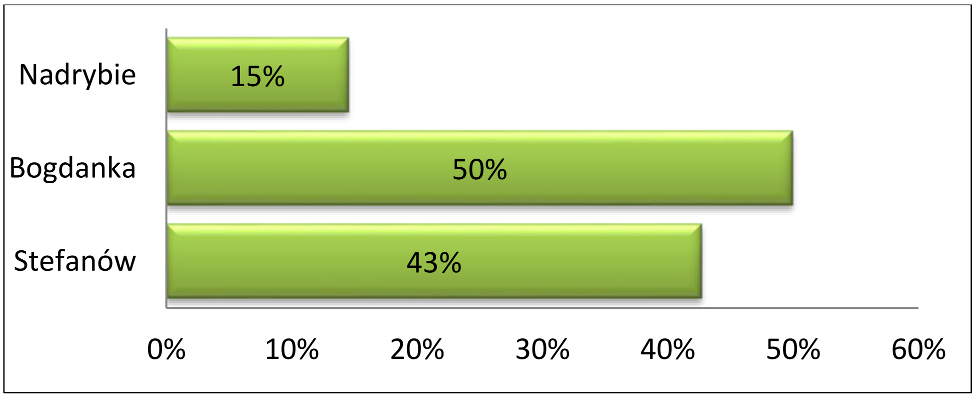

Fig. 4.35. Percentage of correctly filled in the questionnaires by panels (*own study*)

https://doi.org/10.32056/KOMAG/Monograph2022.1

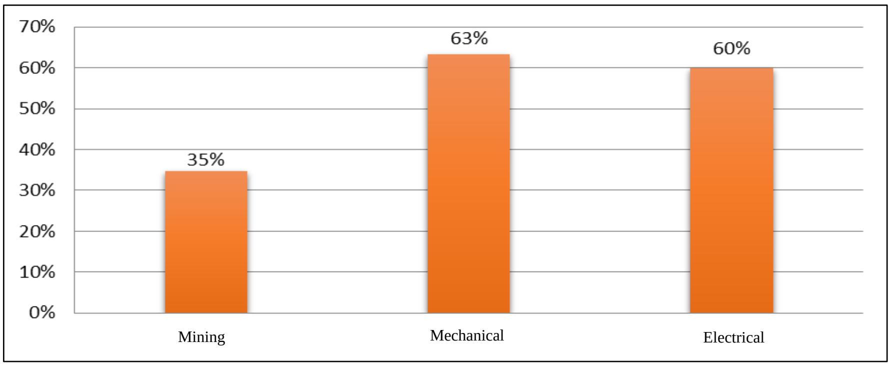

Fig. 4.36. Attendance by department employees (*own study*)

The questionnaire survey was intended for 303 employees of the mining department, 49 employees of the mechanical department, and 45 employees of the electrical department. 35% of miners, 63% of mechanics, and 60% of electricians responded among the surveyed group (Fig. 4.36). 41% of employees on physical workers' positions among 288 and 41% out of 76 persons of medium-level supervisory personnel responded. In the smallest group of the supervisory personnel the attendance was 45% (33 employees) - Fig. 4.37.

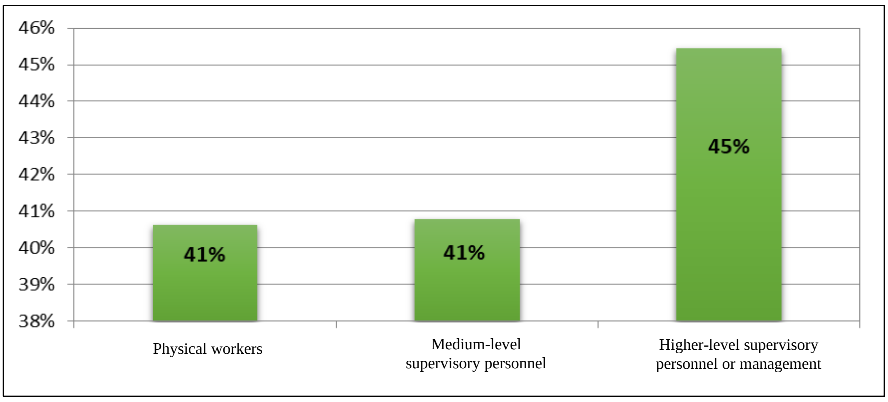

Fig. 4.37. Attendance by job positions (*own study*)

The surveyed persons worked in four mining districts: G-1 (24%), G-4 (24%), G-5 (26%), and G-6 (12%). 4% of the respondents marked more than one district. They were employees of the mechanical and electrical departments, while 12% did not provide this information (Fig. 4.38).

71% of the surveyed persons were employed on the physical workers' positions, 19% were persons from the medium-level supervisory personnel, while 9% were the higher-level supervisory personnel and management (Fig. 4.38).

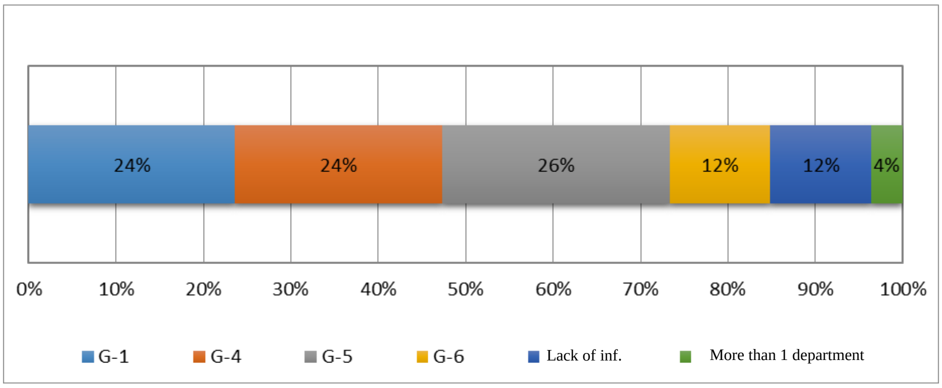

Fig. 4.38. Respondents' structure by positions (*own study*)

The employees of the mining department were 64% of surveyed persons, 19% - of the mechanical one, 16% - of the electrical department, 1% of the persons did not answer this question (Fig. 4.39).

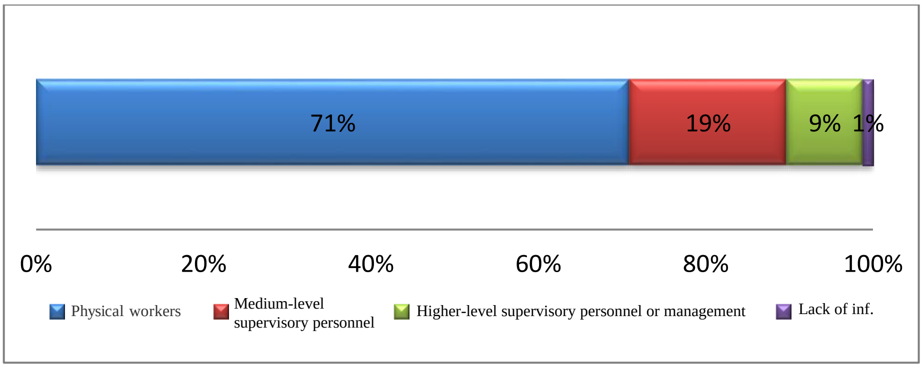

Fig. 4.39. Job positions of surveyed persons (*own study*)

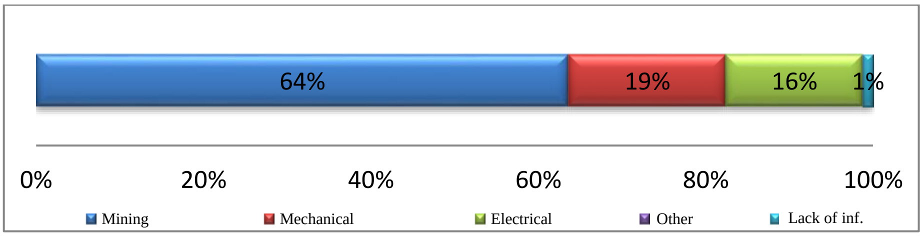

Fig. 4.40. Department of the surveyed persons work (*own study*)

The first question was related to a determination of the factors impact on the unit cost of coal mining from the plough-equipped longwalls. The considered factors were: the level of output, the cleanness of mining and organisation of work. The results were very similar and oscillated around 70% (Fig. 4.41).

https://doi.org/10.32056/KOMAG/Monograph2022.1

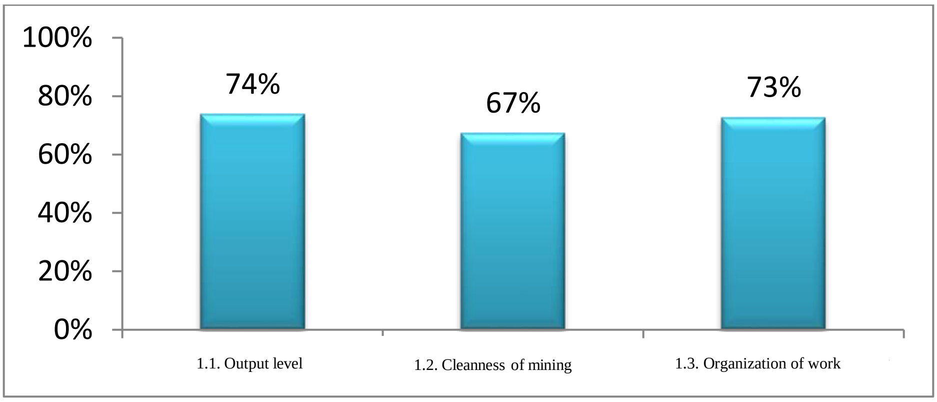

Fig. 4.41. Impact on unit cost (*own study*)

According to the opinions of the higher-level supervisory personnel and management, the organisation of work has the biggest impact on the unit costs (90%), the smallest impact - according to all the respondents - the cleanness of mining. The second question was related to the impact of specific factors on the amount of output from the plough-equipped longwalls. According to the respondents the geological factors have the biggest impact (51%), technical and organisational factors have the impact of 25% and 24%, respectively (Fig. 4.42). All the respondents, irrespective of their position, length of service and department, answered in a similar way.

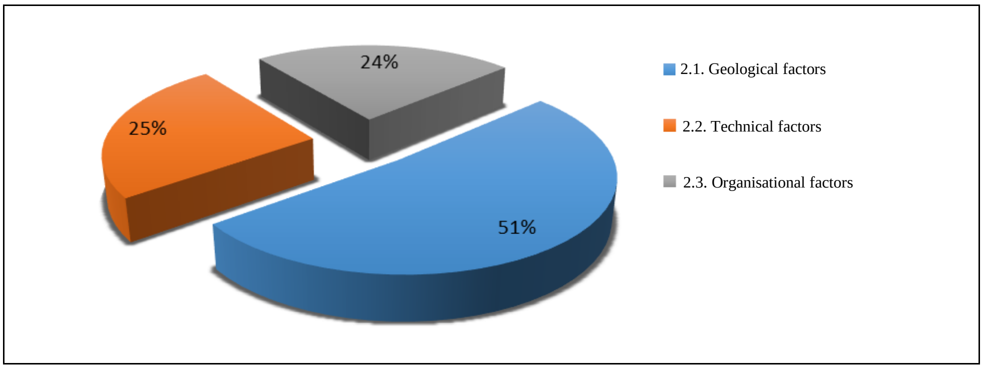

Fig. 4.42. Impact of factors on the amount of output (*own study*)

The first sub-question of question two was related to an impact of geological factors on the amount of output from the plough-equipped longwalls. In this case the largest number of respondents referred to the roof fall - 29%, and to the seam thickness - 28% (Fig. 4.43).

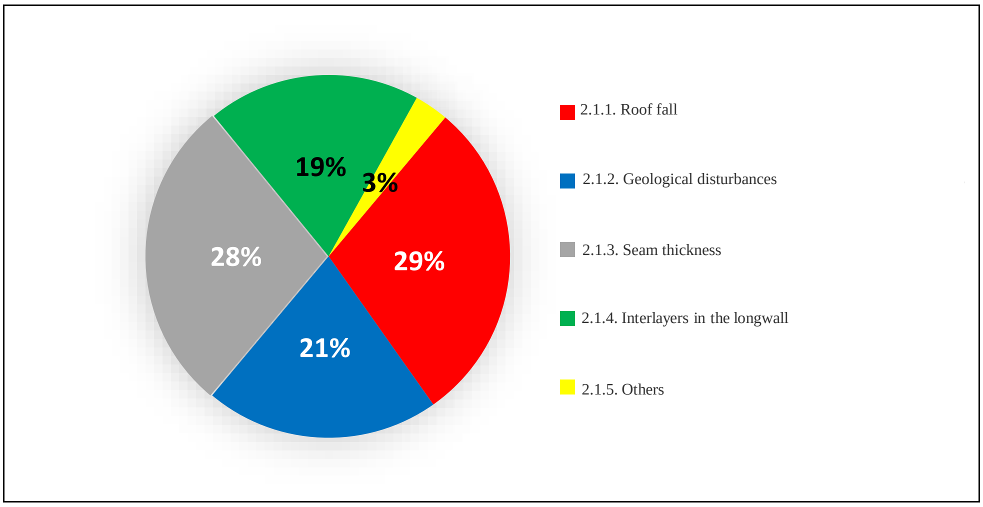

Fig. 4.43. Impact of geological factors on the amount of output (*own study*)

Comparing answers from individual districts with average geological conditions in specific districts it is possible to notice coincidence of factors (Table 4.23, Fig. 4.39).

Parameters describing geological conditions in the mining areas (*own study*)

Table 4.20.

District	Year	Seam height [m]	Dirt band [m]	Seam thickness [m]	Roof fall [m]	Floor ripping [m]	Mining height [m]
G-1	2015	1.34	0.16	1.19	0.11	0.31	1.76
G-4	2015	1.46	0.18	1.28	0.12	0.27	1.84
G-5	2015	1.35	0.18	1.17	0.13	0.21	1.68
G-6	2015	1.50	0.21	1.29	0.21	0.16	1.87

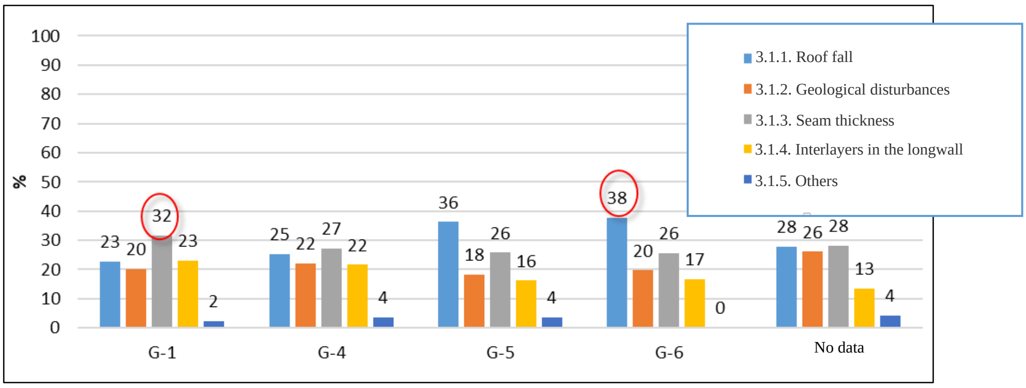

Fig. 4.44. Impact of geological factors on the amount of output by districts (*own study*)

https://doi.org/10.32056/KOMAG/Monograph2022.1

The next sub-question of question two read: which of technical factors affect the amount of output from plough longwalls? In this question 23% of respondents referred to the roof maintenance, while an appropriate quality of spare parts has the smallest impact (12%) (Fig. 4.45).

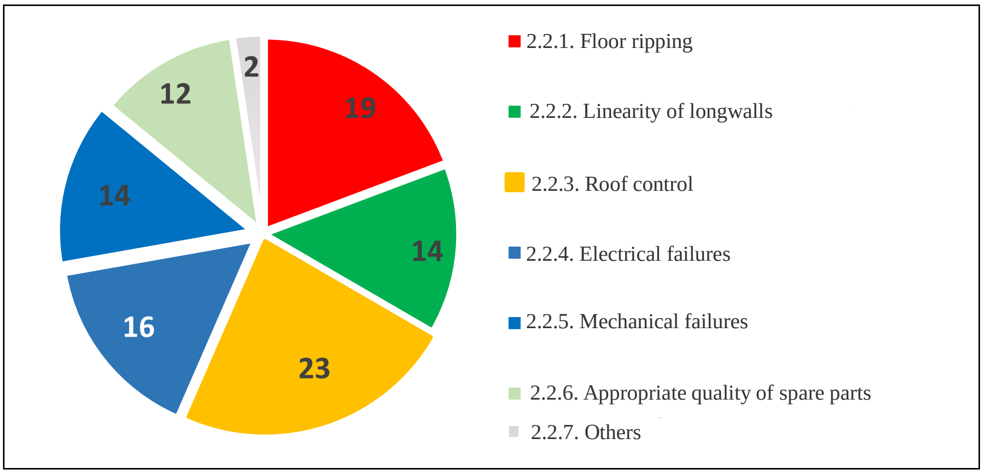

Fig. 4.45. Impact of technical factors on the amount of output (*own study*)

According to respondents the other important technical factors included:

- control of stable corners,
- running the stage loader,
- appropriate and reliable control, time-scale of inspections,
- care of the equipment by the personnel,
- mining failures.

The answers of respondents reflect average parameters describing the mining areas (Table 4.21, Fig. 4.46).

Parameters describing geological conditions in the mining areas (*own study*)

Table 4.21.

District	Year	Seam height [m]	Dirt band [m]	Seam thickness [m]	Roof fall [m]	Floor ripping [m]	Mining height [m]
G-1	2015	1.34	0.16	1.19	0.11	0.31	1.76
G-4	2015	1.46	0.18	1.28	0.12	0.27	1.84
G-5	2015	1.35	0.18	1.17	0.13	0.21	1.68
G-6	2015	1.50	0.21	1.29	0.21	0.16	1.87

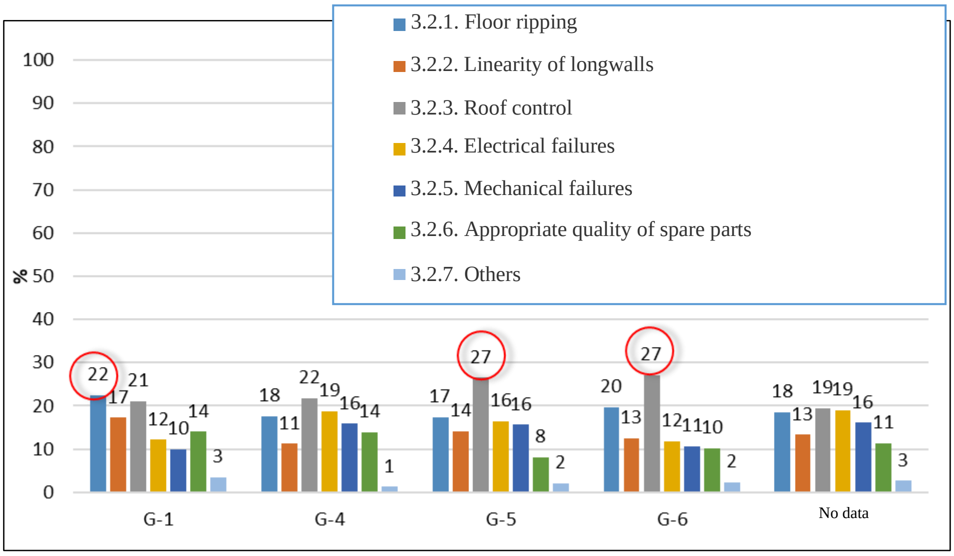

Fig. 4.46. Impact of technical factors on the amount of output by districts (*own study*)

The medium-level supervisory personnel and employees on the workers' positions think that in the group of technical factors the roof control has the biggest impact on the amount of the output from the plough-equipped longwalls.

The higher-level supervisory personnel think that electrical failures have the biggest impact on the amount of output from the plough-equipped longwalls (mean - 24%). According to the mining department the roof control affects the amount of output from the plough-equipped longwalls (24%). According to the mechanical department the floor ripping (23%) and the roof control affect the amount of output from the plough-equipped longwalls (22%). According to the electrical department employees the roof control (20%), the linear course of the face (19%) and the floor ripping have a decisive impact on the amount of output from the plough-equipped longwalls.

In the second question the organisational factors were discussed as the last ones, where the respondents indicated a reduction in the number of employees at the longwalls (30% of answers) as the biggest impact on the amount of output from the plough-equipped longwalls (Fig. 4.47).

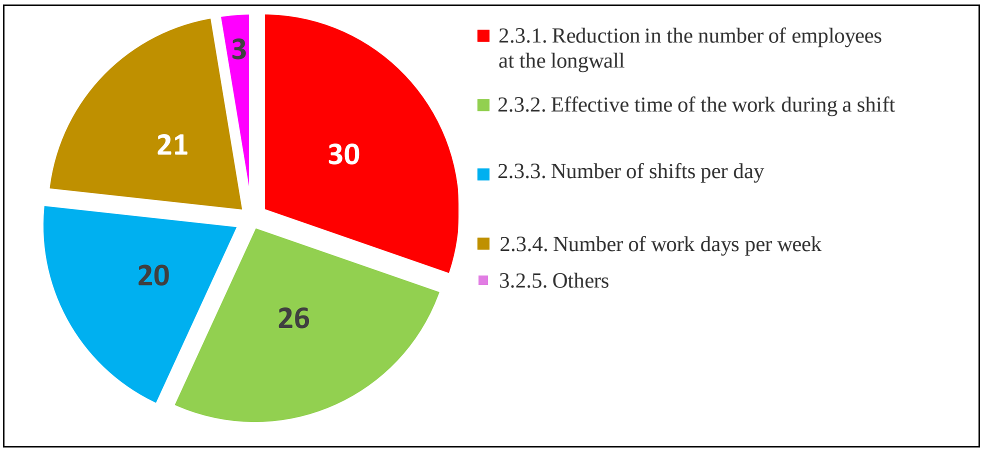

Fig. 4.47. Impact of organisational factors on the amount of output (*own study*)

Other factors referred to include:

- logistics,
- hoisting capabilities,
- emuneration,
- employees' experience,
- a small number of employees,
- climatic conditions in the mine.

The employees of the G-1, G-4, and G-5 districts considered a reduction of the employees' number at the longwall as a decisive impact on the amount of output from the plough-equipped longwalls (more than 30% answers). According to the G-6 district employees the effective time of work during a shift has the biggest impact among organisational factors (29%). According to the higher-level and medium-level supervisory personnel the effective time of work during a shift has the biggest impact (approx. 33%). According to the employees on the workers' positions, a reduction in the employees' number at the longwall has the biggest impact on the amount of output from the plough-equipped longwalls (33%).

The third question was related to the impact of geological, technical, and organisational factors on the cleanness of mining from the plough-equipped longwalls. According to the respondents' opinions in this case the geological factors (62%) have the biggest impact while the organisational factors - 16% have the smallest impact (Fig. 4.48).

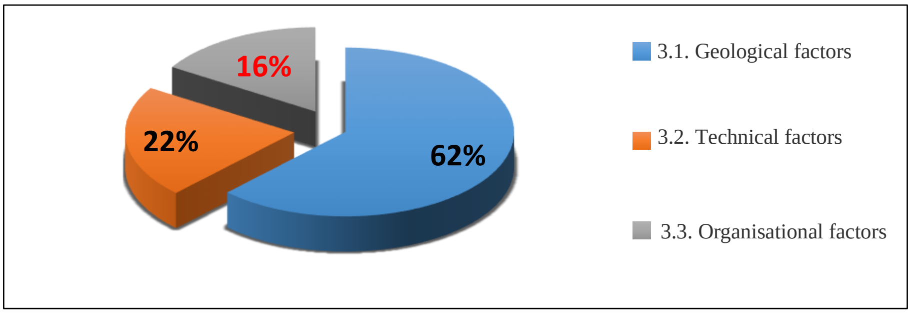

Fig. 4.48. Impact of geological, technical, and organisational factors on the cleanness of mining operations (*own study*)

According to the respondents the roof fall has the biggest impact on the cleanness of mining (29%), while geological disturbances have the smallest impact (21%) among geological factors (Fig. 4.49).

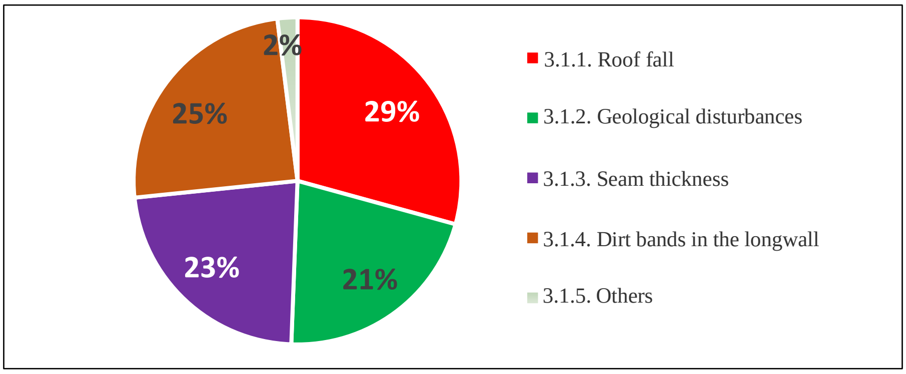

Fig. 4.49. Impact of geological factors on the cleanness of mining operations (*own study*)

According to the G-5 and G-6 district employees the roof fall has the biggest impact on the cleanness of mining from the plough-equipped longwalls (geological factors), acc. to the G-1 district employees the dirt bands in the face have the biggest impact on the cleanness of mining from the plough-equipped longwalls. The assessment of mechanical and electrical department employees is similar (30% and 29%). According to the mining department the roof fall has the biggest impact on the cleanness of mining from the plough-equipped longwalls (32%). Among technical factors, affecting the cleanness of mining, in the respondents' opinions, the floor ripping has the biggest impact (34%), while electrical failures - the smallest one (7%) (Fig. 4.50).

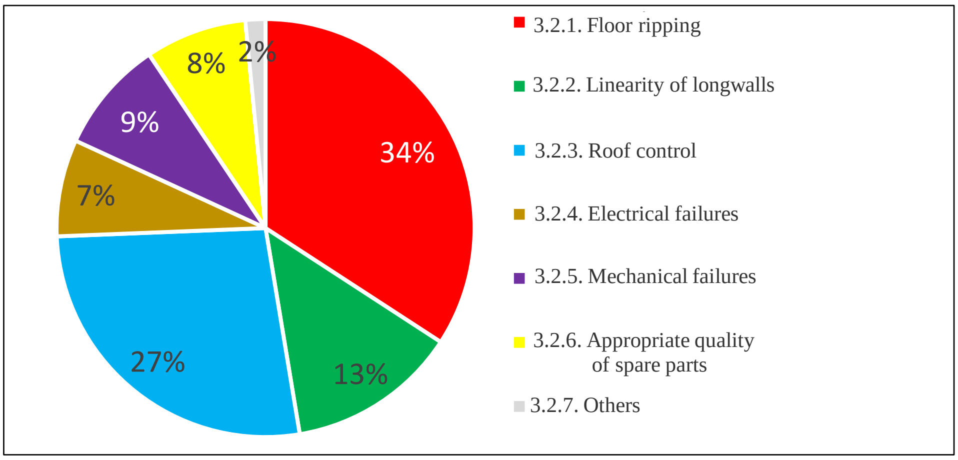

Fig. 4.50. Impact of technical factors on the cleanness of mining operations (*own study*)

Other technical factors:

– deep dinting at sumping-in,
– floor ripping in the headgates and in the tailgates.

According to the respondents from all the mining districts the floor ripping has the biggest impact on the cleanness of mining operations (Fig. 4.51).

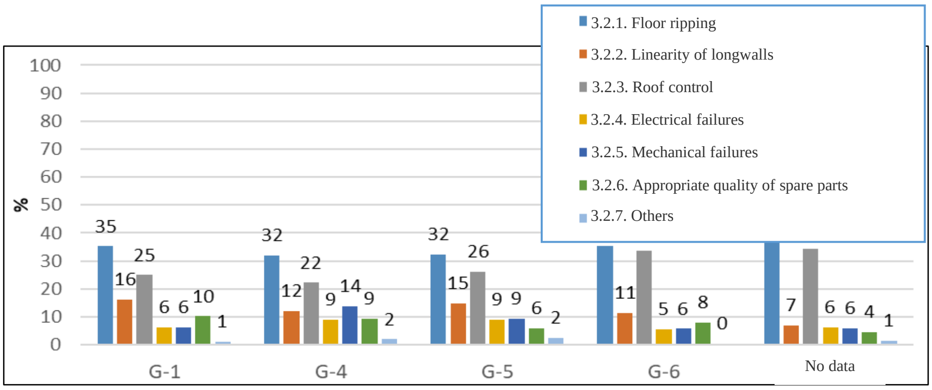

Fig. 4.51. Impact of technical factors by districts (*own study*)

Like in question two, the respondents indicated that the reduction of employees' number at the longwall has the biggest impact on the cleanness of mining operations among organisational factors (38%) (Fig. 4.52). Also other factors have a substantial impact, such as:

– the training of employees,
– the correct situation assessment by the supervisors and cutting machine operators,
– no supports installed under the armoured face conveyor.

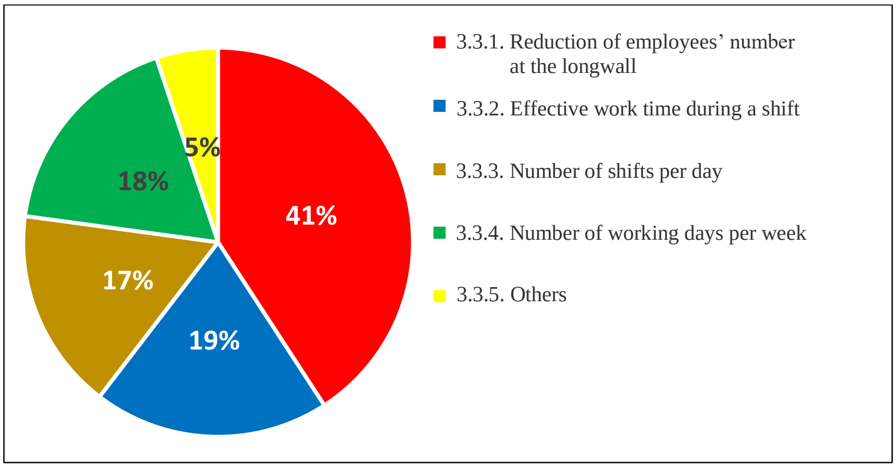

Fig. 4.52. Impact of organisational factors on the cleanness of mining operations (*own study*)

According to the higher-level supervisory personnel the cleanness of mining from the plough-equipped longwalls is seriously affected (organisational factors) by the number of working days in a week (20%). The medium-level supervisory personnel and employees on physical workers' positions said that the reduction of employees' number at the longwall had the biggest impact (approx. 40%).

The next questions of the carried out questionnaire read:

- Question 4 - Does the supervisors and the staff affect a reduction of negative effects of the factors specified below?
- Question 5: What is your assessment of arduousness of the following operations performance in the plough-equipped longwalls, 1.2 m to 1.5 m high?

A 5-degree verbal impact scale was then applied for calculations allowing to analyse the criteria based on weights, where 1 on the point scale corresponds to a very small impact with the weight of 0.06, and 5 - to a very big impact with the weight of 0.42 (Table 4.25). The assessed factors from all the questionnaires were averaged and the obtained values (mean assessments) were used for matrix calculations. Next, a comparison matrix was constructed to assess the significance of criteria, for all the analysed conditions (MP4 and MP5).

Scale of factors - criteria assessment for the questions: 4 and 5 (*own study*)

Table 4.22.

Verbal description	Weight	Point scale
Very big	0.42	5
Big	0.26	4
Average	0.16	3
Small	0.10	2
Very small	0.06	1

Matrix MP4 for the criteria (conditions 4.1……..4.10) specifying, whether the supervisors and the staff have an impact on the reduction of negative effects of assessed factors, is presented below:

$$MP4 = 1 \begin{bmatrix} 1 & 4.1/4.2 & 4.1/4.3 & 4.1/4.4 & 4.1/4.5 & 4.1/4.6 & 4.1/4.7 & 4.1/4.8 & 4.1/4.9 & 4.1/4.10 \\ 4.2/4.1 & 1 & 4.2/4.3 & 4.2/4.4 & 4.2/4.5 & 4.2/4.6 & 4.2/4.7 & 4.2/4.8 & 4.2/4.9 & 4.2/4.10 \\ 4.3/4.1 & 4.3/4.2 & 1 & 4.3/4.4 & 4.3/4.5 & 4.3/4.6 & 4.3/4.7 & 4.3/4.8 & 4.3/4.9 & 4.3/4.10 \\ 4.4/4.1 & 4.4/4.2 & 4.4/4.3 & 1 & 4.4/4.5 & 4.4/4.6 & 4.4/4.7 & 4.4/4.8 & 4.4/4.9 & 4.4/4.10 \\ 4.5/4.1 & 4.5/4.2 & 4.5/4.3 & 4.5/4.4 & 1 & 4.5/4.6 & 4.5/4.7 & 4.5/4.8 & 4.5/4.9 & 4.5/4.10 \\ 4.6/4.1 & 4.6/4.2 & 4.6/4.3 & 4.6/4.4 & 4.6/4.5 & 1 & 4.6/4.7 & 4.6/4.8 & 4.6/4.9 & 4.6/4.10 \\ 4.7/4.1 & 4.7/4.2 & 4.7/4.3 & 4.7/4.4 & 4.7/4.5 & 4.7/4.6 & 1 & 4.7/4.8 & 4.7/4.9 & 4.7/4.10 \\ 4.8/4.1 & 4.8/4.2 & 4.8/4.3 & 4.8/4.4 & 4.8/4.5 & 4.8/4.6 & 4.8/4.7 & 1 & 4.8/4.9 & 4.8/4.10 \\ 4.9/4.1 & 4.9/4.2 & 4.9/4.3 & 4.9/4.4 & 4.9/4.5 & 4.9/4.6 & 4.9/4.7 & 4.9/4.8 & 1 & 4.9/4.10 \\ 4.10/4.1 & 4.10/4.2 & 4.10/4.3 & 4.10/4.4 & 4.10/4.5 & 4.10/4.6 & 4.10/4.7 & 4.10/4.8 & 4.10/4.9 & 1 \end{bmatrix}$$

where:

$$W = [0{,}08_{4.1};\ 0{,}10_{4.2};\ 0{,}09_{4.3};\ 0{,}12_{4.4};\ 0{,}08_{4.5};\ 0{,}08_{4.6};\ 0{,}09_{4.7};\ 0{,}144_{4.8};\ 0{,}11_{4.9};\ 0{,}10_{4.10}]^T$$

and:

- 4.1 - roof fall,
- 4.2 - floor ripping,
- 4.3 - face height,
- 4.4 - straight-line course of the face,
- 4.5 - electrical failures,
- 4.6 - mechanical failures,
- 4.7 - appropriate quality of spare parts,
- 4.8 - organisation of work at the longwall,
- 4.9 - reduction of employees number at the longwall,
- 4.10 - others.

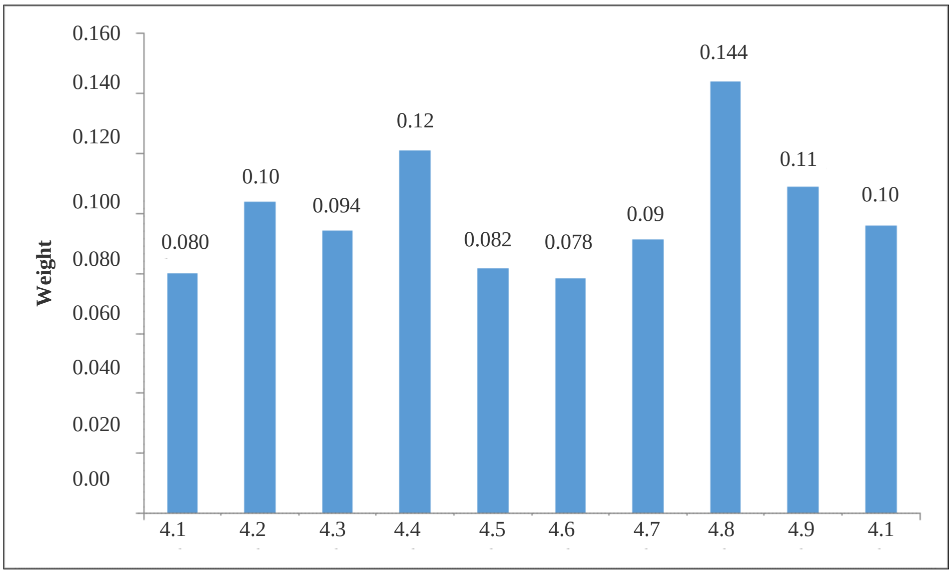

Fig. 4.53. Assessment of the degree of negative effects reduction for individual factors occurrence (*own study*)

The analysis of weight vector components for the assessed conditions (4.1….4.10) shows that the most significant criterion in the assessed groups, on which the supervisors and the staff have the biggest impact, is the quality of spare parts with the weight of 0.144 and the straight-line course of the face - 0.12. Mechanical failures (0.078) and the roof fall (0.08) are the criteria, which are least affected by the supervisors and the staff.

To analyse the question 5 (working in a similar way like with the question 4) matrix MP5 was constructed for the criteria (conditions 4.1……..4.5), defining the arduousness of operations in the plough-equipped longwalls with faces 1.2 m to 1.5 m high.

https://doi.org/10.32056/KOMAG/Monograph2022.1

$$MP5 = \begin{bmatrix} 1 & 4.1/4.2 & 4.1/4/3 & 4.1/4.4 & 4.1/4.5 \\ 4.2/4.1 & 1 & 4.2/4.3 & 4.2/4.4 & 4.2/4.5 \\ 4.3/4.1 & 4.3/4.2 & 1 & 4.3/4/4 & 4.3/4.5 \\ 4.4/4.1 & 4.4/4.2 & 4.3/4/3 & 1 & 4.4/4.5 \\ 4.5/4.1 & 4.5/4.2 & 4.5/43 & 4.5/4.4 & 1 \end{bmatrix}$$

where

$$W = [0{,}198_{4.1};\ 0{,}177_{4.2};\ 0{,}235_{4.3};\ 0{,}207_{4.4};\ 0{,}184_{4.5}]^T$$

and:

- 4.1 – down-time of the plough-equipped longwall,
- 4.2 – an operation of the longwall equipment (sections),
- 4.3 – a performance of work caused by the roof fall,
- 4.4 – a repair of powered roof support unit failure,
- 4.5 – a performance of inspection and overhaul of powered roof support units.

From the above calculations it results that the most significant factor in the assessed group (4.1…4.5) is the performance of work due to the roof fall (0.235). The least important (expressed by the lowest weight values) were the criteria: an operation of the longwall equipment (sections) and a performance of inspection and overhaul of powered roof support units (0.184).

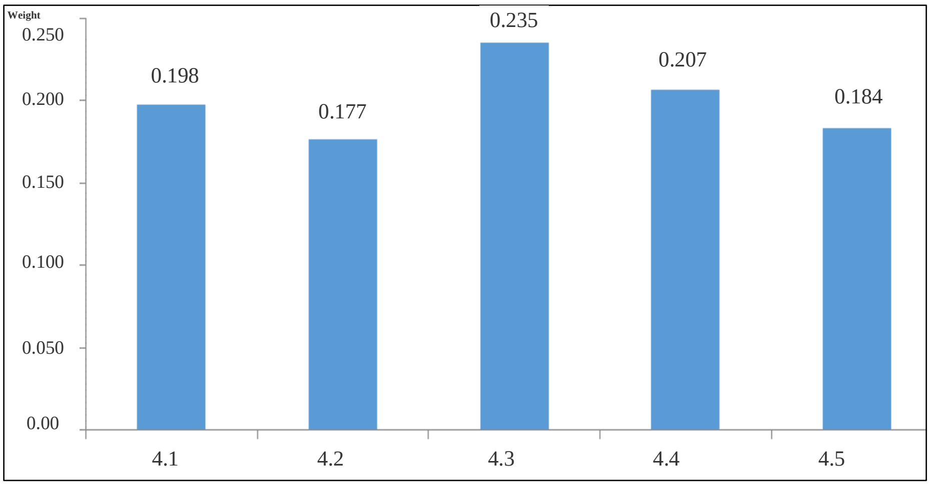

Fig. 4.54. Arduousness of work at the plough-equipped longwalls (*own study*)

The diagrams (Fig. 4.55 and 4.56) present the percentage impact of each individual factor on the ROM pollution and the amount of output from the plough-equipped longwalls.

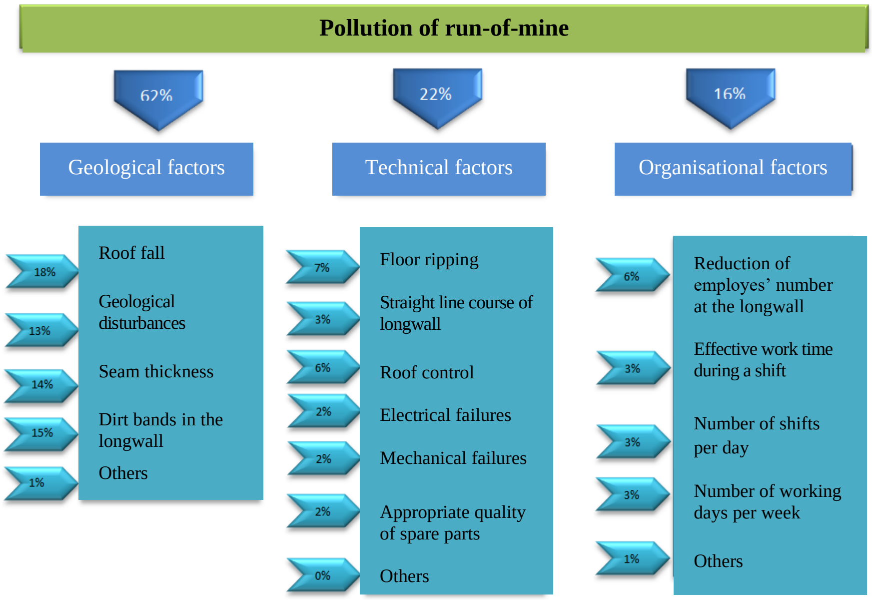

Fig. 4.55. Impact of factors on the ROM pollution for the plough-equipped longwalls (own study)

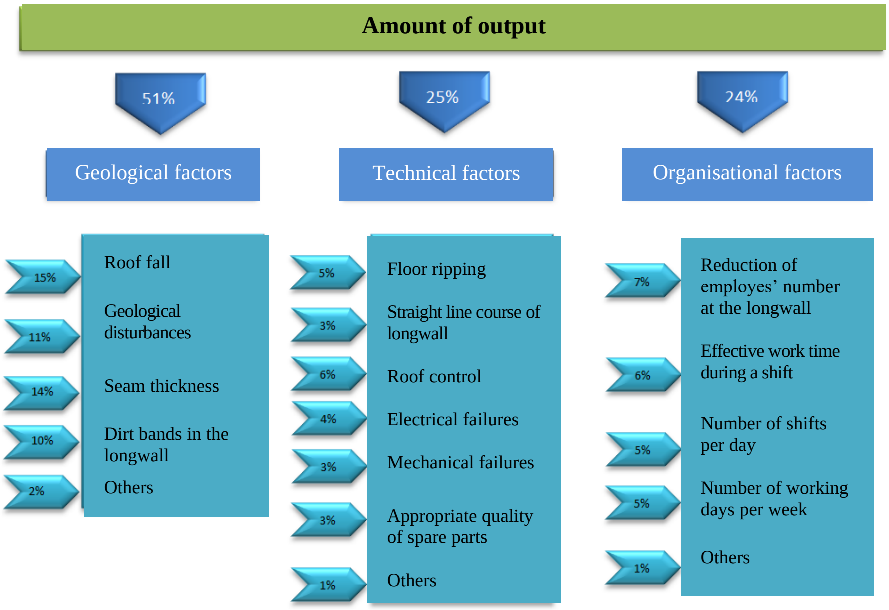

Fig. 4.56. Impact of factors on the amount of output from the plough-equipped longwalls (own study)

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4.4.4. Calculation procedure of the ROM pollution index (WZU) in the case of plough-equipped longwalls

To determine the ROM pollution index (WZU) for the plough-equipped longwalls the previously selected factors (criteria) were used - the most important ones for the considered task, assessed by employees in the questionnaire.

The questionnaire form, developed for the need of WZU index calculation, contained a fixed, selected list of questions and accompanying closed set of answers. Both in the first and in the second part the answers were given in percentages, so that the sum of factors in the assessed groups would be 100%.

Because of a multi-factor and multi-range nature of the task (assessed criteria were difficult to compare) to determine the index a procedure was suggested, using the structure of a multiple criteria task, hierarchical, based on a group of main factors (the first part of the questionnaire, level one of criteria) and detailed factors (criteria) (their developments - the second part of the questionnaire), allowing for a more detailed preference assessment.

The following factors were in the main group:

- 2.1. - Geological,
- 2.2. - Technical,
- 3.3. - Organisational,

Detailed factors in the group - Geological:

- 3.1.1. OS - Roof fall,
- 3.1.2. ZG - Geological disturbances (sandstone lenses, faults),
- 3.1.3. MP - Seam thickness,
- 3.1.4. PnŚ - Dirt bands in the face,
- 3.1.5. Others.

Detailed factors in the group - Technical:

- 3.2.1. PS - Floor ripping,
- 3.2.2. PŚ - Straight-line course of the face,
- 3.2.3. US - Roof control,
- 3.2.4. AE - Electrical failures,
- 3.2.5. AM - Mechanical failures,
- 3.2.6. JCzZ- Appropriate quality of spare parts,
- 3.2.7. Others.

Detailed factors in the group - Organisational:

- 3.3.1. ZLP - Reduction of employees’ number at the longwall,
- 3.3.2. ECP - Effective working time per shift,

- 3.3.3. LZD - Number of shifts per day,
- 3.3.4. LDPwT - Number of working days per week,
- 3.3.5. Others.

Values of individual factors in the separated groups were calculated as a vector of the weights obtained from the matrix calculations (criteria comparison matrices). The values were calculated both for the main group factors, and also for their developments. Final (global) scores for detailed factors were obtained by multiplying vectors of main and detailed factors weights. An identification of partial factors was the next stage at the index calculation. The factors may be stimulants, destimulants, or nominants in nature. Stimulants are the features, whose high values are desired from the general criterion point of view (the higher, the better). Destimulants are the features, whose high values are not desired (the smaller, the better). Nominants are the features, that have a certain optimum level and going away from it (both downwards and upwards) is unfavourable. The following factors were classified as stimulants:

- Group of geological factors:
  - 3.1.1. OS - Roof fall,
  - 3.1.2. ZG - Geological disturbances (sandstone lenses, faults),
  - 3.1.4. PnŚ - Dirt bands in the face,
- Group of technical factors:
  - 3.2.1. PS - Floor ripping,
  - 3.2.4. AE - Electrical failures,
  - 3.2.5. AM - Mechanical failures,
- Group of organisational factors:
  - 3.3.1. ZLP - Reduction of employees' number at the longwall,
  - 3.3.3. LZD - Number of shifts per day,

The following factors were classified as destimulants:

- Group of geological factors:
  - 3.1.3. MP - Seam thickness.
- Group of technical factors:
  - 3.2.2. PŚ - Straight-line course of the face,
  - 3.2.3. US - Roof control.
- Group of organisational factors:
  - 3.3.2. ECP - Effective working time per shift,
  - 3.3.4. LDPwT - Number of working days per week.

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Criteria comparisons for geological, technical, and organisational factors were carried out on the level one. Valuations were carried out and priorities (weights) for the criteria were calculated, using the comparison matrices in pairs. The vector of weights was obtained: $WI = [0{,}62_G; 0{,}21_T; 0{,}16_O]^T$. When analysing components of the vector of weights for three main groups of the criteria, it is possible to state that the geological conditions were the most important criterion in terms of the target achievement (0.62), technical (0.21) and organisational (0.16) ones ranked next.

Weights for the partial criteria were then calculated for three main groups of criteria. The vector of weights with components $WG = [0{,}29_{3.1.1}; 0{,}21_{3.1.2}; 0{,}23_{3.1.3}; 0{,}25_{3.1.4}; 0{,}02_{3.1.5}]^T$ was obtained as a result of the matrix resolving in the group of geological criteria. The roof fall criterion obtained the highest weight (0.29). The factors defined in the questionnaire as others recorded the lowest weight (0.02).

In the group of technical criteria the calculated components of the vector of weights $WT = [0{,}34_{3.2.1}; 0{,}13_{3.2.2}; 0{,}26_{3.2.3}; 0{,}08_{3.2.4}; 0{,}09_{3.2.5} 0{,}08_{3.2.6}; 0{,}02_{3.2.7}]^T$ show that the floor ripping (0.34) and roof control (0.26) are definitely the most important criteria. The quality of spare parts (0.08) and electrical failures (0.08) were indicated as the least important.

Partial assessments of the criteria obtained for the group of organisational factors $WO = [0{,}38_{3.3.1}; 0{,}18_{3.3.2}; 0{,}16_{3.3.3}; 0{,}17_{3.3.4}; 0{,}05_{3.5.5}]^T$ show that the reduction of employees' number at the longwall is the most important criterion (0.38). So-called other factors obtained low weights (0.05), while the remaining factors achieved similar weights: the number of shifts per day (0.16); the number of working days per week (0.17), and the effective working time per shift (0.18).

Local weights in the matrix notation for the level two have the form:

$$WII = \begin{matrix} 3.1.1 \\ 3.1.2 \\ 3.1.3 \\ 3.1.4 \\ 3.1.5 \\ 3.2.1 \\ 3.2.2 \\ 3.2.3 \\ 3.2.4 \\ 3.2.5 \\ 3.2.6 \\ 3.2.7 \\ 3.3.1 \\ 3.3.2 \\ 3.3.3 \\ 3.3.4 \\ 3.3.5 \end{matrix} \begin{bmatrix} 0{,}29 & 0 & 0 \\ 0{,}21 & 0 & 0 \\ 0{,}23 & 0 & 0 \\ 0{,}25 & 0 & 0 \\ 0{,}02 & 0 & 0 \\ 0 & 0{,}34 & 0 \\ 0 & 0{,}13 & 0 \\ 0 & 0{,}26 & 0 \\ 0 & 0{,}08 & 0 \\ 0 & 0{,}09 & 0 \\ 0 & 0{,}08 & 0 \\ 0 & 0{,}02 & 0 \\ 0 & 0 & 0.38 \\ 0 & 0 & 0.18 \\ 0 & 0 & 0.16 \\ 0 & 0 & 0.17 \\ 0 & 0 & 0.05 \end{bmatrix}$$

where:

WII - local vector of the level II.

The weights determined in each matrix are of local importance. Multiplied by weights of appropriate higher level are specified as global weights. The global vector of the level II is described by the matrix:

$$W[1,17]^T = WIxWII$$

where: WI - local vector of the level I,

WII - local vector of the level II:

$$W = [0{,}182_{3.1.1};\ 0{,}134_{3.1.2};\ 0{,}143_{3.1.3};\ 0{,}154_{3.1.4};\ 0{,}011_{3.1.5};\ 0{,}071_{3.2.1};\ 0{,}027_{3.2.2};\ 0{,}055_{3.2.3};\ 0{,}016_{3.2.4};\ 0{,}018_{3.2.5};\ 0{,}017_{3.2.6;};\ 0{,}003_{3.2.7};\ 0{,}061_{3.3.1};\ 0{,}029_{3.3.2};\ 0{,}025_{3.3.3};\ 0{,}027_{3.3.4};\ 0{,}008_{3.3.5}]^T$$

where: W - global vector of the level II.

The components of the global vector of the level II were used to calculate the final ROM pollution index for the plough-equipped longwalls. The calculated index is a sum of the global factors being stimulants, less weights of the factors being destimulants.

The ROM from the plough-equipped longwalls pollution index (WZU) for the LW Bogdanka, calculated on the basis of the on results of questionnaire survey of mine employees, is 0.38. This index, calculated for individual panels, has similar values: the Nadrybie - 0.36, the Bogdanka - 0.38, the Stefanów - 0.38.

Having calculated the index for the entire LW Bogdanka S.A. mine, in the second stage an attempt was made to calculate the index values assuming the maximum staff influence on individual factors, which can reduce the pollution of the ROM from the plough-equipped longwalls. The answers to question 4 of the questionnaire considered the information about the extent to which the staff can influence individual factors, reducing the ROM pollution. Taking the maximum reductions for those factors, which can be affected by the staff, the calculated index of the ROM from the plough-equipped longwalls pollution (WZU) for the LW Bogdanka S.A. was 0.20.

Having considered the results of the questionnaire survey, the adjusted parameters for the longwall advances in the case of individual scenarios were estimated (Table 6.23).

Estimation of longwall advances in thin seams acc. to scenarios (own study)

Table 4.23.

Scenario	Longwall advance [m/day]	Increase in longwall advance [%]
Baseline	13.20	0.0%
Pessimistic	13.83	4.8%
Optimistic	15.18	15.0%
Full optimisation effects	18.48	40.0%

An advance increase of 4.8% corresponds to the effect achievable only by optimised plough operation without any floor ripping. An advance increase by 15% corresponds to the effect achievable due to an elimination of the floor ripping and by a half of effects originating from a frequency reduction of electrical and mechanical failures occurrence. Also 30% of effects, due to proper roof control and 20% of effects due to an elimination of the roof fall were calculated. An increase in the advance by 40% corresponds to cancelling all avoidable unfavourable factors affecting the mining process. This is an extremely optimistic situation.

4.5. Economic efficiency (profitability) model

As it was mentioned, the assessment of the impact of giving up the cutting and extracting of excessive amounts of dirt was carried out in the model based on the net present value (NPV) method. To emphasise the importance of identified benefits (cost savings) from the mining process rationalisation and reduction of the dirt amount a differential approach was applied. The baseline model was the model of mining from the faces in thin seams, at basic mining costs of longwalls by means of a plough. In the model additionally identified cost savings were introduced into the structure of cash flows as well as the changes in the output level resulting from the differences in the advance as a function of the deposit recovery period. These models were separately designed for each research scenario.

The FCFF (*free cash flow to firm*) system was used to calculate the resultant economic-financial measures, taking into consideration:

- the ROM output:
  - the coal output
- the identified cost savings related to a smaller amount of mined dirt,
- revenues,
- the cash operating costs, in particular related to:
  - driving of roadways, opening, and development,
  - equipping the face,
  - the operational phase - mining, estimated for the plough-equipped longwalls,
  - the face liquidation.

The following calculations were conducted on the grounds of the aforementioned assumptions:

- EBIT (earnings before interest and tax), and after the tax deduction the following cash values were obtained:
- NOPAT (net operating profit after tax).
- FCFF (free cash flow to firm).

Then discounting the calculated annual cash flows with the adopted discount rate, the Net Present Value (NPV) was calculated. Table 4.24 below presents the structure of economic efficiency assessment model.

Structure of the economic efficiency assessment model (own study)

Table 4.24.

Calculation periods	Unit	Total (0-25)	0	1	2	3	…	…	20	21	22	23	24	25
			2016	2017	2018	2019	…	…	2036	2037	2038	2039	2040	2041
Total net output for longwalls and roadways	[Mg]													
Revenues	[PLN]													
Mining costs of plough mining	[PLN]													
Amount of avoided dirt	[Mg]													
Total savings	[PLN]													
Total costs	[PLN]													
EBITDA	[PLN]													
Depreciation	[PLN]													
EBIT	[PLN]													
Tax	[PLN]													
NOPAT	[PLN]													
FCFF	[PLN]													
Discount rate	[%]													
DCF	[PLN ‘000]													

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4.5.1. Identification of potential benefits resulting from the cleanness improvement of the mining process

The showed importance of rationalisation of mining from the faces of low thicknesses means the pursuit of minimisation of dirt cutting. The Author is aware of the fact that it is frequently a difficult, and sometimes just impossible task (conditions independent of human activities).

Within the carried out analysis it was found that a smaller amount of dirt, generated at the longwall faces, has a favourable impact both on the pace of the mining (advance) and on the continuity of transport and preparation operations performance. It translates next into the costs of mining, transport, preparation, and management of dirt on the surface, visible especially in a unit-based approach. These benefits, because of their place of origin, may be divided into three main groups (areas):

1. Benefits related to mining the longwalls:

   – an increase in the longwall advance, calculated as an increase in the labour efficiency and a better utilisation of possessed production assets. It translates also to a faster mining of resources: a smaller discount and incurring costs only in the period of shortened mine life (savings in global costs of deposit resources acquisition),
   – a reduction of failures number and time, related to the characteristic of dirt in the face and specific operation of plough systems. It was determined that a smaller amount of dirt and an operation of machinery under a smaller load can translate into a smaller failure frequency, non-incurring high costs of services, repairs, and overhauls as well as outsourced services.

2. Benefits related to the operation of haulage, horizontal transport, and shaft hoisting:

   – a reduction of total cost of horizontal and vertical transport related to a reduction of cutting and transporting excessive rock amounts,
   – a reduction of costs related to an exchange of loading, unloading components and skip parts as a result of their life extension (a reduction of stone amounts featuring higher hardness and large sizes in the shaft feed),
   – a reduction of costs related to an exchange of horizontal haulage belts and rollers exposed to hits from lumps of rock, especially at transfer points of the haulage system (calculated in the analysis only indirectly),

- a reduction of costs related to possible stone separation (probably stopping it entirely in certain areas), allowing to utilise assets - crushers and screens in other places of the mine.

3. Benefits related to the operation of the preparation plant, dirt management on the surface, and sales:

- a reduction of the CPP total costs related to the preparation of limited ROM weight or output of poorer quality,
- an elimination of the costs related to the transport or disposal of dirt on the surface,
- making the preparation process more flexible through easier obtaining of higher quality coal.

Also the CAPEX savings may be identified within the benefits achievable due to cutting smaller amounts of stone (e.g. for the face equipment used more rationally, which could work longer and be less frequently replaced with the new one). However, because of the difficulty in estimating their reliable level, the savings in this field were omitted. The approach to an aggregation of the costs into cost centres, corresponding to recording the process costs at the place of their origination, was used in calculations. Total costs of a specific centre were divided previously, determining the cost component falling per plough-equipped longwalls.

Finally, it was found that the benefits resulting from the improved cleanness of the mining process (a reduction of the dirt amount) would correspond **primarily to all the costs not incurred due to giving up mining, transport and preparation of the dirt left in the rock mass**.

The subject of analysis included also possible effects of cutting costs within underground workshop centres, resulting from potentially lower failure frequency and - as a result - lower service, repair, and overhaul costs. **The value of savings in this field was calculated as 5%.**

It should be clarified that in the case of savings related to the advance increase, the rationalisation of mining was primarily related to the optimal use of machinery technical capacity, which translated into more effective cutting (i.e. with bigger advance) under the conditions of lower resistance in faces. The effects of the advance increase and the impact on costs may be understood in two ways as:

1. An increase in the output at the same fixed cost in the same period.
2. Savings in costs (variable and/or fixed) at a lower output level in the same unit of time.

The latter approach was used for the analysis needs. Achievable benefits are presented in Table 4.25.

Savings for scenarios broken down to cost centres (own study)

Table 4.25.

Specification	Unit	Level of savings in scenario
Savings in the haulage centre	[PLN]	24,612,461.0
Savings in costs of longwalls	[PLN]	50,745,615.0
Savings in the shaft centre	[PLN]	9,847,141.0
Savings in the underground workshops centre	[PLN]	5,570,642.0
Savings in the CPP centre	[PLN]	13,305,146.0

Summarising, the proceeding way at the calculation of economic effects for the individual research scenarios is presented in Table 4.26.

Way of cost savings consideration for the individual scenarios (own study)

Table 4.26.

Scenario	Savings in mining processes and dirt management	Additional savings in operating costs (5%)
As it is now	No	No
Pessimistic	Yes	No
Baseline	Yes	Yes
Optimistic	Yes	Yes
Full optimisation effects	Yes	Yes

4.5.1.1. Other technical-economic assumptions of the assessment model

The period of analysis was a resultant of the forecast based on including the next faces in thin coal seams in the mining process, in accordance with the natural advance of mining fronts in the mine. An assumption was made that the analysis would continue till a recovery of resources in the seams planned for mining by the plough technique and would depend on the estimated advance in the next scenarios. In this way the number of the plough faces, to be started up in this period, was estimated at 33 and the time of analysis - depending on the advance and scenario - would fluctuate between 23 and 26 consecutive yearly periods.

Revenues from coal sales

The sales revenues were calculated as a product of an average price of saleable coal with the 21 GJ calorific value (20% ash content and 0.7% sulphur content) determined as PLN 200/Mg and as an average annual production of saleable coal.

The amount of coal per year corresponded to a product of an average daily coal output from the plough-equipped longwalls during 300 consecutive working days.

Operating costs

The economic efficiency was assessed using the category of cash operating costs. These costs were estimated as the total costs less the depreciation cost in the part assigned to the fixed assets (infrastructure) of the model plough face. An assumption was made that the operating costs of mining from the model plough face in the entire life cycle would comprise:

- the costs of development operations,
- the mining costs of plough mining in the entire life cycle (including equipment installation and liquidation),
- the costs of workings reconstruction,
- the costs of haulage and horizontal transport,
- the costs of shaft winding,
- the costs of auxiliary processes, such as:
  - the ventilation,
  - the dewatering,
  - the OHS preventive actions,
- the costs of services:
  - mechanical,
  - electrical,
  - other,
- other costs of the underground part of mine,
- the costs of coal preparation and dirt management on the surface,
- other costs of mine surface operation,
- the costs of administration,
- the costs of sales.

The costs of driving roadways, opening, and development were determined by means of the ratios for a determined amount of recoverable reserves of the model plough face. The costs of equipment installation and liquidation were estimated based on the costs of three plough faces, which completed their operation. The costs of coal mining from the plough-equipped longwalls included the cost of the mining centre and of driving the roadways. The other cost centres were illustrated via a margin on a tonne of saleable coal. In this way the operating costs, directly related to coal cutting in faces were reproduced, but also the costs of accompanying (supporting) processes, including the maintenance of the permanent mine infrastructure.

Discount rate

In the calculations an assumption was made of invariable discount rate on the level of the weighted Company equity costs plus the value of additional reward (2 pct. points) due to a higher expert assessment of the risk for the plough systems. After rounding this value was determined as 10.0%. The consideration of own equity cost as well as of outside funds corresponds to the assumptions of the free cash flow to firm calculation in the FCFF method.

Other economic-financial assumptions

To keep transparency of the economic efficiency calculation and because of the specific nature of carried out research work, the net working capital (NWC) requirement and its changes over time were not calculated on the expenditure side (no significant fluctuations in the NWC requirement level, year to year, were assumed. Also no charges on the Mining Plant Closure Fund were calculated or the VAT influence.

4.5.2. Assessment results and conclusions

Based on the carried out analysis of economic efficiency Table 4.27 presents the outcomes of the NPV estimation for a given scenario as well as the value of discounted savings. The Net Present Value for individual scenarios ranges between PLN 1,496.0 and 1,777.4 million, while total net benefits range from PLN 74.6 to 281.5 million.

Comparison of economic efficiency assessment results (own study)

Table 4.27.

Scenario type	Specification	NPV [PLN million]	Savings against the current situation [PLN million]
"as it is"	Current situation	1,495.9	0.0
"to be"	Pessimistic scenario	1,570.5	74.6
"to be"	Baseline scenario	1,580.6	84.7
"to be"	Optimistic scenario	1,753.5	257.6
"to be"	Scenario of full optimisation effects	1,777.4	281.5

Table 4.28 specifies other technical and economic-financial variables originating from the models of economic efficiency assessment for the analysed research scenarios. It is possible to notice that the amount of mined coal, the

volume of rock left in the deposit, and the level of revenues and costs remain constant. Instead, there are differences in values of identified savings and, as a result, of accumulated profit, both EBIT and NOPAT. The spread of results, obtained between the '*as it is*' current situation and the scenario of full optimisation effects for 23÷26 (depending on the scenario) consecutive yearly periods of analysis are presented below:

- the amount of rock left in the deposit: 4.3 million Mg,
- the value of total savings: PLN 170.0 million,
- the level of accumulated EBIT: PLN 170.0 million,
- the value of accumulated NOPAT: PLN 384.1 million.

Specification of other decision and output variables in the case of the assessment models for individual research scenarios (own study*)*

Table 4.28.

Specification		As is	Scenario			
			pessimistic	baseline	optimistic	full effects
Coal output	Mg	66.8	66.8	66.8	66.8	66.8
Amount of avoided rock	Mg	0.0	4.28	4.28	4.28	4.28
Total revenues	PLN million	13,369.2	13,369.2	13,369.2	13,369.2	13,369.2
Total savings	PLN million	0.0	84.2	134.9	170.0	170.0
Total operating costs	PLN million	12,713.2	12,713.2	12,713.2	12,713.2	12,713.2
Accumulated EBIT	PLN million	656.0	790.9	791.0	826.1	826.1
Accumulated NOPAT	PLN million	406.5	546.9	551.4	787.2	790.7

Figs. 4.57 and 4.58 present a variability of the accumulated net cash flows for two extreme scenarios - *as it is* and full optimisation effects in the analysis consecutive periods. The difference in the average free net cash flows is approx. PLN 29.8 million, while on the discounted level - approx. PLN 27.7 million.

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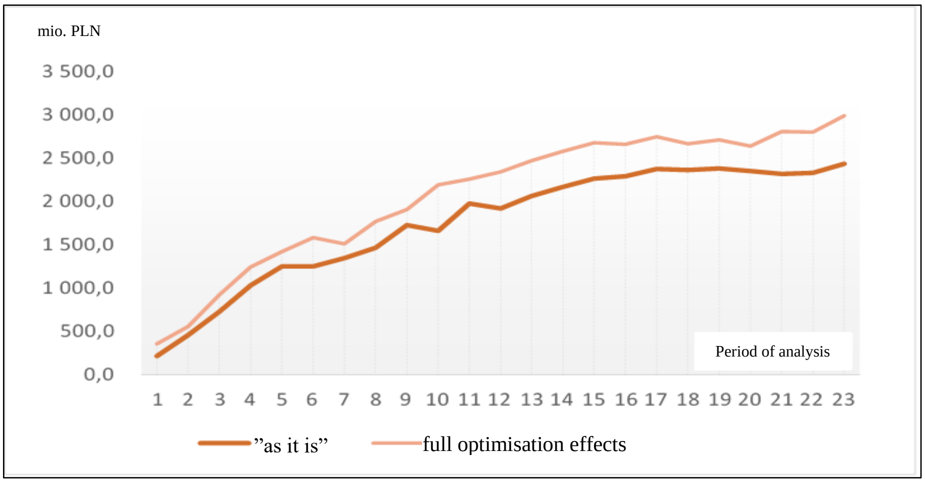

Fig. 4.57. Accumulated free cash flows (FCFF) for the “*as it is”* and full optimisation effects scenarios (*own study)*

Tables 4.29÷4.33 present the results of economic efficiency assessment for all the analysed scenarios. These tables present the most important variables of assessment models.

Results of economic analysis: baseline “*as it is*“ scenario (own study)

Table 4.29.

Periods of calculation	Units	In total (0-25)	0 2016	1 2017	2 2018	3 2019	4 2020	5 2021
Total net output from longwalls and roadways	[Mg]	66 846 115	4 923 351	5 856 113	5 293 245	4 750 332	3 394 128	2 375 609
Revemes	[PLN]	13 369 223 051	984 670 173	1 171 222 536	1 058 648 903	950 066 429	678 825 618	475 121 812
Mining costs of exploitation by means of a plough technique	[PLN]	4 552 784 171	317 492 036	392 290 784	292 836 528	191 750 305	137 724 983	261 571 852
Amount of avoided dirt	[Mg]	-	-	-	-	-	-	-
Total savings	[PLN]	-	-	-	-	-	-	-
Total costs	[PLN]	12 713 177 179	919 257 370	1 109 978 150	939 063 332	767 147 411	548 084 505	554 214 732
EBIDA	[PLN]	2 855 600 542	227 465 946	254 133 529	293 761 383	338 911 773	242 145 003	- 739 659
Depreciation	[PLN]	2 199 554 670	162 053 143	192 889 142	174 175 812	155 992 756	111 403 890	78 353 262
EBIT	[PLN]	656 045 872	65 412 803	61 244 386	119 585 571	182 919 018	130 741 113	- 79 092 921
Tax	[PLN]	249 524 850	12 428 433	11 636 433	22 721 259	34 754 613	24 840 812	-
NOPAT	[PLN]	406 521 022	52 984 370	49 607 953	96 864 313	148 164 404	105 900 302	- 79 092 921
FCFF	[PLN]	2 606 075 692	215 037 513	242 497 095	271 040 124	304 157 160	217 304 192	- 739 659
Discount rate	[%]	10	100%	91%	83%	75%	68%	62%
DCF	thousand PLN	1 495 857 175	215 037 513	220 451 905	224 000 103	228 517 776	148 421 687	- 459 270

Periods of calculation	Units	6 2022	7 2023	8 2024	9 2025	10 2026	11 2027	12 2028
Total net output from longwalls and roadways	[Mg]	3 702 082	4 039 068	4 556 358	2 593 628	4 916 762	2 205 318	3 168 833
Revemes	[PLN]	740 416 302	807 813 686	911 271 542	518 725 682	983 352 409	441 063 678	633 766 503
Mining costs of exploitation by means of a plough technique	[PLN]	309 607 907	326 927 257	218 804 601	346 251 667	199 482 495	301 671 578	200 878 874
Amount of avoided dirt	[Mg]	-	-	-	-	-	-	-
Total savings	[PLN]	-	-	-	-	-	-	-
Total costs	[PLN]	763 710 021	823 654 935	773 632 216	668 007 246	795 640 776	570 875 922	584 543 672
EBIDA	[PLN]	98 673 682	117 318 718	287 467 122	- 63 579 610	349 211 793	- 57 247 843	153 270 038
Depreciation	[PLN]	121 967 401	133 159 966	149 827 796	85 701 953	161 500 159	72 564 402	104 047 206
EBIT	[PLN]	- 23 293 719	- 15 841 248	137 639 326	-149 281 564	187 711 633	-129 812 244	49 222 831
Tax	[PLN]	-	-	26 151 472	-	35 665 210	-	9 352 338
NOPAT	[PLN]	- 23 293 719	- 15 841 248	111 487 854	-149 281 564	152 046 423	-129 812 244	39 870 493
FCFF	[PLN]	98 673 682	117 318 718	261 315 650	- 63 579 610	313 546 582	- 57 247 843	143 917 700
Discount rate	[%]	56%	51%	47%	42%	39%	35%	32%
DCF	thousand PLN	55 698 721	60 203 052	121 905 679	- 26 963 961	120 885 781	- 20 065 020	45 856 614

Periods of calculation	Units	13	14	15	16	17	18
		2029	2030	2031	2032	2033	2034
Total net output from longwalls and roadways	[Mg]	1 758 287	1 312 242	967 882	1 480 455	626 671	898 453
Revemes	[PLN]	351 657 363	262 448 327	193 576 374	296 090 986	125 334 165	179 690 658
Mining costs of exploitation by means of a plough technique	[PLN]	79 213 545	36 990 821	81 564 622	73 693 761	78 840 885	81 604 327
Amount of avoided dirt	[Mg]	-	-	-	-	-	-
Total savings	[PLN]	-	-	-	-	-	-
Total costs	[PLN]	292 021 467	195 687 157	199 160 878	253 889 338	157 146 134	192 145 348
EBIDA	[PLN]	117 363 158	109 835 255	26 224 198	90 878 242	- 11 065 318	17 168 908
Depreciation	[PLN]	57 727 262	43 074 085	31 808 701	48 676 594	20 746 651	29 623 598
EBIT	[PLN]	59 635 896	66 761 171	- 5 584 504	42 201 648	- 31 811 969	- 12 454 690
Tax	[PLN]	11 330 820	12 684 622	-	8 018 313	-	-
NOPAT	[PLN]	48 305 076	54 076 548	- 5 584 504	34 183 335	- 31 811 969	- 12 454 690
FCFF	[PLN]	106 032 338	97 150 633	26 224 198	82 859 929	- 11 065 318	17 168 908
Discount rate	[%]	29%	26%	24%	22%	20%	18%
DCF	thousand PLN	30 713 791	25 582 798	6 277 864	18 032 735	- 2 189 214	3 087 979

Periods of calculation	Units	19	20	21	22	23	24	25
		2035	2036	2037	2038	2039	2040	2041
Total net output from longwalls and roadways	[Mg]	1 058 705	757 548	893 600	1 510 020	715 713	2 218 130	873 584
Revemes	[PLN]	211 740 955	151 509 625	178 720 057	302 003 937	143 142 563	443 626 037	174 716 732
Mining costs of exploitation by means of a plough technique	[PLN]	147 490 606	113 768 038	84 714 365	52 679 425	124 429 765	84 809 435	15 693 711
Amount of avoided dirt	[Mg]	-	-	-	-	-	-	-
Total savings	[PLN]	-	-	-	-	-	-	-
Total costs	[PLN]	277 329 620	208 764 289	195 201 232	236 010 608	213 888 012	353 284 236	120 838 575
EBIDA	[PLN]	- 30 710 587	- 32 151 560	13 020 415	115 609 595	- 47 049 085	163 167 226	82 518 221
Depreciation	[PLN]	34 878 078	25 103 104	29 501 590	49 616 266	23 696 364	72 825 425	28 640 064
EBIT	[PLN]	- 65 588 665	- 57 254 664	- 16 481 175	65 993 329	- 70 745 449	90 341 801	53 878 157
Tax	[PLN]	-	-	-	12 538 732	-	17 164 942	10 236 850
NOPAT	[PLN]	- 65 588 665	- 57 254 664	- 16 481 175	53 454 596	- 70 745 449	73 176 859	43 641 308
FCFF	[PLN]	- 30 710 587	- 32 151 560	13 020 415	103 070 862	- 47 049 085	146 002 284	72 281 371
Discount rate	[%]	16%	15%	14%	12%	11%	10%	9%
DCF	thousand PLN	- 5 021 426	- 4 779 125	1 759 456	12 661 840	- 5 254 355	14 822 969	6 671 281

Results of economic analysis for the pessimistic scenario (own study)

Table 4.30.

Periods of calculation	Units	In total (0-24)	0	1	2	3	4	5
			2016	2017	2018	2019	2020	2021
Total net output from longwalls and roadways	[Mg]	66 846 115	5 578 996	5 200 467	5 293 245	5 194 370	3 482 388	2 548 939
Revemes	[PLN]	13 369 223 051	1 115 799 285	1 040 093 424	1 058 648 903	1 038 873 944	696 477 586	509 787 719
Mining costs of exploitation by means of a plough technique	[PLN]	4 552 784 230	312 363 735	405 431 625	296 185 745	232 879 515	133 089 960	184 780 110
Amount of avoided dirt	[Mg]	4 278 727	382 085	332 224	343 318	335 904	237 794	134 941
Total savings	[PLN]	84 191 906	7 518 230	6 537 125	6 755 416	6 609 530	4 679 037	2 655 214
Total costs	[PLN]	12 713 177 238	992 937 620	1 044 679 852	942 408 376	862 933 567	555 399 311	494 367 890
EBIDA	[PLN]	2 939 792 389	313 920 731	173 378 004	297 171 463	353 185 110	260 147 639	101 836 648
Depreciation	[PLN]	2 199 554 670	183 540 836	171 427 307	174 175 520	170 635 203	114 390 327	83 761 606
EBIT	[PLN]	740 237 719	130 379 895	1 950 697	122 995 943	182 549 908	145 757 312	18 075 042
Tax	[PLN]	232 318 423	24 772 180	370 632	23 369 229	34 684 482	27 693 889	3 434 258
NOPAT	[PLN]	507 919 296	105 607 715	1 580 065	99 626 714	147 865 425	118 063 423	14 640 784
FCFF	[PLN]	2 707 473 965	289 148 551	173 007 372	273 802 234	318 500 628	232 453 749	98 402 390
Discount rate	[%]		100%	91%	83%	75%	68%	62%
DCF	thousand PLN	1 557 456 336	289 148 551	157 279 429	226 282 838	239 294 236	158 769 039	61 100 142

Periods of calculation	Units	6	7	8	9	10	11	12
		2022	2023	2024	2025	2026	2027	2028
Total net output from longwalls and roadways	[Mg]	2 707 085	4 367 185	5 706 750	2 454 170	3 680 199	2 724 209	2 321 016
Revemes	[PLN]	541 417 017	873 436 927	1 141 350 082	490 834 031	736 039 899	544 841 737	464 203 212
Mining costs of exploitation by means of a plough technique	[PLN]	302 000 710	382 871 387	262 705 584	283 869 009	228 513 854	206 401 231	243 175 673
Amount of avoided dirt	[Mg]	154 553	289 739	385 587	170 771	234 913	155 495	123 641
Total savings	[PLN]	3 041 116	5 701 144	7 587 137	3 360 233	4 622 349	3 059 655	2 432 869
Total costs	[PLN]	635 455 722	922 234 913	957 386 817	587 083 626	674 869 026	538 042 548	524 947 199
EBIDA	[PLN]	- 1 712 941	101 040 322	379 190 757	- 11 882 384	186 685 316	99 433 659	17 951 308
Depreciation	[PLN]	89 284 647	144 137 164	187 640 356	81 006 978	120 892 093	89 574 816	76 262 426
EBIT	[PLN]	- 90 997 588	- 43 096 842	191 550 401	- 92 889 362	65 793 222	9 858 844	- 58 311 118
Tax	[PLN]	-	-	36 394 576	-	12 500 712	1 873 180	-
NOPAT	[PLN]	- 90 997 588	- 43 096 842	155 155 825	- 92 889 362	53 292 510	7 985 663	- 58 311 118
FCFF	[PLN]	- 1 712 941	101 040 322	342 796 181	- 11 882 384	174 184 604	97 560 479	17 951 308
Discount rate	[%]	56%	51%	47%	42%	39%	35%	32%
DCF	thousand PLN	- 966 911	51 849 662	159 916 948	- 5 039 291	67 155 705	34 194 353	5 719 840

Periods of calculation	Units	13 2029	14 2030	15 2031	16 2032	17 2033	18 2034
Total net output from longwalls and roadways	[Mg]	1 755 229	964 662	1 834 327	872 449	1 514 335	618 795
Revemes	[PLN]	351 045 794	192 932 405	366 865 372	174 489 770	302 867 067	123 758 971
Mining costs of exploitation by means of a plough technique	[PLN]	84 828 974	48 248 507	73 340 562	78 144 155	84 430 612	145 241 030
Amount of avoided dirt	[Mg]	91 229	55 657	100 990	61 644	102 725	41 086
Total savings	[PLN]	1 795 095	1 095 159	1 987 174	1 212 958	2 021 303	808 443
Total costs	[PLN]	296 390 877	165 853 413	294 790 517	185 283 126	269 193 073	223 123 035
EBIDA	[PLN]	114 015 567	59 905 024	134 246 520	19 171 606	85 516 876	- 78 030 451
Depreciation	[PLN]	57 565 555	31 730 873	60 184 491	28 752 004	49 821 580	20 525 169
EBIT	[PLN]	56 450 012	28 174 151	74 062 029	- 9 580 398	35 695 296	- 98 555 621
Tax	[PLN]	10 725 502	5 353 089	14 071 786	-	6 782 106	-
NOPAT	[PLN]	45 724 510	22 821 062	59 990 244	- 9 580 398	28 913 190	- 98 555 621
FCFF	[PLN]	103 290 065	54 551 935	120 174 735	19 171 606	78 734 770	- 78 030 451
Discount rate	[%]	29%	26%	24%	22%	20%	18%
DCF	thousand PLN	29 919 453	14 365 229	28 768 876	4 172 300	15 577 254	- 14 034 463

Periods of calculation	Units	19 2035	20 2035	21 2035	22 2035	23 2035	24 2035
Total net output from longwalls and roadways	[Mg]	801 375	1 014 878	1 169 726	1 848 200	1 723 054	1 470 067
Revemes	[PLN]	160 274 942	202 975 637	233 945 106	369 640 014	344 610 791	294 013 415
Mining costs of exploitation by means of a plough technique	[PLN]	95 827 068	104 912 864	121 200 141	98 828 197	118 299 516	25 214 464
Amount of avoided dirt	[Mg]	61 612	74 180	82 214	123 322	102 759	100 345
Total savings	[PLN]	1 212 327	1 459 625	1 617 716	2 426 593	2 021 976	1 974 482
Total costs	[PLN]	194 139 074	229 954 934	266 451 241	324 696 295	328 815 510	201 739 676
EBIDA	[PLN]	- 6 249 007	7 955 032	7 772 893	108 202 010	74 525 832	142 414 853
Depreciation	[PLN]	26 402 797	33 474 704	38 661 313	60 831 698	56 708 574	48 166 632
EBIT	[PLN]	- 32 651 804	- 25 519 672	- 30 888 419	47 370 312	17 817 258	94 248 221
Tax	[PLN]	-	-	-	9 000 359	3 385 279	17 907 162
NOPAT	[PLN]	- 32 651 804	- 25 519 672	- 30 888 419	38 369 953	14 431 979	76 341 059
FCFF	[PLN]	- 6 249 007	7 955 032	7 772 893	99 201 650	71 140 553	124 507 691
Discount rate	[%]	16%	15%	14%	12%	11%	10%
DCF	thousand PLN	- 1 021 763	1 182 465	1 050 356	12 186 523	7 944 846	12 640 718

Results of economic analysis for the baseline scenario (own study)

Table 4.31.

Periods of calculation	Units	In total (0-24)	0 2016	1 2017	2 2018	3 2019	4 2020	5 2021
Total net output from longwalls and roadways	[Mg]	66 846 115	5 578 996	5 200 467	5 293 245	5 194 370	3 482 388	2 548 939
Revemes	[PLN]	13 369 223 051	1 115 799 285	1 040 093 424	1 058 648 903	1 038 873 944	696 477 586	509 787 719
Mining costs of exploitation by means of a plough technique	[PLN]	4 552 784 230	312 363 735	405 431 625	296 185 745	232 879 515	133 089 960	184 780 110
Amount of avoided dirt	[Mg]	4 278 727	382 085	332 224	343 318	335 904	237 794	134 941
Total savings	[PLN]	134 937 521	12 049 748	10 477 295	10 827 159	10 593 342	7 499 267	4 255 610
Total costs	[PLN]	12 713 177 238	992 937 620	1 044 679 852	942 408 376	862 933 567	555 399 311	494 367 890
EBIDA	[PLN]	2 990 538 004	318 452 249	177 318 174	301 243 205	357 168 922	262 967 869	103 437 045
Depreciation	[PLN]	2 199 554 670	183 540 836	171 427 307	174 175 520	170 635 203	114 390 327	83 761 606
EBIT	[PLN]	790 983 334	134 911 413	5 890 867	127 067 686	186 533 719	148 577 542	19 675 439
Tax	[PLN]	239 572 754	25 633 169	1 119 265	24 142 860	35 441 407	28 229 733	3 738 333
NOPAT	[PLN]	551 410 580	109 278 245	4 771 603	102 924 825	151 092 313	120 347 809	15 937 105
FCFF	[PLN]	2 750 965 249	292 819 081	176 198 910	277 100 345	321 727 515	234 738 136	99 698 712
Discount rate	[%]		100%	91%	83%	75%	68%	62%
DCF	thousand PLN	1 580 593 980	292 819 081	160 180 827	229 008 550	241 718 644	160 329 306	61 905 056

Periods of calculation	Units	6 2022	7 2023	8 2024	9 2025	10 2026	11 2027	12 2028	13 2029
Total net output from longwalls and roadways	[Mg]	2 707 085	4 367 185	5 706 750	2 454 170	3 680 199	2 724 209	2 321 016	1 755 229
Revemes	[PLN]	541 417 017	873 436 927	1 141 350 082	490 834 031	736 039 899	544 841 737	464 203 212	351 045 794
Mining costs of exploitation by means of a plough technique	[PLN]	302 000 710	382 871 387	262 705 584	283 869 009	228 513 854	206 401 231	243 175 673	84 828 974
Amount of avoided dirt	[Mg]	154 553	289 739	385 587	170 771	234 913	155 495	123 641	91 229
Total savings	[PLN]	4 874 111	9 137 437	12 160 188	5 385 571	7 408 413	4 903 824	3 899 250	2 877 066
Total costs	[PLN]	635 455 722	922 234 913	957 386 817	587 083 626	674 869 026	538 042 548	524 947 199	296 390 877
EBIDA	[PLN]	120 054	104 476 615	383 763 809	- 9 857 045	189 471 379	101 277 828	19 417 690	115 097 538
Depreciation	[PLN]	89 284 647	144 137 164	187 640 356	81 006 978	120 892 093	89 574 816	76 262 426	57 565 555
EBIT	[PLN]	89 164 594	- 39 660 549	196 123 453	- 90 864 024	68 579 286	11 703 012	- 56 844 736	57 531 983
Tax	[PLN]	-	-	37 263 456	-	13 030 064	2 223 572	-	10 931 077
NOPAT	[PLN]	89 164 594	- 39 660 549	158 859 997	- 90 864 024	55 549 221	9 479 440	- 56 844 736	46 600 906
FCFF	[PLN]	120 054	104 476 615	346 500 353	- 9 857 045	176 441 315	99 054 256	19 417 690	104 166 462
Discount rate	[%]	6%	51%	47%	42%	39%	35%	32%	29%
DCF	thousand PLN	67 767	53 613 023	161 644 972	- 4 180 349	68 025 765	34 717 912	6 187 074	30 173 313

Periods of calculation	Units	14	15	16	17	18	19
		2030	2031	2032	2033	2034	2035
Total net output from longwalls and roadways	[Mg]	964 662	1 834 327	872 449	1 514 335	618 795	801 375
Revemes	[PLN]	192 932 405	366 865 372	174 489 770	302 867 067	123 758 971	160 274 942
Mining costs of exploitation by means of a plough technique	[PLN]	48 248 507	73 340 562	78 144 155	84 430 612	145 241 030	95 827 068
Amount of avoided dirt	[Mg]	55 657	100 990	61 644	102 725	41 086	61 612
Total savings	[PLN]	1 755 252	3 184 919	1 944 053	3 239 618	1 295 722	1 943 042
Total costs	[PLN]	165 853 413	294 790 517	185 283 126	269 193 073	223 123 035	194 139 074
EBIDA	[PLN]	60 565 117	135 444 265	19 902 701	86 735 191	- 77 543 173	- 5 518 292
Depreciation	[PLN]	31 730 873	60 184 491	28 752 004	49 821 580	20 525 169	26 402 797
EBIT	[PLN]	28 834 244	75 259 774	- 8 849 303	36 913 611	- 98 068 342	- 31 921 089
Tax	[PLN]	5 478 506	14 299 357	-	7 013 586	-	-
NOPAT	[PLN]	23 355 738	60 960 417	- 8 849 303	29 900 025	- 98 068 342	- 31 921 089
FCFF	[PLN]	55 086 611	121 144 908	19 902 701	79 721 605	- 77 543 173	- 5 518 292
Discount rate	[%]	26%	24%	22%	20%	18%	16%
DCF	thousand PLN	14 506 026	29 001 128	4 331 408	15 772 495	- 13 946 821	- 902 285

Periods of calculation	Units	20	21	22	23	24
		2035	2035	2035	2035	2035
Total net output from longwalls and roadways	[Mg]	1 014 878	1 169 726	1 848 200	1 723 054	1 470 067
Revemes	[PLN]	202 975 637	233 945 106	369 640 014	344 610 791	294 013 415
Mining costs of exploitation by means of a plough technique	[PLN]	104 912 864	121 200 141	98 828 197	118 299 516	25 214 464
Amount of avoided dirt	[Mg]	74 180	82 214	123 322	102 759	100 345
Total savings	[PLN]	2 339 395	2 592 773	3 889 192	3 240 697	3 164 576
Total costs	[PLN]	229 954 934	266 451 241	324 696 295	328 815 510	201 739 676
EBIDA	[PLN]	8 834 803	8 747 951	109 664 608	75 744 553	143 604 947
Depreciation	[PLN]	33 474 704	38 661 313	60 831 698	56 708 574	48 166 632
EBIT	[PLN]	- 24 639 901	- 29 913 361	48 832 911	19 035 979	95 438 315
Tax	[PLN]	-	-	9 278 253	3 616 836	18 133 280
NOPAT	[PLN]	- 24 639 901	- 29 913 361	39 554 658	15 419 143	77 305 035
FCFF	[PLN]	8 834 803	8 747 951	100 386 355	72 127 717	125 471 667
Discount rate	[%]	15%	14%	12%	11%	10%
DCF	thousand PLN	1 313 237	1 182 116	12 332 060	8 055 091	12 738 586

Results of economic analysis for the optimistic scenario (own study)

Table 4.32.

Periods of calculation	Units	In total (0-23)	0	1	2	3	4	5
			2016	2017	2018	2019	2020	2021
Total net output from longwalls and roadways	[Mg]	66 846 115	6 229 762	5 250 907	5 290 933	5 038 849	3 680 960	2 335 567
Revemes	[PLN]	13 369 223 051	1 245 952 411	1 050 181 432	1 058 186 596	1 007 769 897	736 191 981	467 113 355
Mining costs of exploitation by means of a plough technique	[PLN]	4 552 784 217	345 307 277	395 135 722	304 342 003	227 995 782	136 728 921	214 184 092
Amount of avoided dirt	[Mg]	4 278 727	423 803	332 049	343 318	324 587	247 216	127 057
Total savings	[PLN]	170 032 088	16 841 483	13 195 267	13 643 088	12 898 751	9 824 089	5 049 114
Total costs	[PLN]	12 713 177 224	1 105 402 649	1 039 590 765	950 178 357	839 228 189	582 898 315	499 861 449
EBIDA	[PLN]	3 025 632 584	362 350 805	196 806 408	295 743 500	346 969 777	284 015 367	49 191 296
Depreciation	[PLN]	2 199 554 670	204 959 560	173 020 472	174 092 173	165 529 317	120 897 613	76 890 276
EBIT	[PLN]	826 077 914	157 391 245	23 785 935	121 651 327	181 440 460	163 117 755	- 27 698 980
Tax	[PLN]	38 902 957	2 361 402	2 210 922	4 317 039	6 603 377	4 719 754	-
NOPAT	[PLN]	787 174 957	155 029 843	21 575 013	117 334 287	174 837 083	158 398 001	- 27 698 980
FCFF	[PLN]	2 986 729 627	359 989 403	194 595 485	291 426 461	340 366 400	279 295 613	49 191 296
Discount rate	[%]		100%	91%	83%	75%	68%	62%
DCF	thousand PLN	1 753 477 422	359 989 403	176 904 987	240 848 315	255 722 314	190 762 662	30 543 925

Periods of calculation	Units	6	7	8	9	10	11	12	13
		2022	2023	2024	2025	2026	2027	2028	2029
Total net output from longwalls and roadways	[Mg]	3 178 149	5 061 486	4 220 571	3 668 207	3 092 784	3 107 046	2 836 040	987 649
Revemes	[PLN]	635 629 892	1 012 297 121	844 114 120	733 641 440	618 556 842	621 409 118	567 207 904	197 529 855
Mining costs of exploitation by means of a plough technique	[PLN]	346 837 673	326 635 856	314 987 156	242 918 683	221 374 874	216 028 062	253 589 878	53 861 726
Amount of avoided dirt	[Mg]	189 686	340 449	275 021	260 931	190 262	168 079	149 736	55 657
Total savings	[PLN]	7 537 922	13 529 077	10 929 029	10 369 103	7 560 818	6 679 279	5 950 336	2 211 758
Total costs	[PLN]	739 532 506	946 703 690	831 814 876	691 331 542	596 118 029	593 347 280	597 542 220	174 022 975
EBIDA	[PLN]	8 541 556	245 821 695	162 216 519	173 423 002	131 569 835	136 839 042	68 776 736	58 188 409
Depreciation	[PLN]	104 906 248	166 699 186	138 988 246	120 744 000	101 570 204	102 097 925	93 160 715	32 469 771
EBIT	[PLN]	- 96 364 693	79 122 509	23 228 273	52 679 002	29 999 631	34 741 117	- 24 383 980	25 718 638
Tax	[PLN]	-	-	4 968 780	-	6 776 390	-	-	2 152 856
NOPAT	[PLN]	- 96 364 693	79 122 509	18 259 493	52 679 002	23 223 241	34 741 117	- 24 383 980	23 565 782
FCFF	[PLN]	8 541 556	245 821 695	157 247 739	173 423 002	124 793 445	136 839 042	68 776 736	56 035 554
Discount rate	[%]	56%	51%	47%	42%	39%	35%	32%	29%
DCF	thousand PLN	4 821 485	126 145 398	73 357 231	73 548 282	48 113 275	47 961 249	21 914 388	16 231 504

Periods of calculation	Units	14	15	16	17	18	19	20	21	22	23
		2030	2031	2032	2033	2034	2035	2035	2035	2035	2035
Total net output from longwalls	[Mg]	1 834 327	1 481 911	640 787	1 162 083	1 050 991	512 476	1 441 042	1 608 550	1 709 685	1 425 353
Revemes	[PLN]	366 865 372	296 382 200	128 157 459	232 416 584	210 198 194	102 495 171	288 208 377	321 710 011	341 937 100	285 070 618
Mining costs of exploitation by means of a plough technique	[PLN]	67 566 936	85 237 094	109 048 617	132 045 901	94 240 332	149 902 285	63 143 280	112 507 783	107 380 360	31 783 925
Amount of avoided dirt	[Mg]	100 990	102 740	41 086	82 166	82 149	33 105	102 768	102 769	102 759	100 345
Total savings	[PLN]	4 013 253	4 082 770	1 632 713	3 265 200	3 264 519	1 315 566	4 083 878	4 083 918	4 083 538	3 987 619
Total costs	[PLN]	289 541 734	265 606 939	189 632 451	274 306 070	222 889 564	216 103 921	238 974 610	309 318 485	315 893 980	203 336 629
EBIDA	[PLN]	141 558 119	83 582 293	- 38 592 260	- 358 505	25 180 040	- 95 175 504	100 728 665	69 435 408	86 369 420	132 450 961
Depreciation	[PLN]	60 221 229	48 724 262	21 250 019	38 265 782	34 606 891	17 117 680	47 411 021	52 959 964	56 242 762	46 729 353
EBIT	[PLN]	81 336 890	34 858 031	- 59 842 278	- 38 624 287	- 9 426 851	- 112 293 184	53 317 644	16 475 444	30 126 658	85 721 608
Tax	[PLN]	2 410 078	-	-	-	-	-	-	-	2 382 359	-
NOPAT	[PLN]	78 926 812	34 858 031	- 59 842 278	- 38 624 287	- 9 426 851	- 112 293 184	53 317 644	16 475 444	27 744 299	85 721 608
FCFF	[PLN]	139 148 041	83 582 293	- 38 592 260	- 358 505	25 180 040	- 95 175 504	100 728 665	69 435 408	83 987 061	132 450 961
Discount rate	[%]	26%	24%	22%	20%	18%	16%	15%	14%	12%	11%
DCF	thousand PLN	36 642 028	20 008 936	- 8 398 800	- 70 928	4 528 852	- 15 561 955	14 972 674	9 382 846	10 317 472	14 791 879

Results of economic analysis for the full optimisation effects scenario (own study)

Table 4.33.

Periods of calculation	Units	In total (0-22)	0	1	2	3	4	5	6
			2016	2017	2018	2019	2020	2021	2022
Total net output from longwalls and roadways	[Mg]	66 846 115	6 229 762	5 250 907	5 844 998	5 106 843	3 106 043	2 798 584	2 613 740
Revemes	[PLN]	13 369 223 051	1 245 952 411	1 050 181 432	1 168 999 575	1 021 368 573	621 208 503	559 716 719	522 748 069
Mining costs of exploitation by means of a plough technique	[PLN]	4 552 784 217	349 388 623	389 269 082	292 307 910	253 734 338	172 923 997	156 564 818	362 104 584
Amount of avoided dirt	[Mg]	4 278 727	423 803	332 049	381 960	319 927	213 233	158 821	152 614
Total savings	[PLN]	170 032 088	16 841 483	13 195 267	15 178 698	12 713 580	8 473 650	6 311 393	6 064 727
Total costs	[PLN]	12 713 177 224	1 109 483 995	1 033 724 124	1 004 550 945	873 271 800	551 465 175	495 972 552	686 994 206
EBIDA	[PLN]	3 025 632 584	358 269 459	202 673 048	371 864 625	328 577 298	180 376 004	161 985 751	- 71 770 162
Depreciation	[PLN]	2 199 554 670	204 959 560	173 020 472	192 237 298	167 766 945	102 159 025	91 930 191	86 411 249
EBIT	[PLN]	826 077 914	153 309 899	29 652 576	179 627 327	160 810 353	78 216 978	70 055 560	- 158 181 410
Tax	[PLN]	35 426 991	2 361 402	2 210 922	4 317 039	6 603 377	4 719 754	-	-
NOPAT	[PLN]	790 650 923	150 948 496	27 441 654	175 310 288	154 206 977	73 497 224	70 055 560	- 158 181 410
FCFF	[PLN]	2 990 205 593	355 908 057	200 462 126	367 547 586	321 973 921	175 656 249	161 985 751	- 71 770 162
Discount rate	[%]		100%	91%	83%	75%	68%	62%	56%
DCF	thousand PLN	1 777 350 658	355 908 057	182 238 296	303 758 336	241 903 773	119 975 582	100 580 407	- 40 512 385

Periods of calculation	Units	7	8	9	10	11	12	13	14
		2023	2024	2025	2026	2027	2028	2029	2030
Total net output from longwalls and roadways	[Mg]	5 066 631	4 362 394	4 632 315	3 029 436	2 633 968	2 857 493	1 312 242	1 544 159
Revemes	[PLN]	1 013 326 110	872 478 876	926 463 016	605 887 222	526 793 516	571 498 613	262 448 327	308 831 797
Mining costs of exploitation by means of a plough technique	[PLN]	315 378 321	347 349 361	243 735 573	277 589 020	214 399 326	197 166 709	38 403 404	74 015 235
Amount of avoided dirt	[Mg]	342 835	280 289	326 114	178 422	140 487	152 048	75 743	91 591
Total savings	[PLN]	13 623 903	11 138 394	12 959 410	7 090 285	5 582 789	6 042 223	3 009 939	3 639 734
Total costs	[PLN]	936 004 058	883 105 242	807 170 205	645 625 198	534 661 057	543 210 449	197 099 740	261 682 131
EBIDA	[PLN]	257 809 522	144 279 976	284 532 781	66 909 905	84 295 280	128 160 096	111 432 612	101 540 776
Depreciation	[PLN]	166 863 567	143 767 948	152 280 560	99 557 596	86 580 032	93 829 708	43 074 085	50 751 376
EBIT	[PLN]	90 945 955	512 028	132 252 221	- 32 647 691	- 2 284 752	34 330 387	68 358 527	50 789 400
Tax	[PLN]	-	4 968 780	-	-	-	1 776 944	2 152 856	2 410 078
NOPAT	[PLN]	90 945 955	- 4 456 751	132 252 221	- 32 647 691	- 2 284 752	32 553 443	66 205 671	48 379 322
FCFF	[PLN]	257 809 522	139 311 196	284 532 781	66 909 905	84 295 280	126 383 151	109 279 756	99 130 698
Discount rate	[%]	51%	47%	42%	39%	35%	32%	29%	26%
DCF	thousand PLN	132 297 049	64 989 701	120 669 675	25 796 665	29 544 982	40 269 567	31 654 453	26 104 211

Periods of calculation	Units	15	16	17	18	19	20	21	22
		2031	2032	2033	2034	2035	2035	2035	2035
Total net output from longwalls and roadways	[Mg]	940 626	1 462 427	583 274	1 204 704	685 655	2 400 343	1 088 176	2 091 397
Revemes	[PLN]	188 125 271	292 485 390	116 654 732	240 940 860	137 130 964	480 068 530	217 635 144	418 279 402
Mining costs of exploitation by means of a plough technique	[PLN]	125 047 427	75 490 262	145 448 171	91 389 327	148 591 059	103 638 764	128 014 764	50 834 142
Amount of avoided dirt	[Mg]	61 644	102 715	41 075	94 717	41 107	164 429	61 655	141 449
Total savings	[PLN]	2 449 662	4 081 783	1 632 259	3 763 955	1 633 551	6 534 245	2 450 123	5 621 034
Total costs	[PLN]	240 821 848	254 608 837	218 252 933	239 087 119	235 585 174	395 141 826	262 587 473	303 071 138
EBIDA	[PLN]	- 19 229 672	90 120 388	- 80 661 452	45 302 340	- 74 028 979	170 336 925	- 6 574 879	189 430 942
Depreciation	[PLN]	31 017 244	48 162 052	19 304 491	39 684 644	22 791 680	78 875 977	35 927 327	68 601 644
EBIT	[PLN]	- 50 246 916	41 958 335	- 99 965 942	5 617 696	- 96 820 659	91 460 949	- 42 502 206	120 829 298
Tax	[PLN]	-	1 523 479	-	-	-	-	-	2 382 359
NOPAT	[PLN]	- 50 246 916	40 434 856	- 99 965 942	5 617 696	- 96 820 659	91 460 949	- 42 502 206	118 446 939
FCFF	[PLN]	- 19 229 672	88 596 908	- 80 661 452	45 302 340	- 74 028 979	170 336 925	- 6 574 879	187 048 582
Discount rate	[%]	24%	22%	20%	18%	16%	15%	14%	12%
DCF	thousand PLN	- 4 603 431	19 281 269	- 15 958 438	8 148 024	- 12 104 330	25 319 499	- 888 467	22 978 165

Summarising, it was found that during the study all the possible effects resulting from a better plough operation on the seam floor were analysed first of all. The effect of the plough cutting head operation optimisation consists in avoiding the cutting of a part of the deposit (floor), containing small amounts of organic matter (coal), while very significant amounts of rock. This rock has a negative impact on the quality of the ROM originating from the longwall faces. It increases the ash content and thereby reduces the calorific value of the preparation plant feed.

An appropriate data sample was prepared to assess the possible economic effects, resulting from the improved quality of the ROM originating from low longwalls. The data were taken from three longwall faces of the Bogdanka mine, which were in various phases of the mining process.

The assessment of empirical data was started from a statistical analysis to select the optimum model. Based on it, it was possible to link by a mathematical model the longwalls advance with the qualitative structure and the quantity of the ROM, originating from the analysed plough faces. The readings of the SysKon400 type devices, installed on the haulage conveyors in those longwalls, reflected the qualitative and quantitative data.

The statistical analysis revealed poor relationships between the percentage ash content with other quality parameters of the ROM stream in the linear and non-linear models. Therefore, the Monte Carlo simulation and cluster analysis were included in the scope of all the methods.

The model data sample for three combined faces was built by means of the Monte Carlo method, comprising the life cycle of the model plough face. Based on the current data, it was found that the life cycle of the model face would consist of the start-up phase - three weeks, the operational phase of mining - approx. 34 weeks, and the liquidation phase lasting 4 weeks. Overall, the period of life was calculated for 41 weeks.

The Monte Carlo method was used to create a representation of the model plough face.

The cluster analysis allowed then to aggregate the data into bigger and uniform sets, enabling a continuation of analyses based on the characteristics of average selected clusters. As a result, a mathematical model of plough faces advance was constructed. The cluster analysis allowed finally to determine the relationship between the advance and the qualitative parameters of the ROM stream, and in particular with the ash content. The admixture of dirt has a definite impact on the ash content in the feed. The economic effect was estimated in this way, based on calculations of the dirt amount avoided due to an elimination of the floor ripping and potential savings in the process costs. These savings were identified using the current cost structure at the LW Bogdanka SA.

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It is necessary to emphasise the fact that the *in situ* scientific and research work in the Bogdanka mine, as regards longwall faces profiling in thin coal seams, allows to state that there is only a possibility to run the plough on the working floor in a better way - as a method to improve the quality of mined coal ROM. Other reasons (sources) of the ROM pollution, based on the current knowledge and experience, are very difficult to eliminate.

As the assessment issue is complicated and the uncertainty, related to the results achievable in reality, is high, a decision was made to introduce a scenario analysis as an instrument to illustrate the spread of achievable economic effects versus the determined advance increments and the improvement in the mining cleanness. In this way four scenarios were determined, whose economic effects were also compared to the current situation (i.e. without improvements), expressed by means of the "*as it is"* scenario.

The carried out analyses of the economic efficiency of the mining process in the scenarios assuming an improvement in the ROM stream quality resulted, according to the Author's opinion, in achieving significant effects. In particular, for the amount of the coal resources in thin seams of the LW Bogdanka SA of approx. 66.8 million Mg, it is possible to expect that:

– the level of total cost savings can reach PLN 170 million,
– the maximum total profit value (NOPAT) can reach PLN 384.1 million,
– the value of total income effects on the level discounted in the NPV method can reach PLN 281.5 million.

The Author does not decide explicitly, what level of the aforementioned effects is achievable in the LW Bogdanka SA mining practice, which should be subject to a natural verification in the future, once relevant recommendations are implemented. The value of the expected results is undoubtedly affected by the determination to achieve them and a change of qualitative against quantitative criteria importance in the management methods.

The favourable economic results, at small costs of such a solution implementation, make it very interesting and induce to develop the recommendations, which is especially important in the era of the current crisis of hard coal mining sector in Poland.

5. Production Line Management from the Point of View of ROM Quality Stabilisation and Improvement and Economic Effects Maximisation - System Construction Concept

In this study the Author tried to prove that the effective support of mining operations is possible and it is inseparably related to the issues of mining process optimisation and planning, comprising comprehensive management of the mining company resources, including its internal information flows. The conception of the production support system, presented in this chapter, is based on a holistic and integrated approach to the movement of machinery and people, the flow of production, transport and service tasks in terms of production processes planning, organising, controlling, and inspecting functions. Basic tasks of production support system were determined, functioning in an integrated information environment of underground mining plants, comprising:

- mining projects designing and scheduling based on a numerical deposit model, together with:
  - an application of parametric methods to design workings, transport systems, and linear infrastructure objects,
  - a forecast of production results and costs,
  - an optimisation of economic parameters resulting *inter alia* from the development plans, schedule of mining operations, mining technology, selection of equipment, and production organisation methods,
- planning and organising production, service, and transport tasks, together with allocating resources indispensable for their performance (employees, machinery and equipment, as well as materials),
- a coordination and control of production, transport, and service tasks performance, including:
  - reporting the obstacles (failures, stoppages, limitations, delivery delays),
  - monitoring efficiency and productivity ratios (e.g. OEE),
  - dynamic updating of plans and schedules,
  - a circulation of documents (e.g. shipping bills, orders of materials, service assignments, object documentation),
- managing full life cycle of production projects and equipment (from the stage of procurement request notification till the liquidation) resulting *inter alia* in an increased accuracy and analytical usability of the cost assignment system,
- following and managing the production quality based on the data from:
  - the measuring instruments installed on the ROM transport routes,
  - the deposit model,

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  - the qualitative tests gathered in the LIMS (*Laboratory Information Management System*) module,
- integrating, gathering, processing, and reporting multidimensional structures of the data registered in the economic IT area and technical systems for e.g.:
  - a selection of cutting technology, equipment, adaptation of production organisation methods, service and repair strategies for basic machinery and equipment,
  - a preparation of the cost calculations concerning operations and analyses of mining projects and products life cycles costs,
- an optimization of the process of raw material storage, transport, and preparation based on the algorithms and models of advanced process control (APC).

The system proposed in this monograph in the field of defining, managing, updating, and making available the numerical deposit model (NDM) is to support the work of geologists from the Surveying-Geological Departments and to enable an exchange of geological data with other areas of the IT production support system. The main element of the conception in the (NDM) area consists in a numerical deposit model, understood as a digital computer geological model of the mineral deposit, describing the deposit location, its geometry and spatial differentiation of the mineral quality. It enables:

  - to improve the process of deposit geological structure interpretation and documentation,
  - to base the process of mining planning and scheduling on the complete and checked geological data, comprising in particular a precise quality characteristic of the mineral.

The Numerical Deposit Model is based on geological observations and surveys, carried out both on the surface and also in underground mine workings. The developed conception puts big emphasis on the actions aimed at a digitisation of source materials and gathering them in the geological database.

The aim of geological modelling of the deposit consists in as good as possible determination of the deposit geological structure, and the quantity and quality of the mineral within the mine field. The deposit model, understood in this way, is then the basis for all actions related to a creation of mining plans and their optimisation from the point of view of the company economic calculation. The object of geological modelling may comprise, within a selected range, physical properties of the deposit overburden rock as well as of the rock surrounding the coal seams. The model may be also enhanced with some information supplementing the picture of the deposit geological situation, such as e.g. the results of hydrogeological observations.

Such solutions have been implemented in the Polish underground mining industry only for a few years. Bearing that in mind, the Author of this monograph made the following assumptions:

- a numerical deposit model should be an entirely new method of analysis of the deposit geological structure which at present does not have a direct equivalent,
- duties of the geological department were taken as the basis of the current situation,
- the target numerical deposit model should support activities of the geological department related to designing and carrying out geological work as well as to interpreting and documenting the geological deposit structure,
- the most important objective of the numerical deposit model maintenance should consist in its importance from point of view of the mining plant production targets, i.e. new solutions in the field of mining planning as well as output and production planning.

As it has already been mentioned, the Author assumes reversing the organisation of business processes of an underground mining plant, in which the sales planning precedes the coal mining (*Demand Driven Planning*, DDP). That is a reversal of the sequence in relation to the current situation in most mines, in which the mining operations precede the sales planning. Such an important change causes that all the actions affecting a possibility of more efficient performance of operations within the process of mining and production planning become particularly important. Bearing the above in mind, a preparation of real plans of coal mining, preparation and sales will be possible only through their iterative adaptation to one another, so that the assumed figures would be adjusted to the market needs, and at the same time executable at specific economic, geological, and mining conditions. The production quality planning seems to be the key issue to accomplish the goal, which must be finally delivered to the mine organisational units, responsible for mining operations designing, the information on expected parameters of the ROM obtained from mining and from development operations.

Such an approach to the planning process, in the Author's opinion, is possible only if the information on the coal quality is gathered in the system enabling its use within the actions related to the mining operations planning and scheduling. An introduction of this sub-process is to allow as a target - based on carried out simulations - to design optimum space-time recovery of deposits resources adapted to the required commercial parameters of the product.

This conception assumes an introduction of changes improving the main processes identified within the (NDM) area, that is:

- the geological work designing and performance,
- the interpretation and documentation of geological structure of the deposit together with its resources register,
- the support of mining and production planning,
- the support of coal mining operations.

The Author produced a diagram of the main processes in the considered area together with the most important tasks performed within those processes. Also the input and output data from individual tasks were defined. A description of the current situation of processes and tasks, performed in the considered area, was prepared. A few different implementable business solutions were presented, and from among them one solution, adopted as a target solution, was selected and described in detail. Moreover, the study characterises the chosen solution in view of its functionality and it describes the process of its individual components implementation and integration. Also the products of the selected business solution implementation were characterised and its architecture was presented.

Because of a very complex tectonic structure of hard coal deposits mined in Poland and a related necessity of frequent construction of the deposit model, as well as requirements of the areas of mining operations scheduling (MOS) and quality planning (PLN) in the field of the model topicality and possibility of its use in mining scheduling in view of quality, the Author recommends an introduction of specialised IT tools to develop a numerical deposit model, having its own computing-reporting environment, enabling digital geometrical modelling, equipped with complex functions of stratigraphic modelling.

With respect to geological data storage the solution has two variants:

a. the geological database is a module for deposit modelling,
b. the geological database is a separate tool.

In both variants the requirements, related to the geological database, are identical. The main difference between the variants (a) and (b) consists in the degree of geological database integration with the deposit modelling tool. In the variant (a) they are entirely integrated, which facilitates feeding the model with the data and limits the range of necessary integrations, and also simplifies the operation (one graphical environment). However, it is possible to apply the variant (b), if the tool optimal from the customer's needs point of view does not have a module of geological database and in this field it uses the solutions of other producers. In this case the requirements, related to the possibilities in the field of numerical deposit modelling, should override against the advantages resulting from the use of tools having a geological database and a deposit modelling module. In the solution, where the geological database is a separate tool to model the deposit, a dedicated tool is used (CAD), having its own computing-reporting environment.

In the conception, the implementation of a specialised tool (CAD) to create a numerical deposit model, having its own computing-reporting environment and a module of geological database, is the recommended solution in the (NDM) area.

In accordance with the conception, the target solution in the (NDM) area should ensure an accomplishment of the main business objective by an introduction of the deposit information from two components in the management area:

- Geological Database (GDB) being the place of basic geological information on the deposit storage, including the results of hydrogeological observations and the results of deposit and surrounding rocks geomechanical tests together with the information on the results of laboratory tests of geological samples.
- Geological Deposit Model (GDM), which is intended to determine in the best possible way the geological structure of deposit and the mineral quantity and quality.

The GDB ensures a consistency of the source geological information and a quick access to the stored information. The GDM, in turn, is responsible for creating as accurate as possible picture of the deposit geology both in terms of its structure and mineral quality.

The use of geological deposit model assumes:

- the management of Geological Database, comprising :
  - a preparation of the geological data storage standard, defining individual tables for geological data, dictionaries, and data validation rules,
  - a feed of the database with new geological observations and survey results,

  - a creation of specifications, reports, and visualisations based on the data gathered in the geological database.
- the management of Numerical Deposit Model, comprising:
  - a definition of the deposit model,
  - a development of the deposit model,
  - an update of the deposit model.

For the needs of designing and scheduling geological activities, and for the quality planning in accordance with the conception, the geological model is made available within the Numerical Model Management, i.e. the current deposit model is published for a subscribing application from the (MOS) area. The requirements related to the solution architecture are intended to ensure an appropriate level of the solution integration in the (NDM) area with the solutions in other areas, in particular with the (MAP) area. Because of that the chosen solution should have:

- an integrated 3D (CAD) system containing a functionality enabling the designing of mining operations,
- a possibility of calculating volumes and resources broken down to quality categories of the mineral deposit,
- a possibility of reading and saving map files (dwg, dgn),
- a possibility of exchanging geological data with an outside repository as an integral part of the Central Data Warehouse used e.g. in the (MAP) area,
- a possibility of accessing the input data (geological database) and the results of modelling (geological deposit model) from any place in the organisation,
- a possibility of securing full access control defined for a user/group.

Linking of both components (GDB and GDM) with the Central Data Warehouse allows a bidirectional exchange of information with the (MAP) area and it makes the deposit model available to the needs of the (MOS) area. Figure 5.1 presents a schematic diagram of the system structure in the target solution.

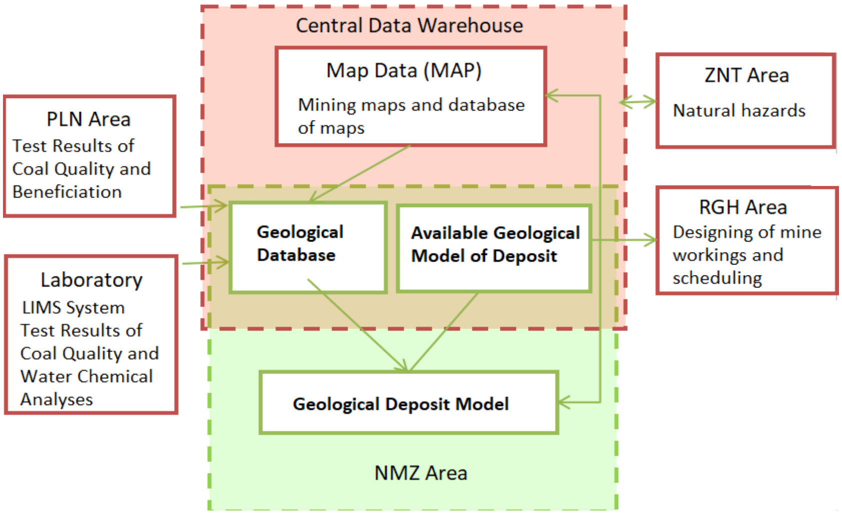

Fig. 5.1. Diagram of target solution in the NDM area and its links with other areas of the production support system (*own study*)

In the target solution proposed in the conception, in the (NDM) area there is no need for the Geological Database (GDB) and the Geological Deposit Model (GDM) integration. Both modules operate in one environment on the same data set. The Geological Database (GDB) is designed as a relational database, existing as an element of the Central Data Warehouse, having a dedicated application to manage the gathered data. Within the Geological Database:

- the scope and nature of stored information is configured (own types of geological data are defined, specific to the enterprise requirements),
- the access control is used, defined for a user or a group of users on the information level (table columns).

Default data structures (data model) are delivered together with the system. They are modified and configured to run the operations of the underground mining plant. Full geological information indispensable for geological observations and surveys is stored in the (GDB), comprising:

- Lithostratigraphic profiles of:
  - the surface boreholes,
  - the underground boreholes,
  - the mine workings.
- Results of coal quality sampling, including:
  - the qualitative chemical, physical, and physicochemical data,
  - the data related to an analysis of coal washability.

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- Hydrogeological information, including:
  - the measurements of the water table height (in boreholes, wells, surface reservoirs),
  - the measurements of the water inflow (to mine workings),
  - the measurements of underground water chemism,
  - the measurements of underground water radioactivity.
- Results of geomechanical tests in penetrometric boreholes.

The introduced data are validated in the Geological Database (GDB). Dictionaries and validation rules are configured in this environment. The following validations are used during the data introduction:

- the dictionary validation - dictionaries of lithological separations are created as well as the names of stratigraphic units and seams,
- the numerical validation - the scope of parameters which are possible to be entered with numerical values, is limited,
- the geological validation (stratigraphic sequence, strata sequence, breaks in the data, etc.) - principles of deposit stratigraphic structure are established,
- the geometrical validation - coordinates of objects stored in a spatial form are verified.

New boreholes are designed and the data are entered at the advance of their drilling in the (GDB). Incomplete boreholes are marked with a special flag and can be excluded from modelling.

2D visualisations together with a configurable data set are created by means of the (GDB) tools. In this way the prepared borehole logs comply with the Polish legislation requirements.

Standard reports exist in the (GDB). They are shaped and defined in detail by the users. The data can be sorted and filtered and in a processed form exchanged with the external tools through interfaces (webservice, flat files, etc.).

The (GDB) environment enables recording and full service of geological samples (approval, data description, making them available) together with a bidirectional integration with the (MAP) area.

5.1. Geological deposit model

In general, two main modelling methods are used in the field of geological modelling, applied separately or together:

- the stratigraphic modelling,
- the block modelling.

A stratigraphic model is the basis of the entire modelling process. The stratigraphic model is developed via an interpolation (extrapolation) and a superposition of coordinates of stratigraphic strata and surfaces findings. It is defined by taking into account the modelling rules, which are introduced by the user, and which comprise:

- the names and stratigraphic sequences of modelled strata,
- the relationships determining deposition trends between neighbouring seams,
- the relationships between individual beds of composite seams (including possible imposed locations of intergrowth lines),
- the erosion boundaries separating consecutive geological stages,
- the surfaces forcing strata deposition trends in individual geological stages (enabling an imposition folds and washouts),
- the types and parameters of interpolators used to calculate strata thicknesses and their data separately,
- the minimum and maximum seam thickness values, which are to be interpolated,
- the geometrical data being a supplement to borehole data, e.g. a location of the outcrop line,
- the assigning faults to individual geological stages or individual surfaces or seams,
- the geographical range of seams and occurrence of surfaces.

Various sets of rules are then used for various seams, so that based on one data set it would be possible to develop separate models in various locations and in various ranges of the deposit stratigraphic profile.

The following data types are processed:

- lithostratigraphic sequences documented in boreholes,
- fault strike lines with marked known throws and inclinations,
- profiles documenting exposures of lithological and stratigraphic strata,
- point findings of individual stratigraphic surfaces,
- graphical elements imposed by the user (own interpretations), forcing certain geometrical parameters of the model, e.g. the set thickness of an indicated stratum in an indicated location, splitting line, range of individual seams occurrence.

Based on the aforementioned sets of data and rules, the software computes a full stratigraphic profile of the deposit. The outcome consists of a stratigraphic table and a set of surfaces (grid) representing all the strata and stratigraphic

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surfaces selected for modelling. The data on seam qualitative parameters and dirt band contents in each seam profile are processed in a similar way.

Various interpolator methods are available to develop grid models, e.g. finite element, inverse distance, kriging, nearest neighbour, and triangulation methods.

5.2. Modelling of the mined deposit disturbances

Because of the tectonic structure complexity of deposits mined in the country, the faults modelling is an important element of the deposit model functionality. Faults are created in the system based on point findings. Such a finding features information about the fault surface location or full information about the location and its measured inclination as well as the fault throw in a specific place. These data are analysed in the system and based on them, the fault findings are correlated and interpreted (throw and angle). The fault line is a broken line, and the fault surface inclination and its throw change along the course of the fault line. Also decaying and branching faults are modelled.

The coal seams splitting is modelled, the splitting line may be created automatically or introduced manually by the user as an element of geological interpretation. Also the cases of seam wedging out and their washouts are modelled, as well as outcrops to the Carboniferous roof or to the ground surface.

5.3. Deposit quality modelling

One of the requirements set to geologists consists in a creation of quality maps: calorific value, sulphur content, ash content, moisture content, density, and possibly other qualitative parameters. The modelling of qualitative parameters is indirectly related also to the (MOS) and (PLN) areas. Averaged samples entered to the (GDB) are used for an interpolation of a specific parameter value between samples, due to which the surfaces originate. Their data reflect values of a given qualitative parameter at any point of the seam. A tool for geostatistic analyses is used to analyse the structure of the specific parameter variability. The developed qualitative models are used to calculate the resources and are also used as the basic data set for the quality planning process within the (MOS) area.

5.4. Hydrogeology and engineering geology

The hydrogeological data are stored in the (GDB). In the deposit model they are used primarily to create various thematic point maps, whose main content consists of the marked measuring point and the measured value. These data are also used to develop models of the water table position in the selected water-bearing levels. These models may be updated with the use of additional lines or interpretation points entered by the user. The system supports the analysis of those surfaces and their graphical interpretation, 3D visualisation of water reservoirs and calculation of their volume. Hydrogeological cross-sections are also created.

Maps of deposit rocks and surrounding rocks strength are created based on the results of geomechanical analyses stored in the (GDB). These are mainly point maps as the geomechanical parameters values are not interpolated.

5.5. Calculation of deposit resources

Calculations of deposit resources are carried out in the system. In the resources calculations the system takes into account a classification because of belonging to the indicated panels, a definition of one or several qualitative parameters range, geological and technological boundaries, and losses.

The inventory of resources is one of many documents created by geologists in the SGD departments. In the suggested solution the specification of resources, necessary to produce the above document, is created within the (CDW) environment taking into account the data comprised by the (GDM) and (GDB).

In the field of resources reporting the process configuration has a multilevel nature. The results of resources calculations are saved by the system in tables, whose layout is adapted to the requirements of a specific report. Specifications are created for the needs of:

– the inventory of resources,
– the report on resources for the State Treasury Agencies (ARP).

The resource reports from the (GDM) are produced on the current deposit model and get to the dedicated area of the (CDW) environment. The results of resource specifications (based on the modelled deposit) provide the basis to update the data in the module to serve resource panels in the (CDW, GDB). The resource data, updated (approved) by the authorised user, are made available (in the CDW analytical and reporting module) in a form enabling a preparation of the final report versions.

5.6. Data visualisation and exchange with other areas

All the operations in the (GDM) are performed in the 3D space. In this way the geological data and individual elements of the model are visualised, as well as specific methods of a graphical presentation of geological data (isoline maps,

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cross-sections). The detailed way of data visualisation is user-defined and comprises e.g. elevations, grid display and transparency.

Isoline structural and qualitative maps are created for the purposes of geological work documenting and reporting. Geological cross-sections (also along broken lines), geological profiles, and boreholes correlation specification in 2D space are created. The prepared printout specifications ensure compliance with the standards used in the company.

The target solution, proposed in the conception in the (GDM) area, utilises the Central Data Warehouse to report and make available the necessary geological contents originating from the (GDM) and (GDB). Visualisations, created in the system, are saved to .dwg files with a full set of attributes enabling an exchange of graphical data with other areas. They are also printed and exported to other data exchange formats, e.g. pdf.

5.7. Mining operations designing and production planning (MOS)

The system, proposed by the Author in the area of mining operations designing, production planning, opening-development operations and mining scheduling (MOS) is to support the process of mining operations designing and scheduling as well as the production scheduling with a possibility of quick and dynamic updating, carrying out simulations and a creation of numerous variants, as well as integrating a given area with other component processes of the Production Support System. These processes are based on the geological deposit model, describing structural and qualitative features of the mined mineral. The area of mining operations designing, production planning, and opening-development and mining scheduling (MOS) is a crucial activity of the mining plant, providing the basis for other areas, e.g. short- and long-term planning of operational activity, planning of all the technological links, cost and revenue forecast, investments, procurement, repairs management and risk management.

Due to very complicated geological and mining conditions in the deposits and related necessity of frequent schedules updating, creating their various variants, and also a flexibility required from the system in the field of designing and scheduling support in terms of quality as well as customers’ requirements and needs, the Author recommends an implementation of a solution comprising an introduction of new tools dedicated to production designing and scheduling in underground coal mines. Such tools are adapted for a creation of short-term schedules including all their details, and they also contain all the required functionalities to create analyses of long-term schedules using single 3D models.

The proposed solution ensures:

- within designing and scheduling of mining operations:
  - an automatic or semi-automatic support of the deposit development designing stage (based on the set parameters),
  - a change of development every single time, and it causes updating of the entire draft production schedule,
  - a visualisation of contents related to natural hazards,
  - setting up the schedule of derived tasks - belt conveyors installation, construction of a roadway support, supply of technological elements with utilities etc.,
  - defining resources (districts, machines), own or subcontractors' resources,
  - defining the productivity from individual panels, for individual machines/districts, defining in various units (Mg/d, m/d),
  - defining the sequence of mining,
  - defining the production targets - quantitative and qualitative,
  - defining the production limitations (e.g. related to transport, to natural hazards in the given panel etc.),
  - forecasting the ROM quality at the entry to the preparation plant,
  - complex, functional mixing allowing to maintain the defined quality of the ROM stream,
  - an export of scheduling results to the database,
  - setting up a long-term schedule keeping all the details required for the short-term schedule (long-term schedule can be a combination of a few short-term ones);
- within reporting the recommended solution enables:
  - an integration with the database environment,
  - acquiring the information on the volume, thickness, and quality of raw coal from the geological model,
  - considering in reports a decline of resources based on the advance of planned opening and development operations,
  - transferring the information on the coal quality directly to the surface,
  - selecting tasks from the schedule to display them graphically in real time,
  - creating a visualisation of the production schedule,
  - exporting the data according to the defined data format, scheduling the operations related to mining (e.g. longwall equipment change),
  - producing operational reports,
  - a schedule visualisation in the form of a Gantt chart (a possibility to configure the Gantt chart appearance and content),

- producing a report for a chosen district (a plan for the district),
- a comparison of the plan with performance in a given period,
- a possibility to display the performance and plan on one chart,
- a possibility to prepare own report templates,
- a cooperation with tools to forecast the costs and to model accrued liabilities.

As it was presented in the conception, the target process of production preparation and planning starts from the analysis of the market in terms of possibility to sell individual product groups. Then the sales division sets up a schedule of commercial products sales, and the Preparation Plant services prepare a production schedule, which is the basis to perform schedules of development and mining operations. This work is necessary to carry out a financial simulation, based on which the profitability of the planned production targets is analysed in the case of defined market needs. If the requirements related to the project profitability are not met, a new sales plan is defined again, as well as new production, development, and mining schedules. Fig. 5.2 presents the diagram of the MOS system structure in the target solution suggested by the Author. The presented target solution in the (MOS) area is characterised by the following features.

- an integration of preparation and sales with planning - (PLN) area,
- an integration with selected solutions from (MAP), (NDM), (RPT), and (M&E) areas,
- an integrity with the database environment,
- a possibility to work on the files of graphical map software,
- a quick and dynamic update of schedules in real time,
- a flexibility in setting up many variants of the schedule and in their analysis,
- complex and functional mixing allowing to maintain the defined quality of the ROM stream,
- reporting according to the description in the (RPT) area, on the SAP system level,
- a comparison of the plan with the performance in a given period (in metres or Mg),
- an export of scheduling results to the database,
- a possibility to prepare own report templates,
- a cooperation with tools to analyse the costs and to model accrued liabilities.

The target solution is a system of mining operations designing and scheduling, consisting of four modules, whose functionality meets the requirements in the field of mining operations designing and scheduling.

The target solution in the (MOS) area should ensure an accomplishment of the main business objective by an introduction of the system of mining operations designing and scheduling, comprising the following modules:

– the mine workings designing module,
– the mining operations scheduling module,
– the module for mixing and optimisation from the quality point of view,
– the module for mining operations revenues and costs forecasting.

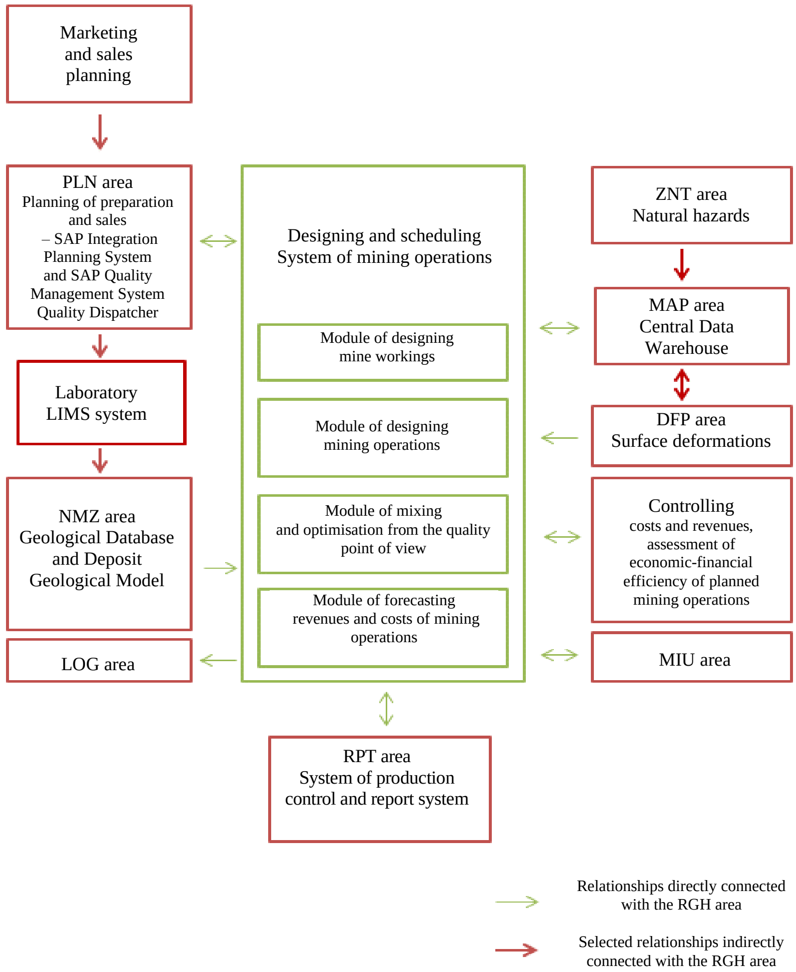

Fig. 5.2. Diagram of production designing and scheduling system in terms of quality and links with other areas (*own study*)

In accordance with the prepared graphical diagram (Fig. 5.2) the solution, dedicated to mining operations designing, production planning, and planned opening, development, as well as mining operations scheduling, should be based on a cooperation with the following areas:

- (NDM):
  - provides the data concerning the deposit model, on which the mining operations designing is based;
- (PLN):
  - enables preparation and sales planning, comprising the required quantities and the quality of raw coal with accuracy to grades satisfying the saleable coal production assumptions,
  - receives information on the possibility of intended plans implementation;
- (M&E):
  - provides the data on machines availability and repairs management,
  - receives schedules of mining operations necessary to set up schedules for longwall equipment installation and removal;
- (RPT):
  - provides data from other areas in the form of reports and exchanges the information on the current situation,
  - receives reports on designing and scheduling;
- (MAP):
  - provides current maps of mine workings and special maps,
  - receives graphical schedules visualisation, including information necessary to forecast the ground deformation;
- (GSD):
  - provides the report on the PZZ and environmental decision conditions;
- Controlling:
  - provides the (MOS) area with information on the unit costs of conducting individual mining operations, which is used for a current assessment of individual mining variants costs, and the information necessary for the process of modelling accrued liabilities as regards the mining operations costs,

- receives the data on the planned mining to evaluate economic-financial efficiency of a given mining plan variant:
  - quantitative information on the ROM,
  - qualitative information on the ROM,
  - technical information on planned opening-development and mining workings,
  - preliminary cost and revenue forecast made in the module for mining operations revenues and costs forecasting, taking into consideration a simulation of accrued liabilities;
- draws attention to the information concerning an assessment of economic-financial efficiency of a given mining plan variant in:
  - the discount methods: NPV and IRR (NPVR),
  - the selected data collected from the Profit and Loss Account and from the Cash Flow Statement.

The performance of above operations, within the cooperation with the Controlling area, requires a preparation of a module (tool) for an assessment of economic-financial efficiency of a given mining plan variant. The target solution in the (MOS) area should comprise a possibility of hard coal analysis in the scope of net and gross output from the point of view of the forecast of coal pollution with dirt.

In general, an assumption is made that the following information will be available from the operations scheduling:

- saleable coal (broken down to grades),
- other commercial products,
- coal pollution (size of aggregate production),
- balance of products and semi-products piles,

used to analyse the costs and sales plans.

The process of mining operations designing, production planning, and opening, development, as well as of mining operations scheduling provides the feedback to other areas and information on non-satisfying of the requirements. A possibility of introducing additional sources of raw coal is also predicted, which originates *inter alia* from the third party companies, to obtain final products of specified quality parameters.

5.7.1. Description of components for the MOS system

The system of mining operations designing and scheduling, presented in the conception, consists of four modules improving the work of the Production Preparation Department:

– the mine workings designing module,
– the mining operations scheduling module,
– the module for mixing and optimisation from the quality point of view,
– the module for forecasting the mining operations revenues and costs.

The system also comprises additional functions supporting a visualisation and management of modules, which:

– allow an interactive creation of animations and design filtering during the schedule visualisation in the separate 3D window,
– allow the authorised users to access files and review them after the software downloading and installing,
– enable integrating many designs into one complex design,
– enable creating printout templates,
– present strategic schedules for investors,
– allow to add many attributes enabling schedules setting up, taking into account all the operations comprised by the production process, e.g. information on machinery and equipment, i.e. the time of machines delivery, assembly time, performance time.

5.7.1.1. Mine workings designing module in the MOS

The basis for this module includes an integration with the numerical deposit model,based on which the designs of mining operations are made. Developments are made in the 3D graphical environment. Due to spatial workings designing a real image (on actual coordinates), is obtained. It allows a better understanding of the strata deposition and the course of planned workings. The designed workings are described by means of user-defined attributes characterising e.g. the type of working, its function, cross-section. The created designs are linked to the existing mine workings using a digital inventory of vector mining maps (described in the MAP area). The same system of coordinates is used in designs, as that one used on mine workings maps.

This module contains the following functionalities ensuring a preparation of mine workings designs, based on digital geological maps containing complete information on the deposit:

- 3D CAD environment, a possibility of downloading and saving AutoCAD dwg files,
- designing of the longwall mining operations in underground hard coal mines,
- the work on a numerical deposit model (grid or block), with an access both to the deposit structure and quality, a full integration with the target solution of the (MOS) area, a visualisation of the model, a calculation of resources, a performance of mathematical operations on the model elements,
- three-dimensional and flat (3D and 2D) designing of longwall faces as well as of development workings,
- projecting a two-dimensional development on the structural surfaces of the deposit model,
- automated, parametric designing of the longwall system development, comprising also an automation of the workings names and a determination of their driving sequence - a possibility of a manual edition of prepared schedules.
- a possibility of introducing an unlimited number of attributes to supplement the database and schedules as well as to create the lists of available values for individual attributes as well as downloading such lists from the Excel and .csv files or from databases,
- automatic assigning of attributes by formulae operating on the deposit model or on graphical properties and other attributes of designed workings,
- manual and automatic defining of relationships describing the sequence of workings driving, creating relationships by means of attributes and also based on spatial searching, enabling a repeatable and controlled creation of complex mining operations, without the need of manual interference or scripts writing,
- creating shapes of mine workings cross-sections and assigning them to the workings,
- creating three-dimensional blocks of mine workings based on defined workings cross-sections,
- creating cross-sections via designing of the working and the deposit model,
- filtering the CAD objects display based on their graphical properties and attribute values,
- performing logic operations on blocks (adding, subtracting, intersection determination),
- an update on the work advance based on the surveying data or the data from the reporting system,

- an interactive integration with the mining operations scheduling module,
- an integration with other systems to import data on the accessible districts and mining machinery.

5.7.1.2. Mining operations scheduling module in the MOS

Short- and long-term schedules should be set up based on the mining development made on the deposit model. In the case of long-term schedules it is possible to create and analyse many variants to determine the optimum solution. The planning process starts from a preparation of a general schedule of mining operations and as the work advances it is then specified depending on the needs. Long-term schedules are set up as first and based on them various variants are analysed through an optimisation in the field of quality and quantity of products due to the production targets. The details related to other technological processes are introduced next, as a result of which closely interrelated short- and long-term schedules originate.

Due to an integration with the mining operations designing module, a dynamic schedules update is possible at a change of the deposit development. Three basic functions may be performed in the scheduling module:

- setting up mining operations schedules,
- analysing many variants of schedules,
- quick and dynamic schedules updating.

The module should contain options of quick setting up development and mining operations schedules, in which some assumptions of the schedules and the results of calculations are based on modelled geological data. The speed of responding to changes in the deposit geological exploration is significant, forcing an update of the designs and schedules, in particular:

- an interactive integration with the mine workings designing module, comprising downloading and modifying the tasks, relationships, and attributes as well as their dynamic update,
- setting up integrated schedules for longwalls as well as opening and development operations including a possibility of their separate reporting,
- producing quantitative-qualitative forecasts,
- forecasting the mining operations costs,
- a visualisation environment comprising the configurable Gantt chart and a tabular report with a possibility of introducing own computation formulae based on the scheduling results,

- filtering the tasks by date, attributes, relationships,
- tasks grouping, tasks sorting, conflict searching, critical path displaying,
- creating own calculation fields, including attributes entered manually (also variable over time) or calculated based on the formula,
- a graphical visualisation of plans and schedules including charts, blocks, and surfaces and their 3D animations, a simple exchange of data from the schedule to a reporting window and a possibility of the data integration with a spreadsheet,
- an integration between production tasks and auxiliary tasks,
- a detailed creation of work calendars and report calendars with accuracy to a shift and a possibility to define calendars for a period of more than 100 years,
- defining the rules in the calendar comprising repeatable or one-off periods without any production or with reduced production capacity,
- defining separate calendars for resources,
- defining resources with a possibility of introducing their productivity,
- defining productivity varying over time or depending on the task attribute values,
- defining relationships *finish to start*, *start to start*, *start to finish*, *finish to finish* or a percentage task advance necessary to start a linked task,
- defining delays between tasks,
- resources balancing considering the task priorities and defined production targets,
- resources balancing allowing to set up a detailed short-term schedule manually and a long-term schedule automatically,
- managing numerous scenarios, creating baseline plans and comparing them with the current plans, comparing the plans with the performance,
- schedules updating based on the current production data,
- an integration with other systems to import data on the accessible districts and mining machinery,
- setting up waste production schedules based on the data from long- and short-term schedules,
- reporting package utilising results of the scenario for quick production of information reports for all the interested persons and for the needs of further planning stages (in accordance with the description in the RPT area),
- defining own report templates.

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This module, combined with the preparation and sales support systems, prepares the data necessary to forecast the revenues and costs of mining operations, in particular:

- the yield of saleable coal from roadway operations for the needs of defining the working driving costs,
- the yield of saleable coal from individual longwalls in years,
- the quality of commercial products.

5.7.1.3. Module for mixing and optimisation from quality point of view in the MOS

Module for mixing and optimisation from the quality point of view is responsible for defining qualitative objectives (also combined with quantitative targets) for the set up schedules and for schedules optimisation, taking into account the intended targets. Qualitative targets for mining operations scheduling originate from the quality planning system in the (PLN) area. At the same time a few qualitative and quantitative targets with diversified priorities are defined for the schedule. For each of the targets a range of selected qualitative parameters is determined, which is required in consecutive computing periods. All the rules, defined previously from the design and mining operations schedule level, are observed during scheduling, hence the intended qualitative targets may be not achievable in the assumed computing periods. At the same time this module provides a tool used for scheduling mining operations, taking into account the intended coal grades, based on the data on the coal washability contained in the deposit model.

5.7.1.4. Module for forecasting mining operations revenues and costs

A preliminary simulation of revenues and costs forecast is made in this module. These simulations are used to assess individual variants of schedules.

On one hand the schedules of longwalls running and output as well as the development schedules are the input data to the model, and on the other hand the information on the unit costs of individual mining operations performance is given. This module should be closely related to the mining operations scheduling module. A substantial amount of information is taken from this module. Its correct preparation significantly improves a possibility of a quick development of simulations related to the mining operations revenues and costs. The following simulations will be run to determine:

- the costs by type (road workings, longwall equipment installation, longwalls mining, longwalls liquidation),

- the costs of accrued liabilities (suspended instalments and paid instalments, balance of accrued liabilities in a given period, state of individual longwalls, charging with the cost of roadways driving and longwall equipment installation),
- the revenue forecasts.

First the controlling activities provide the information on the unit costs of individual mining operations and necessary information to model the accrued liabilities for the mining operations costs. The output data are acquired then from the mining operations scheduling module and from the module for mining operations revenues and costs forecasting, necessary to obtain a complete assessment of economic-financial efficiency of a given mining variant. Next the information, related to the performed assessment of the given mining variant with use of the discount methods, is provided: NPV and IRR (NPVR). The data indispensable to carry out calculations may be taken from the profit and loss account and from the cash flow statement. The above operations within the cooperation with the controlling area require a preparation of a module (tool) for an assessment of economic-financial efficiency of a given mining variant.

5.8. Forecasting the production quality, planning and integration of the planning process on the mining side with the process of coal preparation and sales

The conception of the process of a mining plant production planning and controlling, developed by the Author, responds, first of all, to the market needs and intended sales of the mining plant in individual product groups. Based on these data, it is possible to simulate related needs at all the planning stages, i.e. coal output planning.

As it has already been mentioned, reversing of the mining planning sequence against the current situation, in which mining precedes sales planning, is the basic change proposed by the Author herein. At the same time such a significant change causes that forecasting and controlling of the ROM quality become especially important, in particular in relation to the necessity for such mining planning, which allows optimising the quality of mined coal via a stabilisation of its parameters on the planned level.

Reversing of the mining planning sequence against the current situation is possible only by a development of the system, which provides appropriate tools, supporting manual as well as automatic gathering of geological-mining data related to the carried out mining operations, and also analytical and reporting

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tools allowing to monitor continuously the current situation and to update future operations plans.

In accordance with the presented conception, the area of widely understood quality connects all the areas of the mining company activity with one another (Fig. 5.3).

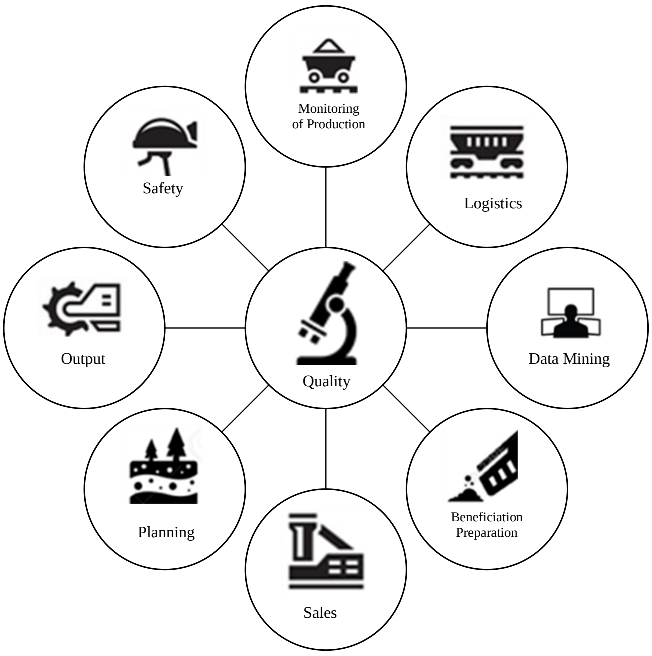

Fig. 5.3. The place of the quality planning area at the contact of all the areas of the mining plant operations (*own study*)

Also an integration among individual elements of the system is extremely important as well as ensuring the appropriate information flow rate among them.

The target solution operates as follows:

- The Quality Control Department fulfils a monitoring function in the process of ROM mining and saleable coal production, and the planning and forecasting activities concerning the raw coal quality are transferred to the Production Preparation Department, where the planning support tools ensure the required functionality in the field of quality planning and forecasting.
- All the ROM haulage routes are monitored in terms of quality and quantity using the installed measuring infrastructure (qualitative parameters analysers and the measuring weighers). The acquired information is made available by means of the IT systems, visualised and

on a current basis controlled by the quality dispatcher. The measurement data are stored in the production reporting system.

- All the mining fronts are monitored with the use of a dedicated measuring system in terms of their shape and course in the 3D space.
- The described system functionalities cooperate with the Laboratory Information Management System (LIMS), supporting processes of coal samples receipt and registration, planning and performance of analytical work as well as recording the analysis results and making them available.

The production reporting area is supplied with information on the quality of semi-products and products, originating from the monitoring activities.

5.8.1. Quality management at the stage of mining and production planning

The quality management at the stage of mining and production planning is closely related to the solution proposed for the area of mining operations scheduling. A conception description, including the solutions adopted therein, is presented below. The mining and production planning is a process, within which, based on the sales plan, the planning is implemented in the field of determining the production level and mining operations designing and scheduling. The preparation and sales planning is usually carried out in the ERP class systems (in the field of sales and preparation planning), combined with the quality control system (in the field of quality management). These systems provide the area of mining operations designing and scheduling with the data on the required quantity and quality of raw coal with accuracy concerning the produced grades, of meeting the production assumptions for the saleable coal.

The designs of mining operations and integrated schedules of development and preparation operations are carried out in the mining operations designing and scheduling system. This system has a full access to complete and current geological information contained in the geological deposit model. The geological deposit model is supplied with the information originating from the deposit exploration (geological boreholes, results of longwalls and roadways profiling, results of deposit sampling) and comprises information on the deposit structure and on its quality.

Mining operations are designed against a background of correctly visualised information on the deposit structure and quality. The design of the deposit opening and development is integrated with the mining operations schedule.

All the information related to the planned ROM amount and quality, including also the forecast of coal pollution with rock, is calculated based on:

– the mining operations design, comprising the shape of the workings and their location in the 3D space,
– the deposit model, containing a spatial description of the deposit structure and quality.

In such an approach the forecast of raw coal quality is an integral part of the mining operations schedule and it is made by the employees of the Production Preparation Department. Mining plans are produced in variants. One of the criteria for individual variants assessment is its compliance with the requirements of the preparation and sales planning system, related to the quantity and quality of raw coal.

The developed variants of mining plans may be optimised in the field of satisfying qualitative, quantitative or other (e.g. cost) targets. Individual targets (one or many targets for a given plan) feature a permissible range of values for the optimised parameter and also a defined priority. The objective of such an optimisation is to indicate the best possible sequence of mining operations within a given design, using all the available resources and considering the limitations resulting from the nature of mining operations (mining system, sequence of workings driving, natural hazards, maximum ground deformations etc.). Because of numerous limitations, related to the carried out mining operations, there may be situations, where in none of the analysed development and schedule variants the set targets will be accomplished. In such a case the appropriate information is provided as a feedback to the preparation and sales planning system, where after plans redefining the planning process starts anew. Consistent sales, production, and mining operations plans are obtained at the output of the mining and production planning process. These plans contain information on the quantity and quality of commercial products and semi-products.

5.8.2. Quality monitoring at the stage of coal mining, its preparation and processing

Coal mining is a process comprising all the actions aimed at a deposit preparation for mining operations, an installation of machines and equipment at longwall faces, mining the deposit resources and transporting the mineral to the surface to the preparation complex. The quality monitoring is defined within this process. The quality monitoring is aimed at a permanent control of the ROM parameters. The monitoring process is carried out on the selected sections of the ROM mining and transporting ‘path’ (longwall, belt conveyors). Such actions allow controlling of the ROM parameters both in the scope of quality (dirt amount, moisture content, calorific value, sulphur, ash) and also of the grades fall out. Appropriate corrective and preventive actions are implemented in the case of deviations from the planned values.

The suggested system is responsible for continuous monitoring of the coal quality parameters. The data is visualised on a current basis in the production parameters monitoring system, in the quality control room, based on the installed underground and surface measuring infrastructure integrated into one consistent system. The system consists of:

- the analysers of coal quality parameters installed on conveyors,
- the weighers installed on conveyors,
- the system for monitoring the height of longwall faces.

The results of all the measurements made within the quality monitoring system are transferred to the SCADA system, and from it - to the acquisition platform of the data repository defined in the (M&E) area. The other quality reporting areas use the data from the quality monitoring via this platform. Special attention should be drawn to the fact that the quality monitoring system is to serve primarily for the current operational correctness control of mining and preparation processes. The results of measurements taken by this system, due to a relatively low accuracy, do not replace the production settlement methods used at present.

The basis of the solution recommended by the Author consists in the establishment of the QUALITY CONTROLLER function and in the creation of a uniform quality monitoring system, consisting of the ROM quality parameters analysers and of the measuring weighers installed on the belt conveyors. Ultimately the number and location of measuring sets are determined in a thorough audit of the ROM transport routes on the level of the project feasibility study by the target concept contractor. The measuring sets are connected to one consistent IT system integrated on the level of a dedicated integration platform. The quality controller carries out continuous monitoring and supervision of the ROM qualitative (ash, moisture content, calorific value) and quantitative parameters in individual haulage points and of individual longwalls heights. He controls also the performance of the quality plan, being an inseparable part of the production schedules and of the mining operations schedules, and responds to a deterioration of raw coal quality exceeding the set standards.

The solution, recommended in the conception assumes a purchase of analysers from one manufacturer and also a purchase of weighers from one manufacturer. Also a system for an automatic control of the ROM stream on the surface originates within the target solution. It is to be used for the haulage management depending on the quality of transported ROM.

Special attention should be drawn to the fact that the results of measurements taken by the automation system are to serve primarily for a current operational

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correctness control of mining and preparation processes. However, because of a low accuracy they do not replace the production settlement methods used at present.

The recommended solution results in reducing the frequency of manual ROM sampling in favour of using a continuous quality measurement system and making results available within the production reporting system. Monitoring of the mining fronts in terms of their shape (at the cross-section), and also the course in space is an additional element of the system. The objective of such a system operation consists primarily in the control of coal seam mining cleanness on a current basis. Such systems are now under development and their commercial versions are not available yet. The data from the system are subject to reporting and a visualisation in the monitoring system installed in the quality control room.

5.8.3. Structure of system for production quality forecasting and for integration of mining, preparation, and sales planning

The target solution in the (PLN) area, presented in the conception assumes an implementation of an automatic system of underground quality monitoring. The information is gathered automatically from the sensors installed on belt conveyors in various places of the ROM haulage from longwalls and in the preparation plant. Fig. 5.4 describes components of the system and its links with other areas of the concept.

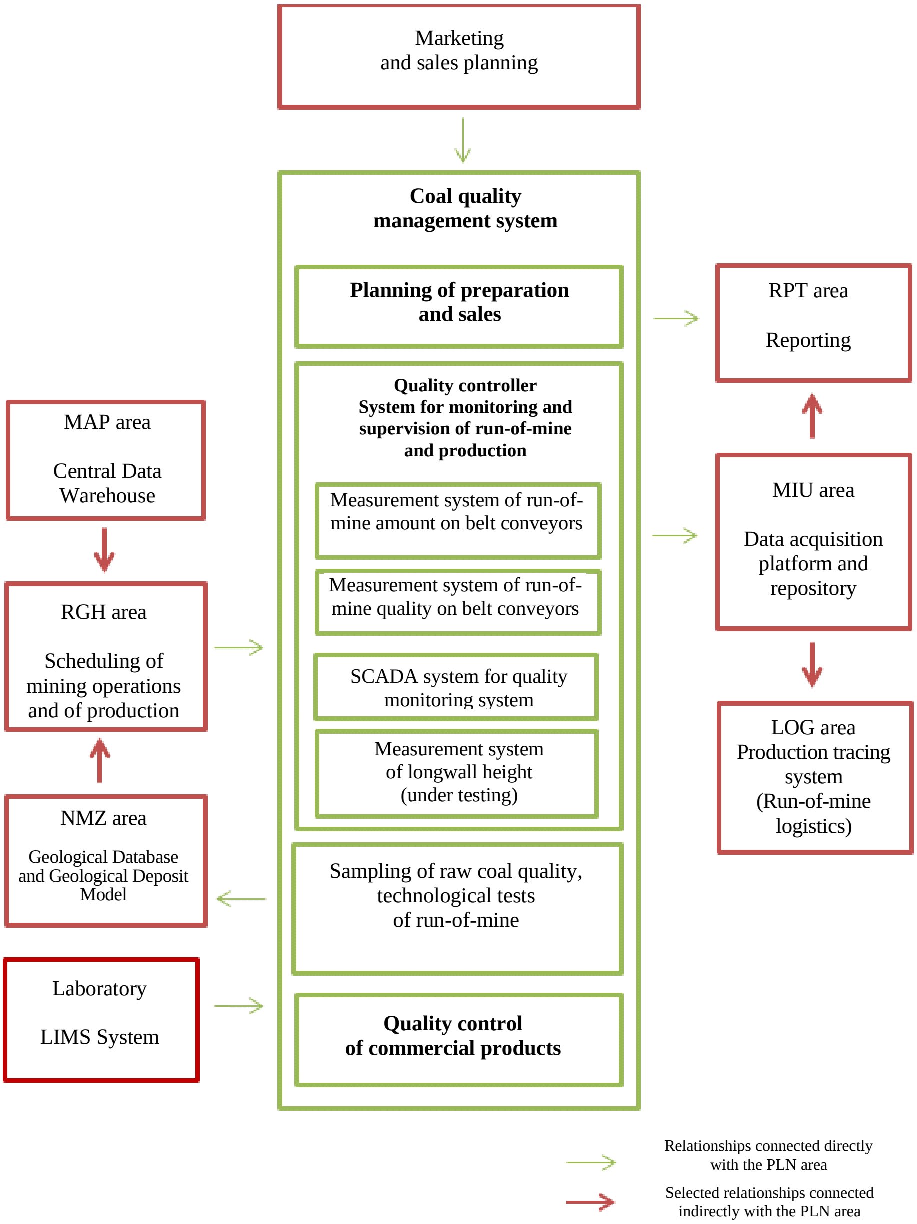

Fig. 5.4. Diagram of coal quality management and its links with other areas of the conception (*own study*)

5.8.4. System components for the QUALITY area

The quality management at the stage of preparation and sales planning specifies the required quantities and qualities of raw coal to meet the production assumptions related to the saleable coal.

These requirements are transferred to the mining operations designing and scheduling system, where mining plans are drawn taking into consideration the requirements with respect to the raw coal quantity and quality.

5.8.5. SCADA system for the quality monitoring area

The SCADA system collects the information from individual quality parameters monitoring devices and visualisse it in real time. The recommended solution does not require a purchase of a new SCADA system. The use of the existing SCADA systems is assumed as well as expanding them to meet the requirements of the QUALITY CONTROLLER.

5.8.6. Raw coal quality sampling

Sampling of raw coal quality is still carried out manually, its performance is aimed at a control of the monitoring system operation correctness and at a calibration of measuring devices.

The results of sampling feed, via the geological database, the geological deposit model with the data on the raw coal qualitative parameters due to which more accurate forecasting of ROM quality is possible, are used. The performed technological analyses of raw coal also feed the geological deposit model with the data on its washability. Based on the test results and coal washability characteristics it is possible to plan the preparation processes in a better way.

5.8.7. Commercial products quality control

The last stage of the quality control process acc. to the proposed conception consists in the commercial products quality control. Qualitative samples are taken in accordance with the assumed schedule and next they are transferred to the laboratory. The results of sampling provide the basis to determine the saleable coal price and to estimate the coal output. The following activities are carried out, among others, within the quality control:

- monitoring of production quality and quantity,
- a determination of monthly quality tasks and analyses of their performance,
- examining secondary quality settlements,

- a supervision of technical infrastructure related to quality control procedures and laboratory tests of coal physical parameters,
- defining unified rules of coal qualitative parameters documenting, complaint procedures, secondary settlements.

Quality control tasks comprise a performance of monitoring of production quality and quantity, aimed at ensuring products quality on the level consistent with the commercial offer. In the (PLN) area the concept assumes a cooperation of the constructed solution with the Central Data Warehouse, functioning as a separate data repository and integrating the contents both from the map resources base (MAP area) and from the geological model base (NDM area). Moreover, a cooperation with the acquisition platform from the data repository (M&E area) is predicted as the place, from which the data is transferred to the (CDW) and to other systems within the considered area, and also to other systems.

In addition, the proposed business solution assumes a purchase of equipment from the field of:

- the analysis of coal quality parameters (analysers),
- the performance of the ROM quantity (weight) measurements,
- the performance of mining fronts shape and course measurements.

An implementation of the above tools may require an expansion of the existing IT network, including the SCADA system, in parallel to the purchase or modernisation of computer units (workstations).

In the present period, difficult for the sector, the basic determinant of a mining plant development direction includes maintaining of high flexibility in the context of products range, production costs level and their unit cost, which are related to an implementation of adaptive planning mechanisms and techniques of operational activity improvement. However, this is an extremely complicated action due to high dynamics of plan changes which are exceptionally unfavourable phenomena in the mining sector. The level of employment and a capital intensive nature of processes combined with long-term planning horizons and strong inertia of the production system make a quick reaction to raw materials prices impossible, causing that such a scenario must be predicted and planned properly in advance. This problem applies similarly to the methods of operational activity improvement (a systematic identification of improvement areas and an effective elimination of losses dispersed through the organisation are generally a long-lasting process). In this context a continuous optimisation of long- and short-term plans from the point of view of changing conditions of carried out business becomes a necessity. This is confirmed by literature

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(Lisowski 2013), in which a broad discussion on management issues was carried out, showing that the plans optimisation as a stage of ‘elementary decision cycle’ in the mining sector is possible and highly demanded.

It is necessary to emphasise the fact that in a general approach the production support system described herein, like any IT solution, may be a tool facilitating and improving the performed tasks. The need to exclude actions for any price in favour of applying only those tools, which provide measurable benefits, without a requirement of a far reaching justification, is extremely important in the current period, difficult for the sector. It should also be highlighted that the effectiveness of production support system strongly depends on the company adaptation capacity, on the will to reorganise with the intention to use practically the defined software functionalities. The production support system adaptation to the market requirements and internal company needs is a long-lasting process, which is worth starting from a review of the best world practices in this area.

It is worth mentioning that only the focus on computerisation, standardisation, and data gathering without any further idea of their practical use to improve efficiency does not bring the expected effects. A limited long-term data use causes their quality deterioration, which may result in undermining the staff trust in the essence of the entire system operation. An implementation of the full PDCA cycle (*Plan Do Check Act*), based on the proper and real data provides measurable effects. It is not simple to achieve this state, because it requires skilful configuring and combining the systems of planning, operational support, analytical and optimisation environments, so the advisability of the IT production support system implementation in the underground mining industry should be considered in the context of target support of those four elementary activities.

6. Summary

This monograph is a summary of the Author's studies carried out on the development of a deep mine model, taking into account specific nature of natural conditions and the requirements of safe, and at the same time economically efficient mining of hard coal deposit in the Lublin Coal Basin at big depths. The conception of an IT system construction, presented in the final part of the monograph, assumes an automatic calculation of the quantity and quality of the ROM and dirt in selected time slots, and after a completion of the simulation an automatic forecast generation of all the parameters related to the implemented mining project, such as the ROM quantity, amount of dirt and coal qualitative parameters.

The Author determined the pollution impact - as qualitative losses - on the efficiency of coal acquisition in the conditions of an underground mining plant using the example of the LW Bogdanka SA as the main objective of the study. Experimental studies and in situ measurements were carried out during the mining operations of a specific coal seam part by means of a longwall system in the Lublin Coal Basin. Geological in situ observations, conducted by the Author, of roadheadings, longwalls as well as headgates and tailgates allowed to develop a conception of a system for monitoring, on a current basis, the cleanness of deposit mining under the conditions of the Lublin Coal Basin.

The Author analysed the reasons of the ROM pollution origination in the process of hard coal mining. The impact of such elements as the deposit opening structure, the dirt amount, geological-mining conditions of the deposit location, technique and technology of mining on the cleanness of coal mining was considered. Based on the LW Bogdanka SA experience, the Author presented technical possibilities enabling to reduce the amount of dirt both in the case of longwall faces as well as roadways. The Author performed also an assessment of the ROM dilution impact on the economic efficiency of the production process of an underground mining plant. The effects, resulting from better running of a plough head on the seam floor were primarily analysed in the Author's study. The effect of the machine trajectory optimisation consists in avoiding the cutting activity of a part of the deposit (floor), containing small amounts of organic matter (coal), while very significant amounts of rock. This rock has a negative impact on the quality of the ROM from longwall faces; it causes an increase of the ash content and thereby a reduction of energy value of the feed directed to the preparation plant.

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To evaluate possible economic effects, resulting from an improvement in the quality of the ROM from low longwalls, an appropriate input database was prepared for a statistical analysis, which originated from three longwall faces in the thin seam of the LW Bogdanka SA, being in various phases of the mining process.

An assessment of the empirical data was started from a statistical analysis, directed at a selection of the optimum model, based on which it is possible to model an advance of longwalls depending on the qualitative structure and the quantity of ROM from the analysed plough faces. The readings from the SysKon400 type devices, installed on the haulage conveyors of those longwalls, were the qualitative and quantitative data.

The empirical data, obtained from the IT systems, featured diversity, high variability and specificity characteristic for a given phase of mining carried out in plough faces of thin seams.

The choice of statistical methods was aimed at a determination of empirical data forecasting capacity and of a possibility to develop a theoretical model describing the relationships among the ROM qualitative features, output from longwalls, failure statistics, advances or the avoidable dirt amount, which can be identified based on an assessment of geological profiles from longwalls sampling. A preliminary assessment of the data decided about the legitimacy of the data statistical analysis methods use only in the basic scope. Descriptive statistics were prepared for each longwall. The correlation analysis, regression analysis, and variance analysis were performed.

The statistical analysis revealed a low interrelation of the percentage ash content with other qualitative parameters of the ROM stream in linear and non-linear models, therefore for the needs of forecasting calculations the Author applied the Monte Carlo simulation and the cluster analysis. A model data sample for three combined faces was built with use of the Monte Carlo method, comprising the life cycle of a model plough face. Based on the current data, it was found that the life cycle of the model face consisted of the start-up phase (3 weeks), operational phase of mining - approx. 34 weeks and the liquidation phase lasting 4 weeks. Altogether, the period of the model face life was determined as 41 weeks.

The cluster analysis allowed to aggregate the data into larger and more uniform sets, enabling a continuation of the study. As a result of the partial loss of relationships between single observations, a mathematical model of the advance was developed as a function of model qualitative parameters (in particular the ash content). An optimisation of the plough head movement was related to savings in the costs of mining and supporting processes. These savings were identified using the current cost structure at the LW Bogdanka SA. It is necessary to emphasise the fact that the *in situ* scientific and research work, carried out in the Bogdanka mine conditions as regards as longwall faces profiling in thin coal seams, allowed to state that there was only a possibility to run the plough on the working floor in a better way - as a method of improving the quality of mined coal ROM. Other reasons (sources) of the ROM pollution, based on the current knowledge and experience, are very difficult to eliminate.

As the assessment issue is complicated and the uncertainty related to the results achievable in reality is high, the Author made a decision to introduce a scenario analysis as an instrument (method) to illustrate the spread of achievable economic effects versus the determined advance increment and improvement in the mining cleanness. In this way four scenarios were determined, whose economic effects were also compared to the current situation (i.e. without improvements), expressed by means of the *as it is* scenario.

The carried out analyses of economic efficiency resulted, in the Author's opinion, in achieving significant effects. In particular, for the coal resources in thin seams of the LW Bogdanka SA of approx. 66.8 million Mg, it is possible to expect that:

– the level of total cost savings can reach PLN 170 million,
– the maximum total operating profit value (NOPAT) can reach PLN 384.1 million,
– the value of total income effects on the level discounted in the NPV method can reach PLN 281.5 million.

The Author does not decide explicitly, what level of the above effects is achievable in the LW Bogdanka SA mining practice, which should be subject to a natural verification in the future, once relevant recommendations are implemented. The value of expected effects is undoubtedly affected by the determination to achieve them and a change of qualitative against quantitative criteria importance in the management methods.

Favourable economic results, at small costs, of such a solution implementation make it very interesting from the economic point of view and induce to recommend solutions enabling an achievement of indicated economic effects. This is especially important in the era of current hard coal mining crisis in Poland.

In the monograph the Author presented his own conception of a production process automation and monitoring, which he implemented in the LW Bogdanka SA, consisting of a few interconnected components. It resulted from the experience gathered during an implementation of large research projects for the KGHM Polska Miedź S.A., JSW S.A., and Tauron Wydobycie S.A. In accordance with the developed conception the Integration Platform is the central element of the system, being a significant tool for the information integration, enabling a standardised exchange of data between systems built in various technologies, or using various communication protocols.

The integration platform, suggested for the Bogdanka mine, uses the SOA (*Service Oriented Architecture*) technology, which is a recognised architecture in contemporary systems, allowing to provide the application functionality as a service. It enables creating new systems on the basis of already existing services, without the need for a generation of a new application code or reducing this action to a significant extent. A service approach to the solution architecture and data integration facilitates the management of connections between the systems exchanging the data between themselves. Moreover, the mutual network of connections between systems is simplified and the impact of changes (modifications) in one system, on the functioning of the entire IT environment processing the data, is substantially limited. In addition, because of the SOA approach, it is possible to locate quickly incorrectly operating services (functionalities of individual systems). The versatility of the Integration Platform allows to use it also for an integration of the data between systems other than those, which are used in the proposed solution.

So-called Technical Data Warehouse is an additional component of the Integration Platform, differing from a standard definition of this system. The task of this component consists in gathering the data from industrial automation systems and in preliminary processing of received signals, including the signals correlation, their interpretation and adjustment in the case of delivery of data previously not defined for an interpretation. The necessity to standardise the data transfer from the control systems to the Technical Data Warehouse by means of the OPC UA (*Open Platform Communications Unified Architecture*) standard is the key assumption of the presented conception.

The Support System for the Mine Operations Maintenance is another important element of the proposed solution. From the point of view of provided functionality this is the most important component of the solution. The task of this system consists in gathering information about the production resources of the mine, which require sustained maintenance. This tool supports the management of overhauls, maintenance activities, or repairs. Two basic types of equipment can be distinguished as those to be managed with use of the Support

System for Operations Maintenance, i.e. the equipment managed by electricians and the equipment managed by mechanics.

The assumption is that the Support System for Mine Operations Maintenance comprises by its actions all the areas of the mine in such a way that the data from the areas are available only for the users working in those areas.

The Support System for Mine Operations Maintenance during its operation gathers large amounts of data, entered both automatically and manually. An effective analysis of information, held in the system, can substantially improve the quality of made management decisions based on the results of suggested solution operations. The building of the Centre for Advanced Data Analysis (CADA) is suggested for processing large amounts of data, gathered both in the Technical Data Warehouse, the Support System for Mine Operations Maintenance and also in the existing area systems, whose data are correlated with the other components of the proposed solution.

The Underground Industrial Automatic Control Systems are a significant source of data both for the Technical Data Warehouse and for the Support System for Mine Operations Maintenance. The suggested standardisation of data making available and integrating from the existing SCADA systems is aimed at the direct reading of information on the operation of machinery and equipment in the monitored objects. The use of the aforementioned OPC UA standard is proposed. In accordance with the proposed conception the OPC UA servers, integrating the data, are to be connected with central components of the solution via the Integration Platform and they enter the Technical Data Warehouse for preliminary processing of the data from the mining plant operations.

The mechanisation and automation degree of basic processes, carried out in the mine, simply forces the necessity to combine the production management and reporting areas with the area of machinery and equipment operations maintenance. This operation is the foundation of the presented conception from the point of view of the TPM (*Total Productive Maintenance*) methodology implementation.

The automation and monitoring conception of the production process in the LW Bogdanka SA, presented herein, assumes that the hitherto functionality of the LW Bogdanka SA technical and visualisation systems layer is not subject to a change and still comprises the online support of the operational supervision. The basic difference includes the parallel performance of making the data available to the production support system, operating in a longer time horizon. This action does not involve dispatchers, but analysts and planners. The data exchange among the areas should comprise information on the course of production, the process of equipment operation, the movement of ROM, the movement of people and the state of transport orders execution. A development and implementation of permanent mechanisms oriented onto making the data available from the SCADA environment to the intermediating layer systems, e.g.: MES (*Manufacturing Execution System*), CMMS (*Computerised Maintenance Management Systems*) are crucial factors that condition the effectiveness of the entire solution.

The conception of building a decision support system in the field of production activity, using IT solutions and production monitoring, presented by the Author, is the foundation of one of the main LW Bogdanka SA strategic objectives, functioning under the slogan of building so-called Intelligent Solutions Mine. The Intelligent Solutions Mine, as meant by the LW Bogdanka SA, is a series of innovative technical solutions, that should result in the mining effectiveness increase, simultaneously ensuring the safety of people working underground and minimisation of negative environmental impact. Having that in mind, the basic assumptions of the constructed system were developed within the R&D work carried out by the mine and the MEERI PAS. The initiatives related to the following areas were kept at the forefront:

– the deposit management,
– the opening scheduling, a development and mining operations,
– the production resources management,
– the production processes monitoring,
– the registration of the company assets combined with the coordinated procurement policy.

At the same time the solution architects were obliged to consider the specificity of the mining plant, participating in the project, selecting together the system components, which ultimately will ensure:

– a flexible system development,
– an integration with the existing solutions,
– a possibility of operation both with the present and the target financial-accounting system.

An achievement of the following business effects was the measure of the planned implementation successes:

– achieving standardisation in the field of all the dispatcher reporting tasks,
– direct monitoring of production processes in the full scope on the Management Board Office level,

- an increase in the production processes monitoring – a reduction of a failure risk,
- an increase in the machinery and equipment availability, meaning an increase in the production and sales volume,
- shortening the time of response to failures.

The presented conception of the LW Bogdanka SA production processes automation and monitoring is the heart of the Intelligent Mine idea, for which an establishment and appropriate legitimisation of fully decision-capable Centre for Advanced Data Analysis (CADA), in which the important information from production monitoring and visualisation systems is analysed and reported to the mine management on a current basis, is crucial.

There is no doubt that reacting in real time to irregularities, if any, as well as making quick and right decisions, results in obtaining a real picture of the production capacity and of a possibility to improve the profitability, without any significant disturbance to functioning of the Mining Plant Operations Dispatcher.

The studies carried out by the Author confirm the statement that the coal pollution is an unfavourable process, having a negative impact on the ROM extraction efficiency, however, there are possibilities to control the amount of pollution, both through the use of available technological and technical solutions, changing the process of deposit mining and the scale of pollution impact on the efficiency of the ROM extraction, which can be controlled and evaluated with support of appropriate IT solutions.

An important aspect of this research work consisted in the assessment of possibilities to use the knowledge on the forecast of the ROM pollution in the process of mining production scheduling. As it was found, the advance in the field of mining production planning and processes computerisation allows to model the deposit form and its qualitative parameters efficiently.

Presenting the conception in this monograph, the Author fully recognises the scientific achievements comprised by the hitherto publications from the studied issue field - the achievements, which directly or indirectly created the environment enabling to prepare principles of the ROM quality management system in a hard coal mine.

The monograph gives foundations for mining systems designing and the ROM pollution forecasting, depending on mining-geological conditions of extraction operations. It is also possible to use the developed methodology for practical purposes in the case of a mine with other geological-deposit conditions and for various mining systems.

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References

[1] Abigmow J.M., Owobienko B.K. 1971: Osnowy ekonomiczeskoj ocenki potier rudy pri otkrytom sposobie razrabotki. Izd. „Nauka”, Leningrad;

[2] Agoszkow M.I. 1974: Techniko-ekonomiczeskaja ocienka izwleczenia polieznych iskopajemych iz niedr. Izd. „Niedra” Moskwa;

[3] Agoszkow M.J., Panfiłow E.J. 1973: Jedinaja kłasyfikacja potier twiordych polieznych iskopajemych pri rozrabotkie miestorożdienii, Gornyj Żurnał nr 3;

[4] Agoszkow M.J., Rysow P.A. 1967: Doskonalenie metody liczenia strat i zubożenia kopaliny przy wybieraniu złóż rud systemami z zawałem, Gornyj Żurnał nr 3;

[5] Ajdukiewicz Z. 1957: Optymalna wielkość kopalni, Drogi postępu w górnictwie. Zjazd Naukowo-Techniczny - Katowice. Warszawa;

[6] Annels A.E., 1996: Ocena złóż surowców mineralnych - podejście praktyczne 1st Edn., Chapman and Hall, UK;

[7] Bajkow B.N. 1973: Socjalno-ekonomiczeskije problemy tacjonalnowo ispolzowania niedr. Izd. Niedra, Leningrad;

[8] Bajkow B.N. 1978: Opriedielenie potier i razubożenia na karierach cwietnoj metalurgii, Izd. „Niedra” Moskwa;

[9] Bajkow B.N., Kuczko W.S. 1974: Techniczno-ekonomiczeskije normirowanie potier i razubożenia poleznych iskopajemych pri dobyczi, Izd. „Niedra” Moskwa;

[10] Biezrukowa L.G., Małkin A.S. 1979: Ekonomiczeskaja ocienka rieszienij pri projektirowanii i razwitii szacht. Izd. „Niedra” Moskwa;

[11] Bindels G, 1964: Ein Verfohren zur Bestimmung des technisch und wirtschaftlich gunstigsten Zuschnitts von Flózbetrieben mit anschliesseender laufender Uberwachung durch die Plankbstenrechnung. Gluckauf, nr 23;

[12] Bock I., 1996: Selective blast mining in gold mines, Journal of the Southern African Institute of Mining and Metallurgy, pp 183-254;

[13] Bock I., Jagger L., Robinson R.E. 1998: An economic model for gold and platinum mining using selective blast mining, Journal of the Southern African Institute of Mining and Metallurgy, 98(2), pp 75-84;

[14] Brewis A.A.C. 1995: Narrow vein mining I: Steep veins. Mining Magazine, 173, p. 116-130;

[15] Bromowicz R. 1963: Metoda wyznaczania optymalnej wielkości obszaru górniczego kopalni elementarnej. Zeszyty Problemowe Górnictwa, tom 1, zeszyt 2;

[16] Brzychczy E. 2012: Metoda modelowania i optymalizacji robót eksploatacyjnych w wielozakładowym przedsiębiorstwie górniczym. Kraków: Wydawnictwa AGH. Seria Rozprawy i Monografie, nr 245;

[17] Butra J. 1981: Wpływ wybranych czynników i parametrów eksploatacji systemami komorowo-filarowymi na straty eksploatacyjne i zubożenie rudy - praca doktorska, Politechnika Wrocławska, Wrocław 1981

[18] Campbell G. 1994: Geophysical contributions to mine-development planning: A risk reduction approach, in C.R. Anhacusser, ed., Proceedings XV CMMI Congress: SAIMM Symposium Series, pp. 283-325;

[19] Chudek M., Dąbrowski J. 1979: Model matematyczny metody optymalizacji struktury i wielkości modernizowanej kopalni węgla kamiennego. Projekty - Problemy. Budownictwo Węglowe, nr 9;

[20] Chudek M., Paździora J., Dąbrowski J. 1979: Techniczne warianty struktury i wielkości modernizowanej kopalni węgla kamiennego. Projekty - Problemy. Budownictwo Węglowe, nr 7-8;

[21] Chugh Y, Moharana A, Patwardhan A. 2004: An analysis of the effect of out-of-seam dilution on coal utilization. Proceedings of the VI International Conference on Clean Technologies for the Mining Industry, University of Concepción, Chile. University of Concepción, Chile, pp 18–21;

[22] Chugh Y. P., McGolden M., Hirschi J., Thatavarthy K., Moharana A., Roberts E., Meyer B., Carty R., 2002: Development and demonstration of advanced mining technologies to reduce production costs. Illinois Clean Coal Institute ICCI Project Number: 01-1/1.1 D-5;

[23] Chugh Y.P., McGolden M. 2003: Development and demonstration of advanced mining technologies to reduce production costs. Final Technical Report, USA;

[24] Chugh, Y. i A. Moharana, 2005: Zubożenie w podziemnym górnictwie węgla w USA - wpływ na produkcję i przeróbkę węgla oraz zagospodarowanie odpadów. Proceedings of the International

Conference on Mineral Processing Technology, January 6-8, 2005, New Dehli, India, pp: 10-20;

[25] Cieszkowski H., Krajewski St., Siewierski St. 1984: Technologia eksploatacji złóż rud miedzi w LGOM zwłaszcza grubych. Materiały konferencyjne, Polkowice;

[26] Claes F., Rusche H. 1975: Die Berechnung von Flozbetriebskosten mit einem Programmsystem fur die Zeschittsplanung, Guckauf Forsch-H-e, nr 2;

[27] Clark L., Pakalnis R. 1997: An empirical design approach for estimating unplanned dilution from stope hangingwalls and footwalls, in Proceedings 99th Canadian Institute of Mining and Metallurgy (CIM) Annual General Meeting, Calgary (Canadian Institute of Mining and Metallurgy: Montreal);

[28] Clark, L.M., 1998: Minimizing dilution in open stope mining with a focus on stope design and narrow vein longhole blasting. Master of applied science. Praca dyplomowa. University of British Columbia, Canada;

[29] Cyrnek C. 1974: Prognozowanie nakładów inwestycyjnych w zakresie górniczych robót udostępniających i przygotowawczych w zależności od tempa realizacji w cyklu budowy głębinowych kopalń węgla kamiennego. Zeszyty Naukowe AGH, Górnictwo zesz. 59, Kraków;

[30] Cyrnek C., Soliński I. 1977: Próba optymalizacji cyklu budowy głębinowej kopalni z uwzględnieniem rozwoju projektowanego wydobycia. Kwartalnik AGH „Górnictwo", zesz. 4;

[31] De Wet J.A.J., Hall D.A. 1994: Interpretation of the Oryx 3-D seismic survey. In C.R. Anhaeusser ed., Proceedings XV CMMI Congress: SAIMM Symposium Series, S14, 3, pp. 259-270;

[32] Denk D. 1977: Untersuchungen uber den optimalen Zuschnitt von Strebbetrieben. Gluckauf Forsch-H-e, nr 1;

[33] Dorstewitz G. 1967: Moderne Verfahren der wirtschaftlichen Analyse von Bergbaubetrieben Gluckauf, nr. 19;

[34] Drewniak R., Rosielski J. 1980: Zasady ustalania strat zasobów w procesie eksploatacji jako element racjonalnej gospodarki zasobami węgla kamiennego. Prace Głównego Instytutu Górnictwa - seria dodatkowa. Katowice;

[35] Dyczko A. 1998: Aktualna sytuacja ekonomiczna i perspektywy rozwoju KGHM Polska Miedź S.A. Kwartalnik AGH „Górnictwo" rok 22, zeszyt 2;

[36] Dyczko A. 2002: Techniczno-ekonomiczne aspekty zubożenia złóż rud miedzi na przykładzie kopalń KGHM Polska Miedź S.A. Wyd. Gospodarka Surowcami Mineralnymi Tom 18 Zeszyt specjalny, Kraków;

[37] Dyczko A. 2004: Niektóre aspekty zubożenia urobku w kopalniach rud miedzi. Materiały Konferencyjne Gospodarki Surowcami Mineralnymi. Wydawnictwo IGSMiE PAN, Kraków;

[38] Dyczko A. 2006: Cienkie pokłady – rola i miejsce w bazie surowcowej Polski. Wydawnictwo Gospodarka Surowcami Mineralnymi. Tom 22. Zeszyt specjalny, Kraków;

[39] Dyczko A. 2007: Thin coal seam, their role in the reserve base of Poland International Mining Forum 2007, pp 81-89, Taylor & Francis Group plc, London, UK;

[40] Dyczko A. 2009: „Informacyjni technologii u polskomu girnictwi – wid idei do prakticznoj realizacji”. III Mieždunarodnaja Naukowo-Prakticieskaja Konferencija „Szkoła Podzemnoj Rozrabotki”, 13-19 września 2009 r., Gaspra, Krym, Ukraina; Narodowy Uniwersytet Górniczy, Dniepropietrowsk, p. 156-166;

[41] Dyczko A. 2013: W kierunku inteligentnej kopalni przyszłości. Prezentacja wygłoszona podczas Międzynarodowej konferencji ECOFORUM „Po pierwsze środowisko” Lublin;

[42] Dyczko A. 2014: Rola informatyki w inteligentnej głębokiej kopalni przyszłości. Szkoła Eksploatacji Podziemnej, Kraków – prezentacja;

[43] Dyczko A. 2016: System wspomagania zarządzania w LW „Bogdanka” SA – fundament budowy Kopalni Inteligentnych Rozwiązań. Wiadomości Górnicze, nr 3, s. 230–241;

[44] Dyczko A. i in. 2013: Strategia informatyzacji LW Bogdanka SA – materiał nie publikowany;

[45] Dyczko A., Dunst N., Galica D. 2014: Wykorzystanie narzędzi informatycznych do modelowania złoża węgla kamiennego, 2014, nr 10, 458-472. Wiadomości Górnicze;

[46] Dyczko A., Galica D., Kowalczyk I. i inni 2015: Opracowanie metodyki wykonywania i analizowania dołowych obserwacji geologicznych pod kątem obniżenia ilości skały płonnej i podniesienia dokładności prognoz jakości urobku. Kraków. Praca niepublikowana;

[47] Dyczko A., Galica D., Kudlak Ł.: 2014: Wybrane aspekty zastosowania narzędzi informatycznych w projektowaniu

i harmonogramowaniu produkcji górniczej, nr 9, 448-457. Wiadomości Górnicze, Katowice;

[48] Dyczko A., Galica D., Sypniowski S. 2012: Deposit model as a first step in mining production scheduling. CRC Press. Geomechanical Processes during Underground Mining: School of Underground Mining;

[49] Dyczko A., Galica D., Sypniowski S., Szot M. 2013: Planowanie i harmonogramowanie produkcji górniczej w LW „Bogdanka" S.A., Wiadomości Górnicze 2013, 64 (7-8), 422-426;

[50] Dyczko A., Kicki J., Myszkowski M., Stopa Z., Tor A. 2011: Nowe spojrzenie na technikę i technologię eksploatacji cienkich pokładów węgla kamiennego. Materiały IMF 2011. Wyd. Fundacja dla AGH. Ark. Wyd. 26. Kraków;

[51] Dyczko A., Kicki J., Myszkowski M., Stopa Z., Tor A. 2013: New Techniques and Technologies in Thin Coal Seam Exploitation. International Mining Forum. Wyd. CRC Press/Balkema;

[52] Dyczko A., Kicki J., Stopkowicz A. 2007: Wykorzystanie narzędzi informatycznych w zarządzaniu gospodarką złożem w kopalniach rud. Materiały Konferencyjne. Górnictwo cynku i ołowiu na początku XXI wieku, s. 111-120, ZGH Bolesław 23.11.2007 r.;

[53] Dyczko A., Kołomański D., Kowalczyk I. 2016: Modelowanie blokowe skał stropowych pokładów węgla LW Bogdanka SA, Zeszyty Naukowe Instytutu Gospodarki Surowcami Mineralnymi i Energią Polskiej Akademii Nauk, Kraków, nr 92, s. 427–438;

[54] Dyczko A., Kopacz M. 2008a: „Influence of impoverishment on forming characteristics of mining investment projects and selected production figures exemplified by a copper deposit" 21st World Mining Congress, Session Economics, management and design in the mining industry, Kraków;

[55] Dyczko A., Kopacz M. 2008b: „Wpływ zubożenia na kształtowanie wartości górniczych projektów inwestycyjnych oraz wybranych wielkości produkcyjnych na przykładzie złoża rud miedzi, Gospodarka Surowcami Mineralnymi t. 24 z. 4/2, s. 251-264, wyd. IGSMiE PAN, Kraków;

[56] Dyczko A., Kopacz M., Polak R., 2013: Opracowanie dynamicznego modelu zarządzania wartością i strategią LW „Bogdanka" SA z uwzględnieniem wpływu kluczowych projektów inwestycyjnych na wartość przedsiębiorstwa - Wstępne studium wykonalności projektu:

INTELIGENTNA KOPALNIA system zarządzania produkcją w LW Bogdanka SA Kraków – materiał niepublikowany;

[57] Dyczko A., Kowalczyk I., Mól D.: 2016: Ciągła kontrola parametrów jakościowych węgla uzupełniona wynikami profilowań geologicznych i modelowania złoża środkiem do obniżenia ilości skały płonnej w LW Bogdanka SA, Wiadomości Górnicze nr 7-8;

[58] Dyczko A., Krawczyk A. 2011: Geomatyka górnicza – praktyczne zastosowania. Monografia. Wyd. Fundacja dla AGH. Ark. Wyd. 26. Kraków;

[59] Dyczko A. 2018: Metodyka oceny wpływu zanieczyszczenia urobku na efektywność procesu produkcji węgla kamiennego na przykładzie LW Bogdanka SA, praca doktorska AGH Kraków, niepublikowana;

[60] Dziura T., Zapotocki A. 1976: Badania górniczego wykorzystania złoża rud miedzi w kopalniach LGOM, Zeszyt Naukowy Politechniki Śląskiej nr 70, Gliwice;

[61] Elbrond J. 1994: Economic effects of ore losses and rock dilution, Canadian Institute of Mining and Metallurgy (CIM) Bulletin, 87(978):131-134;

[62] Fiszel H. 1969: Teoria efektywności inwestycji i jej zastosowanie. PWN, Warszawa;

[63] Gałąź H. 1994: Analiza wyników produkcyjnych ściany 4/I wyposażonej w wysokowydajny kompleks urabiająco-odstawczy i ich wpływu na poziom jednostkowych kosztów oddziałowych KWK Bogdanka;

[64] Gałąź H., Stachowicz St. 1994: Wzrost efektywności KWK Bogdanka SA w warunkach gospodarki rynkowej. Materiały z konferencji naukowo-technicznej nt.: Przemysł wydobywczy gwarantem efektywności krajowej gospodarki”. Kraków;

[65] Gałąź H., Stachowicz St. 1995: Organizacja pracy dla frontu eksploatacyjnego o wydobyciu 10 000 ton na dobę na bazie krajowego wyposażenia mechanizacyjnego w KWK Bogdanka SA – Biblioteka Szkoły Eksploatacji Podziemnej, Nr 11. Szczyrk;

[66] Gamow T.A. 1973: Ekonomiczeskaja ocenka miestorożdieniji cwietnych metałłow, Niedra, Moskwa;

[67] Gawroński G., Kozek B., Stachowicz St. 1988: Porównanie intensywności zaciskania wyrobisk chodnikowych prowadzonych w zrobach i pokładzie w kopalni K-1 w Bogdance – Wiadomości Górnicze, Nr 10/88;

https://doi.org/10.32056/KOMAG/Monograph2022.1

[68] Gazda W. 1971: Metoda minimalizacji kosztów wybierania pokładu węgla z pola wybierkowego. Praca doktorska. Politechnika Śląska, Gliwice;

[69] Głuch P., Kosiński Z., Limburski T., Stachowicz St. 1987: „Obudowa połączeń wyrobisk korytarzowych" – Zeszyty Naukowe Politechniki Śląskiej w Gliwicach, Nr 157;

[70] Goszcz A. 1971: Wydobycie i wydajność ściany kompleksowo zmechanizowanej jako funkcja długości ściany. Wiadomości Górnicze, nr 1, Katowice;

[71] Grudziński Z. 2009: Propozycje struktur cenowych dla węgla kamiennego energetycznego i węgla brunatnego. Polityka Energetyczna 12 (2/2), s. 159-171, Kraków;

[72] Grudziński Z. 2012: Metody oceny konkurencyjności krajowego węgla kamiennego do produkcji energii elektrycznej. Kraków: IGSMiE PAN. Seria, Studia, Rozprawy i Monografie, nr 180, Karków.

[73] Henning, J.G., H.S. Mitri, 2007: Numerical modelling of ore dilution in blasthole stopping. International Journal of Rock Mechanics and Mining Sciences 44(5):692-703, DOI:10.1016/j.ijrmms.2006.11.002

[74] Horak G. 1966: Techniczno-ekonomiczne warunki eksploatacji grubych pokładów węgla kamiennego. Wyd. „Śląsk", Katowice;

[75] Hurysz J., Sikora W. 1963: Wpływ technicznych parametrów ścian zawałowych na koncentrację wydobycia. Przegląd Górniczy, nr 9, Katowice;

[76] IGSMiE PAN 2008: Analiza stanu techniki strugowej na świecie i możliwości jej wdrożenia w LW Bogdanka SA, praca niepublikowana;

[77] Ingler D. 1984: Mining Methods, Rock Dilution and Ore Losses;

[78] Janik G., Kuś R. 1992: Zintegrowany system komputerowego wspomagania produkcji w zakładach górniczych;

[79] Jawień M. 1965: Metoda wyznaczania optymalnej wielkości kopalni typu jednostkowego wraz z określeniem wielkości tego typu kopalń dla wybranych złóż polskich. Zeszyty Naukowe AGH, z. 62, Kraków;

[80] Jawień M., Hajdasiński M. 1973: Optymalizacja rozwoju górnictwa paliw stałych w świetle rachunku efektywności przedsięwzięć i zamierzeń inwestycyjnych. Zeszyty Naukowe AGH, Górnictwo z. 47;

[81] Jaworski B., Kozek B. 1987: Optymalizacja parametrów techniczno-organizacyjnych oraz doboru mechanizacji robót w ścianach o wysokiej wydajności w oparciu o rachunek kosztów oddziałowych:

[82] Jurdziak L., Kawalec W. 2011: Wirtualna kopalnia - nowe możliwości zapewnienia bezpiecznej i efektywnej ekonomicznie eksploatacji. W: X Szkoła Geomechaniki 2011: jubileuszowa międzynarodowa konferencja: materiały naukowe, Gliwice-Ustroń, 18-21 października 2011. Zeszyty Specjalne Wydziału Górnictwa i Geologii Politechniki Śląskiej, s. 93-111;

[83] Jureczko J., Buczek H., Drewniak R. i in. 1986: Analiza i ocena gospodarki zasobami złóż kopalin na przykładzie wybranych kopalń węgla kamiennego, miedzi, cynku i ołowiu część II: Analiza gospodarki zasobami złóż węgla kamiennego, październik 1986 r., Katowice;

[84] Kaiser P., Henning J., Cotesta L., Dasys A. 2002: Innovations in mine planning and design utilizing collaborative immersive virtual reality (CIRV). 104th CIM Annual General Meeting, 28 April – 1 May 2002, Vancouver, BC;

[85] Kamionka M. 1971: Minimalne opłacalne wydobycia ze ścian z obudową zmechanizowaną. Wiadomości Górnicze, nr 3, Katowice;

[86] Kamionka M., Wasilewski K. 1967: Wpływ prędkości postępu frontu eksploatacyjnego na efekty ekonomiczne obudowy zmechanizowanej w ścianach. Przegląd Górniczy, nr 3, Katowice;

[87] Kamionka M., Zbyradowski T. 1967: Określenie optymalnej długości ściany. Mechanizacja Górnictwa 1967, nr 4;

[88] Kamionka M., Zbyradowski T. 1969: Określenie optymalnych parametrów ścianyw zależności od warunków naturalnych i systemów mechanizacji. Praca naukowo-badawcza ZKMPW, nr 68, Gliwice;

[89] Kamionka N., Zemła P. 1972: Efektywność ekonomiczna ścianowych systemów eksploatacji „od pola" i „do pola". Ekonomika Mechanizacji i Automatyzacji Górnictwa;

[90] Kapageridis I. 2005: Input space configuration effects in neural network-based grade estimation. Computers Geosciences, 31: 704-717;

[91] Kapłunow R.P. 1938: Klasyfikacja i metodyka określania strat przy urabianiu złóż rud, Gornyj Żurnał nr 10;

[92] Kapłunow R.P. 1948: Woprosy gornowo dieła, Uglietechizdat, Moskwa;

https://doi.org/10.32056/KOMAG/Monograph2022.1

[93] Karbownik A. 1978: Struktura procesu projektowania w górnictwie. Projekty - Problemy. Budownictwo Węglowe 1978, nr 11;

[94] Karbownik A. 1981: Podstawy teorii projektowania. Zagadnienia wybrane dla kierunków górniczych. Skrypty Uczelniane Politechniki Śląskiej nr 1030. Gliwice;

[95] Karbownik A. 1982a: Analiza ekonomicznych kryteriów oceny decyzji projektowych i inwestycyjnych. Budownictwo Węglowo. Projekty - Problemy, nr 8;

[96] Karbownik A. 1982b: Podstawy projektowania kopalń. Cz.1. Skrypty Uczelniane Politechniki Śląskiej nr 1096, Gliwice;

[97] Karbownik A. 1987: Podstawy teorii projektowania. Zagadnienia wybrane dla kierunków górniczych. Politechnika Śląska. Gliwice;

[98] Karbownik A., Pogonowski T., Piwko S., Łyczbiński R. 1978: Analiza optymalizacyjna parametrów geometrycznych i produkcyjnych ściany zmechanizowanej wykonana za pomocą EMC. Zeszyty Naukowe Politechniki Śląskiej w Gliwicach, S. Górnictwo, nr 79, Gliwice;

[99] Karbownik A., Pogonowski T., Sznurawa J. 1975: Optymalizacja parametrów ścian zmechanizowanych z wykorzystaniem EMC. Projekty - Problemy. Budownictwo Węglowe, nr 4;

[100] Karbownik A., Pogonowski T., Sznurawa J. 1979: Dobór najkorzystniejszych parametrów geometryczno-produkcyjnych ściany zmechanizowanej. Zeszyty Naukowe Politechniki Śląskiej w Gliwicach, S. Organizacja, Nr 8, Gliwice;

[101] Kicki J., Dyczko A. 2008: A Mining Software System at “Bogdanka” S.A. Underground Coal Mine – an Important Implementation of an Integrated Mineral Deposit Management System in the Polish Mining Industry. International Mining Forum, s. 141-147, Taylor & Francis Group, London, UK;

[102] Kicki J., Dyczko A. 2009: Informatyka w polskim górnictwie - gdzie jesteśmy? Materiały Konferencyjne Szkoły Eksploatacji Podziemnej, Kraków;

[103] Kicki J., Dyczko A. 2012: Planowanie produkcji górniczej z wykorzystaniem rozwiązań IT. Wiadomości Górnicze 2012, 63 (7-8), 386-397, Katowice;

[104] Kicki J., Dyczko A., 2010: Przyszłość rozwiązań informatycznych w górnictwie – systemy Business Intelligence (BI) i nie tylko. Wiadomości Górnicze, nr 7-8/2010. pp. 406-415, Katowice;

[105] Kicki J., Dyczko A., Timler M. 2009: „Koncepcja automatyzacji i monitoringu procesu produkcyjnego w kopalni podziemnej. Materiały Konferencyjne międzynarodowego Kongresu Górnictwa Rud Miedzi – Perspektywy i wyzwania. 24-26 września, Lubin;

[106] Kicki J., Sobczyk E.J. i in. 2016: Prawne, techniczne, ekonomiczne i środowiskowe aspekty gospodarki skałą płonną w kopalniach węgla kamiennego. Wyd. IGSMiE PAN, Kraków;

[107] Kindla L, Kozdrój M., Sitko W. 1968: Wpływ przerw w pracy na efektywność produkcji przodków górniczych. Zeszyty Naukowe Politechniki Śląskiej, S. Górnictwo, Nr 29, Gliwice;

[108] Knissel W., Jurgen H. and Fahlbusch M. 1995: Significance of Selective Mining Exploitation for Economical and Environmentally Beneficial Underground Ore Mining. In. Jnl. Min. Resources Engineering, Imp. Coll. Press, 5, 2, pp. 165-174;

[109] Konopko W., Hryniak E., Kula B., Kurzeja A. 1979: Optymalna długość ścian zawałowych. Biuletyn GIG, nr 1, Katowice;

[110] Konstantynowicz E. 1971: Geneza cechsztyńskich złóż miedzi w Polsce. Prace Naukowe Instytutu Górnictwa Politechniki Wrocławskiej nr 4, Wrocław;

[111] Kopacz M. 2015a: Ocena kosztów gospodarki skałą płonną w funkcji zmiennego poziomu współczynnika uzysku węgla netto na przykładzie kopalni węgla kamiennego, Gospodarka Surowcami Mineralnymi – Mineral Resources Management, tom 31, zeszyt 3, strony 121-144, Kraków;

[112] Kopacz M. 2015b: The impact assessment of quality parameters of coal and waste rock on the value of mining investment projects - hard coal deposits, Mineral Resources Management, vol. 31, Issue 4, p. 5-30;

[113] Kot A. 1972: Ustalenie kryteriów obliczania zasobów przemysłowych węgla kamiennego. Praca doktorska, Gliwice;

[114] Kowalczyk I., Galica D., Dyczko A., Kołomański D., Mól D. 2016: Model geologiczny złoża kopaliny – sposób konstrukcji, rola i znaczenie w procesie planowania i harmonogramowania eksploatacji. Zeszyty Naukowe Instytutu Gospodarki Surowcami Mineralnymi i Energią Polskiej Akademii Nauk, Kraków, nr 92, Kraków;

[115] Krasiczyński K. 1972: Ocena efektywności komorowo-filarowych systemów eksploatacji stosowanych w LGOM. Rozprawa doktorska,

https://doi.org/10.32056/KOMAG/Monograph2022.1

Akademia Górniczo-Hutnicza im. St. Staszica w Krakowie. Instytut Górnictwa Podziemnego.

[116] Krasiczyński K. 1974: Sposób obliczania strat rudy i metalu w procesach eksploatacyjnych. Cuprum nr 2, Wrocław;

[117] Krasiczyński K., Majchrowicz R., Wilczyński W. 1974a: Celowość wybierania złoża miedzi o miąższości poniżej 2,7 m komorowo-filarowymi systemami eksploatacji. Komunikat nr 58 Instytutu Górnictwa Politechniki Wrocławskiej, Wrocław;

[118] Krasiczyński K., Majchrowicz R., Wilczyński W. 1974b: Wpływ geometrii systemów komorowo-filarowych z zawałem stropu na straty eksploatacyjne rudy w filarach podporowych, komunikat nr 59 Instytutu Górnictwa Politechniki Wrocławskiej, Wrocław;

[119] Krupiński B. 1963: Zasady projektowania kopalń. Cz. 1 i 2, Katowice;

[120] Kryj K. i in. 2011: Problem ekonomicznych skutków urabiania pozapokładowej skały płonnej w kopalniach węgla kamiennego. Środkowo-pomorskie towarzystwo naukowe ochrony środowiska, Tom 13;

[121] Kustra A., Sierpińska M. 2013: Realizacja funkcji budżetowania i kontroli w cyklu życia pola eksploatacyjnego w kopalni węgla kamiennego. Przegląd Górniczy, t. 69, nr 9, s. 94–98, Katowice.

[122] Lachman K. 1978: Optymalizacja budowy poziomu kopalni w warunkach Lubelskiego Zagłębia Węglowego. Praca doktorska AGH, Kraków;

[123] Lama R.D. 1977: Principles of underground coal mine design - an approach (part 1). Colliery Guardian, nr 5;

[124] Lisowski A. 1981: Gospodarka zasobami złóż surowców mineralnych. Przegląd Górniczy, nr 6, Katowice;

[125] Lisowski A. 2001: Podstawy ekonomicznej efektywności podziemnej eksploatacji złóż. Katowice – Warszawa. Wyd. PWN;

[126] Lisowski A. 2013: Górnictwo węgla kamiennego w Polsce: krytyczna ocena sposobu przeprowadzenia rynkowej transformacji i dyskusja problemów wciąż oczekujących na rozwiązanie 2006-2013. Wyd. Główny Instytut Górnictwa, Katowice;

[127] Lorenz i in. 2002: Lorenz, U., Blaschke, W., Grudziński, Z. Propozycja nowej formuły sprzedażnej węgla energetycznego przeznaczonego dla energetyki zawodowej. IGSMiE PAN. Seria, Studia, Rozprawy i Monografie, nr 112, Kraków;

[128] Lubosik Z. 2009: Geoinżynieryjne i ekonomiczne kryteria eksploatacji węgla kamiennego z resztkowych parcel pokładów. Prace Naukowe GIG. Górnictwo i Środowisko, nr 3, Katoiwce;

[129] Łokszin B.S., Korobki W.J. 1968: K opriedieleniju oczistnowo zaboja pri wyjemkie połogich ugolnych plastow ugliedobywajuszczimi kompleksami. Ugol, nr 10;

[130] Magda R. 1982: Koncepcja czasoprzestrzennego odwzorowania procesu wybierania pokładu węgla. Sesja Naukowa AGH - Wybrane Materiały Konferencyjne, Kraków, s. 245 – 257;

[131] Magda R. 1983: Zasady integracyjnej metody modelowania procesu wybierania pokładów węgla. Kwartalnik AGH - Górnictwo, vol. 7, nr 3, s. 237 – 249;

[132] Magda R. 1984: Odwzorowanie wartości kapitałowej nakładów finansowych związanych z robotami przygotowawczymi i eksploatacyjnymi w ramach pojedynczego pola ścianowego z uwzględnieniem oprocentowania ciągłego. Kwartalnik AGH - Górnictwo, vol. 8, nr 2, s. 169 – 187;

[133] Magda R. 1990a: Racjonalizacja rozcinki pola eksploatacyjnego na pola ścianowe. Górnictwo - Kwartalnik AGH, 1990, nr 4;

[134] Magda R. 1990b: Analityczne modelowanie kosztów wybierania pól ścianowych w polu eksploatacyjnym. Górnictwo - Kwartalnik AGH, nr 4;

[135] Magda R. 1994: Wytyczne racjonalnego projektowania pól ścianowych. Szkoła Eksploatacji Podziemnej. Wydawnictwo CPPGSMiE PAN, Kraków;

[136] Magda R., Domański J. 1991: Racjonalizacja wybranych parametrów pól ścianowych. Archiwum Górnictwa, 1991, nr 2;

[137] Magda R., Franik T. 1989: Wykorzystanie techniki komputerowej w projektowaniu para. Zeszyty Naukowe AGH, S. Górnictwo, z. 144;

[138] Magda R., Franik T., Domański D. 1992: Ocena istotności wpływu wybranych czynników techniczno-organizacyjnych na optymalne parametry geometryczne pól ścianowych. Archiwum Górnictwa nr 3;

[139] Magda R., i in. 2002: Magda R., Magda R., Woźny T., Kowalczyk B. i in. 2002: Racjonalizacja modelu i wielkości kopalń węgla kamiennego w warunkach gospodarczych początku XXI wieku. Wyd. AGH, Kraków;

[140] Mielnikow N.W. 1973: Racjonanoje ispolzowanie mineralnych resursów, Gornyj Żurnał nr 1, Moskwa;

https://doi.org/10.32056/KOMAG/Monograph2022.1

[141] Mielnikow N.W. 1974: Buduszczije gornyja raboty, Gornyj Żurnał nr 2, Moskwa;

[142] Ministerstwo Środowiska. Rozporządzenie Ministra Środowiska. [Online] Dostępne w: http://web.archive.org/web/20051227032310/ http://www.mos.gov.pl/1akty_prawne/rozporzadzenia_ms/05.128.10 75.pdf [dostęp 31 czerwca 2015 r.];

[143] Nazarchik A.F. 1972: Issledowanija effektiwnosti razrabotki żylnych mestorożdenij. Izd. „Niedra" Moskwa;

[144] Nieć M. 1982: Geologia kopalniana. Wyd. Geologiczne, Warszawa;

[145] Nieć M., Górecki J., Sermet E., Szczepański J., Ślusarczyk G., Borowicz A. 2015: Metodyka rozpoznawania i dokumentowania złóż kopalin oraz geologicznej obsługi kopalń. XVI seminarium w ramach III Polskiego Kongresu Górniczego R. 56 nr 5, s. 21–25.

[146] Noppe M. 2003: Pomiary i kontrola zubożenia w czasie podziemnej eksploatacji węgla. In: Proceedings of the 5th mining geology conference. Bendigo, Australia; p. 243–9;

[147] Nowak A., Chmiel P. 1981: Ogólny model wyposażenia technicznego przodków ścianowych w kopalniach węgla kamiennego. Projekty - Problemy. Budownictwo Węglowo, nr 3;

[148] Pakalnis R. 1986: Empiryczne projektowanie komór w kopalni Ruttan. Praca doktorska University of British Colombia, pp: 276;

[149] Pakalnis R. C., Poulin R. and Hadjigeorgiou J. 1995: Quantifying the cost of dilution in underground mines, Mining Engineering, 47(12), pp1136-1141;

[150] Palarski J. 2009: Problemy górnictwa węgla kamiennego". Wiadomości Górnicze nr 1 s. 2-14;

[151] Parysiewicz W., Wolski J. 1962: Optymalne parametry ścian zmechanizowanych. Przegląd Górniczy, nr 11;

[152] Pogonowski T. 1971: Model teoretyczny cyklu budowy głębinowej kopalni węgla kamiennego. Zeszyty Naukowe Politechniki Śląskiej, s. Górnictwo;

[153] Pogonowski T., Karbownik A. 1976: Wyznaczanie optymalnych parametrów pola eksploatacyjnego dla potrzeb projektowania górniczego. Zeszyty Naukowe Politechniki Śląskiej, s. Górnictwo, zesz. 69, Gliwice;

[154] Polskie Stowarzyszenie Wyceny Złóż Kopalin. Kodeks Wyceny Złóż Kopalin. [Online] Dostępne w: http://www.polval.pl/kodekspolval.pdf [dostęp 31 czerwca 2015 r.];

[155] POLVAL 2008, Polskie Stowarzyszenie Wyceny Złóż Kopalin. Kodeks Wyceny Złóż Kopalin. [Online] Dostępne w: http://www.polval.pl/kodekspolval.pdf [dostęp 31 czerwca 2015 r.];

[156] Popov G. 1971: The Working of Mineral Deposits. 2nd Edn., Mir Publishers, Moscow;

[157] Porzycki J., 1972: Budowa geologiczna złoża węgla kamiennego w okolicy Łęcznej. Kwart. Geol., 16, 2: 486-487.

[158] Praca Zbiorowa Instytutu Górnictwa Politechniki Wrocławskiej 1972: Opracowanie metodyki określenia strat złożowych i eksploatacyjnych oraz zubożenia w kopalniach Lubin i Polkowice, Wrocław;

[159] Prawo geologiczne i górnicze. Ustawa z dnia 09.06.2011 (Dz. U. Nr 163/2011, poz. 981);

[160] Przybyła H.I., Chmiela A. 2007: Organizacja i ekonomika w projektowaniu wybierania węgla. Wyd. Politechniki Śląskiej, Gliwice;

[161] Rabsztyn J. 1970a: Matematyczne metody obliczania długości i wybiegu ścian. Przegląd Górniczy, nr 11;

[162] Rabsztyn J. 1970b: Matematyczne metody oblizania długości i wybiegu ścian. Przegląd Górniczy, nr 4;

[163] Rabsztyn J., Kozdrój M. 1967: Mechanizacja urabiania i ładowania węgla. Wyd. Śląsk, Katowice;

[164] Rajwa S. 2007: Wpływ wybranych wyników geoinżynieryjnych na proces przygotowania produkcji w polskich kopalniach węgla kamiennego. Katowice. Prace Naukowe GIG. Górnictwo i Środowisko, nr 4, Katowice;

[165] Revey G. F. 1998: Effects and control of overbreak in underground mining, Mining Engineering, 50(8): 63-67;

[166] Rokita J., Strzoda J. 1970: Analiza ekonomiczna w kopalniach węgla kamiennego. Wyd. Śląsk, Katowice;

[167] Rokita J., Strzoda J. 1976: Metody oceny efektywności produkcji kopalń węgla kamiennego. Wyd. Śląsk, Katowice;

[168] Rżewski W.W. 1964: Otkryte gornyje raboty w słożonych usłowiach, Niedra, Moskwa;

https://doi.org/10.32056/KOMAG/Monograph2022.1

[169] Saeedi G., Shahriar K., Rezai B., Karpuz C. 2010: Numeryczne modelowanie zubożenia pozapokładowego w eksploatacji ścianowej systemem od granic. Int J Rock Mech Min Sci, 47 (2010), pp. 533–543;

[170] Saeedi Gh, Rezai B, Shahriar K., Oraee K. 2008: Szacowanie poziomu zubożenia pozapokładowego w eksploatacji ścianowej i jego wpływ na uzysk w zakładzie przeróbczym w kopalni węgla Tabas. In: Proceedings of the international seminar on mineral processing technology, pp. 370–73, Trivandrum, India;

[171] Saginow A.S., Kwon S.S. 1972: Nowyj mietod issliedowanija i racjonalnyje sposoby wskrytija szachtnych poliej. Izd. „Niedra", Moskwa;

[172] Saginow A.S., Kwon S.S., Adiłow K.N. 1974: Mietody analiza i optimizacji tiechnołogiczieskich schiem ugolnych szacht. Izd. „Niedra", Moskwa;

[173] Scoble M. J., Moss A. 1994: Dilution in underground mining: Implications for production management, Mineral Resource Evaluation II: Methods and Case Histories, Geological Society Special Publication, 79:95-108 (Geological Society Publishing House: London);

[174] Sharp W.R. 1974: Design of an underground mine layout. Bureau of Mines Report of Investigations, nr 7828;

[175] Siska L., Vitek A. 1975: Nektere nove poznatky v projoktovani dolu. Uhli 1975, nr 4;

[176] Sikora W. (red.): Lokowanie odpadów górniczych i elektrownianych w kopalniach. Zeszyty Naukowe Politechniki Śląskiej zeszyt nr 224 Gliwice 1995

[177] Sitko W. 1973: Liniowy model matematyczny rozkładu zadań wydobywczych w polu eksploatacyjnym. Zeszyty naukowe Politechniki Śląskiej, S. Organizacja, z.3, Gliwice;

[178] Sitko W. 1976: Ekonomika górnictwa. Skrypty Uczelniane Politechniki Śląskiej nr 596, Gliwice;

[179] Sitko W., Chmiela A., Kozyra J. 1973: Projektowanie organizacji wybierania węgla kamiennego. Politechnika Śląska, Gliwice;

[180] Sobczyk E.J. 2009: Uciążliwość geologiczno-górniczych warunków eksploatacji węgla kamiennego i jej wpływ na gospodarkę złożem. Seria Studia, Rozprawy i Monografie, nr 150, IGSMiE PAN, Kraków;

[181] Soja J. 1964: Teoretyczne i praktyczne wyznaczenie wybiegu ścian zmechanizowanych na podstawie pracochłonności. Praca doktorska. Politechnika Śląska, Gliwice;

[182] Soliński I. 1977: Metoda oceny efektywności procesu budowy górniczego zespołu produkcyjnego w warunkach Lubelskiego Zagłębia Węglowego. Praca doktorska AGH Kraków;

[183] Stachowicz J. 1975: Optymalizacja procesu wybierania złoża w czynnej kopalni głębinowej węgla kamiennego w ujęciu planowania perspektywicznego i taktycznego, w świetle teorii gry z naturą. Zeszyty Naukowe Politechniki Śląskiej nr 429. s. Organizacja zesz. 6;

[184] Stachowicz St., Kosonowski J., Kozek B. 1995: Charakterystyka zaciskania wyrobisk korytarzowych kamiennych w KWK Bogdanka. Wiadomości Górnicze, Nr 5/95;

[185] Stachowicz St., Krukowski R. 1994: Stosowanie obudowy kotwiowej w świetle badań w warunkach „in situ" w KWK Bogdanka. Materiały konferencyjne z IV Konferencji „Obudowa kotwiowa w kopalniach węgla kamiennego" Ustroń, kwiecień 1994;

[186] Stokes W. P. C. 1994: Computer asisted mine scheduling. Mining Magazine;

[187] Suchan S. 1974: Metoda optymalizacji wymiarów pól eksploatacyjnych i sposobu ich przygotowania w kopalniach głębinowych. Zeszyty Naukowe AGH, s. Górnictwo zesz. 53;

[188] Suorineni F.T., Tannant D.D., Kaiser P.K., 1999: Określenie zrzutu uskokowego w otwartych komorach Int. J. Rock Mech. Min. Sci., 36: 891-906;

[189] The Coal Resource 2008: https://www.worldcoal.org/sites/default/files/resources_files/coal_resource_overview_of_coal_report%2803_06_2009%29.pdf

[190] Turek M. 2013: System zarządzania kosztami w kopalni węgla kamiennego w cyklu istnienia wyrobiska wybierkowego. Wyd. Difin, Warszawa;

[191] Villaescusa E. 1996: Excavation design for bench stoping at Mt Isa mine, Queensland, Australia, Transactions of the Institutions of Mining and Metallurgy, Mining Technology, A1-A10;

[192] Villaescusa E. 1998: Geotechnical design for dilution control in underground mining, in Proceedings Seventh International Symposium on Mine Planning and Equipment, pp 141-149. Balkema, Calgary;

https://doi.org/10.32056/KOMAG/Monograph2022.1

[193] Walisko J. 1962: Korzystna długość ścian zmechanizowanych. Przegląd Górniczy 1962, nr 12;

[194] Wang J. 2004: Wpływ naprężeń, podcinania, robót strzałowych i czasu na stabilność komór i zubożenie urobku. Praca doktorska University of Saskatchewan, Saskatoon;

[195] Wanielista K. 1979: Ocena ekonomiczna strat eksploatacyjnych i zubożenia rudy. Rudy i metale nieżelazne nr 9;

[196] Wanielista K. 1986: Ekonomika w górnictwie. T. I i II. Skrypty Pol. Wrocławskiej, Wrocław;

[197] Wawrzyniak E. 1976: Zubożenie kopaliny w procesie eksploatacji. Cuprum nr 4, Wrocław;

[198] Węgierski J., Wolski J. 1964: Ekonomiczna efektywność budowy kopalń węgla kamiennego w planowaniu perspektywicznym. Przegląd Górniczy 1964, nr 9;

[199] Wilkinson W.A. 2010: Benefits of building efficient mine planning process. Mining Engineering September;

[200] Włoszek J. 2002: Deponowanie odpadów w podziemnych kopalniach węgla kamiennego podczas eksploatacji z zawałem stropu. Wyd. IGSMiE PAN Kraków;

[201] Wodarski K., Bijańska J. 2014: Ocena ekonomicznej efektywności i ryzyka eksploatacji resztkowych złóż węgla kamiennego na wybranym przykładzie. Przegląd Górniczy 9, s.86-89;

[202] Wolski J., Pogonowski T. 1968a: Metody rachunku ekonomicznej efektywności inwestycji w projektowaniu kopalń węgla kamiennego. Zaszyty Naukowe Politechniki Śląskiej, s. Górnictwo zesz. 30, Gliwice;

[203] Wolski J., Pogonowski T. 1968b: Projektowanie optymalnej wielkości i modelu głębinowej kopalni węgla kamiennego. Cz.1. Skrypty Uczelniane Politechniki Śląskiej nr 214, Gliwice;

[204] Wright E. A. 1983: Dilution and mining recovery – Review of the fundamentals, Erzmetall, 31(1);

[205] www.syskon.eu

[206] Wyra P., 1981: Wpływ wybranych parametrów pól eksploatacyjnych na optymalne ze względu na koszt wydobycia 1 t wybiegi ścian w nowo projektowanych kopalniach. Przegląd Górniczy 1981, nr 2.

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Methodology for Run-of-Mine Quality Management in a Hard Coal Mine

Abstract

The Monograph presents the results of the research work, conducted by the Author, on the assessment of the hard coal dilution impact on the efficiency of production process in the LW Bogdanka SA and on a development of a system for the mining production management support with use of IT solutions and monitoring of production processes oriented onto the ROM quality stabilisation and improvement. The scope of the analysis required mathematical modelling and a development of methods for an estimation of qualitative parameters meeting the needs of coal mining sector.

The ROM dilution impact on the economic efficiency of an underground mining plan production process is highlighted. The Author's concept of the production process automation and monitoring, which has been implemented in the LW Bogdanka SA, consisting of a few interconnected components, is described. The Integration Platform forms a central component of the presented solution, enabling the information integration oriented onto a standardised exchange of data among systems, using various communication protocols. The Author included his most important observations and conclusions resulting from the carried out research investigations and a multi-year scientific experience gathered due to an implementation of research projects at the KGHM Polska Miedź S.A., JSW S.A. and Tauron Wydobycie S.A.

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