

# **Risk Assessment Methods in Estonian Oil Shale Mining Industry**

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**Risk Assessment Methods in Estonian Oil Shale Mining  
Industry**

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Declaration:

Hereby I declare that this doctoral thesis, my original investigation and  
achievement, submitted for the doctoral degree at Tallinn University of Technology  
has not been submitted for any academic degree.

Sergei Sabanov

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## LIST OF ORIGINAL PUBLICATIONS

PAPER I. Pastarus, J.R.; **Sabanov, S.**, 2005. A method for securing working mining block stability in Estonian oil shale mines. Proceedings of the Estonian Academy of Sciences. Engineering, 11, pp. 59 - 68.

PAPER II. Nikitin, O., Pastarus, J., **Sabanov, S.**, 2006. Roof bolting analysis at "Estonia" oil shale mine. 15th International Symposium on Mine Planning and Equipment Selection, Torino, Italy. (Eds.) M. Cardu, R. Ciccu, E. Lovera, E. Michelotti. Italy: FIORDO S.r.l. - Galliate (NO) (2), pp. 1441 - 1446.

PAPER III. Pastarus, J.; **Sabanov, S.**; Nikitin, O., 2006. Risk assessment of blasting works in Estonian oil shale mines. 15th International Symposium on Mine Planning and Equipment Selection, Torino, Italy. (Eds.)M. Cardu, R. Ciccu, E. Lovera, E. Michelotti. Italy: FIORDO S.r.l. - Galliate (NO), (2), pp. 1273 - 1278.

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PAPER V. **Sabanov, S.**; Pastarus, J.; Nikitin, O.; Väli, E., 2007. Risk assessment of pillars bearing capacity under rock dump in Estonian mine "Viru". In: Proceedings of the Sixteenth International Symposium on Mine Planning and Equipment Selection and the Tenth International Symposium on Environmental Issues and Waste Management in Energy and Mineral Production. Bangkok, Thailand. Reading Matrix Inc USA, 2007, pp. 966 - 972.

PAPER VI. **Sabanov, S.**; Pastarus, J.-R.;Nikitin, O; Väli, E., 2008. Risk assessment of vibration impact on roof and pillars stability in Estonian underground mines. In: 5th International Symposium "Topical problems of education in the field of electrical and power engineering". Doctoral school of energy and geotechnology: Tallinn University of Technology, 2008, pp. 57-61.

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PAPER VIII. **Sabanov, S.**; Sokman, K., 2008. Technological and environmental aspects of assessment of a combination of different mining methods used in Estonian oil shale industry. Oil Shale, 2008, Vol. 25, No. 2 Special, pp. 163-173

PAPER IX. **Sabanov, S.**; Pastarus, J.-R.; Nikitin, Väli E., 2008. Risk assessment of Surface miner for Estonian Oil-shale mining industry. The 12 th Internationale conference of Internationale Association for Computers Methods and Advances in Geomechanics. IACMAG 12. Goa, India (accepted)



# 1 INTRODUCTION

## 1.1 Background

*The actuality* of the thesis consist on that in Estonia the risk assessment methods are used in different branches of industry, but the number of references on solution of mining problems is limited. Investigations have shown that the above-mentioned methods are applicable for solving complicated mining problems. However, prevention of hazardous situations is of moral, ethical and economical character rather than facing the adverse consequences. In the stage of design, the above-mentioned methods give a unique possibility to predict negative consequences and allow to work out tools to avoid and mitigate hazard factors. The risk assessment methods provide information about mining and its influence on the environment. Having received the information, the management of a mine can come to adequate political and strategic decisions. The methods can favourably reflect problems at the local and regional level.

Various processes in a mining industry often become dominating in the most dangerous technological and geotechnical factors, which can pose a hazard to people and their work, stop the production, cause economic damage to the enterprise and environmental. Providing of favourable conditions for the enterprise in conformity with technological and ecological safety defines the necessity of risks assessment. The risks assessment method provides information on the processes of a mining industry, helps people make adequate decisions and apply them in practice.

*The bases of investigation* were served worldwide experience which has shown that the risk assessment method is a very powerful tool to solve complicated industrial problems. It allows qualitative and fast finding of optimal variants for solving the existing problems. This methods was successfully used in Safety Dam Assessment [ICOLD 1999]; Assessment and Management of Roof Fall Risks in Underground Coal Mines [Duzgun 2004]; Performance Evaluation, Safety Assessment and Risk Analysis for Dams [Hoeg 1996]; Risk Analysis Safety Assessment for Use at Swedish Dams [Graham 1995]; Environmental-Impact Assessment for the Proposed Oil-Shale Integrated Tri-Generation Plant [Jaber 1999]; On the Quantitative Definition of Risk [Kaplan 1981]; Hazard Identification for Rock Engineering [Naismith 1998]; Health and Safety in Canadian Mining Industry [Gibbs 1978]; Landslide Risk-systematic Approaches to Assessment and Management [Einstein 1997]; Background for Spatial Differentiation in Life Cycle Assessment – the EDIP 2003 methodology [Hauschild 2004]; An Analysis of the Geomechanical Factors Influencing Coal Mine Roof Stability in Appalachia [Karmis 1984]. Using these methods available, one can make prognosis and flexibly resolve question of different complexities. These methods can also be used in the planning, exploitation and closure stages of projects. Conventional theories do not enable to solve these tasks. Nowadays, under the conditions of economising usage of mineral resources [Reinsalu 2002; Valgma 2007], the use of such special approach has become topical.

## 1.2 Objectives

*The aim* of the investigation is determination of applicability of the Risks Assessment technique for the conditions of the Estonian oil shale mining industry. The main purpose is to elaborate the modified methodology of risks assessment based on the data of Estonian oil shale mining industry for various mining processes and problems that cannot be decided by means of other methods.

*The tasks of the study are:*

- Investigation of long-term stability of underground facilities.
- Development of method for mining blocks stability control.
- Elaboration of a risk assessment method applicable to selective extraction and transportation of mineral resources.
- Determination and elaboration of safety factors and parameters for underground mining advanced technology efficiency.
- Defining environmental impact resulting from different mining processes.

The statistical data, which have become available during the last 40...50 years, are based on the experience obtained on about 400 mining blocks in an area of more than 100 km<sup>2</sup>. They provide a good basis for the elaboration of the methods of risk assessment, and for applying the results to solving the mining and environmental problems.

## 1.3 Problems of oil shale mining

Estonian oil shale of various qualities are used for generation of electric power and in shale oil processing, however the different mining and excavation methods and accompanying development processes has different influence on environment. Oil shale mining can disturb the landscape and hydrological regime through the pollution of water and air arising from the use of different extraction methods. On the other hand, the commercial (mineable) oil shale deposit is located in a densely populated and rich farming area. Extraction of a mineral resource is accompanied by many technological processes each of which has distinctive conditions and its own disturbance on the mining work. Failure in mining can result in a hazard to workers or stop the extraction of mineral resource thereby depriving the enterprise of profit.

*For the solution of problems* relating to the mining industry, there are various conventional methods available; however, the Risk Assessment methodology shows the greatest promise for defining hazardous influences. It allows to develop methods for avoiding or reducing negative factors. The method helps quickly, conveniently and qualitatively solve, operate, find optimum variants for existing problems, to predict with the minimal expenses during the project stage how to flexibly avoid the subsequent problems.

*The methods* of risks assessment based on statistical analysis and consist of into three general approaches depending on the type and quality of the assailable data: analytical approach uses logical models for calculating probabilities, empirical approach uses existing databases to generate probability and judgmental

approach uses experience of practicing engineers in guiding the estimation of probabilities. Theoretical estimation helps systemizing chosen calculation methods.

#### 1.4 Contribution

*The originality* of risk assessment methods for the Estonian oil shale industry consist of a unique system which starts from the initial analysis of rock mass properties, followed by the assessment of technological problems and ends with an environmental impact analysis. The dominating factors exerting a significant effect on the mining process were established. The updated model of risk estimation allows to reveal the duration of processes and to carry out qualitative sampling of the most suitable variants. The risk mitigation technique of pillars failure and losses of mineral resources was elaborated.

*Practical importance* of the thesis involves the following problems: stability of a mining block, application of advanced mining technology; extraction of mineral resources, loading, transportation and their influence on the environment. The risk assessment can be used for different purposes and at different levels: as a basis for decision-making when selecting among different remedial actions for a mined out area within time and financial restraints; to relate ground surface subsidence risk levels to acceptable risk levels established by the society for other activities. The results of the risk assessment are of particular interest for practical purposes. Risk assessment methods are applicable in various fields of mining production.

*The adequacy* was estimated by theoretical calculations, expert estimation and analysis of results, which were very close to experimental data. For determination of the efficiency and adequacy of the elaborated methods, risk assessment for advanced technology was performed. The elaborated risk assessment methods can be used in different geological conditions, where the room and pillar mining system is used, and with other mining technologies. The methods gives an opportunity to find a better way for planning new underground and open (surface) mining activities in accordance with environmental performances and economic profit of enterprises.

Basic material of the thesis is published in 24 papers in the scientific journals Proceedings of the Estonian Academy of Sciences. Engineering, 2005 and Oil Shale 2007, 2008, but also reported on the 15th International Symposium on Mine Planning and Equipment Selection, Torino, Italy; 16th International Symposium on Mine Planning and Equipment Selection; 10th International Symposium on Environmental Issues and Waste Management in Energy and Mineral Production. Bangkok, Thailand 2007; 4th and 5th International Symposiums "Topical problems of education in the field of electrical and power engineering", Kuressaare, Estonia 2006, 2007; 15th Meeting of the Association of European Geological Societies 2007; 5th and 6th International Scientific and Practical Conferences. Rezekne, Latvia 2005, 2007; XIII Congress of International Society for Mine Surveying (ISM) Budapest, Hungary 2007; International Oil Shale Conference. Amman, Jordan., 2006; 27th Oil Shale Symposium, Colorado School of Mines, Golden, Colorado.

## 2 GEOLOGY AND CURRENT MINING

The most important mineral resource of Estonia is a specific kind of oil shale. The commercially important oil shale bed is located in a densely populated and rich farming district in the northeastern part of Estonia. It stretches 200 km from west to east and 50 km from north to south. The oil shale layers form a flat bed with a slight southward inclination (2–3 m per km). The depth of the oil shale bed varies: at the northern border of the field it occurs immediately under the Quaternary sediments, while at the southern border it is at a depth of 100–150 m. The thickness of the commercial oil shale bed decreases somewhat westward and southward from the central part of the deposit [Pastarus 2005, 2006, 2007].

The oil shale layers occur among the limestone interlayers in the Middle Ordovician Kukruse Regional Stage. It is a stratified sedimentary rock, rich in the organic matter – kerogen (15–46 %). The content of carbonates ranges from 26–57% and that of clastic material from 18–42%. The commercial oil shale bed consists of six indexed oil shale seams – A to F (from the bottom upwards). Mechanical characteristics of the oil shale and limestone seams are quite different. The compressive strength of oil shale is 16–40 MPa, while that of limestone is 40–85 MPa. The volume density is 1500–1900 kg/m<sup>3</sup> and 2200–2600 kg/m<sup>3</sup>, respectively. The strength of the rocks increases in the southward direction. For this reason, the stability of pillars is difficult to predict [Pastarus 2005, 2006, 2007].

Surface mining is carried out in open casts with the maximum overburden thickness of 30 m. Draglines with 90 m boom length and 15 m<sup>3</sup> bucket sizes are used for overburden removal. Hard overburden consists of limestone layers and is blasted before excavation. Oil shale layers are either blasted or broken by ripping (low-selective mining). The disadvantage of ripping is excessive crushing of oil shale by bulldozer crawlers. The excavated rock is transported by trucks to the processing or crushing plant depending on opencast [Nikitin 2007].

Underground oil shale mining is carried out by the room-and-pillar method with blasting. The method is highly productive, easy to mechanize, and relatively simple to design. It gives the extraction factor of about 80%. However, the structure of the exploitable oil shale bed makes the rocks more difficult to break from the total massive. In underground mines, the oil shale is extracted at a depth of 35–75 m. The field of an oil shale mine is divided into panels, which are subdivided into mining blocks, approximately 300–350 m in width and 600–800 m in length each. A mining block usually consists of two semi-blocks. The room is very stable when it is 6–10 m wide. However, in this case the bolting must still support the immediate roof. The pillars in a mining block are arranged in a singular grid. Its height corresponds to the thickness of the commercial oil shale bed, which is approximately 2.8–3.8 m. Actual mining practice has shown that pillars with a square cross-section suit best. The cross-sectional area of the pillars is 30–50 m<sup>2</sup>, depending on the depth of the oil shale bed [Pastarus 2005, 2006, 2007].

### 3 RISK ASSESSMENT METHODS

Risk assessment defined in its broadest sense, deals with the probability of any adverse event [SENES, 2000]. Various types of risk considered in the mine project life circle include the engineering risk, human health risk and ecological risk. Risk assessment is the process of deciding whether the existing risks are tolerable and risk control measures are adequate. It incorporates the phases of risk analysis and risk evaluation. Risk analysis enables one to find out how safe the mining is, and risk evaluation – how safe should it be. The primary steps of a risk assessment are presented in Figure 1.

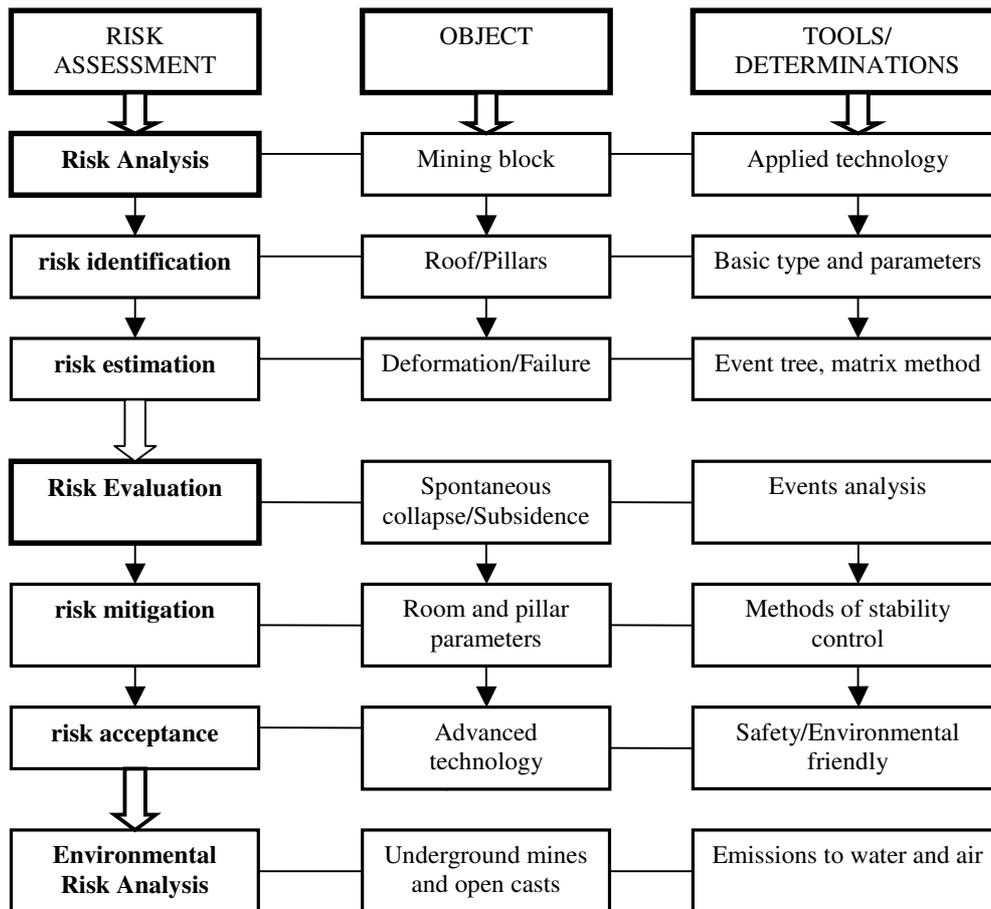


Figure 1. Risk assessment.

#### 3.1 Risk Analysis

Risk analysis is used for performing safety assessment for many different mining systems. Risk analysis includes risk identification and risk estimation. The description of the system, scope and expectations of the risk analysis should be defined at the outset. An iterative approach should be adopted with qualitative

methods being employed at the early stages of the process. If more information becomes available, use of quantitative analyses is required [Graham, 1995].

Mining includes separate stages of production with different destination and place of performance. The technological scheme characterizes processes and specifies the order of works performance in time, the mode of their carrying out and means of their realization. The technological scheme depends on extraction methods. For this reason, there is a great variety of possible combinations of processes in the excavation field.

### **3.1.1 Collapse of mining block**

In risk analysis the object of investigation a mining block was chosen (Figure 1). The processes in pillars and overburden rocks have caused unfavourable environmental side effects accompanied by significant subsidence of the ground surface. The subsidence of ground surface results from the collapse of pillars. Ground surface subsidence can cause soil erosion and flooding, swamp formation, agricultural damage, deforestation, changes in the landscape, lowering of the ground water level and formation of unstable cavities.

#### **3.1.1.1 Applied technology scope**

Underground mining is carried out by the room-and-pillar method with blasting. The depth of subsidence depends on the thickness of the extracted seam. The dimensions of the actual roof and pillars depend on the quality of mining works and applied technology. Blasting can significantly disturb the dimensions of pillars and the stability of roof. Consequently, pillar and roof sizes vary from place to place within a mining block. If the difference between the designed and actual parameters is large enough, the mining technology is disturbed; spontaneous collapse is likely to occur or the losses in pillars increase.

### **3.2 Risk identification**

Risk identification is the process of determining potential risks and it starts with the source of problems, or with the problem itself. Failure can be described on many different levels. Conceptualization of the different possible failure modes for a mining system is an important part of risk identification. One should first take into account as many types of failure as possible. The initial list can then be reduced by eliminating those types of failures considered implausible. Figure 2 presents factors contributing to the mining block collapse and ground surface subsidence.

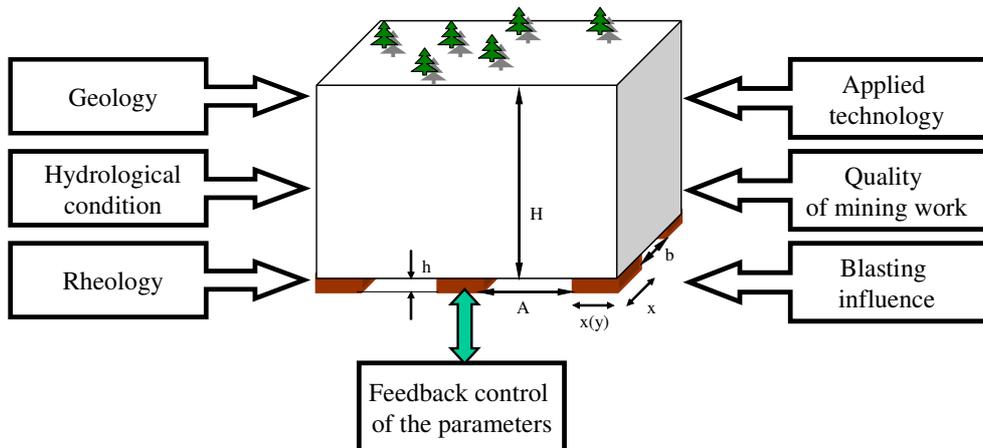


Figure. 2. Factors contributing to the mining block collapse and surface subsidence

Identification is based on the information available on the excavation depth, mining system (room and pillar or longwall mining), rock mass properties, underground water conditions and on the technology used.

### 3.2.1 Roof and pillars

Using the room and pillar methods with blasting, it is important to take into consideration the parameters of the roof and pillars and all possible ways of their disturbance. The structure consists of limestone and oil shale layers with different physical-mechanical characteristics. Since the strength of the rocks can increase with depth, the stability of the pillars and roof is difficult to predict.

#### 3.2.1.1 Basic type and parameters

For safety roof control it is necessary to consider the immediate roof exfoliation levels, anchor bolt parameters and anchor lock type, supporting grid, critical arch of roof, tectonic joints arrangements, upstreaming mining water etc. The pillars stability condition is sensitive to safety factor, geometrical form, height, cross sectional area, moisture content and rheological properties.

### 3.3 Risk estimation

Risk estimation entails the assignment of probabilities of the events and responses identified under risk identification. The assessment of appropriate probability estimates is one of the most difficult tasks of the entire process. Tools that are often used to help in risk estimation are fault trees and event trees. Probability estimation can be grouped into three general approaches depending on the type and quality of the available data: analytical approach uses logical models for calculating probabilities, empirical approach uses existing databases to generate probability, judgmental approach uses experience of practicing engineers in guiding the estimation of probabilities [Graham, 1995].

### 3.3.1 Deformation of roof

The risk factor is suggested for use in the safety control of the roof supported by anchor bolts. The risk factor is determined by the ratio of initial tightening (I) to critical (C), taking into account the inflow water pressure (W) and the roof stability coefficient (K). In the conditions of the Estonian oil-shale mines, the risk factor is acceptable, if the limits are  $2 < R < 4$ . Other values can be regarded as dangerous or demanding additional control. The roof stability coefficient (K) depends on the spacing interval between tectonic joints shown in Table 1.

$$R = (I + WS) K / C, \quad [1]$$

where R is the risk factor, I - initial tightening, W- water inflow [Talve L. 1975], S- square of pressure on anchor (depending on the supporting grid), C - critical tightening (1.5 t), K-roof stability coefficient [Mining Law regulation, 2004].

Table 1. Inflow water pressure and roof stability coefficient for different geological condition

Condition	Inflow water pressure, t/m <sup>2</sup> (W)*	Roof stability coefficient, (K)**	Spacing interval between tectonic joints, m**
Normal	0	1	20
Average	0.1	1.2	10-20
Low stable	2.5	1.45	10
Unstable	5	1.82	3-5

\*- Talve L. 1975

\*\* - Mining Law regulation, 2004

#### 3.3.1.1 Matrix method in the deformation of roof

Risk estimation for immediate roof deformation in Estonian oil shale mines is presented by severity scale in Table 2. Laminated roof deformation on the basis of plate's hypothesis obtained by the experimental data of the Institute of Mining Survey (VNIMI) in St. Petersburg and the Estonian Branch of A. A. Skotchinsky Institute of Mining Engineering (IGD, Moscow, Russia) [Andreev 1987; Seleznev 1961].

Table 2. Severity Scale

Point	Criteria	Description	Exfoliation level, mm ***	Process
5	Severe	Catastrophic (very harmful or potentially fatal; great efforts needed for its correction and recovery)	110 -	full destruction
4	Serious	harmful, but not potentially fatal, difficult to correct but recoverable	61-110	steady-state creep
3	Moderate	somewhat harmful, correctable	31-60	transient creep
2	Mild	little potential for harm, easily correctable	11-30	elastic deformations
1	Harmless	no potential for harm, correctable	0-10	instant deformations

\*\*\*- IGD, Moscow, Russia; VNIMI.

For determination of impact propagation, the Boundaries Scale presented in Table 3 is suggested to use. Using the formula of the critical width (PAPER I), the number of rooms for typical underground condition of Estonian oil shale mines was received.

Table 3. Boundaries Scale

Point	Criteria	Description	Number of rooms effected
5	Local	Impact migrates on the ground surface	>35
4	Not confined	Impact migrates outside the critical area	10-35
3	Confined	Impact migrates off-site of the rooms row	6-9
2	Isolated	Impact is contained in a small area	1-5

The risk significance is determined by multiplied criteria of the Severity and Boundaries points: (2-4) – Minimal. Special measures are not required; (6-8) – Low. Special measures for the reduction of risk are required; (9-12) – Moderate. Measures of organizational character are required (additional equipment, improved technology, parameters changing); (16-20) – High. The analysis of the reasons and existing security measures is carried out. Actions for the prevention of similar cases are made. An information card is compiled; (25) – Catastrophic. Work stops. The analysis of risk is immediately carried out. Additional actions are developed.

Additional instructing or training is carried out. Works cannot be begun while the risk is not reduced.

### 3.3.2 Failure of pillars

Prognostication of pillars durability without taking into account rheological properties is a difficult task. The experimentally obtained factor of rock durability change in time ( $k_t$ ) is used for the calculation of pillars parameters. The changes of the critical value  $k_t < 0.44$  can result in the creep of rock and increasing collapse probabilities.

#### 3.3.2.1 Event tree

As a starting point in constructing an event tree, a case of possible collapse is initiated and then a systematic analysis of the different possible outcomes of the sequences is provided. Event tree determines the probability of pillars collapse during the prescribed time period for different conditions and parameters, which can be estimated using the judge-mental procedure and taking into account the statistics.

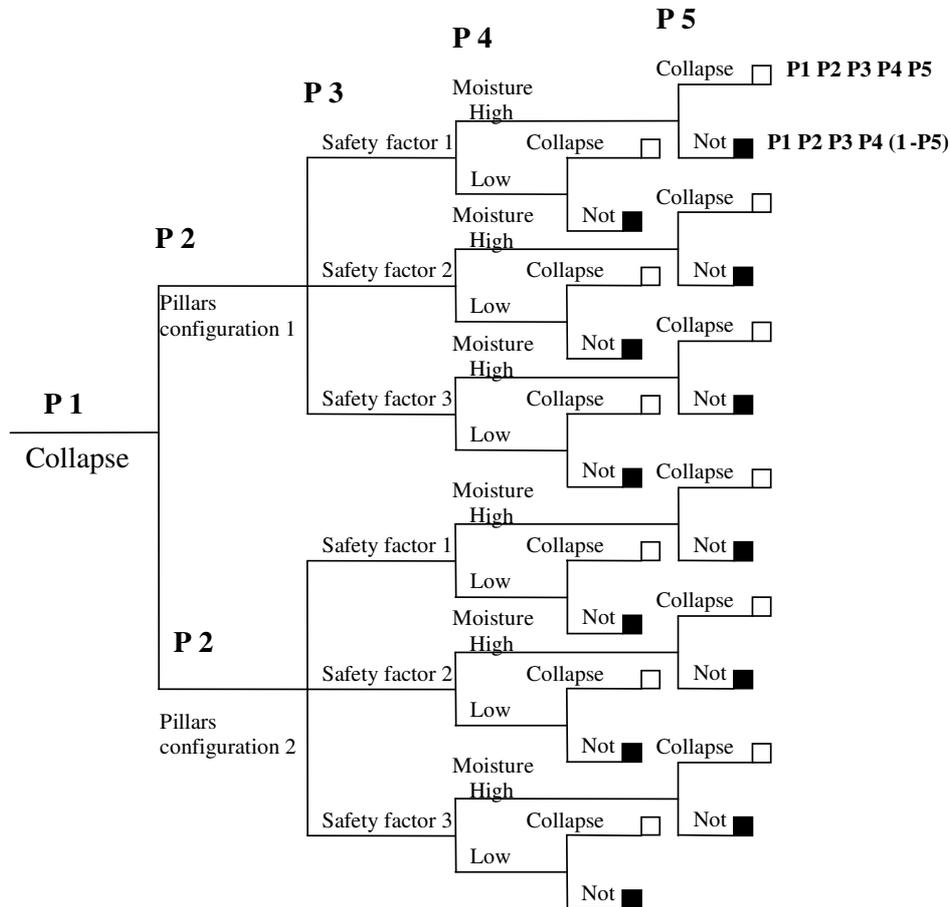


Figure 3. Event tree of collapse probability.

For clearer calculation of pillars long-term stability, an acoustic emission method using the Kaiser Effect is available.

### 3.4 Risk Evaluation

Risk evaluation is the process of examining and judging the significance of risk. Risk evaluation stage is the point at which values and judgments enter the decision process, by including consideration of the importance of the estimated risks. Risk evaluation is fundamental to risk assessment and risk-based decision making. The principal role of risk evaluation in risk assessment is the generation of decision guidance against which the results of risk analysis can be assessed. It requires a statement of the owner's safety management principles and of the values and preferences of the public (prevailing financial, legal and regulatory conditions). The risk evaluation process should be clearly communicated to all interested groups. The extent, to which each of these basic principles apply depend on the nature of the risk assessment [ICOLD, 2006]. Risk evaluation includes risk mitigation and risk acceptance.

#### 3.4.1 Spontaneous collapse of mining block and ground surface subsidence

The first spontaneous mining block collapse and ground surface subsidence in an Estonian oil shale mine took place more than forty years ago [Pastarus 2005, 2006, 2007]. Since then, 18 % of mining blocks have experienced spontaneous collapses. Up to the present, more than 70 spontaneous collapses of mining blocks have been recorded. Figure 4 shows that the majority of collapses occurred during the first 30 months of pillars lifetime, and then the number decreased to the point of 60 months. The collapses, registered after that period, took place in the locations with complicated geological conditions.

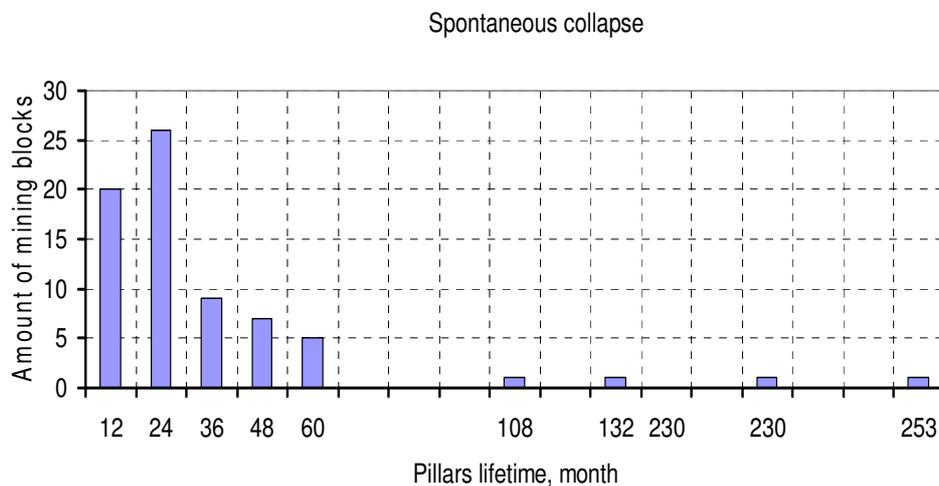


Figure 4. Amount of collapsed mining blocks in time

Based on statistical data and analysis of pillars lifetime, it was established that during the above-mentioned time period the collapsed pillars expose a logarithmic normal distribution. Summarising this data, it is possible to assume that with a great probability a mining block collapse during 60 months is due to the decrease in the pillars cross-sectional area and increased chamber size.

#### **3.4.1.1 Events analysis**

For the Estonian oil shale mining industry, the risk assessment embodies the above-mentioned consideration and the regulations of the Mining Law and Law on Mineral Resources. For spontaneous collapse in a working mining block, the risk might be expressed in terms of the number of deaths per year, the probabilities of the frequency of events causing fatalities. For subsidence in environmental perspective, the preferred choice could be the lowest risk. Evaluation of subsidence includes the depth and area, and the importance of the disturbed landscape.

### **3.5 Risk mitigation**

Risk mitigation is a selective application of appropriate techniques and management principles to reduce either the likelihood of an occurrence or its consequences, or both. If the calculated risk of the existing system is judged to be too high, alternatives are proposed to reduce the risk of failure. These alternatives are incorporated into the risk model and re-evaluation is conducted to estimate their impact [ICOLD, 2006]. After repeated study the decision makers can be provided with suitable alternatives and their estimated costs for consideration in improving overall mining system safety.

#### **3.5.1 Improvement of room and pillars parameters**

The method of backfilling is suggested to avoid collapses and decrease the losses of mineral resources [Sabanov 2008b]. Backfilling allows to increase the capability of pillars thereby reducing the losses of oil shale and to restore, in a certain measure, filtration, hydrodynamical and aerodynamical properties of the geological environment. Moreover, backfilling with limestone, which is one of oil shale mining by-products, will help to avoid waste rock generation on the ground surface and decrease environmental pollution.

##### **3.5.1.1 Methods of stability control**

For risk mitigation, the geometrical parameters of pillars can be changed using the method elaborated for the monitoring of the stability of a working mining block (PAPER I).

Calculations are performed considering the concept of critical area, conditional thickness and sliding rectangle methods. The working mining block stability and losses in pillars depend on the actual parameters of the pillars and roof. The changes in the conditional thickness of parameters *in situ* conditions in a mining block are illustrated in Figure 5 A.

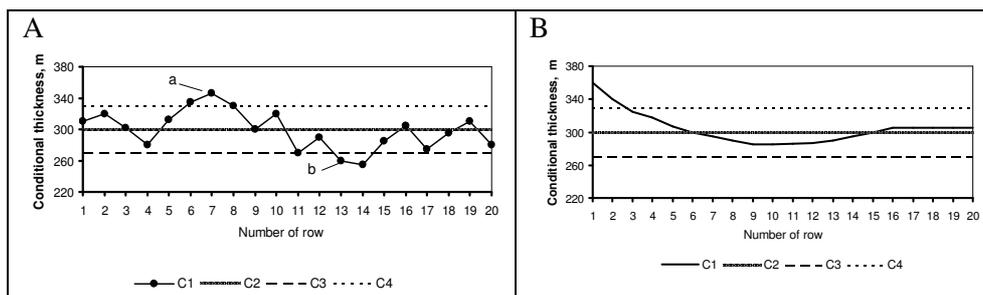


Figure 5. Deviation of the conditional thickness parameters from the designed value. *a* – collapse is possible; *b* – losses in mineral resources have increased; *C1* – actual conditional thickness in the critical area; *C2* – designed value of the conditional thickness; *C3* – lower limit; *C4* – upper limit.

As is seen, the conditional thickness parameters fluctuate around the designed value. If the conditional thickness parameters remain between the lower and upper limits, the mining block is stable and losses are minimal. If the conditional thickness is beyond the limits, the collapse is likely to occur (area *a*) or the losses increase (area *b*). Figure 5 B shows a quick and smooth approach of the conditional thickness to the designed values obtained by using formula [4].

The feedback of the real situation in a mining block is guaranteed by a monitoring system. For mining block monitoring the conditional thickness difference criterion is suited. It insures the stability of a mining block against random deviation of the pillars and roof parameters. Considering the applied mining technology, it is possible to increase or decrease the length of pillars guaranteeing the stability of the working mining block.

The load on the pillar depends on the real parameters of the pillar and roof. Conditional thickness is a geometrical parameter which considers the depth of excavation and the parameters of the pillars and roof [Talve L. 1978, Stetsenko 1981]. It represents the height of a prism whose cross-section equals the pillar's cross-section area. The support coefficient and conditional thickness are presented by the following formula [Talve L. 1978, Stetsenko 1981] (1, 2):

$$K = \frac{S_p}{S_r}, C = \frac{H}{K}, \quad [2]$$

where *K* – support coefficient; *C* – conditional thickness, m; *S<sub>p</sub>* – cross-sectional area of a pillar, m<sup>2</sup>; *S<sub>r</sub>* – roof area per pillar, m<sup>2</sup>; *H* – thickness of the overburden rocks, m.

Conditional thickness contains sufficient information and is suitable for stability calculations. Conditional thickness is related to the load on a pillar as follows:

$$\sigma = C\gamma, \quad [3]$$

where  $\sigma$  – normal stress at the top of a pillar, Pa;  $\gamma$  – weight density of the overburden rocks, N/m<sup>3</sup>.

When the conditional thickness exceeds the limit, a collapse is likely to occur. It is possible to increase the cross-sectional area of the next row pillars to avoid the collapse. For practical application the author has elaborated a method for determining the parameters of the next rows basing on the following formula:

$$S_n = \frac{C_r S_{pr}}{C_{pr}}, \quad [4]$$

where  $C_r$  – actual conditional thickness, m;  $S_{pr}$  – designed value of the cross-sectional area of the pillars, m<sup>2</sup>;  $C_{pr}$  – designed value of the conditional thickness, m.

This method guarantees a quick and smooth approach of the conditional thickness and the cross-sectional area of the pillars to the designed values [Pastarus 2005] The method is applicable in different geological conditions, where the room-and-pillar mining is used (PAPER I).

### **3.6 Risk acceptance**

Risk acceptance is an informed decision to accept the likelihood and the consequences of a particular risk. For failure events with no potential fatalities or irreparable damage to the environment, the target annual failure probability may be decided basing exclusively on economic considerations and corresponding risk analysis. A target level of  $10^{-3} \dots 10^{-2}$  rather than  $10^{-6} \dots 10^{-5}$  may be a reasonable criterion [Hoeg 1996].

#### **3.6.1 Advanced technology**

Room-and-pillar mining is a highly effective and easily applicable working method in Estonian underground conditions, but it needs additional improvement to increase oil shale extraction. Selective mining with backfilling can resolve this problem effectively. Selective extraction allows reduction of rock mass volumes during the loading, transportation and enrichment processes. Thus, about 23 % of the limestone accompanying the extracting processes will be left in the mine for backfill in the excavated areas [PAPER VII].

##### **3.6.1.1 Safety and environmental friendly technology**

Technology can be accepted under safety control for people taking individual risk. The average conditional probability of death due to roof failure is the expected loss of life divided by the miners at risk. The limit value of average individual risk is  $10^{-6}$  per annum. Environmentally friendly and highly effective technology can be accepted under an average conditional probability of the quantity of collapses and ground surface subsidences.

### **3.7 Environmental Risk Analysis**

Mining methods in the Estonian oil shale industry depend, to a great extent, on the depth of the deposit. Based on deposit parameters, the technology where different equipment and extraction methods are used can vary significantly. The Life Cycle Assessment (LCA) tool can be used to analyze and assess the environmental impact of the oil shale mining industry. The inventory data base represents in detail the mining system that comprises the excavation processes description and analysed by classification and characterization methodology of LCIA [ISO 1997, 1998, 2000].

#### **3.7.1 Underground mines and open casts**

Description of unit processes presents a general overview of mining, according to which a technology is applied and a certain kind of equipment is used in excavation processes. The emission values are converted into impact category indicator results by multiplying the emission values by the corresponding characterisation factors [ISO 14040]. The inputs and outputs for all technological chains of underground mines and open casts must be identified.

##### **3.7.1.1 Emissions to water and air**

Annual outlets of mining water and air serve as the measure of emission. In the case of open casts the emission to air is calculated from diesel combustion emission of working machines and from explosion works [Sabanov, 2006]. For stripping, cutting, drilling, loading, transportation and recultivation processes, gaseous emission from diesel combustion must be calculated. Supply unit of blasting operation, the production of ammonium nitrate from ammonia and nitric acid  $\text{NH}_4\text{NO}_3$ , calculate emission from an explosion process. On the other hand, some machines work only on electricity; there is no emission from diesel combustion and, therefore, calculated emissions attribute to power generation (PAPER VIII).

## 4 APPLICATION OF RISK ASSESSMENT IN OIL SHALE MINING

### 4.1 Analysis of long-term stability of underground facilities

#### 4.1.1 Comparison of unconfined compressive strength and acoustic emission of Estonian oil shale

The purpose of this study was to confirm the existence of acoustic emission Kaiser Effect (KE) in Estonian oil shale, for comparison analysis with data received from working mining pillars. The KE of acoustic emission, a phenomenon with a potential for *in-situ* stress estimation can be used for quantifying the damage levels of pillars, and possibly even to measure the state of stress within a pillar. The main role of measurements would be confirmation of estimated stresses, as the estimation is quite simple in regions of sedimentary rocks. The performed tests showed that the Kaiser Effect does exist in oil shale material, but the low material strength also lowers the feasible stress limit for KE-based stress measurement. Tests were made with inspection of the formula for changes in the Estonian oil shale bed.

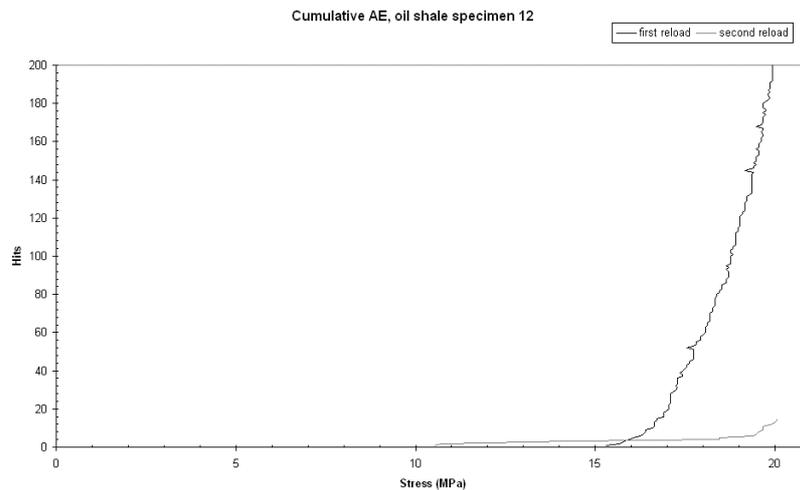


Figure 7. Acoustic emission graphs.

The sorted acoustic emission graph from specimen 12 demonstrates a clear Kaiser effect in the first reloading (previous peak stress was 15 MPa) (Figure 7). This specimen was tested for KE with one hour delay between pre- and reloading. The acoustic emission results show that the crack initiation stress of oil shale is 15-17 MPa, in a direction perpendicular to foliation. The unconfined compressive strength of oil shale makes 37 MPa.

The performed tests still have lots of uncertainties, as the amount of testing was limited. The existence of a Kaiser effect in oil shale has been confirmed, but measuring the *in-situ* stresses would require intact samples large enough for sub-sampling in multiple directions. In addition, the pillars of a room and pillar mine

tend to be damaged by blasting and by the induced loads, which can complicate the sampling process further.

Based on the formula of long-term rock durability, the time-dependent strength decay factor  $k_t$  reaches its limit value of 0.44 when the pillars service life  $t$  approaches infinity ( $t \rightarrow \infty$ ) [Mining Law 2004]. If  $k_t < 0.44$  creep of rock will occur [Undusk 1998]. As results from the acoustical emission tests, the crack initiation stress level was at  $k_t=0.43$ . The differences between the results of crack initiation stress from the old and new blasting technologies were not detected.

On the basis of these results, AE method could be used to estimate the long-term rock durability in the conditions of the Estonian oil shale while designing new mines. Reliability studies of the method would, however, demand further investigation of the Kaiser Effect in Estonian oil shale and limestone, in addition to trial measurements.

#### **4.1.2 Evaluation of pillars long-term durability**

Analysis of bearing capacity of pillars with life-time 22 years arranged under formed rock cone-shape dump was made. The allowable rock cone-shape dump volume was calculated from the actually existing pillar safety factor in this block of rooms. By way of checking calculations, the critical pillars bearing capacity, which will guarantee a control collapse and uniform subsidence, was received. In a practical way, the safety (stable) rock dump height, volume and area for avoiding unexpected deformation were determined [Sabanov, 2007a]. These practical experiments allow assuming that the properties of pillars in the mining block with normal conditions do not significantly change during 264 months. The laboratory tests confirm empirical formula of the long-term rocks strength in the mining conditions of Estonia (Figure 8). For determination of rock durability, samples were taken from pillars with different life time. Oil shale properties depend on the moisture content and can change considerably. The determined relation between the compressive strength and moisture content has practical significance for operative estimation of pillars durability under the local or periodical dampness change.

The results obtained from a laboratory test in the Department of Mining (MI, 2007), Tallinn University of Technology and the experimental data based on the dependence of rock strength in time [Undusk 1998] are presented in Figure 8.

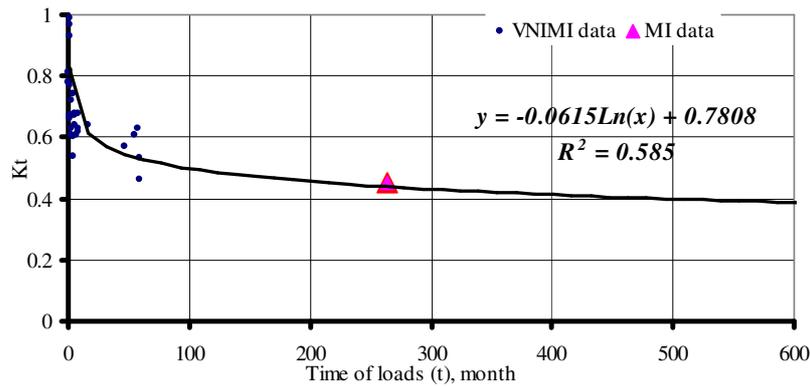


Figure 8. Dependence of rock strength in time  $Kt$ .

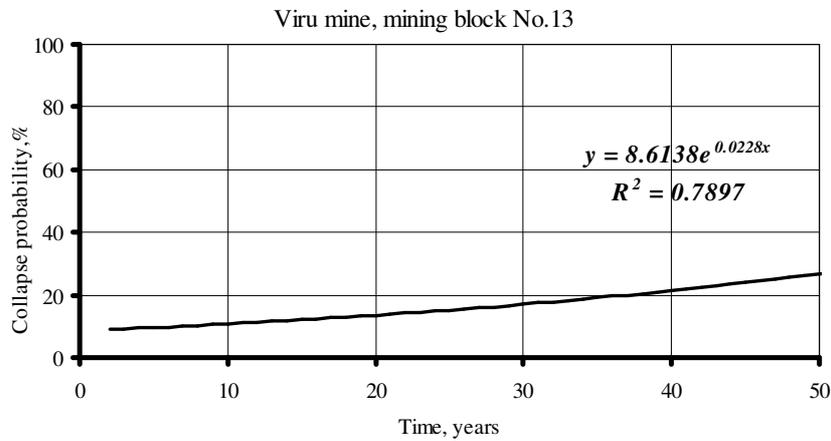


Figure 9. Collapse probability of the left semi-block of room's No. 13

Based on the data from the right semi-block no. 13 and calculation of probabilities for mining block collapse [Reinsalu, 2002], the collapse probability of the left semi-block no. 13 in the *Viru* mine was defined (Figure 9). The result can be used for determination of the collapse probability in the mining blocks of rooms with similar parameters and mine-geological conditions.

## 4.2 Risk assessment of mining processes

### 4.2.1 Selective mining

The next application of risk assessment relates with a high-selective oil-shale mining technology using the surface miner (SM) Wirtgen 2500SM. This study addresses the risks associated with productivity and cutting quality on example of the Estonian oil shale deposit in areas with complicated layering conditions. Main aspects influencing the efficiency of the combine work concern

the duration of the processes. Cutting different layers, track dumper loading (waiting), manoeuvres and maintenance processes are the most important factors. The event tree indicates the probabilities of the SM processes and spent time. Surface miner higher productivity in testing phase (IV) was achieved on account of 100 % „windrowing” method. The high cutting performance can be explained with the absence of waiting time. To determine a suitable variant comparison analysis with maximal possible productivity received during the tests was made (PAPER IX). This information allows finding an adequate decision to improve the quality of the processes [Sabanov, 2008c].

The results obtained in the frames of this project can be used in different industrial sectors. The main applications can be use in the surface mining and road construction sectors (PAPER IX).

#### **4.2.2 Transportation**

The oil shale industry of Estonia provides a significant contribution to the country's economy, but economically viable transportation of oil shale to consumers is impossible without advanced railway network. Railway transportation of oil shale is indispensable [Pastarus 2007] (PAPER IV). The very important tertiary process on the surface – transportation of minerals (oil shale) from mines and open casts to the consumer – was chosen to carry out the risk analysis.

The main quantitative approach used in risk analysis/assessment is the fault/event tree method. This method was selected as the most appropriate one for the analysis/assessment of the risk of the railroad transport system. In the first stage of the project, the time factor was taken into consideration. For probability determination the empirical approach was used. It utilizes the existing data to generate probable estimates based on historical frequencies.

The event tree for oil shale transport processes indicates the probabilities of the transport processes and spent time. It is possible to select different pathways and to determine the probability of one. It requires the independence of these factors. It means that the sum of the probabilities of these pathways gives us the total probability.

The fault tree allows determining time deviations from the mean value. Zero is taken as the mean value of the time. Minus before numbers indicates a decrease in the value, plus – an increase. The sum of the selected pathways determines the full-time deviation from the mean value. For instance, two different pathways are considered (variants A and B) (Table 4).

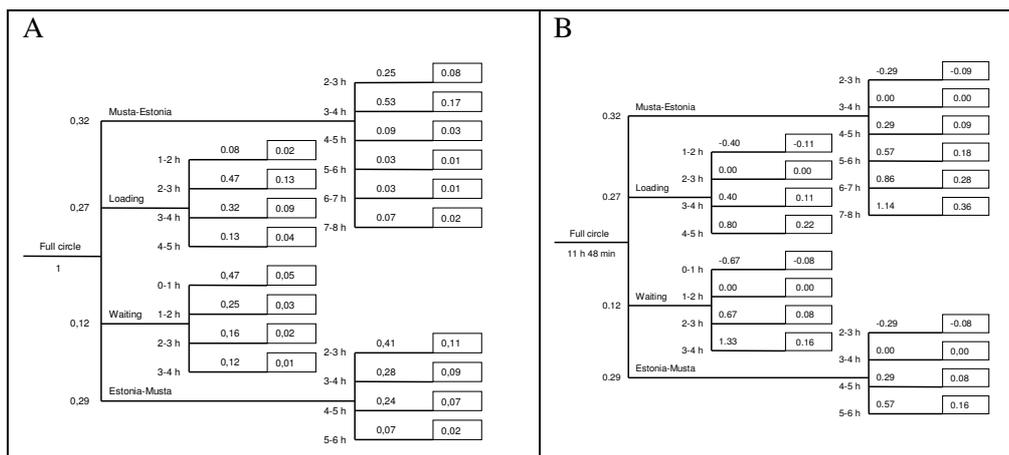


Figure 10. A) Event tree and B) Fault tree

Table 4. Example of the use of the event and fault trees (Figure 10)

Process	Selected time, h		Event tree		Fault tree	
			Pathway probabilities		Deviation from the mean value	
	Variant		Variant		Variant	
	A	B	A	B	A	B
Empty run	3 - 4	6 - 7	0.17	0.01	0.00	0.28
Loading	1 - 2	2 - 3	0.02	0.13	-0.11	0.00
Waiting	3 - 4	0 - 1	0.01	0.05	0.16	0.08
Loaded run	4 - 5	2 - 3	0.07	0.11	0.08	0.08
Total	-	-	0.27	0.30	0.13	0.12

Selected pathways give different values of the probabilities and deviations from the mean value (Table 4). One can see that the probability of selected pathways is 0.27 (variant A) and 0.30 (variant B), and deviation from the mean value is 0.13 and 0.12, respectively. The weight of each process in the full transport cycle is shown. Having this information, a specialist can come to an adequate decision and improve the quality of railway transport between two stations.

### 4.3 Risk assessment of advanced technology

#### 4.3.1 Advanced technology and problems

During the course of the last five years, in the oil-shale mining a new blasting technology with great entry advance rates (EAR) was introduced at experimental mining blocks. The main operations carried out in rooms (6-7 m in width) involve undercutting, drilling of blast-holes, blasting, rock mass loading on the conveyor and roof bolting. The advanced technology is based on an improved drilling-and-blasting method applying bulk emulsion explosive instead of a packaged ones (Nobelit 2000), change from 2.0 m to 4.0 m boreholes on new undercutting method and automatization of the roof drilling-bolting process with a

roof bolting machine. The aim of undercutting is to gain additional free space in the oil shale bed, which increases the effect of blasting. The old undercutting technology is based on bottom cutting with the help of the cutter which gives a horizontal cut, 15 cm high and 1.4–2.0 m deep, into the bottom layer A. In the case of the new undercutting technology, six large holes, up to 4.7 m deep and 280 mm in diameter, are drilled into the central oil-shale layer C. Roof bolter and face drilling machines are operating with remote controls that provide great safety conditions at a working place.

With the improved technology, the EAR reaches 4 m, which is two times as much as the conventional technology can guarantee, but explosive volume increases up to two times and explosion occurs during 4.5 seconds (the period is ~15 times longer than with the old technology). Under such technology the dimension of pillars reduced up to 20-30 % during six months after blasting operations on account of pillar parts flake away [Sabanov 2007]. As a result of greater advance rates, situations with unsupported room length up to 5.5 m with decreasing the stability of immediate roof can be expected. Increased explosive volumes can contribute to reducing of anchor tightening. In some places with complicated mine-geological conditions of the chamber blocks the entry advance rates 4 m was not achieved [Nikitin, 2005, 2006, 2007].

#### **4.3.1.1 Roof and pillars deformation**

The width of the room is determined by the stability of the immediate roof (IR). With advanced technology the entry advance rates reached 4 m. As a result of such greater EAR, the situations with unsupported room length up to 5.5 m with decreasing the stability of IR can be expected. The aim was to determine the main parameters for supported immediate roof deformation in areas with great entry advance rates. A comparative analysis with experimental data received from VNIMI and IGD Research Institute [Andreev 1987; Seleznev 1961] and data from working mining block using advanced technology (Figure 11) was made with the help of table 2. As results, the influence of advanced technology on the immediate roof stability estimated by the deformation criterion was not greater than at using the old technology (PAPER VII).

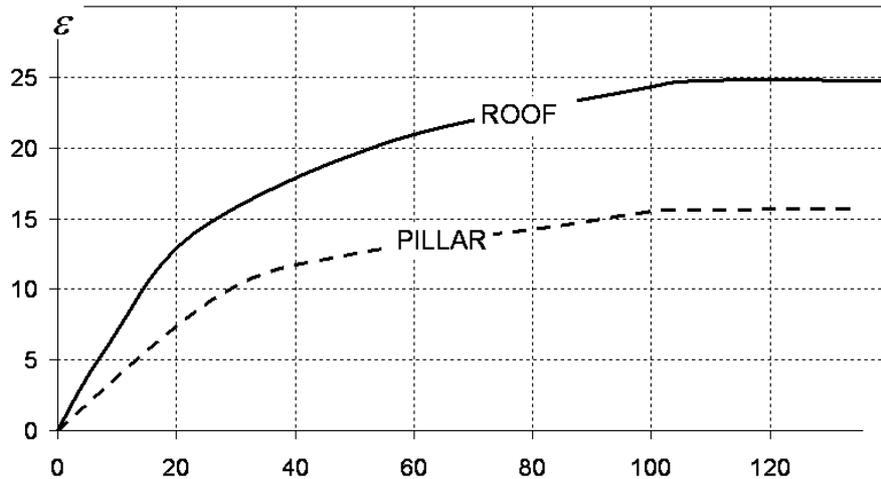


Figure 11. Roof-to-floor convergence curves,  $t$ -time (days),  $\varepsilon$ -deformation (mm)

Immediate roof deformation was evaluated with point 2 (Table 2) by Severity scale criteria and the total amount of inspected rooms with point 2 (Table 3) by Boundaries scale criteria. Total risk magnitude equals 4 – low.

#### 4.3.2 Blasting influence on anchor-bolt tightening

The target was determination of available blasting (seismic) influence on the anchors tightening. During this part of the study the actual field installation of the anchors at working face was tested to make sure that they were properly tightened after blasting. The bolt installation parameters like torque and pre-load were controlled by mechanical torque wrench and load cell with digital data logger. The previous experience has shown that the anchor must be tightened up to a tensile stress of 75% of the yield strength to resist blasting vibration and to prevent inadvertent yielding of the fastener through torque measurement inaccuracies [Allik 1964, 1975]. The higher load will increase the seismic resistance of the anchor-bolt, but the bolt will yield and unload if its yield point is inadvertently exceeded. *In-situ* tests for blasting dynamical influence on anchor-bolt were done. In working faces under average geological conditions (immediate roof with average stability) different types of anchor-bolts were installed.

Blasting operations were carried out on the same day after installation. The drop of anchors loading was about 23% (Figure 12).

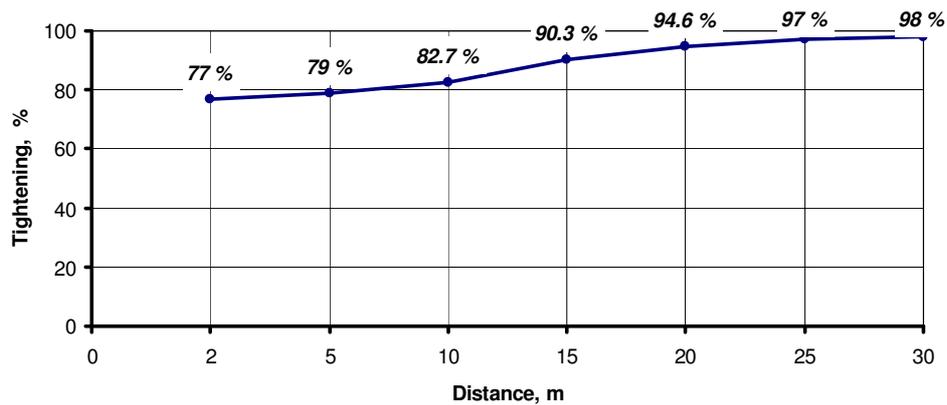


Figure 12. Dependence of tightening drop on the distance from face to bolt

With the bolt-to-face distance greater than 10 metres, the impact of blasting operations and rheologic parameters decreased. When the distance exceeded 30 m, the variations in the anchor loading process stopped (PAPER II).

On the bases of experimental results the probabilities of decreasing of anchor-bolt tightening under using an emulsion explosive were determined (Figure 12). The results can be useful in further planning of anchor-bolt grid.

#### 4.3.3 Emulsion explosive productivity

In an area with complicated geological conditions, where emulsion explosives were used, the entry advance rates reduced from 4 m to ~ 2.3 m in 17 % of revealed cases.

Based on observation results, it was established that under complicated geological conditions the probability of entry advance rate decrease  $P = 0.1445$  (Figure 13). Figure 14 A demonstrates normal face with correct made boreholes. The water flowing from a borehole complicates charging of borehole with emulsion explosive and reduces the quality of blasting operation. If a tectonic joint is present, the impact of blasting realization occurs only before it, but not further (Figure 14 C). Undulating surface of the face (Figure 14 B) and different density of rock massive with karst, in their turn, lessen the quality of blasting operation.

Under normal geological conditions, the probability of failure achieved  $P = 0.0225$ . The main reason was operator's failure  $P = 0.0166$ . At input of a hose for emulsion explosive delivery in boreholes, there was a stack detonator in cracks. In this case there was direct and uniform blast initiation that influenced the result. The length of the cut holes was practically always less than that of the boreholes (Figure 14 B). On account of the restricted free space for machine moving, non-parallel boreholes were made (deviation from the project blasting pattern) and the failure of the machine equipment existed as well. Emulsion explosive temperature and pressure did not considerably affect blasting works. The results of measurements showed that detonation velocity storage in underground conditions did not practically influence the properties of emulsion explosive.

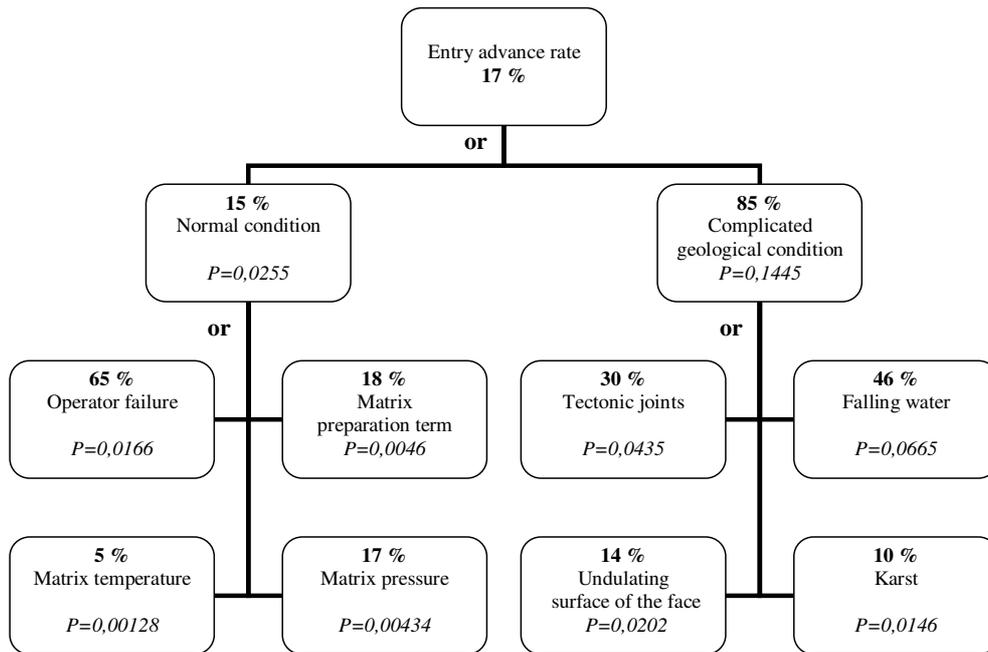


Figure 13. Event tree. % - Contribution of likely failure modes. P - Probability of adverse event.

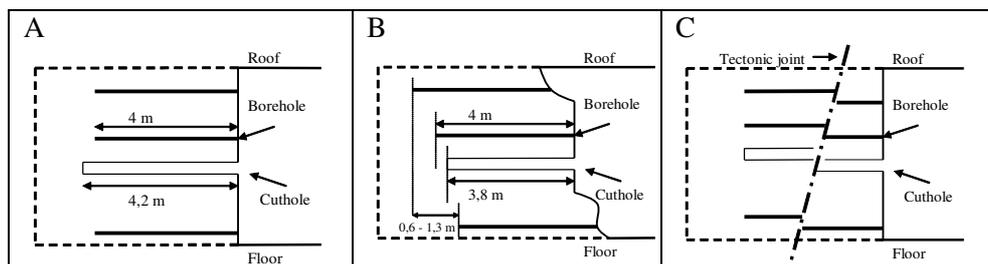


Figure 14. Schematic layout of the faces. A – normal face with suggested longer cut hole; B – undulating face with different lengths of boreholes, C – subsidence face due to a tectonic joint.

The influence of complicated geological conditions on the rate of entry advance with using emulsion explosive was determined. Under normal mine-geological condition (Figure 14 A) a more effective blasting pattern and parameters accuracy of face preparation were also defined. Some methods aimed at avoiding and mitigating the influence of blasting operation were proposed (PAPER III).

#### 4.3.4 Blasting influence on pillars dimensions

The aim was to determine the coefficient of emulsion explosive influence on the dimensions of pillars. The measurement of pillars showed that the project dimensions considerably differ from the real values. As a result of the application of emulsion explosive, the breaking of pillar side exceeded the corresponding value accepted for the cartridge explosive technology. In the inspected mining block in some areas the mine-geological conditions were very complicated due to karst, streaming water, and tectonic joints. The distance between tectonic joints in areas with complicated geological conditions was 3-10 m. Breaking of pillar parts was more intensive during three-four weeks after blasting operations, then slowed down or stopped. In normal mine-geological conditions the deviation of pillars dimensions was similar to that accepted for cartridge explosive technology. At the sites with complicated mine-geological conditions of the mining blocks, deviation of pillars dimensions from the project value achieved 30 % (Figure 15) [Sabanov 2007a].



Figure 15. The area of complicated geological conditions in mining block.

Table 6 presents coefficients for the influence of explosive operations ( $q$ ) for different mine-geological conditions using advanced technology. The coefficient can be applied for four different types of mine-geological conditions – the normal, average, low stable and unstable mine-geological conditions classified by the distance between tectonic joints. The distance accepted by the Mining Law instruction [Mining Law 2004] (\*\*).

Table 6. Coefficients of explosive operation influence

Mine-geological conditions	Coefficient of explosive operation influence, (q)		Distance between tectonic joints, m (**)
	h=2.8 m	h=3.8 m	
Normal	0.6	0.8	20
Average	0.7	1.0	10-20
Low stable	0.9	1.4	10
Unstable	1.2	1.6	3-5

The data obtained experimentally for different mine-geological conditions allows to consider the influence of blasting operation on the dimensions of pillars. The equation for calculation of pillars dimensions was supplemented by author with coefficients for calculation of parameters accuracy for advanced technology. Coefficients need to update and additional experimental inspections.

## **5 ENVIRONMENTAL RISK ANALYSIS**

Final part made with technological and environmental impact assessment. In Estonia, oil shales of different quality are used in the power plants to generate electricity and in shale oil processing; however, the different excavation methods and accompanying development processes have various emissions. Oil shale mining can significantly disturb the environment as a result of water and air pollution derived from the different extraction methods. However, waste generation as well as land-use impacts will be of greater concern than emissions to the water and atmosphere. Life Cycle Assessment (LCA) has proved to be one of the most attractive approaches for sustainability of the mining industry, which has used several environmental and economic indicators to assess its performance. The methodology comprises finding the best available way according to environmentally friendly technology [Amman 1999; Callow 1998; DETR 2000; Durucan 2006; Haes 2002; Jaber 1999]

The method of technological and environmental assessment of a combination of impacts arising from different mining processes gives an opportunity to find better ways for planning new mines in accordance with environmental performances for oil shale deposit conditions [Sabanov, 2008c] (PAPER VIII).

## 6 CONCLUSION

- The applicability of Risks Assessment technique for the conditions of the Estonian oil shale mining industry was determined. The modified methodology of risks assessment for various mining processes and problems, decision of which is impossible with other methods was elaborated. The method allows quickly, conveniently and qualitatively to solve, operate, find optimum variants for existing problems, to predict with the minimal expenses during the project stage how to flexibly avoid the subsequent problems.
- The method of technological stability control was developed. Work out formula to improvement pillars parameters. For estimation long-term rock durability under conditions of the Estonian oil shale for planning new mines design acoustical emission (Kaiser Effect) method is suggested.
- For selective mining of commercial oil shale and transportation systems the method of risk analysis was elaborated. Elaborated modified methods using fault tree gives information about the deviation of the processes and possibility determination of suitable variant. Basing on the excellent results of this investigation, it is recommended to use the applied methods for the whole network of transportation from mines and open casts to consumers and in selective mining processes.
- For advanced underground mining technology efficiency the safety factors and suitable parameters using risk assessment methodology was determined.
  - Immediate roof stability estimated by the deformation criterion and was classified by severity and boundaries (area of impact) scale.
  - Blasting (seismic) operation influence on anchor-bolt tightening and probabilities of decreasing tightening under using emulsion explosive were determined. The results can be useful in further planning of anchor-bolt grid.
  - Probabilities of influence of complicated geological condition on entry advance rate under using emulsion explosive were determined. Received probabilities of disturbance factors will help in mitigation process to avoid the negative influence on quality of blasting operation.
  - Received by experimental ways data for different mine-geological condition allow considering influence of blasting operation on pillars dimension. In formula for calculation pillars dimension was added coefficient to improve accuracy parameters for advanced technology.
- By the method of technological and environmental analysis of a combination of different mining processes emissions to water and air were determined. This method gives opportunity to finds better way for new mines planning in according with environmental performances for oil shale deposit conditions.
- The elaborated modified risk assessment methods represent a complicated system, which starts from initial analysis of rock mass properties, continues with the study of technological problems and ends with environmental impact analysis. Such methods enable solution of problems, decision of which is impossible with other methods. The presented risk assessment methods can be used in different geological conditions and also with other mining technologies.

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## ABSTRACT

In the Estonia the risk assessment methods are used in different branches of industry, but the number of references on solution of mining problems is limited. Investigations have shown that the above-mentioned methods are applicable for solving complicated mining problems. Prevention of the hazardous situations is more moral, ethical and economical than facing the adverse consequences. Risk assessment methods give the information about mining and its influence on the environment. In the stage of design, give unique possibility determinate negative consequences and allow work out means to avoidance and mitigation hazard factors. Having received the information, management of a mine can come to adequate political and strategic decisions. The methods can favorable reflect total problem from local to regional stage.

The purpose of the investigation is determination of applicability of risks assessment technique for conditions of Estonian oil shale mining industry and elaboration the modified methodology for various mining processes and problems, decision of which is impossible to be carried out by other methods.

The methods of risks assessment based on statistical analysis and used three general approaches depending on the type and quality of the assailable data: analytical, empirical and judgmental. Theoretical estimation helps systemizing chosen calculation methods.

The applicability of Risks Assessment technique for conditions of Estonian oil shale mining industry was determined. Elaborated modified risk assessment methods consist of complicate a system which starts from initial analysis of rock mass properties, continuation of technological problems and ends with environmental impact analysis, and has possibility solving of problems, decision of which is impossible to be carried out by other methods.

The modified risk assessment methodology successfully applied to solving following problems: stability of a mining block, extraction of mineral resources, loading, transportation and their influence on the environment. For determination of the efficiency and adequacy of the elaborated methods, risk assessment for advanced technology, designed mining blocks in the mine and new stripping mining in the open cast was performed. The elaborated risk assessment methods can be used in different geological conditions and also with other mining technologies. Such methods are universally applicable in various fields of mining production. The methods gives opportunity to find better way for new underground and open (surface) mining planning in according with environmental performances and economical profit of enterprises.

# KOKKUVÕTE

## RISKI HINDAMISE METOODIKA EESTI PÕLEVKIVITÖÖSTUSES

Maavarade kaevandamisprobleemide lahendamisel on otstarbekas kasutada riskide hindamise meetodeid, eriti, kui traditsiooniliste teooriate ja meetodite kasutamine ei anna häid tulemusi. Ohtlike olukordade vältimine võrreldes nende tagajärgede likvideerimisega on moraalsem, eetilisem ja majanduslikult kasulikum. Riski hindamine kasutamise otstarbekust on näidanud maailma praktika. Riski haldamise meetodeid käsitletakse maailmas ja Eestis erinevates tööstusharudes, kuid viited mäenduslike probleemide lahendamiseks on tagasihoidlikud. Riskide hindamise meetodi eeliseks on, et see võimaldab hinnata projekte juba varases staadiumis ja kõrvaldada seal olevad puudused. Meetod annab informatsiooni kaevandamisest ja nende mõjust keskkonnale ning võimaldab ettevõtte juhtkonnal vastu võtta adekvaatseid poliitilisi ja strateegilisi otsuseid.

Töö eesmärgiks on kontrollida olemasolevate riskihindamise meetodikate kasutatavust Eesti põlevkivitööstuses. Töötada välja uued uurimismeetodid erinevate protsesside kohta.

Riski hindamise meetodika põhineb statistilisel analüüsil. Sõltuvalt andmete kättesaadavusest on kasutatud analüütilist ja empiirilist lähenemist ning eksperthinnanguid.

Riski hindamise meetodika Eesti põlevkivitööstuses on keeruline süsteem, mis algab mäemassiivi omaduste uurimisest ja lõpeb keskkonnamõju hindamisega. Selline meetodika on universaalne, kasutatav erinevates tööstusharudes. Doktoritöö praktiline väljundid seisneb soovitusel: kuidas korraldada kambriploki stabiilsuse seiret, kasutada uusi tehnoloogiaid, paremini väljata, laadida ja vedada põlevkivi, vähem mõjutada keskkonda. Kuna kaevandamise tulemused ja mõju keskkonnale on teada, siis on lihtne kontrollida saadud tulemuste adekvaatsust ja vajadusel väljatöötatud meetodikat täpsustada. Töösse minevate kambriplakkide riskide hindamise meetodika täpsustamine toimub Estonia ja Viru kaevanduse tingimustes, kus kasutatakse käesoleval hetkel olevat kamberkaevandamise tehnoloogiat. Väljatöötatud riski hindamise meetodika on kasutatav kaevandamisviisi moderniseerimisel ja uute tehnoloogiate väljatöötamisel. Keskkonnamõju hindamine võimaldab valida parimat tehnoloogiat ja minimaalset negatiivset keskkonnamõju. Loodud riskide hindamise meetodika on kasutatav ka põlevkivimaardlate teistes geoloogilistes tingimustes, kus on kasutusel kamberkaevandamisviis ja rakendatavad ka teiste kaevandamistechnoloogiate puhul.

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# A METHOD FOR MONITORING WORKING MINING BLOCK STABILITY IN ESTONIAN OIL SHALE MINES

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**Abstract:** This paper deals with stability analysis and monitoring methods of working mining blocks in Estonian oil shale mines, where the room-and-pillar mining with blasting is used. The pillars are arranged in a singular grid. Calculations are performed considering the concept of critical area, conditional thickness and sliding rectangle methods. The results are presented by conditional thickness contours. The feedback of the real situation in a mining block is guaranteed by a monitoring system. For practical application the conditional thickness difference criterion is suited. It allows elaborating means to stop the negative processes in pillars and roof, and guarantees stable parameters and minimal losses of the mining block. The method is applicable in different geological conditions, where the room-and-pillar mining is used.

**Key words:** working mining block, room-and-pillar mining, numerical modeling, stability, critical area, conditional thickness, monitoring.

## Introduction

The most important mineral resource in Estonia is a special kind of oil shale. It is located in a densely populated and rich farming district. The structure of the productive oil shale bed makes the rock difficult to break from the total massive. This is also one reason why shearer mining has been shortly used. Underground oil shale production is obtained by room-and-pillar method with blasting. This method is cheap, highly productive, easily mechanize, and relatively simple to design.

It has become apparent that the processes in overburden rocks and pillars have caused unfavorable environmental side effects accompanied by significant subsidence of the ground surface. They cause a large number of technical, economical, ecological and juridical problems. On the other hand, the collapse in a mining block stops the mining works. The first spontaneous collapse of pillars and surface subsidence in an Estonian oil shale mine took place in 1964. Up to the present, 73 collapses on the area of 100 km<sup>2</sup> have been recorded.

Design of mining block parameters is based on the instruction used in Estonian oil shale mines [1]. The actual roof and pillar dimensions depend on the quality of mining works. Consequently, pillar and roof sizes vary from place to place within a mining block. If the difference between designed and actual parameters is large enough, the mining technology is disturbed: spontaneous collapse is likely to occur or the losses in pillars increase.

The aim of the investigation was to work out the monitoring method and means to stop negative impact on the environment and mining.

The study is based on the complex method including:

- Investigations of in situ conditions;
- Theoretical investigations;
- Modeling on PC.

For stability analysis and monitoring the concept of critical area, conditional thickness and sliding rectangle methods have been used [2, 3, 4, 5]. They are suited for modeling on PC. Visual Basic for application in Excel and MapInfo was used for numerical modeling. The results are presented by conditional thickness contours in a mining block map, which allow determining the potential collapse parameters: the location and area. For practical application the conditional thickness difference criterion is suitable. For these calculations the conditional thickness limits for critical area are determined. If the conditional thickness lies between the limits, a mining block is stable and losses are minimal. If the conditional thickness goes beyond the limits, the collapse is likely to occur or the losses increase. A method has been worked out for designing the dimensions of the pillars of the next rows so that the pillars could quickly reach conditional thickness close to the calculated value. In this case the stability of a mining block and minimal losses are guaranteed.

The stability monitoring method is applicable in different geological conditions, when the room-and-pillar mining is used. The applicability of the method has been demonstrated.

## **Geology**

The commercially important oil shale bed is situated in the north-eastern part of Estonia. It stretches from west to east for 200 km, and from north to south for 30 km. The oil shale bed lays in the form of a flat bed having a small inclination in southern direction. Its depth varies from 5 to 150 m. The oil shale reserves in Estonia are estimated approximately at 4 thousand million tons.

The oil shale seams occur among the limestone seams in the Kukruse Regional Stage of the Middle Ordovician. The commercial oil shale bed and immediate roof consist of oil shale and limestone seams. The main roof consists of carbonate rocks of various thicknesses. The characteristics of various oil shale and limestone seams are quite different. The strength of the rocks increases in the southward direction. The compressive strength of oil shale is 20-40 MPa and that of limestone is 40-80 MPa. The volume density is 1.5-1.8 Mg/m<sup>3</sup> and 2.2-2.6 Mg/m<sup>3</sup>, respectively. The calorific value of dry oil shale is about 7.5-18.8 MJ/kg depending on the seam and the location in the deposit.

## **Current mining system**

In Estonian oil shale mines the room-and-pillar mining system with blasting is used. It gives the extraction factor about 80%. The oil shale bed is embedded at the depth of 40-70 m. The field of an oil shale mine is divided into panels, which are subdivided into mining blocks, approximately 300-350 m in width and 600-800 m in length each. A mining block usually consists of two semi-blocks. The height of the room is 2.8 m. The room is very stable when it is 6-10 m wide. However, in this case the bolting must still support the immediate roof. The pillars in a mining block are arranged in a singular grid. Actual mining practice has shown that pillars with a square cross-section (30-40 m<sup>2</sup>) are best. A work cycle lasts for over a week.

## General theoretical background

The stability and losses of a mining block depend on the correct choice of the pillar and room sizes. For these calculations, the instruction for Estonian oil shale mines was elaborated [1]. It includes the calculation methods for the optimum parameters of the roof, pillars and support, based on long-term investigations in oil shale mines. On the other hand, it is known that the actual dimensions of the roof and pillars depend on the applied technology and quality of mining works. Consequently, the pillar and roof sizes vary from place to place within a mining block. Due to the complicated structure of the pillars and roof, their stability analysis and monitoring demand special calculation methods. For the analysis the concept of critical width, methods of conditional thickness and sliding rectangle were used and they suitable for modeling on personal computer.

The pillar load depends on the width of the mining block, so the concept of the critical width is to be used. The critical width is the greatest width that the rock above the mine can span before its failure, or, if there are pillars, the width we must mine before the pillars accept the full weight of the overlying materials [3]. In fact, the best indicator of critical width in a given situation will be provided from old mine maps, by records of failure and surface subsidence, and from measuring roof-to-floor convergence in the mines. For Estonian oil shale mines it is presented by the following formula [4, 5]:

$$L \geq 1.2H + 10 \quad (1)$$

where  $L$  – critical width, m;  $H$  – thickness of the overburden rocks, m.

In the three-dimensional case, the critical width becomes the critical area. The latter is the minimum area where the destruction of the pillars and surface subsidence is possible. Likely enough, the collapse begins in one critical area (potential collapse center) and then extends to the barrier pillars.

The load on the pillar depends on the real parameters of the pillar and roof. Conditional thickness is a geometrical parameter which considers the depth of excavation and the parameters of the pillars and roof [2, 6]. Geometrical interpretation of support coefficient conditional thickness is given in Figure 1.

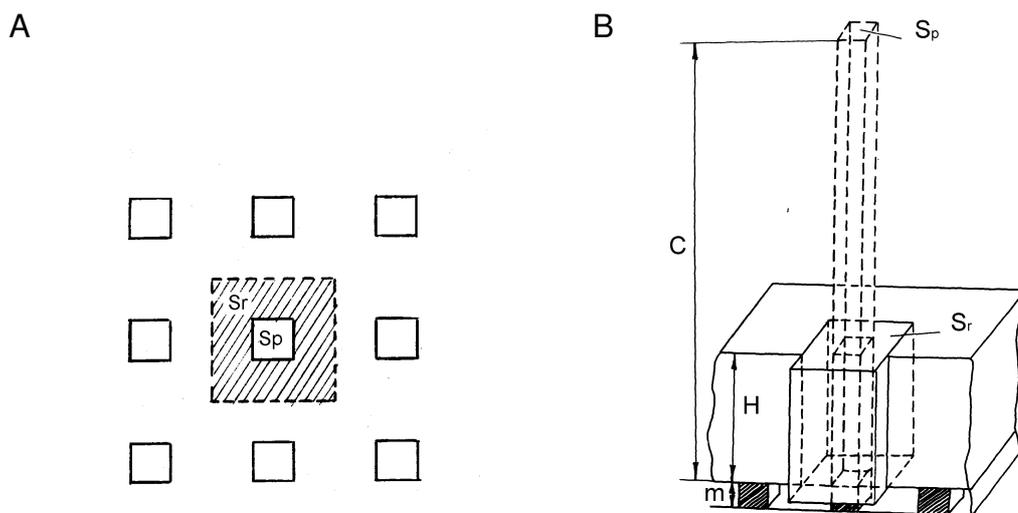


Figure 1. Geometrical interpretation of support coefficient and (A) conditional thickness (B)

It represents the height of a prism whose cross-section equals the pillar cross-section area. The support coefficient and conditional thickness are presented by following formulae [2, 6]:

$$K = \frac{S_p}{S_r}, C = \frac{H}{K} \quad (2)$$

where  $K$  – support coefficient;  $C$  – conditional thickness, m;  $S_p$  – cross-sectional area of a pillar,  $m^2$ ;  $S_r$  – roof area per pillar,  $m^2$ ;  $H$  – thickness of the overburden rocks, m.

Conditional thickness includes sufficient information and is suitable for stability calculations. Conditional thickness is related to the load on a pillar as follows:

$$\sigma = C\gamma \quad (3)$$

where  $\sigma$  – normal stress at the top of a pillar, Pa;  $\gamma$  – weight density of the overburden rocks,  $N/m^3$ .

If the load is too much, a sudden failure of the pillars is likely to occur. Conditional thickness for the critical area can be expressed by the following equation [7]:

$$C_c = \frac{H_a L^2}{\sum S_{pi}} \quad (4)$$

where  $C_c$  – conditional thickness of the critical area, m;  $H_a$  – average thickness of the overburden rocks in the critical area, m;  $S_{pi}$  – cross-sectional area of the  $i$ -th pillar in the critical area, m.

By the sliding rectangle method, the conditional thickness of the critical area must be determined for all positions inside a mining block. The results are presented by conditional thickness contours. The relative uncertainty in conditional thickness is 1.5% at the 95% confidence level. The presented method allows determining the center and area of a potential collapse in a mining block.

The working mining block stability and losses in pillars depend on the actual parameters of the pillars and roof. The changes in conditional thickness parameters of in situ conditions in a mining block are illustrated in Fig.2.

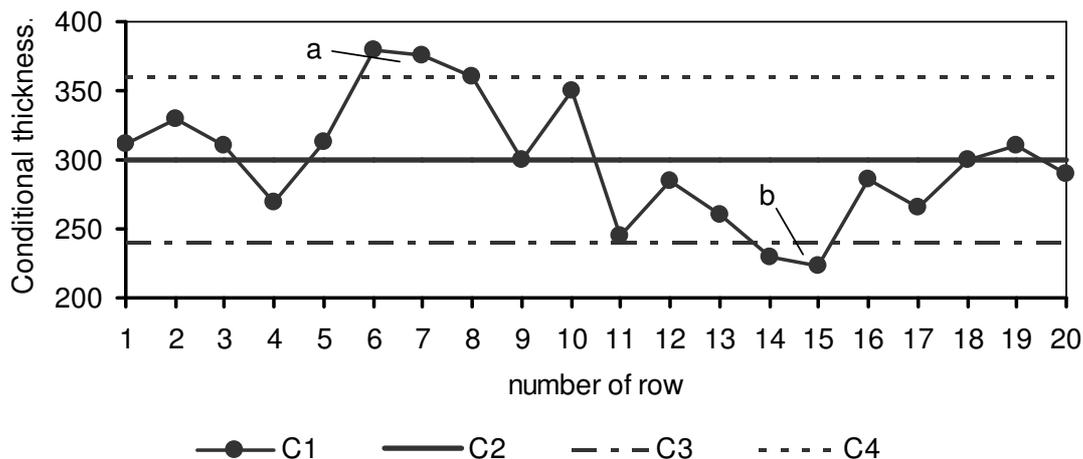


Figure 2. Deviation of the conditional thickness parameters from designed value  
a – collapse is possible; b – losses in mineral resources are increased; C1 – actual conditional thickness in the critical area; C2 – designed value of the conditional thickness; C3 – lower limit; C4 – upper limit.

One can see that the conditional thickness parameters fluctuate around the designed value. If the conditional thickness parameters remain between lower and upper limits, the mining block is stable and losses are minimal. If the conditional thickness is out of the limits, the collapse is likely to occur (area a) or the losses increase (area b).

The feedback of the real situation in a mining block is guaranteed by a monitoring system. For mining block monitoring the conditional thickness difference criterion is suited. It insures the stability of a mining block against random deviation of the pillars and roof parameters. Consideration the applied mining technology it is possible to increase or decrease the length of pillars guaranteeing the stability of the working mining block. The method to design pillars of the next rows is presented in Figure 3.

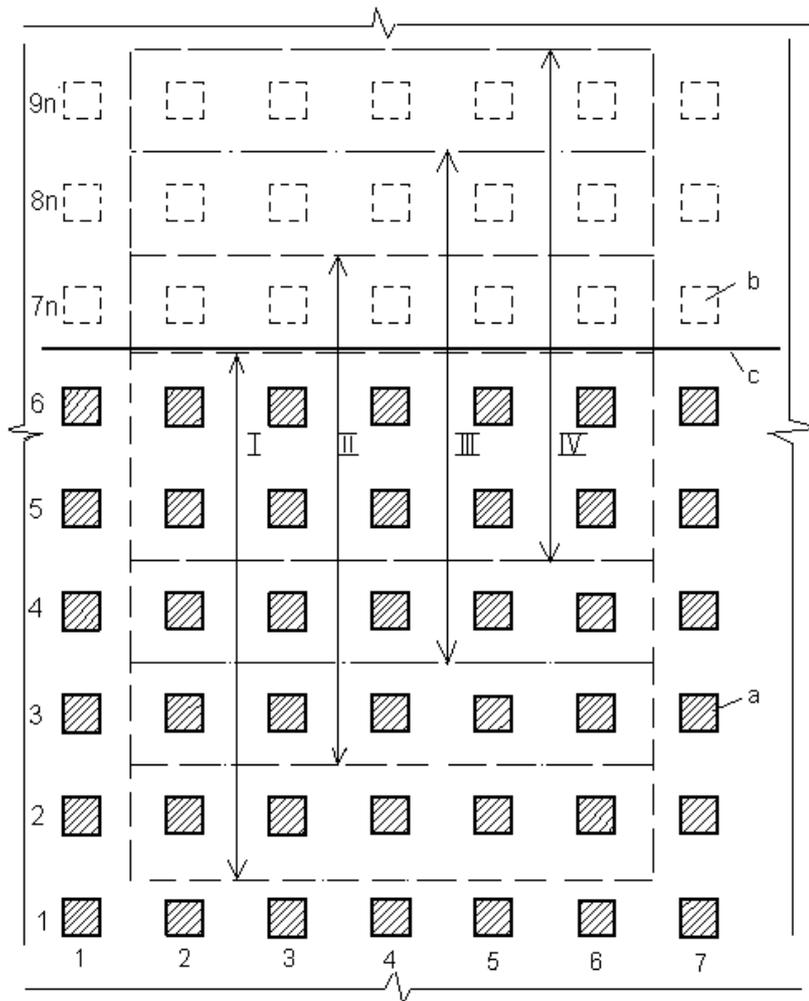


Figure 3. Method to design dimensions of the new rows

a – actual pillars; b – designed pillars; c – stope; I – critical area where the conditional thickness exceeds the limit; II, III, IV – critical area for design the parameters of new rows of the pillars.

In the critical area (I) the conditional thickness exceeds the limits. We must choose cross-sectional area of the pillars of the new row (7n) so, that the conditional thickness (critical area II) remains between the upper and lower limits. If it is not possible, we must change the parameters of the pillars of next row (8n, critical area III) and repeat this process until the purpose is reached. The methods and detailed description of the process based on the actual data is presented in the next paragraph.

## Results

The applicability of the method is demonstrated on the example of the Estonia mine, mining block No.705. The commercial oil shale bed of the thickness of 2.8 m is embedded on the depth of 53 m. The mining block is bordered by the barrier pillars. The optimum conditional thickness (designed value) is 267 m. The dimensions of the critical area exceed 74x74 m.

Investigations of in situ conditions have shown that by using blasting works, the random deviation of the actual pillar and roof parameters from the designed ones does not exceed  $\pm 1$  m. Statistical analysis showed that on average the deviation of the conditional thickness in sliding rectangle differs from the designed one  $\pm 7\%$ . The conditional thickness for the upper limit is 284 m and for the lower one 250 m.

Monitoring of the working mining block shows that in the left semi-block there appears a potential collapse area of the conditional thickness  $C=310$  m (Fig.4).

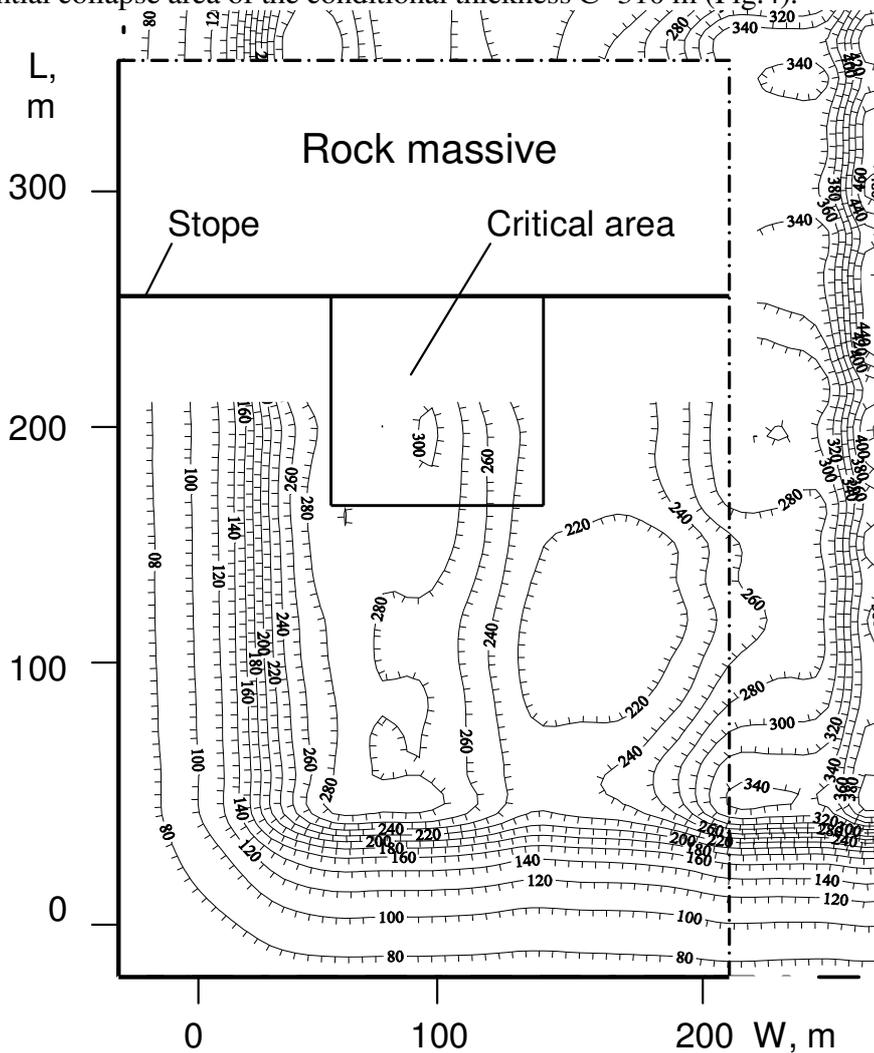


Figure 4. Monitoring of the working mining block. Estonia mine, block No.705

It is clear that the conditional thickness exceeds the limit and the collapse is likely to occur. It is possible to increase the cross-sectional area of the next row pillars to avoid collapse. The design of the pillar dimensions may be performed by three methods.

a) The cross-sectional area of pillars in the critical area is constant and equals the designed value. The conditional thickness varies (Fig.5).

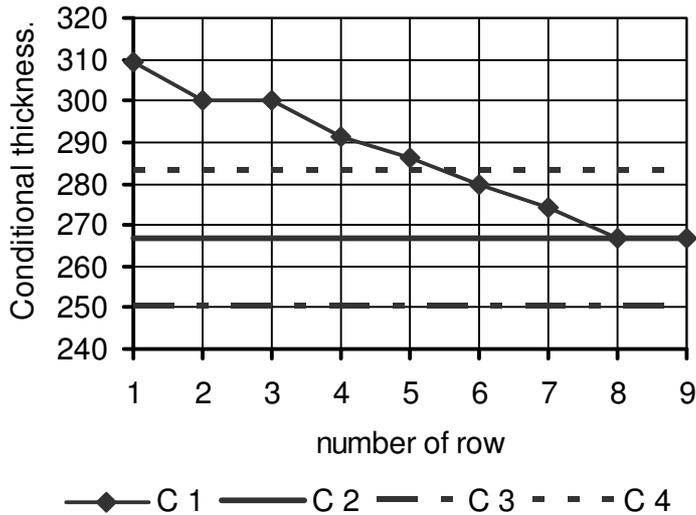


Figure 5. Changes in the conditional thickness values in the critical area at a constant cross-sectional area of the pillars  
 C1 – actual conditional thickness in the critical area; C2 – calculated value of the conditional thickness; C3 – lower limit; C4 –upper limit.

In this case the conditional thickness approaches slowly the designed value, and there appears a very large potential collapse area. The stability of a mining block is not guaranteed. The method is not applicable for practical purposes.

b) The conditional thickness in the critical area is constant and equals the designed value. The cross-sectional area of the new pillars in the next rows varies (Fig. 6).

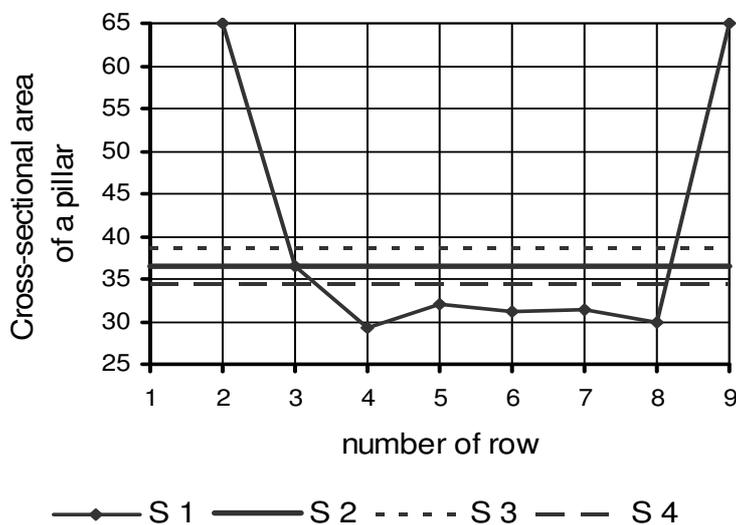


Figure 6. Changes in the cross-sectional area of the pillars at a constant conditional thickness  
 S1 – actual cross-sectional area of the pillar; S2 – calculated value of the cross-sectional area of a pillar; S3 – upper limit; S4 – lower limit.

It is clear that the cross-sectional area of the pillars of the first designed row is very large. The cross-sectional area of the next row pillars is small (rows from 4 to 8). The use of this method is technologically complicated.

c) For practical application a method is elaborated to determine the parameters of the next rows basing on the following formula:

$$S_n = \frac{C_r S_{pr}}{C_{pr}} \quad (5)$$

where  $C_r$  – actual conditional thickness, m;  $S_{pr}$  – designed value of the cross-sectional area of the pillars,  $m^2$ ;  $C_{pr}$  – designed value of the conditional thickness, m.

This method guarantees a quick and smooth approach of the conditional thickness and the cross-sectional area of the pillars to the designed values (see Fig.7 and 8).

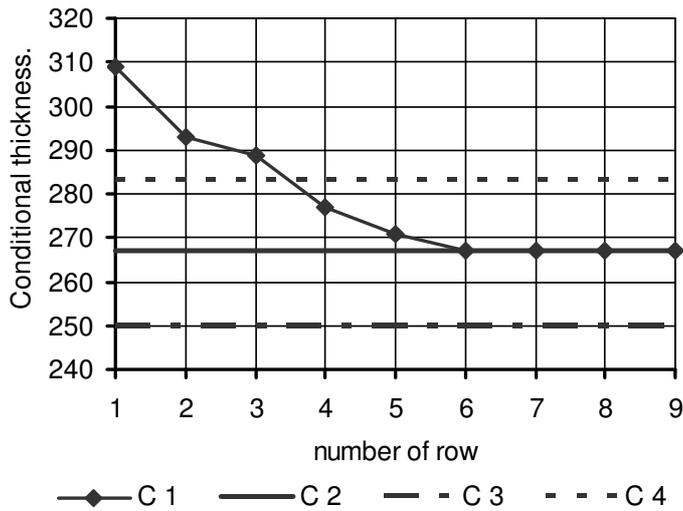


Figure 7. Character of the approach of the conditional thickness to the designed value  
C1 – actual conditional thickness in the critical area; C2 – calculated value of the conditional thickness; C3 – lower limit; C4 – upper limit.

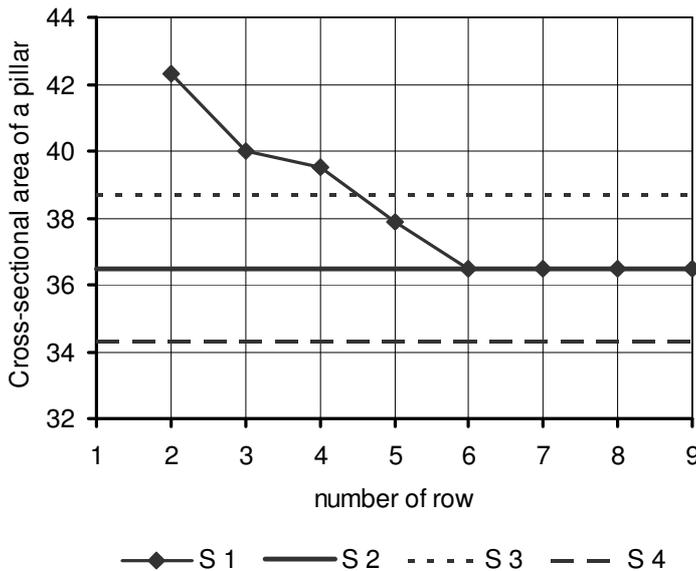


Figure 8. Character of the approach of the cross-sectional area to the designed value  
S1 – actual cross-sectional area of the pillar; S2 – calculated value of the cross-sectional area of a pillar; S3 – upper limit; S4 – lower limit.

It is obvious that the values of conditional thickness and dimensions of cross-sectional area of the pillars quickly approach the calculated value. In this case one must modify only the pillar parameters of the next three rows. There appears a potential collapse area, but it is small and not dangerous. The presented method has shown good results and is applicable for practical purposes.

## Conclusions and recommendations

The following conclusions and recommendations can be made.

1. In Estonian oil shale mines the room-and-pillar mining system with blasting is used. The pillars with a square cross-section arranged in a singular grid suit best. The actual sizes of the roof and pillars depend on the quality of mining works, and may deviate from designed ones. In this case the processes in pillars and overburden rocks may cause a large number of technical, economical, ecological and juridical problems.
2. The concept of critical area, conditional thickness and sliding rectangle methods perform calculations. The results are presented by conditional thickness contours in a mining block map allowing determining the potential collapse parameters.
3. A monitoring system is proposed to estimate the stability of a mining block and losses in pillars. For practical application the conditional thickness difference criterion is suited, which allows to work out means to stop negative processes in pillars and roof.
4. The stability of the pillars and roof, and the extraction of the maximum amount of oil shale are guaranteed, if the average conditional thickness in the critical area is in the range of  $\pm 7\%$  from the designed value. If this value exceeds these limits, one must increase or decrease the pillar cross-sectional area in the next rows. Mining technology allows changing the length of the pillars.
5. The monitoring methods are applicable in different geological conditions where the room-and-pillar mining is used. The applicability of these methods has been discussed.

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PAPER II. Nikitin, O., Pastarus, J., Sabanov, S. (2006). Roof bolting analysis at "Estonia" oil shale mine. In: Mine planning and equipment selection 2006: 15th International Symposium on Mine Planning and Equipment Selection, Torino, Italy, 20-22 September 2006. (Eds.)M. Cardu, R. Ciccu, E. Lovera, E. Michelotti. Italy: FIORDO S.r.l. - Galliate (NO), 2006, (2), 1441 - 1446.

# Roof bolting analysis at “Estonia” oil-shale mine

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**ABSTRACT:** This paper analyses some parameters of the anchor roof bolting technology for a new room-and-pillar method at “Estonia” mine. This technology is based on a blasting method applying emulsion explosives instead of packaged ones, change from 2.0 m to 4.0 m boreholes and on a new large-hole undercutting method. This new technology allows to reach entry advance rates up to 3.8 m. Because of such great advance rates the situations with unsupported room length up to 5.5 m decreasing the roof stability can be expected. Roof-bolting carriage with remote control is used for roof bolting automatization. The analysis of the different types of anchor bolts based on in-site underground tests is presented in this study. The risk factor assists to determine roof stability for different geological conditions. Measures on mitigation and avoidance of negative influence of blasting works are presented. For complicated geological conditions and high water regime at room height 2.8 m the risk of roof collapse is considerably high and room height of 3.8 m is highly recommended in such cases.

## 1 INTRODUCTION

This year, 2006, Estonian oil shale mining industry celebrates the 90th Anniversary. At the beginning of this year the first billion tonne of oil shale has been mined at Estonian oil shale deposit since 1916. Nowadays the main goal of the oilshale industry is to preserve its competitive ability in the market of power resources. Relatively cheap and high safety oil shale mining is capable to guarantee this competitiveness in the nearest future. At Estonian oil shale mines room-and-pillar method with blasting is used (Nikitin, 2005). It gives an extraction factor of 70–80%. For the last two years a new technology has been tested at “Estonia” mine. The new technology is based on a blasting method applying emulsion explosives instead of packaged ones, change from 2.0 m to 4.0 m boreholes and on a new large-hole undercutting method. This new technology allows to reach entry advance rates up to 3.8 m. As a result of such advance rates the situations with unsupported room width × length up to 7.5 × 5.5 m decreasing the roof stability can be expected. The main operations carried out in rooms (6-7 m width) include undercutting, drilling of blast holes, blasting, rock mass loading on the conveyor and roof bolting. Anchor bolts installed manually must still support the immediate roof.

## 2 THE ROOF BOLTING OVERVIEW

The first research of anchor bolting at Estonian oil shale mines was made in 1953 by Prof. Semenovskiy. Since 1959 anchor bolting has been applying at Estonian oil shale deposit as immediate roof support element. From 1959 till 1969 about 4.9 mln.t. of oil shale were mined by room-and-pillar method and more than 0.7 mln. anchors were installed (Allik, 1975).

At present annual production of oil shale is about 14 mln.t. and 1.1 mln. anchors are being set to support the immediate roof. It means that depending on the geological conditions 1 anchor needs to be set to mine 6-7 tons of oil shale.

The most commonly used anchor bolts in spite of high material capacity do not conform with some geological conditions and modern technical requirements. One of such nonconformance is fully

automatized anchor setting process which in case of conventional anchors KMA (Fig. 2) can't be utilize due to specific design of expansion unit. That's why new anchor suitable for automatic installation were tested during this study (Fig. 2). Comparing with conventional anchors (KMA) with greater bearing and lesser metal capacity, they will be suitable for roof bolting process automatization. Since 2002 by way of anchoring equipment drill carriages MBK of special design made on the basis of Russian face drilling machine BUA-3C have been in operation (Fig. 1). But the carriage can only drill and anchor setting-tightening is carried out by hand, whereas electric spanners are used to fasten screw nuts. The anchor expansion unit is fixed in harder limestone layer G/H (Fig. 1).

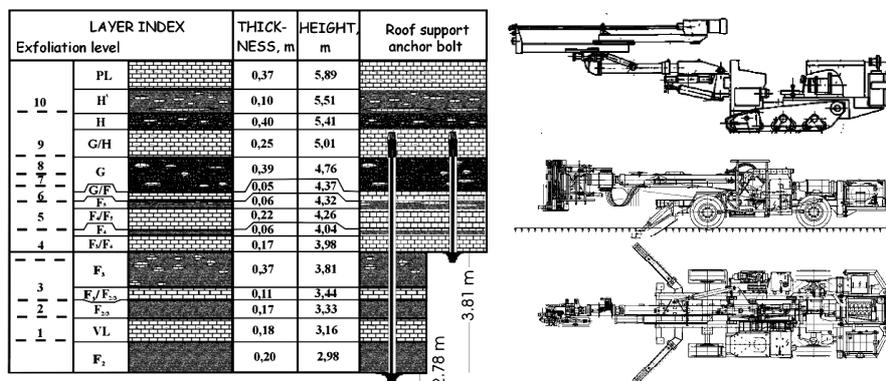


Figure 1. Immediate roof with available exfoliation levels (left hand), roof drilling carriage MBK (right hand above) and SMAG roof bolter FA523V and 2.78-3.81m is room heights.

Ensuring of high job safety, especially under complicated geological conditions fully automatized roof bolting process is very important nowadays. On this way development by using roof bolter machine, like SMAG (Germany) roof bolter FA523V is possible. Remote controlled roof bolter with torque of 0-300 Nm (Fig. 1) has been in operation since 2004 at "Estonia" mine. Thanks to this, anchor installing (MTA-bolt type, see Fig. 1, 2) productivity increases by 75% with greater safety conditions. It ensures that the problem of bolts installation process can be solved completely in the nearest future.

### 3 TARGETS AND EQUIPMENT

This study was carried out by two steps. The target of the first step was determination of the axial load dependence on the installation torque for the different types of anchors and conditions (Fig. 2). The main target of the second step was determination of available blasting (seismic) influence on the anchors tightening. During this part of our study we needed to test the actual field installation of the anchors at working face to make sure that they were properly tightened after blasting.

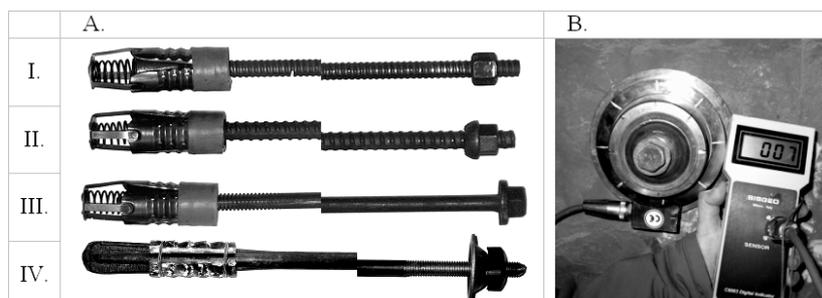


Figure 2. Anchor bolts, tested during study (A.) and load cell with digital data logger SisGeo (B). I.-IV tested anchor types: Unibar, Gewi, MTA (with Duraset expansion unit); KMA correspondingly

The bolt installation parameters like torque and pre-load were controlled by mechanical torque wrench and load cell with digital data logger. According to the compliance certificates the accuracy of the torque wrench is 2% and for load cell is less than 0.5% F.S. Mechanical properties of the tested anchors are shown in Table 1.

Table 1. Mechanical properties of tested anchors

Anchor type	GEWI	Unibar	KMA	MTA
Yield strength, MPa	550	650	400	370
Ultimate tensile strength, MPa	700	700	550	540
Nominal diameter $d_n$ , mm	16.0	13.5	20.0	18.0
Pitch, mm	6.9	4.5	2.5	2.5

The cold form thread pattern (LH-Unibar) and hot-rolled bars (RH-Gewi; Tread Bar) give closer tolerance between thread and nut. These features provide improved load retention capability, compared to conventional (lathe cutting) threaded bar (RH-KMA; MTA).

#### 4 THE STUDY RESULTS

The most practical and commonly accepted equation for describing the process of tightening is shown below (Bonenberger, 2003). There are three components of total input torque ( $M$ ). They are pitch torque ( $M_p$ ), thread torque ( $M_t$ ) and bearing surface torque ( $M_B$ ).

$$M = M_p + M_t + M_B = F \left( \frac{P}{2\pi} + \frac{\mu_t R_t}{\cos \alpha} + \mu_B R_B \right) \quad (1)$$

where  $F$  is the tensile force generated in the bolt as a result of tightening,  $P$  is the thread pitch,  $\mu_t$  and  $\mu_B$  is the thread and bearing surface coefficients of friction,  $R_t$  is the radius at which frictional force in the threads is assumed to act,  $\alpha$  is the thread flank angle,  $R_B$  is the radius at which frictional force between the fastener and bearing surface is assumed to act.

The results of the torque measurements and calculations are presented in table 2 below.

Table 2. Torque Distribution Table

Anchor type	GEWI	Unibar	KMA	MTA
torque needed to overcome (nut-) boltface friction $M_B$ , %	54	54	51	51
torque needed to overcome thread friction $M_t$ , %	29	28	41	41
torque needed to extend the fastener $M_p$ , %	17	18	8	8
tightening torque to pre-load (30kN) an anchor $M$ , Nm	214	127	144	156
$\pm$ , Nm	33	24	28	30
torque coefficient (clean conditions), $k=M/F/d_n$	0.41	0.35	0.23	0.29

Data calculated in table 2 show that required tightening torque to pre-load (30kN) Gewi anchor is greater in comparison with others tested anchor types (Fig. 2a). Realized torque, needed to extend the fastener for Gewi and Unibar is two times greater than for others due to the greater thread flank angle and pitch step. It means that time for tightening is 2-3 times lesser than for KMA and MTA anchors.

The figure 2 illustrates torque-tension relationships for different conditions. It is known that variation in nut factor can cause significant change in bolt tension for a given torque. If the contact surface condition is changed from clean to rust (for KMA), the force in the anchor will change from 39 kN to 15 kN for the same torque 200 Nm (Fig. 2b.). Variations of  $\pm 0.05$  in torque coefficient ( $k$ ) are routinely experienced, but in different surface conditions the larger variations are not unexpected.

The previous experience shown that anchor must be tightened up to a tensile stress of 75% of the yield strength to resist blasting vibration and to prevent inadvertent yielding of the fastener through torque measurement inaccuracies (Alik, 1964). The higher load will increase the seismic resistance of the bolt, but the bolt will yield and unload if its yield point is inadvertently exceeded. In-situ tests for blasting dynamical influence on anchors were done in four rooms. In three working faces under average geological conditions (immediate roof with average stability) 21 Gewi and 27 MTA anchors were installed. Eight of them, MTA anchors during the first six days after installation were under reologic influence only (Fig. 3b, bold dotted line). At that, the anchors tightening decreased under the reologic influence in average by 10%, and at the end of 35 days period to average 11%. In the fourth room with weak roof 11 MTA anchors were tested.

Blasting operations were carried out on the same day after installation –the first time period (Fig. 3b, I.). As you can see the average drop in the anchors loading were about 18%. During the next two weeks the pillar and rooms around it were formed completely –the second time period (Fig. 3b, II.), and due to roof deformation the anchor loading increase up to 95%.

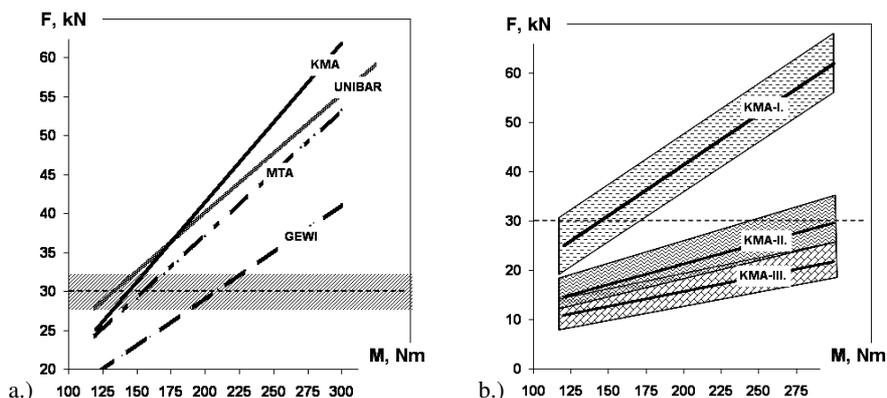


Figure 2. Torque-tension relationships for different types of anchors (a.) and conditions (b.).

where KMA-I; II; III. are the KMA anchor surface conditions: clean with new anchor, nut and bearing plate; with rusted nut and new anchor and bearing plate; all rusted.

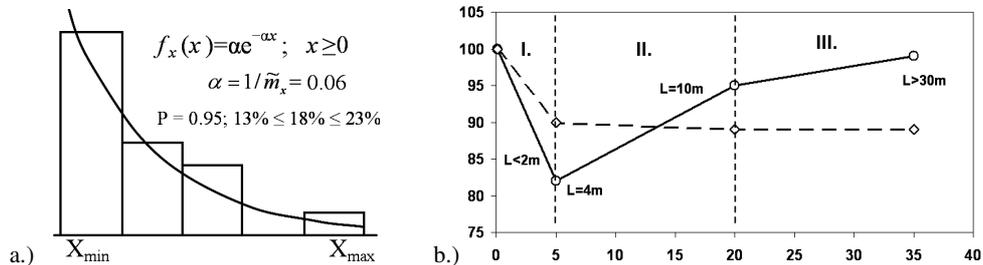


Figure 3. Creep behaviour in tightening (a.) on blasting works, on time and bolt-to-face distance influence (b.).

where on Figure a.) is the exponential distribution graphic for tightening decreasing after first blasting; P is the probability; on Figure b.) X-axis is a time in days; Y-axis is the tightening from pre-loaded in %; numbers I-III. are the time periods; L is the bolt-to-face distance for the given period; dotted bold line is the tightening decreasing on reologic factors (non-blast influence), bold line is the tightening dependence on blasting works, geometric and reologic factors.

We can conclude that if the bolt-to-face distance ( $L$ ) is greater than 10 meters blasting operations and reologic parameters influence will decrease. At the end of third time period (Fig. 3b, III.) bolt-to-face distance was more than 30 m, and the variation in anchor loading process stopped. Analysis of received data showed that percent of tightening decreased for Gewi, MTA and KMA anchors and can be illustrated by average dependence, presented on Figure 3b and is characterised by exponential distribution with  $\alpha = 0.06$  (Fig. 3a).

The Unibar anchor was very sensible (frail metal) to installation angle and crushed on wide range of the loads about 5.8-9.6 ton. On loads about 5.6-6.5 ton, that is practically equal to anchor expansion unit bearing capacity in limestone rocks (70kN), the permanent elongation by 0.22-0.33% was registered. It's obvious that safety factors and functional working loads are at the discretion of the project design engineer, however in-situ loads should never exceed 80% of the published ultimate bar strength.

## 5 RISK ANALYSIS

Scope and expectation of the risk analysis defined influence of blasting operation on tightening decreasing. Assumption of the proposed risk analysis was reason of tightening decreasing for complicated geological condition (cracks arrangement) and high water regime (water pressure) (Tab. 3). The initial tightening of anchor does not remain constant and vary with installing and reological conditions, like following: process of creep, seismic influences from blasting, exfoliation and displacement of caving in breeds.

Bearing capacity of anchor is in direct dependence on initial tightening. Room width increasing and decreasing in tightening causes of roof exfoliations inside and above of supporting zone. Reduction of initial tightening up to 1.0-1.5 t increases zone of blasting influence in two times and intensity of displacement in three times. In exfoliation cavities water accumulates from water-bearing layers, that additionally preload an anchors and weakening the roof. Under the low water pressure tightening decreasing by 5 - 25 % additionally and in complicated geological conditions can make up to 50-100% from initial. In most cases roof collapses occurred along room axis and depending on arrangement of cracks. As collapse prevention methods density of anchoring increasing and chamber width reducing are very useful. Direction of working chambers under corner  $45^\circ$  to cracks reduces the failure probability also (Parusimov et al. 1960).

On experimental data it is certain, that initial tightening should make 3.0 t and anchor loading falling up to 1.0-1.5 t will lead to full loss of anchor bearing capacity. The risk factor is determined by ratio of initial tightening ( $I$ ) to critical ( $C$ ), taking into account inflow water pressure ( $W$ ) and roof stability coefficient ( $K$ ). For conditions of "Estonia" oil-shale mine the risk factor is normal, if the limits are  $2 \leq R \leq 4$ . Other values can be regarded as dangerous or demanding the additional control. Roof stability coefficient ( $K$ ) depends on spacing interval between tectonic joints shown in Table 3.

$$R = (I + W)K / C \quad (2)$$

where  $R$  is the risk factor,  $I$  is the initial tightening,  $C$  is the critical tightening (1.5 t),  $W$  is the inflow water pressure,  $K$  is the roof stability coefficient.

Table 3. Inflow water pressure and roof stability coefficient for deferent geological conditions

Conditions	$W, t/m^2$	Roof stability coefficient, $K$	Tectonic joints spacing, m
Normal	0.0	1.00	$\geq 20$
Average	0.1	1.20	10-20
Low stable	2.5	1.45	5-10
Unstable	5.0	1.82	$\leq 5$

## 6 CONCLUSIONS

The clearly defined thread design of Gewi is more robust and less prone to damage comparing with others. For rolled bars no reduction in strength as a result of threading and full cross sectional area is utilized. Threads for Gewi and Unibar are specially designed with a rugged thread pitch wide enough to be fast under job site conditions and easy to assemble. Underground tests showed that these anchors can be tightened by three times faster than MTA and KMA. Thanks to this the anchor installation productivity will increase by 4-7% especially at places, where anchors are installing manually. The Gewi and Unibars anchors has a continuous rolled-in pattern of deformations along its entire length which allows anchorage hardware or couplers to thread onto the bar at any point; it is possible to unify anchor lengths in underground mines.

Following the previous study it can be said that work is in progress and Gewi, MTA or Unibar (required for more detailed research) anchors capability for automatized installation was proved. The economical viability can be decisive key of anchor type choosing in the nearest future.

The risk factor assists to determine roof stability for different geological conditions. Measures on mitigation and avoidance of negative influence of blasting works are presented. For complicated geological conditions and high water regime at room height 2.8 m the risk of roof collapse is considerably high and rooms height of 3.8 m is highly recommended in such cases.

### 6.1 Acknowledgment

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# Risk assessment of blasting works in Estonian oil shale mines

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**ABSTRACT:** Underground oil shale production in Estonia is obtained by room-and-pillar method with blasting. This method is cheap, highly productive, easily to mechanize and relatively simple to design. Packaged explosives are used for blasting work. The length of boreholes and entry advance rate is about 2.0 m. New mining technology is based on a blasting method to move from packaged to emulsion explosives (NOBELIT 2000U) from 2.0 m to 4.0 m boreholes (ATLAS COPCO machine) and new undercutting (SMAG machine) method. With such equipment new technology the entry advance rates reached 4.0 m., but in 17 % revealed case entry advance rates reduce to 1.3-2.5 m.

This paper deals with the risk assessment of blasting work applied different type of blasting passport for new room-and-pillar mining technology with modern machines at Estonian oil shale mines.

This study addresses risk associated with quality emulsion explosive, drilling and undercutting works for entry advance rates. Results of detonation velocity for emulsion explosive matrix are presented. For risk estimation the event tree is used. Investigation showed that the likelihood and the consequences of the risk are not acceptable. The results of the risk assessment are of practical interest for practical purposes.

## 1 INTRODUCTION

The most important mineral resource in Estonia is a special kind of oil shale. The structure of the productive oil shale bed makes the rock difficult to break from the total massive. Underground oil shale production is obtained by room-and-pillar method with blasting. This method is cheap, highly productive, easy to mechanize and relatively simple to design.

The main problems of the current technology are great volume of blasting operations, low mobility and concentration of works due to the small entry advance rates (1.5-1.7 m per blasting). Machine BUA- 3C guarantees length of a borehole up to 2 m. To increase the effect of blasting, undercutting is used. It is based on the bottom cutting (cutter URAL-33), which gives horizontal cut into the bottom layer A, 15 cm high and 1.8 m deep. Packaged explosives are used for blasting.

New machinery (ATLAS COPCO BOOMER 281-DC11) and modern technology should guarantee greater entry advance rates about 4 meters. Blasting method uses the emulsion explosives NOBELIT 2000U in boreholes of 4.0 m deep and 38 mm diameter. Undercutting method is based on the drilling (SMAG FA523V) of six holes into the central oil shale layer C (Nikitin & Sabanov, 2005). Their depth is 4.5 m and diameter 280 mm (Fig. 1). Three various blasting pattern have been tested and estimated.

New blasting modules UG 547 and UG 2000 (mixing-and-charging modules) in mining blocks No. 3104 and 3105 (Estonia mine) were tested, where mining depth is 62 m. Investigation showed that in practice the entry advance rates is reduced up to 1.6-2.5 m, which is caused by different factors. Conventional theoretical basis does not enable to solve these problems. Only data received by means of experience and practical can be acceptable. Data can be used to solve the problems blasting, using the risk management/assessment methods (Hoeg, 1996).

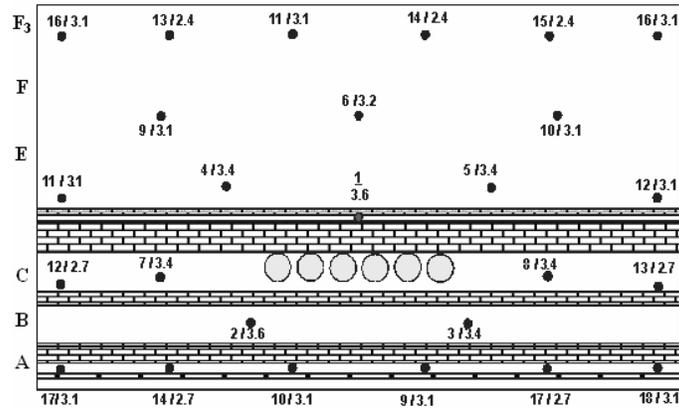


Figure 1. Commercial oil shale bed and boreholes and cut holes pattern.

## 2 GEOLOGICAL CONDITION AND MINING SYSTEM

The commercial oil shale bed and its immediate roof consist of oil shale and limestone seams. There are six commercial important oil-shale seams that are specified from the bottom to the top by the indexes from A to F (Fig. 1). There are often tectonic joints in the rock massive. The oil shale bed is embedded at the depth of 40-70 m. The main roof consists of carbonate rocks of various thickness. The characteristics of various oil shale and limestone seams are different. Compressive strength of oil shale is 20-40 MPa and that of limestone 40-80 MPa. The volume density is 1.5-1.8 Mg/m<sup>3</sup> and 2.2-2.6 Mg/m<sup>3</sup>, correspondingly.

Room-and-pillar mining system with blasting is used at Estonian oil shale mines. The height of the room is 2.8 m. In case of weak immediate roof the height of the room is up to 3.8 m.

## 3 EMULSION EXPLOSIVE CHARACTERISTIC

NOBELIT 2000 is an emulsion explosive, which has been elaborated for blasting works under conditions of Estonian oil-shale deposit (dust explosion hazard). It is waterproof, entirely fills a borehole, high level of safety, with a small sensitivity to mechanical and temperature loads. Orica Eesti Company produces explosives for Estonian open-pits and mines. Parameters of NOBELIT 2000 are presented in Table 1.

Table 1. Parameters of NOBELIT 2000

Parameters	Value
Blasting energy, kJ/kg	3191
Evolving gas volume, l/kg	929
Energy density, kJ/kg	792
Detonation velocity, km/s	3.5 – 4.5
Density, g/sm <sup>3</sup>	0.85
Cutoff diameter, mm	0.85
Brisance, mm	18

Preparation time of the explosive is more than 45 minutes. Short-delay detonators (250 ms) DYNADET with series of 1 to 18 are used (Fig.1). Total blast duration is 4.5 seconds.

#### 4 RESULTS OF INVESTIGATION

To determine detonation velocity boreholes drilling was made under observable charges. The initial temperature of “fresh” (mixing-and-charging module UG 547) emulsion explosive was 25 °C and this one for “old” (mixing-and-charging module UG 2000, explosive period of storage 45 days in underground conditions before blasting work) was 13.4 °C. The temperature of surrounding rocks was 9.4 °C and air 9.9 °C. 3.5 kg of emulsion explosive were charged into each borehole. To determination detonation velocity sensitive element was placed into boreholes, located parallel by cut holes and zero delay of detonator operation. It represents steel stick 1500 mm of length and 4 mm diameter. Two fiber-optic cables are attached along the stick. Emulsion explosive are injected into control boreholes then sensitive element is put into charging area. To determine detonation velocity, EXPLONET FO 2000 was used. The results of the investigations are presented in Table 2. Emulsion explosives “old” in boreholes 1-3 and “fresh” in boreholes 4-6 were charged.

Table 2. Parameters of NOBELIT 2000

Location	Density, g/sm <sup>3</sup>	Detonation velocity, m/s
Borehole 1	0.82	3875
Borehole 2	0.82	3888
Borehole 3	0.82	3677
Borehole 4	0.70	3968
Borehole 5	0.86	3992
Borehole 6	0.76	3952

It has been established, that emulsion explosive storage time (45 days) practically has no influence on detonation velocity. After single borehole explosion, there were not traces of burning and rests of emulsion explosive in a visible part. It is established, that borehole with external diameter 36 mm complicates input the hose in one. To improve the blasting works it is necessary to use the diameter of boreholes 38-40 mm.

#### 5 RISK ANALYSIS

New technology applied in Estonian oil shale mines has revealed technical, organizational, and human circumstances problems (Pastarus & Sabanov, 2005). Main question come, when expected results of entry advance rate to 4.0 m was not achieved. Scope and expectation of the risk analysis were defined at the first stage of blasting work testing. Identification geological condition to which the study relates was made. Assumption of the proposed risk analysis was reason deterioration quality of emulsion explosive. The second case – drilling machine operator failure.

#### 6 FAILURE MODE IDENTIFICATION

Quantitative estimation of the blasting works with long boreholes results ( Kaplan & Garrick 1981).  
Estimation of faces preparation (depth of drilling, arrangement of cut holes, arrangement of boreholes) and registration of deviations.  
Assessment of mixing-and-charging module works (pressure in hose, temperature of matrix, and productivity of the pump).  
Quality estimation of boreholes charging.  
Application of different blasting patterns 75 kg, 75kg (modified) and 79.5 kg per one face.  
Estimation of emulsion explosive preparation time influencing on the quality of blasting work.

## 7 RISK ESTIMATION

Under complicated geological conditions probability of entry advance rate decrease made  $P = 0.1445$  (Fig. 3).

If the stream of water followed from tectonic joints to boreholes, it would complicate emulsion explosives charging. It depends on how much emulsions explosive will be charged into boreholes. On account of tectonic joints influence blasting realization occurred only before it, but not further (Fig. 2c).

At input of a hose for emulsion explosive delivery in boreholes there was a stack detonator in cracks. In this case there was direct and uniform blast initiation that influenced on the result.

Of course undulating surface of the face (Fig. 2b) and different density of rock massive with karst bring own harm to blasting.

Under normal geological condition operator's failure appeared: the cut holes length practically was always less than boreholes length (Fig. 2b). On account of the restricted free space for machine moving non parallel boreholes were made (deviation from project blasting pattern) and the failure of the machine equipment existed as well.

Emulsion explosive temperature and pressure did not affected blasting works considerably.

Measuring detonation velocity standard of emulsion explosive quality have been established within 45 days period of underground storage.

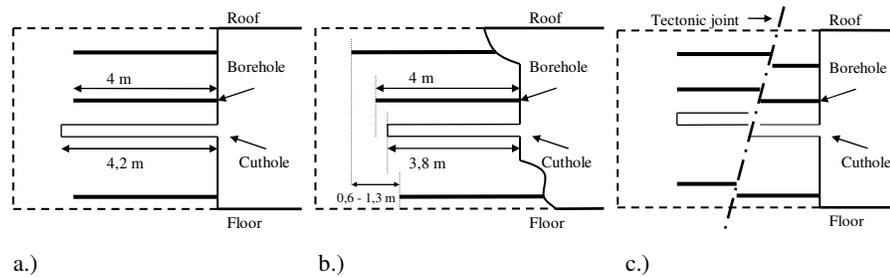


Figure.2. Schematic layout of the holes, faces and geological conditions. (a.) – normal face with suggested longer cut hole; (b.) –undulating face with different length of holes; (c.) –subsidence face because of tectonic joint.

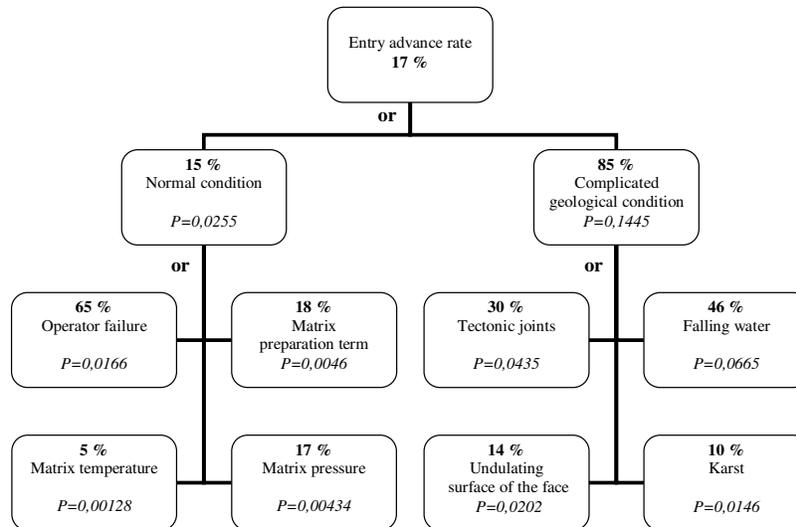


Figure 3. Event tree. % - Contributing of likely failure modes, P - Probability of adverse event.

## 8 RISK EVALUATIONS

Blasting pattern with 79.5 kg charge has high parameters on entry advance rate, lower specific charge, is less sensitive to quality of drilling and forms a face well (the average specific charge of emulsion explosive is  $0.93 \text{ kg/m}^3$ , average entry advance rate is 3.58 m).

Blasting pattern with 75 kg charge - stable, has a wide scatter of parameters (the average specific charge of emulsion explosive is  $1.11 \text{ kg/m}^3$ , average entry advance rate is 3.0 m).

Modified blasting pattern with 75 kg charge (modified) has better parameters, than pattern 75 kg with uniform distribution of charges, but also stable parameters (the average specific charge of emulsion explosive  $0.90 \text{ kg/m}^3$ , the average entry advance rate 3.41).

The deviation from parallelism of boreholes direction around cut holes renders insignificant influence on the charge size 79.5 kg and influences on the charge 75 kg. The last is connected with small charges around cut holes.

The emulsion explosive preparation time for blasting less than 40 minutes changes for the worse parameters of the blasting work, at preparation more than 2.5 hours quality of emulsion explosive does not worsen (there was an opinion, that long time of preparation leads to the bad results of blasting).

Application of matrix with temperature above  $30^{\circ} \text{C}$  leads to deterioration of the results.

Pressure in mixing-and-charging module hose from above 1.7 MPa worsens the results of blasting.

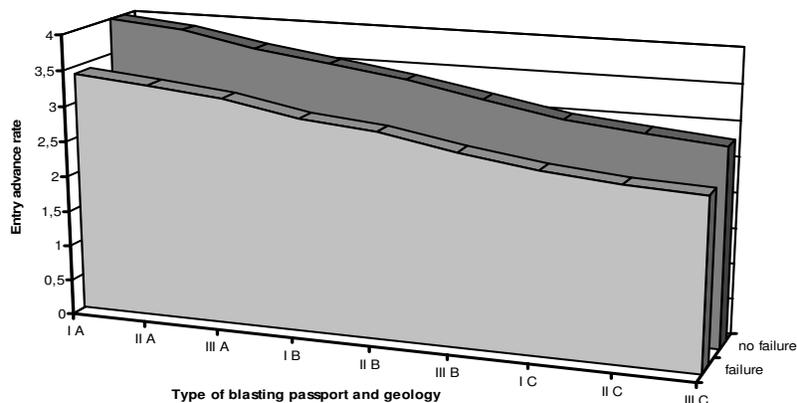


Fig.4. Entry advance rate deviation from different geological condition and blasting pattern. I – 79.5 kg, II – 75 kg (modified), III – 75 kg; A – normal geology, B – medium, C – complicated.

## 9 RISK MANAGEMENT

To organize manufacturing of a new matrix so that by the moment of delivery to mine its temperature was not above  $25^{\circ} \text{C}$ .

To equip charging module with hose having graduation of length for operative estimation of borehole depth and updating charge size at deviation of borehole length from design.

Not to suppose simultaneous preparation of the crossed faces. To maintain direction of cut holes along a line of face advance direction, to supervise parallelism of cut holes groups drilling (the factor defining the subsequent quality of face drilling).

Depth of cut holes should be 4.2 m (at formation of inrushes during undercutting, depth of inrush is a part of cut holes depth) (Fig. 2A).

It is not supposed to change boreholes position located around the cut holes including layer F2.

To use the pattern with size of charge 79.5 kg.

After crossing to the following chamber first of all it is necessary to charge 5-6 holes in a pack A, and then all other face.

At pressure in hose from above 1.7 MPa to replace a mixer.

It is not supposed:

Reduction of charge in the holes located around cut holes (exception - cases when borehole is drilled inside inrush, in this case the weight of the charge in kg can be reduced by amount equal to the depth of inrush in meters).

Change arrangement of detonator delay series.

Blasting earlier than in 40 minutes after the termination of charge forcing (reduction of time to 20 - 30 minutes can be admitted at temperature of the matrix more than 25<sup>0</sup> C).

## 10 CONCLUSIONS AND RECOMMENDATIONS

New oil shale mining technology with blasting was investigated at "Estonia" mine of Eesti Põlevkivi Ltd. Experiments showed that entry advance rate 4 m was not reached in 17% of cases. For analysis, the risk analysis/assessment methods were used.

Investigations in situ conditions showed that complicated geological conditions disturb the quality of blasting works: presence of tectonic joints in rock massive and the falling water into the boreholes. Second influence factor is quality of blasting works: blasting passport parameters deviation from project. Some mitigation method to reduce the negative influence of these factors on the quality of blasting works is presented.

Applicability of risk analysis/assessment methods for blasting works are demonstrated for practical problems solving.

## 11 ACKNOWLEDGEMENT

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PAPER IV. Pastarus, J-R.; Sabanov, S.; Tohver, T. (2007). Application of the railway transport risk assessment methods in Estonian oil shale industry. *Oil Shale*, 24, 35 - 44.

## APPLICATION OF THE RISK ASSESSMENT METHODS OF RAILWAY TRANSPORT IN ESTONIAN OIL SHALE INDUSTRY

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*The paper deals with risk analysis/assessment problems in Estonian oil shale industry. Investigations are focused on application of these methods for railway transport from mine to consumer. Various factors relevant to oil shale transport have been determined. For risk estimation an empirical approach, the event/fault tree is used. It allows to determine probability of deviations of the process duration from the mean value for different pathways. The obtained information affords specialists to improve the quality of the railway transport. The analysis shows that the used method is applicable in conditions of Estonian railway systems. The results of the investigation are of particular interest for practical purposes.*

### Introduction

In Estonia a specific kind of oil shale kukersite is the most important mineral resource. Oil shale reserves are estimated to be approximately four billion tonnes. 85% of mined oil shale is used for generation of electric power and a large share of thermal power, and about 15% goes for shale oil production. Oil shale industry of Estonia provides a significant contribution to the country's economy, but economically viable transportation of oil shale to consumers is impossible without advanced railway network. Railway transportation of oil shale is indispensable. It is cheap and highly productive.

Transportation of oil shale from mines and open casts to consumers causes a lot of technical and economical problems. Conventional theoretical basis does not allow to solve these problems. Available data give a good basis for elaboration of the concept and methods of risk analysis/assessment. The results can be used to solve the problems of transportation.

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This study addresses the risks associated with oil shale loading and transportation, evaluation of the usability of the method and estimation of the probability of failure without detailed assessment of its consequences being the primary objects of interest. The study is based on the literature and on the Estonian experience. As an example, the risk analysis/assessment method has been applied to study transportation in Estonian oil shale mines. To simplify this task, the track between the stations *Musta*, *Raudi* (Estonia mine) and *Musta* was considered.

Risk assessment/management involves judgments about taking a risk, at which all parties must recognize the possibility of adverse consequences which might materialize [1, 2]. Prevention of hazardous situations is more moral, ethical and economical than facing the adverse consequences. Risk assessment method gives information about the transportation system. The information obtained could help the management of the mining company to come to adequate political and strategic decisions. The concept and methods of risk analysis/assessment can be used for different purposes and at different levels: at the stage of transport system design; as the basis for decision-making when choosing between different remedial actions for transportation system within temporal and financial restraints [1]. The risk analysis/assessment method is the most powerful tool for solving complicated mining problems.

Risk analysis involves the use of available information to provide the transportation system for a risk. Various factors relevant to oil shale transportation are determined in the present study. Probable risk analysis is a more rational basis for evaluation. The event/fault tree is used for risk estimation. Having obtained the risk information and knowing the risk evaluation criteria, we come to a decision.

The analysis shows that the risk analysis/assessment methods used are applicable for transportation systems. The results of the risk assessment are of particular interest to be used in practice.

## Theoretical background

In Estonia, like in other countries, risk management methods are used in different branches of industry and for many different technical systems. Irrespective of terminology, there is a general agreement on the basic requirements [1, 3, 4]. Terminology and risk management method used in the frame of this project are presented below.

Risk management is systematic application of the management policies, procedures and practices for identifying, analyzing, assessing, treating and monitoring the risk [1, 4]. Having obtained the risk information, a decision-maker must come to a decision. The primary steps of the risk management are presented in Fig. 1.

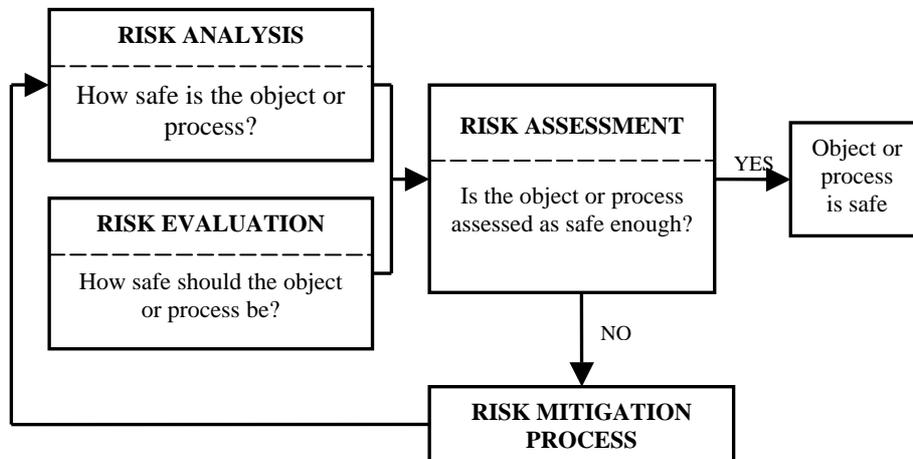


Fig. 1. Risk management process

Risk assessment is the process of deciding whether existing risks are tolerable and risk control measures adequate [1, 4]. It involves making judgments about taking the risk (whether the object or process is assessed as safe enough), and all parties must recognize that the adverse consequences might materialize, and owners will be required to deal effectively with the consequences of the failure event. Risk assessment incorporates the risk analysis and risk evaluation phases.

Risk analysis is the process of determining how safe the object or process is. Risk analysis contains the following steps: scope definition, hazard identification, and risk estimation. The description of the system, scope and expectations of risk analysis should be defined at the outset. An iterative approach should be adopted with qualitative methods being employed at the early stages of the process. If more information becomes available, the use of quantitative analysis is required.

Risk identification is the process of determining what can go wrong, why and how. Failure can be described at many different levels. Conceptualization of different possible failure modes for a technical system is an important part of risk identification. At first, as many types of failure as possible should be taken into account. The initial list can then be reduced by eliminating those types of failures which are considered implausible.

Risk estimation entails the assignment of probabilities to the events and responses identified under risk identification. The assessment of the appropriate probability estimates is one of the most difficult tasks of the entire process. Fault/event trees [1] are the tools often used in risk estimation. Probability estimation can be performed according to three general approaches depending on the type and quality of the available data:

1. Analytical approach uses logical models to calculate probabilities.
2. Empirical approach uses existing databases to establish probability.
3. Judgmental approach uses the experience of practical engineers in guiding the estimation of probabilities.

Attaining an exact value of probability at examining technical systems and processes is not a realistic expectation.

Risk evaluation is the process of examining and judging the significance of risk. It must answer the question how safe the process or object should be. It is based on the available information, including consideration of the importance of the estimated risks and the associated social, environmental and economic consequences. The principal role of risk evaluation in risk assessment is the generation of decision guidance against which the results of risk analysis can be assessed.

Risk acceptance is an informed decision to accept the likelihood and the consequences of a particular risk. In some countries, there is a certain risk level which is defined as the limit of unacceptable risk. For failure events with no potential fatalities or irreparable damage to the environment, the target failure probability may be decided exclusively basing on economic considerations and corresponding risk analysis [2].

Risk mitigation is a selective application of appropriate techniques and management principles to reduce either likelihood of an occurrence or its consequences, or both [1, 3–5]. If the calculated risk of the existing system is judged to be too high, alternatives are proposed to reduce the risk of failure. After repeated study the decision-makers can be provided with suitable alternatives and their estimated costs for consideration in improving overall technical system safety.

### **Applicability of risk analysis/assessment methods in mining**

Worldwide experience has shown that the risk analysis/assessment method is a very powerful tool to solve complicated industrial problems. Conventional theories do not enable to solve these tasks. In the world the risk analysis/assessment methods are used in different branches of industry, but the number of references on solution of mining problems is limited. Investigations have shown that the above-mentioned methods are applicable for solving complicated mining problems. All underground and surface processes in a mine are presented in Fig. 2.

One can see that the stages of the mining process are at different levels and of different importance. Each process will be subjected to risk analysis/assessment. The very important tertiary process on the surface – transportation of minerals (oil shale) from *Estonia* mine to the consumer – was chosen to carry out the risk analysis.

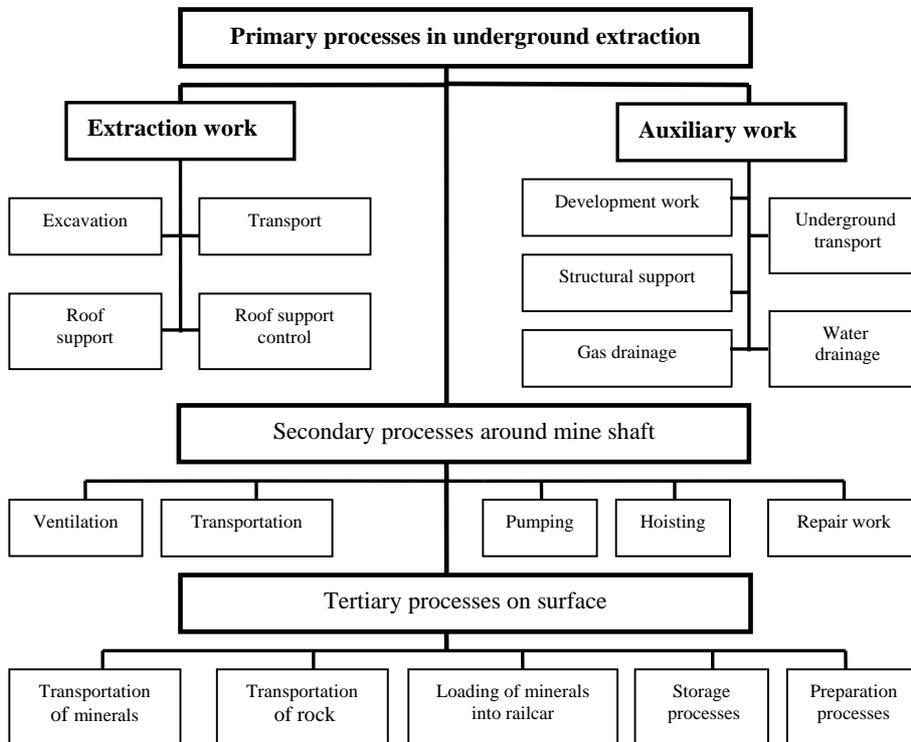


Fig. 2. Processes involved in underground mining

### Network of railways

The network of railways between the mines, open casts and consumers is complicated. To simplify the task, the track between the stations *Musta*, *Raudi* (*Estonia* mine) and *Musta* was considered (Fig. 3).

Cars are unloaded at *Musta* station. An empty train unit comes from *Musta* station to *Raudi* station (*Estonia* mine) where the oil shale loading process takes place. The loaded train unit goes back to *Musta* station. The distance between the stations is 44.7 km. There are four stations which prolong the transportation time in the track.

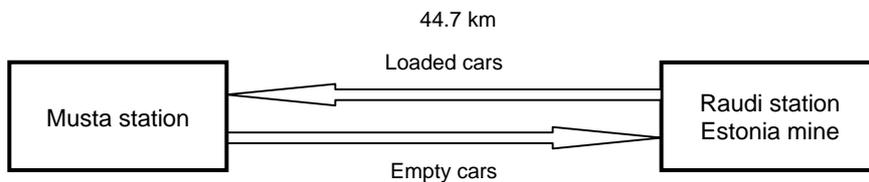


Fig. 3. Network of railways

## Factors contributing to the transport process

Railroad is a complicated system, the efficiency of railway transport depends on many factors. Some factors relevant to transport processes are presented in Fig. 4.

Main aspects influencing the efficiency of the transport work concern the duration of the processes. Empty and loaded run, loading and waiting processes are the most important factors. It is reasonable to perform the analysis of the transport processes during two weeks. Investigations have shown that duration of the processes differs on a large scale (Table 1).

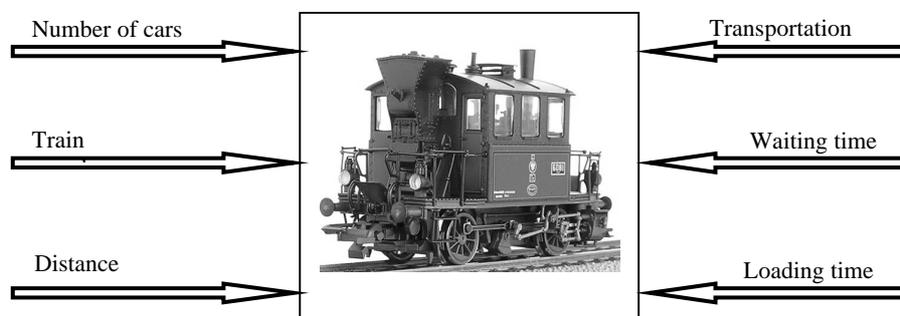


Fig. 4. Factors contributing to the transport process

Table 1. Duration of the process

Process	Duration of the process, h		
	Average	Minimum	Maximum
Empty run ( <i>Musta-Estonia</i> )	3.5	2.2	8.2
Loading	2.8	1.5	5.0
Waiting	1.3	0.2	4.2
Loaded run ( <i>Estonia-Musta</i> )	3.5	2.0	5.3
Total	11.1	5.9	22.7

## Results

The main quantitative approach used in risk analysis/assessment is the fault/event tree method. This method was selected as the most appropriate one for the analysis/assessment of the risk of the railroad transport system. In the first stage of the project time factor was taken into consideration. For probability determination the empirical approach was used. It utilizes the existing data to generate probable estimates based on historical frequencies.

Figure 5 presents the event tree for oil shale transport processes indicating the probabilities of the transport processes and spent time. It is possible to select different pathways and to determine the probability of one. Full-time probability in the event tree is settled by “OR gate” [5, 6]. It requires the independence of these factors. It means that the sum of the probabilities of these pathways gives us the total probability.

Figure 6 presents the fault tree that allows to determine time deviations from the mean value. Zero is taken as the mean value of the time. Minus before numbers indicates a decrease in the value, plus – an increase. The sum of the selected pathways determines the full-time deviation from the mean value.

Application of the event and fault trees is presented in Table 2. For instance, two different pathways are considered (variants A and B).

Selected pathways give different value of the probabilities and deviations from the mean value. One can see that the probability of selected pathways is 0.27 (variant A) and 0.30 (variant B), and deviation from the mean value is 0.13 and 0.12, respectively. The weight of each process in the full transport cycle is shown. Having this information, a specialist can come to an adequate decision and improve the quality of railway transport between the stations *Musta* and *Raudi* (*Estonia* mine).

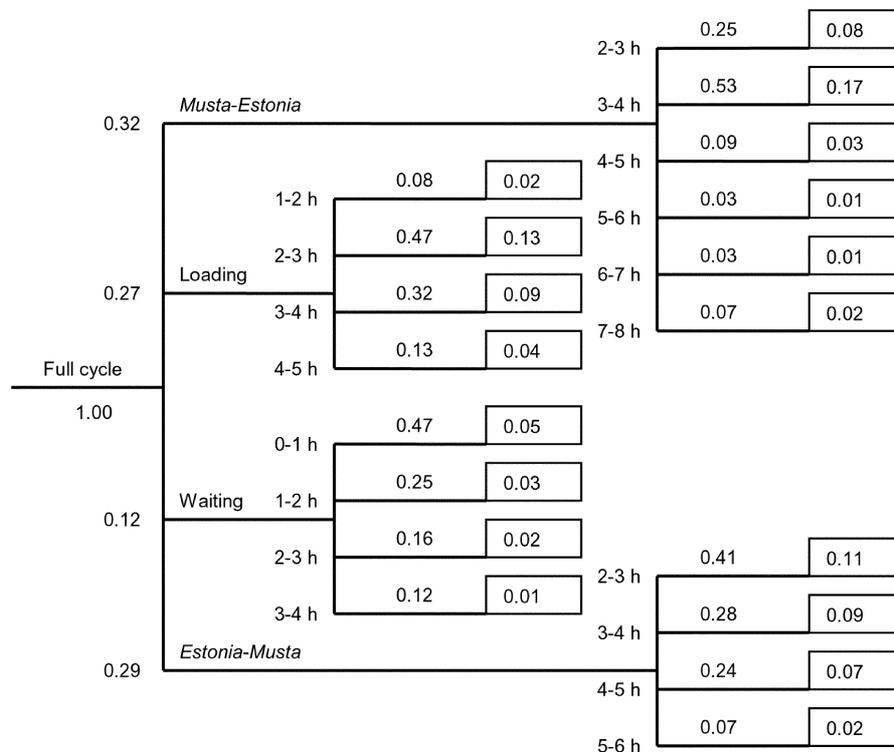


Fig. 5. Event tree

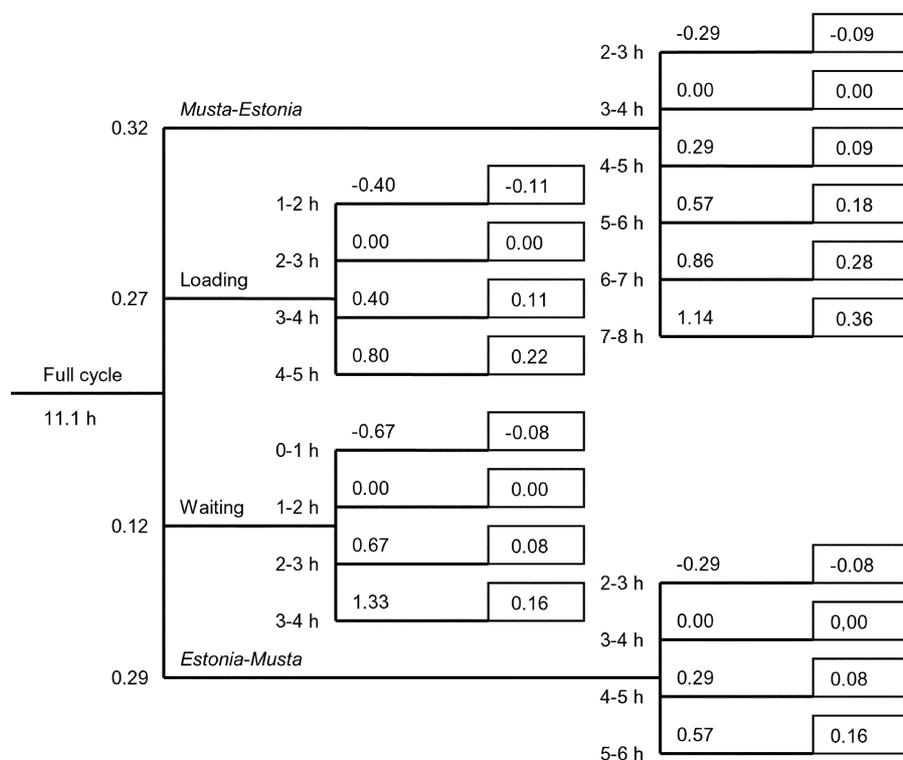


Fig. 6. Fault tree

Table 2. Example of the use of the event and fault trees (Figures 5 and 6)

Process	Selected time, h		Event tree		Fault tree	
			Pathway probabilities		Deviation from the mean value	
	Variant		Variant		Variant	
	A	B	A	B	A	B
Empty run ( <i>Musta-Estonia</i> )	3-4	6-7	0.17	0.01	0.00	0.28
Loading	1-2	2-3	0.02	0.13	-0.11	0.00
Waiting	3-4	0-1	0.01	0.05	0.16	-0.08
Loaded run ( <i>Estonia-Musta</i> )	4-5	2-3	0.07	0.11	0.08	-0.08
Total	-	-	0.27	0.30	0.13	0.12

It may be concluded that the applied methods give excellent results. They are suitable to perform the investigations for the network of railways on the tracks between the mines and open pits belonging to Estonian Oil Shale Company and consumers.

## Conclusions

As a result of this study, the following conclusions and recommendations can be made:

1. In Estonia oil shale is the most important mineral resource. Railway transportation of oil shale is indispensable. Transportation of oil shale from mines and open casts to consumer by railway causes a lot of technical, economical, ecological and juridical problems.
2. The present study addresses the risk associated with transportation time. The primary interest of this study concerns evaluation of the usability of the method and evaluation of the probability of transportation time without a detailed assessment of the consequences.
3. Various factors relevant to transport have been determined. The event tree determines the probability of the efficiency of the transport system. The fault tree gives information about the deviation of the transport time from its mean value.
4. The risk analysis/assessment method is a powerful tool to solve complicated problems in the railway transport. The analysis shows that the used methods are applicable in conditions of Estonian railway systems. The results of the investigation are of particular interest for practical purposes.
5. Basing on the excellent results of this investigation, it is recommended to use the applied methods for the whole network of railways from mines and open casts to consumers.

## Acknowledgments

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## **Risk assessment of pillars bearing capacity under rock dump in Estonian mine “Viru”**

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### Abstract

This paper deals with risk assessment of pillars bearing capacity under rock dump formation in area allocated above chamber blocks number 13, 24 and 25. The specified chamber blocks were excavated by room and pillars method 23 years ago. Calculation of allowable rock dump volume is made from actually existing pillar safety factor in these chamber blocks. By way of checking calculations were received critical pillars bearing capacity which will guarantee control collapse and uniform subsidence. By the practical way determined safety (stable) rock dump height, volume and area for avoidance unexpected deformation. These practical experiments allows assuming that a pillars property in the mining block with normal condition has no significant changes during 22 years and laboratory test confirm empirical formula of the long-term rocks strength in Estonian mining condition. It is taken into account safety parameter for control subsidence of mining block.

**Keywords:** rock strength, pillar, subsidence.

### Introduction

The commercially important oil shale is located in the north-eastern part of Estonia and excavated by underground and open-cast methods. In “Viru” mine oil shale production is obtained by room-and-pillar method with blasting. It gives an extraction factor of 74 – 81 %. This method is highly productive, easily mechanize, and relatively simple to design. In “Viru” mine mining blocks approximately 300-400 m in width and 600-800 m in length each, which usually consists of two semi-blocks. The height of the room is 2.8 m and stable when it is 6-10 m wide. However, in this case the bolting must still support the immediate roof. The pillars in a mining block are arranged in a singular grid and have a square cross-section 30-45 m<sup>2</sup>.

The total amount of oil shale extraction is about 2.5 million tons annually. The losses of oil shale in mining are about 25 % (caused by un-mined supporting pillars). Per one tonne of extracted oil shale half ton waste is generated in the process of oil shale enrichment. The total volume of waste generated on “Viru” mine landfills is 35 million tons.

Up to the present about 73 pillars spontaneous collapses accompanied significant subsidence of the ground surface on the area of 112 km<sup>2</sup> has been occurred. They cause a large number of environmental problems. On the other hand, on account of restricted area for waste generation and to reducing environmental problems were decision formate rock

dump above developed mining blocks. This solution help to avoid spontaneous collapse in mining blocks and control the ground surface subsidence in inspected area [1].

The mining block 13 was in stable condition from year 1985. When height of the rock dump was exceeding 20 m subsidence begins. Other two mining block 24 and 25 nowadays are stable with rock dump height 12 m.

### **Risk analysis of long-term rock strength**

Rock strength data in the key importance for the choice of the sizes of constructive elements used in room-and-pillar mining. Without taking into account the rheologic properties of rock, in particular the character of the change in their long-term durability, the calculation of the sizes of rooms and pillars for a certain term is impossible [2].

The character of changes in time of oil shale bed and roof limestone strata is described with sufficient accuracy by the following empirical formula of The State Research Institute of Mining Geomechanics and Mine Surveying (VNIMI), St. Petersburg has been accepted as a calculating method:

$$kt = \alpha + \beta (1/(1+t))^m \quad (1)$$

Equation (1) was developed on the basis of observation materials collected in oil-shale mines. Factor  $\alpha$  showing the rate of stabilized strength is averaged  $\alpha = 0.44$ ;  $\beta$  and  $m$  demonstrate the decrease in the rock strength intensity,  $\beta = 1 - \alpha = 0.56$  and  $m = 0.6$ . The precision of the formula could reach  $\pm 30\%$  (average  $\pm 12\%$ ) with estimated standard deviation 0.0908. The formula describes a hyperbolic dependence according to which the value of the factor  $Kt$  decreases from  $kt = 1$  (basic strength if  $t = 0$ ) to  $kt = 0.44$  (stabilized strength if  $t = \infty$ ). From the equation (1) long term rock strength determine as dependence:

$$R_t = ktR_0 \quad (2)$$

The basic concept of the VNIMI method is that two strength features characterize the rock pillar: basic strength and stabilized strength. The basic one characterizes rocks at fast loading, e.g. at pressure testing. Under constant pressure the current strength of rock decreases, and in a while it will equal the stabilized strength [3].

Results received from laboratory test in the Department of Mining (MI), Tallinn University of Technology and the experimental data bases on dependence of rock strength in time (VNIMI) presented in Figure 1.

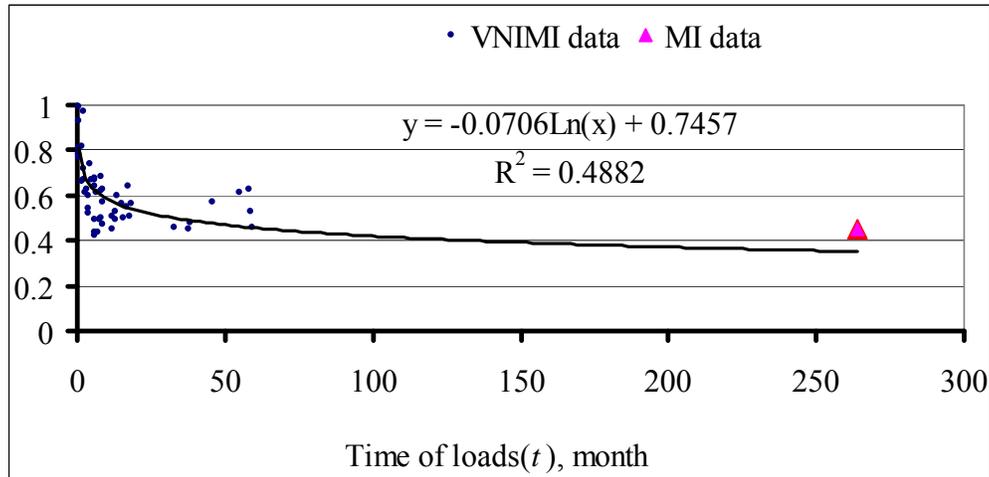


Figure 1. Dependence of rock strength in time.

Figure 1 showed changes of rock strength after 264 month, which can confirm empirical formula (1). Laboratory tests (MI) showed that current compressive strength equal  $R_t = 9.99$  MPa under the water content 12%.

### Risk evaluation of pillars bearing capacity

Basing on the instruction for Estonian oil-shale mines [3] from the formula for square pillar parameters, current strength can be obtained by following formula:

$$R_t = \frac{n\gamma Hh(x^2 + x(A+b) + Ab)}{0.3(x^3 + 2.33x^2(h-1.29q) + 3xq(1-1.56h) + 2.33q^3(h-0.43))} \quad (3)$$

Table 1. Chamber block 13 parameters

x	5÷6	m	Pillars width
b	7÷8	m	Chamber length
A	7÷8	m	Chamber wight
q	0.6	m	Blasting influence on pillars side
h	2.83	m	Height of the room
$R_t$	11.8083	MPa	Pillars current strength
n	1.3		Pillars safety factor
H	44.4+20	m	Overburden thickness + Rock dump
$\gamma$	0.0227	MN/m <sup>3</sup>	Overburden rock density

Determination of pillars lifetime produced from previous formula:

$$t = (8.96/R_t - 7.04)^{5/3} - 1 \quad (4)$$

By checking calculation results the pillars current strength make  $R_t = 11.81$  MPa. Calculated  $t$  equal tree month showed real behaviour of rock dump subsidence, which occurred during this time period.

### The technology of rock dump formation above excavated area

Rock dump development above 13 chamber block has served reason of the overburden thickness partial subsidence in the right semi-block. Than the rock dumping has stoped the situation in 13 chamber semi-block stabilized, that confirms opportunity develop rock dump by small platform to achieve uniform subsidence on calculated critical area.

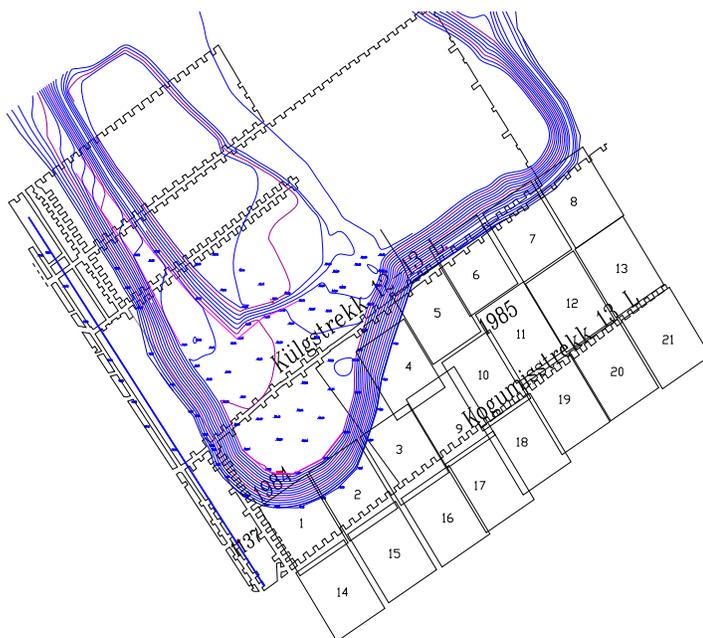


Figure 2. Rock dump above chamber block 13

The purpose of this project was technological develop of safe parameters and methods for loading rock dump loading on a basis of surface stability to provide safety subsidence. The pillar load depends on the width of the mining block, so the concept of the critical width is to be used. The critical width is the greatest width that the rock above the mine can span before its failure, or, if there are pillars, the width we must mine before the pillars accept the full weight of the overlying materials. For Estonian oil shale mines it is presented by the following formula [4, 5]:

$$L \geq 1.2H + 10 \quad (5)$$

Where  $L$  – critical width, m;  $H$  – thickness of the overburden rocks, m.

Using the formula (5) was received critical area for rock dump formation. Embank of rock dump realize by small platforms. Monthly volumes of rock waste on small platforms

total 40 000 m<sup>3</sup>. After the calculation of pillars long-term bearing capacity using empirical formulas (1-4) were received safety parameters of allowable rock dump volume. The calculated area of small platforms is 75 x 80 m with height 20 m will guaranty stability during three month and after this period the platform must be finished and enclosed. On enclosed area the period of massive movement and surface subsidence begins. At the moment of rock dump further development the subsidence must totally complete and the next floor is formed on a top of the first floor with the stable ground.

### **Estimation of safety parameters and management for mining block control subsidence**

The period of dangerous deformations lasts 1-1.5 months and occurs very actively - speed of ground surface subsidence 110 mm per day. Subsidence during the next 2 - 3 months makes 30 mm. These additional subsidences occur non-uniformly, on separate sites non-uniformity is close to critical, that can adversely affect ground constructions.

Observation of pillars at the moment of destruction beginning from collection drift in chamber block 13 was made after the two month of rock dump formation. The square pillars cross-sectional area was 36 m<sup>2</sup> with distance between them about 7 m. After loading increasing on mining block the pillars parts exfoliation are formed. The roof on the collection drift has a lot of tectonic joints supported by converted timber. In the left semi-block located under rock dump strong deformation of sights is observed. Destruction of parts up to 1.5 m is observed on all perimeters of pillars. The greatest destructions were observed in the middle of pillars side.

Table 2. Data for safety parameters estimation

$h_k$	3.0	m	Sediments
$H_k$	41.3	m	Carbonate rock thickness
$h$	2.83	m	Height of pillar
$\varphi$	50°		Sediments movement angle
$\delta$	70°		Carbonate rock movement angle
$B$	5	m	Safety berm width
$a$	37°		Rock dump movement angle

Dangerous zone borders influence of underground developments on a surface are determined concerning a line external security (barrier) of pillars row on corners of movement in carbonate rock and sediments.

In the chamber block safety zone border are determined by lines of crossing layer with the planes down under movement angels through borders of the protected area.

Around of rock dump, through its angular points, is frame the rectangular which sides arranged in a direction of mining developments borders, in parallel the sides of the received rectangular is frame safety berm (the reserve area) which external borders are borders of the protected area [3].

Minimally admissible distance from object up to border protected zones (top view) is determined under the following formula (6). For reviewed rock dump condition minimally admissible distance equal 23.6 m (Figure 3)

$$D = h_k \operatorname{ctg} \varphi + (H_k + h) \times \operatorname{ctg} \delta + B \quad (6)$$

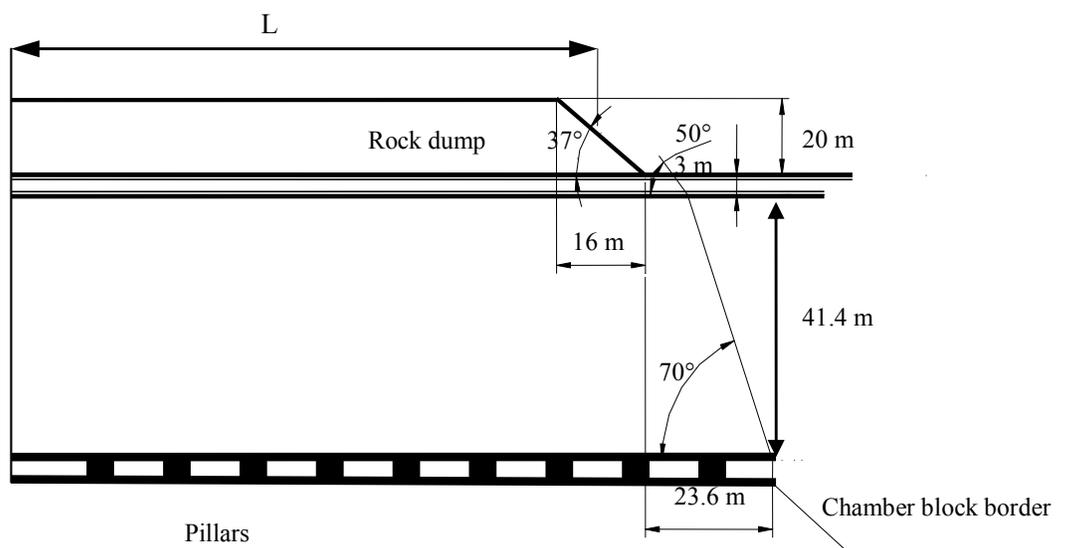


Figure 3. Scheme of minimally admissible distance from mining block up to border rock dump.

Underground development under rock dump protection from working developments by the concrete constructions prevent from distribution of an air shock wave roof caving. Constructions are represented by the strengthened concrete barricade and strong metal gates.

## Conclusion

By way of checking calculations were received critical pillars bearing capacity which will guarantee control collapse and uniform subsidence. By the practical way determined stable rock dump height, volume and area for avoidance unexpected deformation. These practical experiments allow assuming that a pillars property in the mining block with normal condition has no significant changes during 22 years and laboratory test confirm empirical formula of the long-term rock strength of Ordovician rocks in conditions of the Baltic Oil Shale Basin.

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PAPER VI. Sabanov, S.; Pastarus, J.-R.; Nikitin, O; Väli, E. (2008). Risk assessment of vibration impact on roof and pillars stability in Estonian underground mines. In: 5th International Symposium "Topical problems of education in the field of electrical and power engineering". Doctoral school of energy and geotechnology: Tallinn University of Technology, 2008, 57-61.

## Risk assessment of vibration impact on roof and pillars stability in Estonian underground

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### Abstract

*The processes of immediate roof exfoliation and pillars collapse accompanies by significant subsidence of the ground surface. Ground surface subsidence causes soil erosion and flooding, swamp formation, agricultural damage, deforestation, changes in landscape, ground water level decreasing and the formation unstable cavities. During the period of four last years the oil-shale mining at experimental mining block introduced by new blasting technology with great entry advance rates (EAR). With such improved technology the EAR reached 4 m that is two times greater than conventional technology can guarantee, but emulsion explosive volume increase up to two times and explosion occurs during 4.5 seconds (~15 times longer than old technology). As a result of such greater advance rates the situations with unsupported room length up to 5.5 m with decreasing the stability of IR can be expected. Analysis of the immediate roof (IR) stability by the deformation criteria for new room-and-pillar mining technology with modern machinery in “Estonia” mine is presented this paper. The analysis of IR stability based on an in-site underground testing by the leaving bench-mark stations and convergence measurements. The target of this study is to determine the impact of vibration on roof and pillars stability using risk assessment method. Risk analysis of available earthquake influence on mining block is presented in this paper.*

### Keywords

*Deformation, room-and-pillar mining, immediate roof, stability, risk analysis peak particle velocity.*

### Introduction

Four last years the oil-shale mining at “Estonia” mine introduced with new blasting technology with great entry advance rates (EAR). With such improved technology the EAR reached 4 m, that is

two times greater than conventional technology can guarantee. The average productivity of such technology about 3000 m<sup>3</sup> of rock mass per day. The main problems of old technology are the great volume of blasting operations, low mobility and concentration of loading works due to the small entry advance rates (EAR), about 1.5-1.7 m per blasting. One of the ways to improve the quality management system in nowadays situation is high safety drilling-and-blasting mining technology application with greater EAR and daily output.

During the last 2004 year period was tested new technology in two mining blocks 3103 and 3104 in “Estonia” mine [1]. The geological conditions were quite different. The typically excavation height is about h=2.8 m, but on the case of weak IR conditions, like in our blocks, it can be up to 3.8–3.9 m. Roof support is to be achieved by usage of the Steeledale SCS roof bail type anchor bolts [2].

In this case expander plug (anchor lock) must be fixed in harder limestone layer G/H. It improves roof control significantly, reducing bolt-to-face distances and exposure of unsupported roof. The width of the room is determined by the stability of the immediate roof. As a result of such greater EAR the situations with unsupported room width × length up to 7 × 5.5 m with decreasing the stability of immediate roof can be expected. The analysis of IR stability based on an in-site underground testing by the leaving bench-mark stations (BMS) and convergence measurements (Fig 1).

### 1. Prediction of stability using roof-to-floor convergence data

Laminated roof deformation on the basis of plate’s hypothesis by the experimental data of Institute of Mining Surveying (VNIMI) in St. Petersburg and Estonian filial of A. A. Skotchinsky Institute of Mining Engineering (IGD, Moscow, Russia) presented on figure 2. [3, 4].

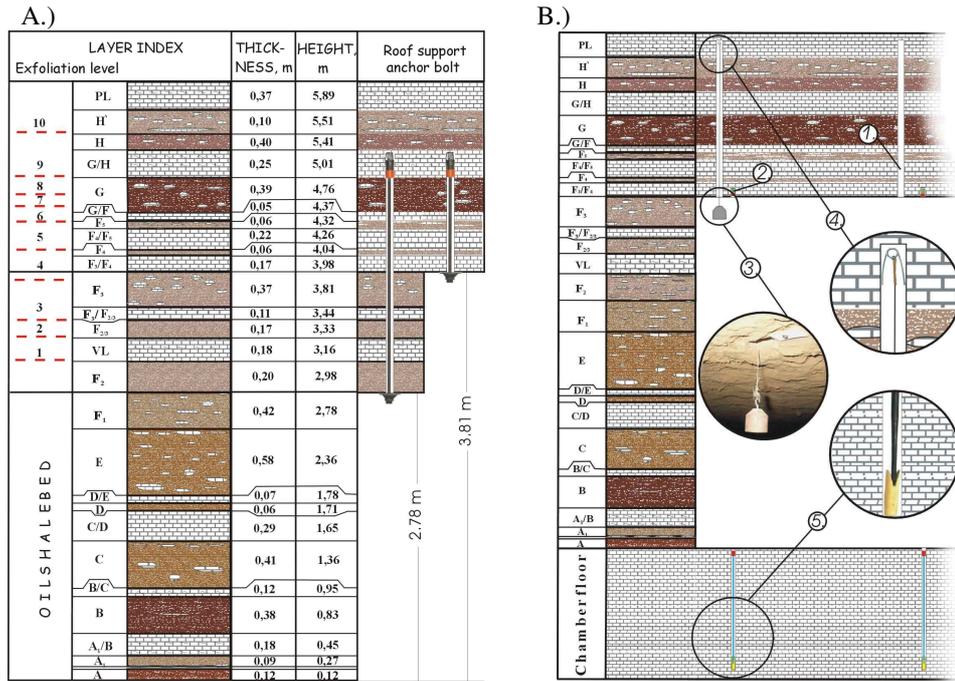


Fig. 1. Structural cross-section with determined IR exfoliation levels (A.) and scheme of bench-mark stations in the roof/floor (B.) Where, 1.- bore hole for stratascop; 2.-bench-mark station on the roof; 3.-bob for rope-bench-mark station; 4. - rope-bench-mark station; 5. - bench-mark station in the floor.

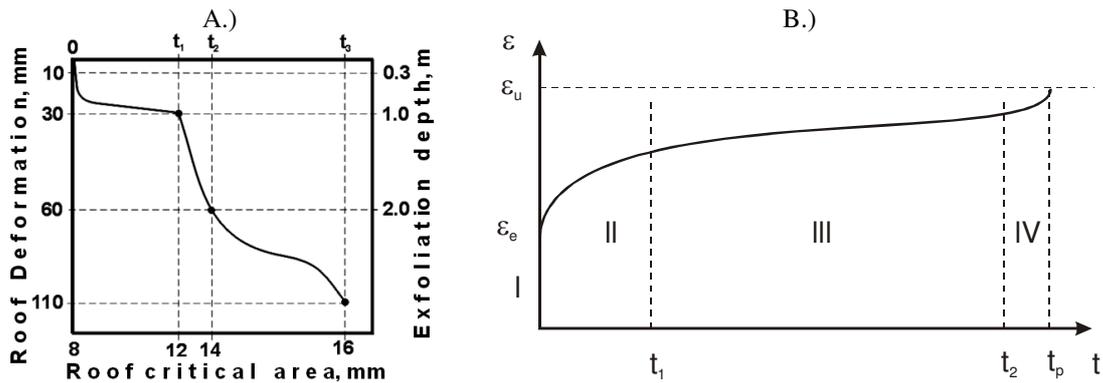


Fig. 2. Roof-to-floor convergence curve by the VNIMI and IGD data (A.) and typical curve for long-term stability analysis (B.) [3] Where,  $t$  – time;  $t_p=t_3$  – time at failure;  $\varepsilon$  – deformation;  $\varepsilon_u$  – ultimate deformation at failure;  $\varepsilon_e$  – elastic deformation; I – elastic deformation  $\varepsilon_e$ ; II – transient creep  $\dot{\varepsilon} < 0$  ( $\dot{\varepsilon}$  – deformation rate); III – steady-state creep  $\dot{\varepsilon} = \text{const}$ ; IV – transient creep  $\dot{\varepsilon} > 0$

In general case for Estonian oil-shale deposit it is possible to allocate four stages in this process. During short time interval after the first blasting there are instant deformations (ID) up to 10 mm. Then during the time (duration depends on geological conditions) there are two processes: increase of elastic deformations (ED) due to rheological processes, blasting work and entry advance, and also increase of creep deformations (CD) up to the cracks formation moment at  $t = t_1$ , when  $\varepsilon=20-30$  mm. Then instead of a plate the arch on three hinges is formed completely. The time period from  $t_1- t_2$  is a transient creep (TC) period due to a partial crushing of average and left/right hinges of an arch, till the moment of the crushing termination, when  $\varepsilon=60$  mm. During the period  $t_2-t_3$

there is a steady-state creep (SSC) in hinges up to their full crush at the  $t_3$ , when  $\varepsilon=110$  mm and full loss of the roof bearing capacity (full destruction up to depth 2-3,5 m) is happen. Duration of these time periods  $t_0-t_3$  depends from many geological (loading, capacity, cracks, etc.) and technological (roof critical area, type of explosives initiation, advance rate, supporting and etc.) factors that present difficulties for dependence  $\varepsilon=f(t)$  finding.

During in-site testing 16 pair of BMS-s was installed and 19 holes were viewed by the stratascop in two mining blocks (3103 and 3104) with different geological conditions (with weak and average stable IR) [5]. The results of IR (on the center of the room) and pillars ( $S=45-50m^2$ ) average deformation

presented on figure 3. The critical areas (L) of the

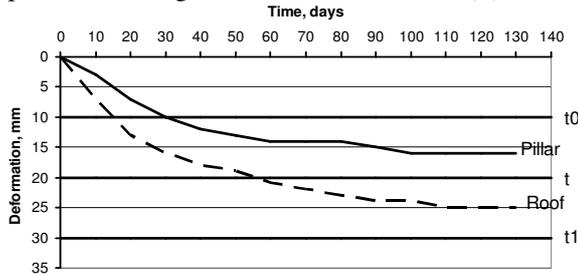


Fig. 3. Roof-to-floor convergence curves in mining blocks 3104 and 3105

By the VNIMI and IGD data the roof failure is happen (depth of failure  $\approx 2.0-3.5$  m) when deformation is  $f_{max}$  equal  $6.3L=8.84A+5.3$ , mm, where A is room width. For our conditions,  $f_{max}$  equal  $8.84 \cdot 7.0 + 5.3 = 67$  mm. From the comparison table on figure 5 you can see that received experimental data are much closed to the data of VNIMI and IGD. Its mean that the improved technology influences on immediate roof stability estimated by the deformation criterion is not greater than with old technology. Analysis of immediate roof failure cases during the experiment shown that depth of failure about 8-10cm when  $\epsilon=0.4f_{max}$  is possible. Then after IR unsupporting the failure on this depth can be expected with great probability.

By the way of exfoliation level (EL) or depth (hv) and deflection rate (DR) determination we can estimate the effectiveness of anchor bolting and supporting pattern. Deflection rate of the system "anchor-roof" by the anchor torque (M, N\*m) measurements was in average 1.3 mm/t, where loads on used anchors (N, t) was determined by the empirical formula  $N=0.2722M$ . On this case DR is a parameter of IR deformation after the vertical load on anchor increasing by one ton. [6, 7].

Measured immediate roof deformation by installed in rooms BMS evaluated by Severity scale criteria. Evaluation of total amount inspected rooms made by Boundaries scale.

#### Severity

5 = Severe - catastrophic (very harmful or potentially fatal; great effort to correct and recover) (110 -, mm) –  $t_{failure}$

4 = Serious-harmful, but not potentially fatal, difficult to correct but recoverable (61-110, mm) –  $t_3$

3 = Moderate – somewhat harmful, correctable (31-60, mm) –  $t_2$

2 = Mild - little potential for harm, easily correctable (11-30, mm) –  $t_1$

rooms for our conditions were about 11-12 m.

1 = Harmless – no potential for harm, correctable (0-10, mm) –  $t_0$

#### Boundaries

5 = Local – impact migrates on ground surface

4 = Not confined - impact migrates outside critical area. (25-30 rooms)

3 = Weakly confined – impact migrates off-site one row of the rooms

2 = Confined – impact migrates off-site four rooms, but is contained in small area

1 = Isolated – impact is contained (one room)

As results receive controllability criteria equal Severity scale multiply on Boundaries scale. (1-10) controllable - process under control; (11-15) influenceable - process controlled by changes of technology; (16-20) process is not controlled. For our experimental mining blocks process was under control.

## 2. Risk analysis of earthquake influence on rock massive

During the short period 21.01.2005-04.02.2005 in Baltic region, three earthquakes were registered. Basic precondition to consideration of this paper has served jumping characteristic of absolute deformation near pillar after earthquake.

21.09.2004 in the second part of afternoon in Tallinn area registered earthquake shocks. It has also registered in Poland, Belorussia, Russia, Austria, Latvia, and Lithuania with earthquake magnitude 4.4 [8]. The Kaliningrad earthquake parameters are: date= 21-Sep-2004; 11:05:03.3; lat= 54.78 lon= 20.29; depth= 15km; ms: 4.1/2; mb: 5.7/3. Geophysicist of Estonian Center of Geology Olga Heinlo said DELFI, that earthquake magnitude in Estonia could be about 3. It was registered two epicentres of earthquake shocks in Kaliningrad area with magnitude 5.2. Director of Latvian State Service of Geology Maris Seglinsh have made statement that significant earthquake magnitude observed in north-western part of Estonia at 16.45.

Knowing velocity of massive fluctuations (acceleration) at which there are the pressure causing infringements or collapse in mining developments, it is possible to judge comparative stability at unitary influence on them of seismic loadings, and seismo-explosive shock waves outside of operative range. On such data it is possible to estimate admissible and critical peak particle velocity at which mining development stability is lost.

Table1. Data of earthquakes during 21.01.2005-04.02.2005

Magnitude:	mb 3.8	mb 4.3
Region:	BALTIC STATES-BELARUS-NW RU	BALTIC STATES-BELARUS-NW RU
Date Time:	29.01.05 at 13:17:48.0 UTC	27.01.05 at 14:07:26.7 UTC
Location:	58.96 N ; 22.70 E	57.23 N ; 25.15 E
Depth:	25 km	25 km
References:	128 km W Tallinn ; 5 km SW Kärkla	73 km NE Riga ; 12 km SW Cesis

By the researches results of Ural University admissible peak particle velocity at supporting by the timbering, strengthened by anchors makes 0.9 m/s and critical 1.2 m/sec [9]. On Estonian standards, the same requirements shown as well for railway tunnels and subway overpass [10].

Critical peak particle velocity on USSR standards for underground constructions with service life up to  $t = 4-10$  years make no more than 0.12 m/c, and for  $t \leq 3$  years no more than 0.48 m/sec [11]. In Estonia, the maximal resolved peak particle velocity for open-casts boards makes 0.48 m/sec.

Knowing the basic rock physic-mechanical properties, such for example as velocity of longitudinal wave's distribution  $V_p$ , ultimate

$$V_d = V_p \times \sigma_r / E = 1053 \times 2500000 / 7100000000 = 0.37$$

$$V_d = V_p \times \sigma_r / E = 1700 \times 3500000 / 7100000000 = 0.84$$

Hence, critical velocity of massive displacement for industrial layer in conditions of Estonian oil-shale deposit will make 0.4 – 0.8 m/sec.

### 3. Richter Magnitude and TNT Equivalent

The Richter magnitudes based on a logarithmic scale (base 10). It's means that for each next number you go up on the Richter scale, the amplitude of the ground motion recorded by a seismograph goes up ten times. By the data of Michigan Technological University, magnitude 8 earthquake releases as much energy as detonating 6 million tons of TNT [15]. This statement is based on the empirical formula:

$$\log(E) = 1.5M \quad (2)$$

Where, M- magnitude and E-energy [16].

The calculation offered by the American Institute of Makers of Explosives (IME), USA, based on the following formula to recalculation of TNT equivalent [17]:

$$TNT = \frac{MQ}{4,186 \times 1090} \quad (3)$$

The blasting energy of Nobelit 2000 Q Nobelit 2000 = 2600 kJ/kg, and QTNT = 1090 kcal/kg or 4.186\*1090 kJ/kg. Then to one kg of TNT corresponds about 1.6 kg of Nobelit 2000.

### 4. Determination of the Peak Particle Velocity

It is obvious, that peak particle velocity PPV is in direct dependence on such parameters, as distance up to explosion, quantities blasted explosives on delay unit, the basic physical and mechanical properties of the rock. Formula PPV, which apply practically all over the world, in a general view looks as follows:

extension strength  $\sigma_r$ , Young module E, it is possible to calculate critical peak particle velocity  $V_p$  under the formula [12]:

$$V_d = V_p \times \sigma_r / E \quad (1)$$

According data from Institute of Oil-shale during the experiment at „Tammiku” mine (mining block №2) the velocity of longitudinal seismic waves was 1700 m/sec [13]. According to measured velocity of longitudinal seismic waves by experts of Japanese firm KOMATSU in 2002 on "Narva" open-pit the separate industrial layers velocity was from 1039 to 2000 m/sec [14]. According to the report of Institute of Oil-shale, the Young module for layer C (one of the weak) is  $E \approx 7100$  MPa and  $\sigma_r \approx 2.5-3.5$  MPa.

$$\left. \begin{array}{l} \\ \\ \end{array} \right\} 0.4-0.8, \text{ m/sec}$$

$$PPV = A \left( \frac{D}{\sqrt{W}} \right)^{-n}, \text{ mm/sec} \quad (4)$$

Where, A- degree of damping of PPV; n - exponent depending on explosive properties; W - explosive quantity, and D - distance.

According to work of MSc. Tomberg for blasting in Estonian underground conditions (ammonite 6ZV) factors have following values  $A=1748$ ;  $n=1.25$  [10].

### 5. Risk estimation of underground construction stability

Every 100 years in Estonian territory occur about 12 earthquakes with magnitude  $2.38 \leq 2.7 \leq 3.02$  ( $p=0.95$ ) and 1-2 with magnitude 3.5–3.9 [18]. Last earthquakes in Estonia territory have been recorded in area of islands Hiiumaa and Osmussaare, and distance from them up to Estonia mine about 250 km. We shall determine earthquake magnitude in area of these islands, capable to influence stability of underground constructions.

$$PPV = 1748 \left( \frac{250 \times 10^3}{\sqrt{9 \times 10^8}} \right)^{-1.25} = V_d = 0.12 \text{ m/sec} \quad (5)$$

Using the formula PPV received  $W = 9 \times 10^8$  kg that corresponds to magnitude  $\sim 7.5$  [19]. It is necessary to note that fact, that the given formula is rather conservative at distance more than 30 m. The formula application for greater distances can lead to probable deviation more than 5%. For the more exact estimation, it is necessary to consider such basic earthquake characteristics as depth of epicentre, amplitude, frequency, structure of overburden and mechanical parameters of the rocks. By the calculation result, we can conclude that probability of earthquake influence on underground construction in Estonian oil shale mines is insignificant. By the made calculation of PPV,

earthquake influence on underground construction during the experiment can be excluded. But in case if earthquake magnitude will make 4 and epicentre occur directly on underground construction (less 10 m) it will produce dangerous influence on mining block stability. On earthquake magnitude 6 safety distances for mining block must exceed 27 km, magnitude 7 – 150 km and magnitude 8 – 850 km.

## 6. Conclusions

1. Immediate roof stability estimated by the deformation criterion is not greater than with old technology.
2. Analysis of immediate roof failure cases during the experiment shown that depth of failure about 8-10cm when  $\varepsilon=0.4f$  max is possible.
3. By the made calculation of PPV, earthquake influence on underground construction during the experiment can be excluded. But in case if earthquake magnitude will make 4 and epicentre occur directly on underground construction (less 10 m) it will produce dangerous influence on mining block stability.

## Acknowledgement

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PAPER VII. Sabanov, S.; Pastarus, J.-R.; Nikitin, O. (2008). Risk assessment of feasibility of roadheaders in Estonian underground mining. Oil Shale (accepted)

## **RISK ASSESSMENT OF FEASIBILITY OF ROADHEADERS IN ESTONIAN UNDERGROUND MINING**

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*This paper deals with the risk assessment method of roadheader feasibility in underground conditions of Estonia mine. In modernization of Estonian underground mining, roadheaders that extract oil shale selectively play the most important role. Selective extraction allows reduction of rock mass volumes during the loading, transportation and enrichment processes. Thus, about 23% of limestone extracted together with oil shale will be left in the mine for backfilling the excavated areas. Backfilling increases carrying capacity of pillars reducing losses of oil shale and restores, in a certain measure, filtrational, hydrodynamical, and aerodynamic properties of the geological environment.*

*For selective extraction four variants of different excavation thicknesses, depending on geological conditions, have been proposed. Risk analysis allows comparison of the advantages and disadvantages of full and selective extraction. Risks of oil shale losses during selective extraction are estimated using the event tree. Preliminary calculations have shown sustainability of roadheaders for selective extraction under the mining and geological conditions of Estonia mine.*

### **Introduction**

In Estonian underground mining, modernization process concerns, first of all, machines that extract oil shale: continuous miners and roadheaders.

The main goal of using roadheaders and continuous miners is improvement of the extraction process accompanied by replacement of blasting works by selective cutting of rock, introduction of flexible and mobile mining-transportation systems with exploitation of highly mechanized and highly productive machines which can be used in several parts of the mine; optimisation of room and pillar dimensions.

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Nowadays all parameters of a mining block are calculated for a long period, and this reduces the extraction factor to about 10%. Consequently, in the case of the method proposed by us [1], the main requirement is long-term stability of the main roof and greatest extraction factor enabled by the new flexible technology.

In this study the risk assessment method was used for determination of feasibility of roadheaders and continuous miners in conditions of Estonian mines. It was shown that roadheaders are most applicable at selective extraction and continuous miners are preferable for full extraction. Selective extraction has been studied for four types of layer thickness with two possible variants. Selective extraction allows to reduce rock mass volumes during loading, transportation and enrichment processes. Risk analysis enables to compare advantages and disadvantage of full and selective extraction [2]. Besides, backfilling offers the possibility of reducing pillar dimensions and minimises losses of oil shale reserves. Risk estimation of oil shale losses in the case of selective extraction has been made with application of event tree.

### **Overview of performance of roadheaders and continuous miners**

Extremely powerful rock-cutting machines designed for continuous excavation of roadways, tunnels and chambers use no explosives. Powered electrohydraulically and emitting no fumes, these machines are used extensively in mining of coal and other mineral resources and in underground construction projects, where their ability to excavate the desired profile without causing harmful vibrations is highly valued for both environmental and safety reasons [3].

The roadheader family also includes separate, multi-purpose hydraulic cutting heads for mounting on excavators (Fig. 1A). Roadheaders for mining are equipped with powerful, geometrically optimized, transverse cutter heads proven to give the best cutting performance in a wide range of rock formations. Mounted on an extremely robust hydraulic boom (telescopic on some models), they offer a rugged, reliable, highly productive solution for development and direct production duties [3].

Robotic continuous miners are now being developed for more automatic operations (Fig. 1B). These offer a vision of the standard mining method of the future: "intelligent" mining machines are completely controlled by computers, with sensors that pinpoint the positions of all moveable parts, and onboard control systems that run the equipment and collect data on the seam. A robotic miner would have its own navigation and guidance systems, as well as internal diagnostics to spot problems and video equipment to allow continuous monitoring of the mining operation by highly trained personnel located in a safe position either underground or on the surface [4].

(a)



(b)



Fig. 1. a) Roadheader; B) Continuous miner.

In today's world, hundreds of kilometers of tunnels are being excavated for mining and construction purposes. Parallel to the rapid increase in urbanization, the need for tunnels in transportation and infrastructure has also increased. On the other hand, there is a tendency towards underground production methods in mining due to the environmental restrictions and the decrease of mining resources close to the surface of the earth. For economic reasons, early commencement of production is required in underground mining operations. Therefore, mechanized excavation systems have become more advantageous than the conventional methods in such mining projects. Roadheaders occupy an exceptional place among the other mechanized excavation machines. Besides driving tunnels, they have received widespread applications for production purposes at excavating of coal, evaporates, industrial minerals and metallic ores. The performance of roadheaders has been investigated for formations of various types. Their initial investment costs are lower than those for the full-face excavation machinery. They are also flexibly equipped to excavate galleries in various shapes. However, they are not suitable for hard cutting conditions being more preferable for excavating stable rocks of

low to medium hardness. Roadheaders are generally classified with respect to their weight as being of light, medium, heavy, and extra heavy types. Heavy types can be used in rocks of higher strength, as the weight is proportional to cutting head power and boom forces. Machine weight of more powerful machines is greater, due to increased boomreaction forces. Otherwise, machine stability is negatively affected and instability may occur. On the other hand, the increase in weight causes a rise in the initial cost of the machine and also problems of sinking of the machine in wet ground. To increase machine stability, side and rear stabilizer pistons are generally used. Side stabilizers may not be useful in tunnels of wide profile. The stability of a roadheader during operation is vitally important for an effective and continuous cutting process [5].

Some researchers have addressed the importance of stability and compared longitudinal- and transverse-head-type roadheaders. As for the transverse cutting-head type, the main component of the resultant cutting force acts vertically on the head. Hence, the transverse cutting-head type is more sensitive to the stability in the vertical direction. On the contrary, the longitudinal-head-type machine is more sensitive to the stability in the horizontal direction. The longitudinal-head-type machines are said to be unable to utilize the full weight of the machine, accordingly they are claimed to require 20–25% more weight than transverse-type machines. It is also reported that, being of the same cutting power, transverse-type roadheaders can cut rock of higher strength than the longitudinal-head-type machines owing to stability considerations [6]. However, it is noted that vertical stability should also be considered at comparison of the stability of roadheaders, since the longitudinal-head-type roadheader can cut vertically as well [5].

### **Risk analysis of roadheader applicability under conditions of the Estonia mine**

Risk analysis for roadheaders is carried out for two methods of extraction. The first method is full extraction and the second one – selective extraction, which allows to reduce rock mass volumes at loading, transportation and enrichment processes. Using selective extraction limestone in the quantity of about 23 vol.% mined together with oil shale can be left in mine for backfilling the excavated areas. It reduces the volume of stored rock waste on the ground surface and decreases harmful influence on the environment. Figure 2 demonstrates that limestone content of layers A, A/A1 and A1 makes 52% of the total amount of rock. On the other hand, extraction layers B, B/C and C containing only 13% limestone are preferred to be mined. Since the diameter of cutting head does not enable to cut separately the 6-cm layer D, C/D, D and D/E must be cut together and left for backfilling.

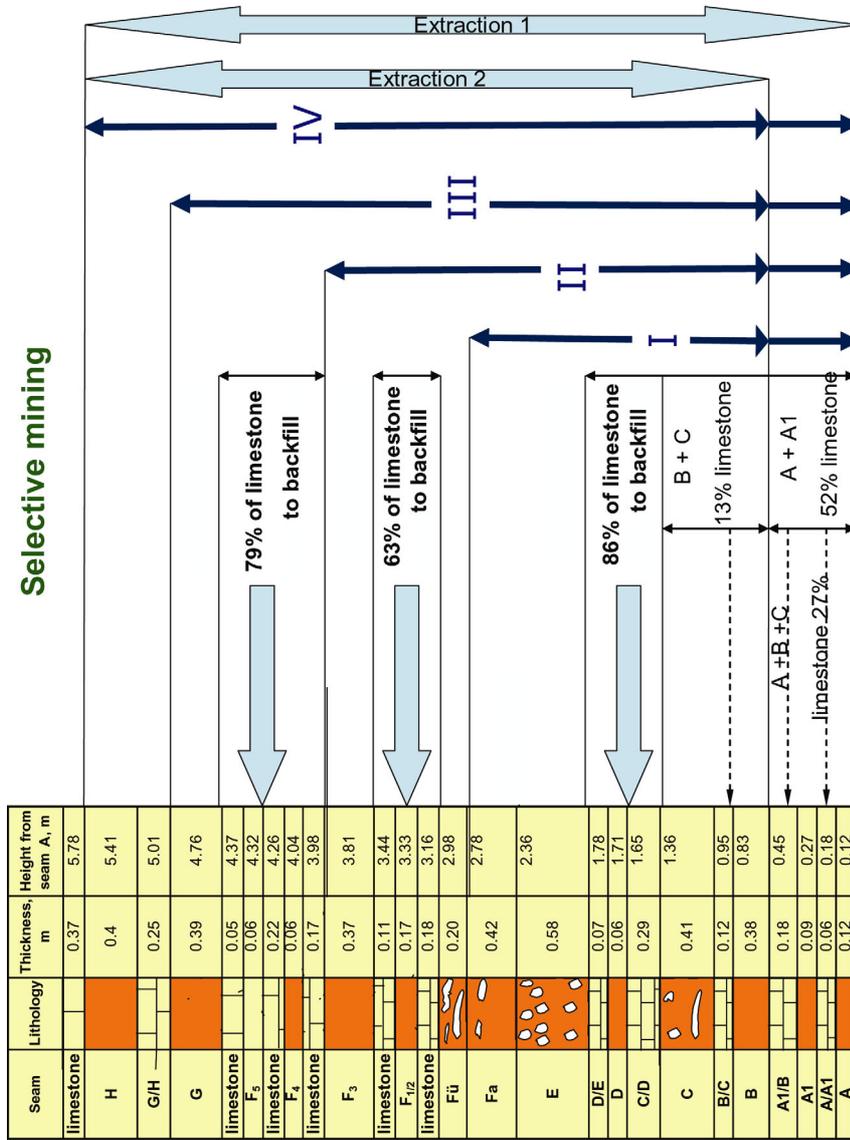


Fig. 2. Cross-section of oil shale commercial bed and extraction possibilities at selective mining.

Four variants of various thickness of development are offered for selective extraction (Fig. 2). The first variant (I) applies to commercial thickness 2.78 m. In this case layers C/D, D, D/E of the thickness 42 cm, that makes 14% of the total amount of rock, are excavated and transported to the mined-out area for backfill. It means that pillars can be designed to have in places of backfilling smaller cross-sectional area that, in turn, reduces oil shale losses with pillars. At excavation height of 4.76 m (III) and 5.41 m (IV), the machine may not perform successfully due to problems arising from machine stability, which adversely affects productivity by 20–30%. However, at the height of 4.76 m selective extracting of limestone layers C/D, D, D/E, F1/F3 and F3/G is carried out in the total thickness of 144 cm, that makes 30% of the total rock amount offering a favorable opportunity to reduce losses with pillars. Limestone content of shale for enrichment makes 8%.

When extracting the layers of thickness 4.76 m anchors must be 60% shorter. One of the adverse factors is choice of such a conical corner of cutting head which will cut in the top part without curvatures, in order to prevent dangerous cusps in the roof. For that reason cusps will appear on the floor, and their elimination will demand additional time. Table 1 presents the amounts of limestone at extraction of different thicknesses. For limestone transportation to waste, additional expenses are needed.

*Table 1. Percentage of limestone going to waste*

Variant of extraction thickness	Selective extraction 1, %. Start from layer A	Height of extraction from layer A, m	Selective extraction 2, %. Start from layer B	Height of extraction from layer B, m	Full extraction, %
I	12.1	2.78	4.0	2.33	24.2
II	9.4	3.81	3.6	3.36	26.5
III	7.6	4.76	2.8	4.31	30.5
IV	11.3	5.41	7.5	4.96	31.4

### **Risk estimation of four variants of extraction thickness**

Risk estimation entails the assignment of probabilities to the events – responses – identified under risk identification. The results of investigation are presented in Figures 3 and 4. The event tree demonstrates probability of risk magnitude (P) for selective extraction thicknesses 1 and 2 (Fig. 2). Figures 3 and 4 show four variants of extraction thicknesses (I, II, III, and IV). Risk magnitude is the result of multiplication of two components. One of them is probable volume of limestone, which demands additional expenses for transportation to waste generation. The second one is losses of thin oil shale layers, which will be left in mine to backfilling. Small number of risk magnitude enables better choice of variants. Selective extraction 2 of thickness I demonstrates smaller risk magnitude on account of less probability of additional transportation of waste to enrichment. Maximal risk

magnitude of selective extraction 1 reaches  $P = 0.0074$  in thickness IV on account of higher waste volume. Summarising of event tree data shows clearly that selective extraction 2 yields better results than selective extraction 1, but for actual mining and geological conditions ability of the floor (layer A1/B) to carry the load of roadheader must be calculated. Higher collapsibility of limestone layer A1/B results in its rapid destruction and causes problems of maneuvering and stability.

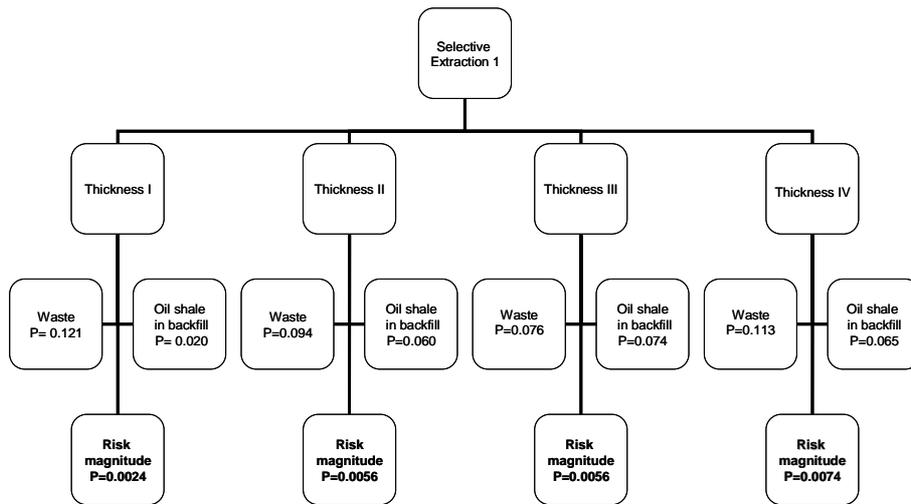


Fig. 3. Event tree for selective extraction 1.

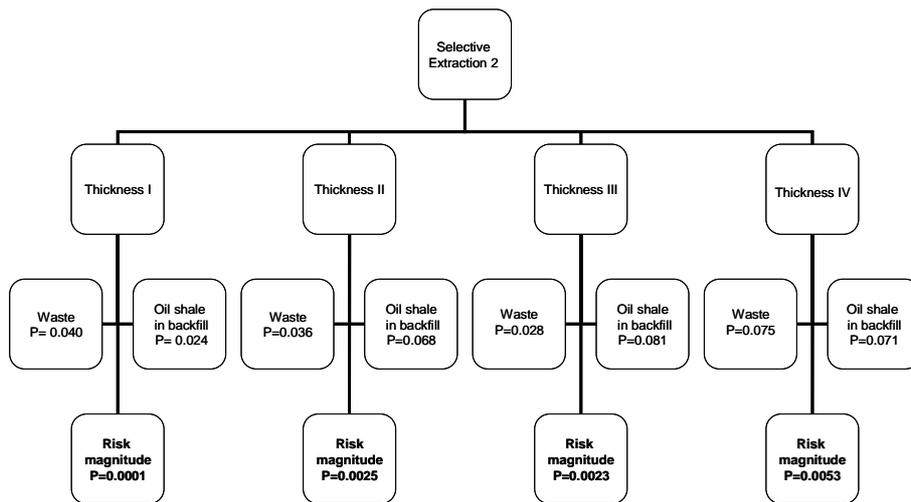


Fig. 4. Event tree for selective extraction 2.

### **Risk evaluation of selective extraction**

Cutting process must be started from the suitable soft middle layer E-Fa and continued to the next layers depending on chosen extraction type. Maximal cutting thickness of cutting head diameter ought not exceed 1200 mm (to avoid extraction of limestone from other layers) [2].

Extracted rock mass from layers C/D, D, D/E, F<sub>1</sub>/F<sub>3</sub> and F<sub>3</sub>/G will be left to backfill by powerful loading machines in an area specially prepared for this purpose where pillars are weak. The average amount of oil shale in backfill is about 22%.

Immediate loading and transportation of cut mined rock will be carried out by powerful LHD machines with diesel drive. In the case of rock mass transportation delay, roadheader can work without losses in productivity [7].

Productivity may be reduced to 20–30% if the thickness of separated layer is small (420 mm) (Fig. 2 C/D, E, D/E) by comparison with cutting head diameter 1200 mm [7].

Productivity of a roadheader depends on its type and supplied value. To reach greatest productivity, it is suggested to use twin boom in Estonian mining and geological conditions. Deviations in productivity may occur in places where geological conditions are complicated, and a clear result could be achieved only experimentally. The machinery of roadheaders showed largest applicability for selective extraction method. For full extraction a method using continuous miners is suggested [7].

### **Applicability of roadheaders in Estonian geological conditions**

The more widespread use of mechanical excavation systems is a trend set by increasing pressure on the mining and civil construction industries to move away from the conventional drill and blast methods to improve productivity and reduce costs. Roadheaders are the most widely used underground partial-face excavation machines for soft to medium-strength rocks, particularly for sedimentary rocks. They are used for both development and production in soft rock mining industry. In addition to their high mobility and versatility, investment costs of roadheaders are generally lower than those for most other mechanical excavators. Because of higher cutting power density due to a smaller cutting drum, they offer the capability to excavate rocks harder and more abrasive than their counterparts, such as the continuous mineral and the borers [8].

For determining cuttability of oil shale and its interlayers and concretions (Fig. 5), a SDM-1 dynamometric drill was used combined with ASR instrumentation developed by A. A. Skotchinsky Institute of Mining Engineering (Moscow) and Donetsk Coal Mining Institute for testing composite bed rocks [9].

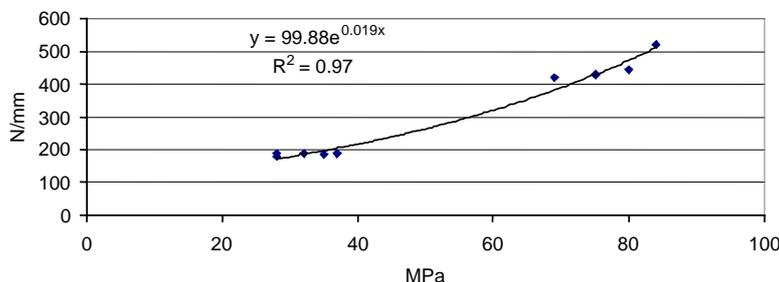


Fig. 5. Cuttability of different layers of commercial oil-shale bed.

Figure 5 demonstrates cuttability of oil shale and limestone layers of the Estonian oil-shale deposit. Variation coefficient of oil shale cuttability for separate layers is 14.1% and that of limestone 11.4%, respectively [9].

## Conclusions

Overview of continuous miners and roadheaders demonstrates the best applicability of roadheaders for selective extraction, while continuous miners suit for full extraction under the mining and geological conditions of Estonian underground mines. Cuttability of commercial oil-shale bed was demonstrated at using of both types of machines.

Basing on event tree data, selective extraction 2 demonstrates better results than selective extraction 1. For actual mining and geological conditions the ability of floor (layer A1/B) to carry the load of machines must be calculated. The more collapsible limestone layer A1/B may be destructed more rapidly causing problems with maneuverability and stability.

Selective extraction allows to reduce rock mass volumes during the loading, transportation and enrichment processes. Thus, about 25% of limestone accompanying oil shale at the extraction processes will be left in the mine for backfill in the excavated areas. Backfilling allows to increase capability of pillars thereby reducing losses of oil shale and restoring in a certain measure filtrational, hydrodynamical, and aerodynamic properties of the geological environment. Such a way of development will considerably reduce the amounts of rock waste stored on the ground surface and decrease harmful impact on the environment.

## Acknowledgements

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## **TECHNOLOGICAL AND ENVIRONMENTAL ASPECTS OF ASSESSMENT OF A COMBINATION OF DIFFERENT MINING METHODS USED IN ESTONIAN OIL SHALE INDUSTRY**

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*Oil shales used in Estonian power plants to generate electricity and in oil production are of different quality. Different excavation methods in use and accompanying development processes are accompanied by various emissions that can pollute water and air. However, generation of waste as well as impact on land use are of greater concern than emissions into the water and atmosphere.*

*Life Cycle Assessment (LCA) has proved to be one of the most attractive approaches to characterize sustainability of mining industry, as several environmental and economic indicators are used to assess its performance. The methodology enables to choose the best available environmentally friendly technology.*

*As shown by investigations, the mining processes exerts smaller effect on acidification, terrestrial eutrophication and ecotoxicity than production of auxiliary materials and transportation of oil shale to customers. Other impacts considered and discussed are ground surface subsidence, land use for deposited wastes and mine water pollution.*

*Assessment of the impact caused by a combination of different mining processes gives the opportunity to find a better way for planning new mines in accordance with environment protection measures in the area of the Estonia oil shale deposit.*

### **Introduction**

Environmental impacts associated with oil shale preparation and production are variable, as environmental impacts of mining methods used to extract oil shale using opencast or underground techniques are different [1]. The objective of this research was to consider all activities that usually take place around a mining site.

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In this study, the Life Cycle Assessment (LCA) tool is used to analyze and assess the environmental impact of oil shale mining. The inputs and outputs of all the technological chains of mines and opencasts under investigation are determined. It means that also transportation and production of essential auxiliary materials (e.g. explosive, steel) are included in the life cycle system [2].

The company *Eesti Põlevkivi Ltd.* – company of oil shale production and distribution owns two operational mines (Estonia and Viru) and two operational opencasts (Narva and Aidu). The annual extraction of oil shale is 10–14 million tonnes, with 47% extracted by opencast methods and 53% by underground methods. The losses in underground mining are about 20–30% (room-and-pillar method) and 5–10% by opencast mining. Opencast mining is carried out at depths of 5–20 m and underground mining – 20–70 m.

Nowadays new highly effective and environmentally friendly technologies are partially integrated into the mining processes. In the Estonia mine new technology is based on a blasting method applying emulsion explosives instead of packaged ones, change from 2.0 m to 4.0 m boreholes, and on a new large-hole undercutting method using modern machines. Loading and transportation of blasted mined rock is carried out by powerful LHD machines with diesel drive. The average productivity of such technology is 2000–4000 m<sup>3</sup> of rock mass per day [3]. Using in opencasts continuous surface miner Wirtgen 2500SM for selective mining enhances the quality of oil shale. Through the cutting quality the mineral resource utilisation is more effective and environmental impact is less. The disturbing impact of drilling-blasting operations in quarries and open pits next to densely populated areas causes vibration, dust and noise emissions which are arguments to stop operations where blasting is used. High-selective technology of surface miner is prospective owing to reduced dust and noise, as well as non-existent vibration [4].

The functional unit of the system under investigation is one tonne of oil shale. The functional unit is a reference unit, for which the inventory and impact assessment results will be presented, making it possible to compare the results of new and old mining technology. The aim of this study is to identify the differences between the environmental impacts caused by oil shale different extraction methods and production of auxiliary materials and transportation of oil shale to customers, and to give an overview of the environment protection measures of oil shale mining.

## **The methodology**

The choice of mining methods in Estonian oil shale industry depend, to a great extent, on deposit depth. As deposit parameters determine the use of different equipment and extraction methods, the technologies can vary significantly [5]. The inventory data base represents a detail description of

the mining system that comprises the description of excavation processes and analysis by classification and characterization methodology of LCIA [6]. These phases correspond to the methodology of LCIA recommended by the International Organization for Standardization [6]. The analysis involves data collection and description of unit processes for calculation procedures. The data collection includes all emissions associated with the oil shale excavation processes [7]. Descriptive information of unit processes is a necessary tool for evaluation operation option and environmental impact [8]. Description of unit processes presents a general overview of mining – what technology is applied and what equipment is used in the excavation processes.

At selection of impact categories and classification, appropriate impact categories were chosen and the collected inventory data (= environmental interventions such as emissions, land use and resource extractions) were classified into the selected impact categories according to their cause – effect relationships [14]. At characterisation, the chosen characterisation factors enable an aggregation of emissions within each impact category. The emission values are converted into impact category indicator values by multiplying the initial data by the corresponding characterisation factors [8] (Table 1).

Acidification ( $\text{SO}_2$ ,  $\text{NO}_x$  (expressed as  $\text{NO}_2$ ),  $\text{NH}_3$ ) refers to the wet or dry deposition of acidic substances of anthropogenic origin on the earth's surface and is commonly called acid rain, but it includes also acid snow and acid fog. Acid rain is able to mobilize metals and other acid-soluble compounds from soil. Acids dissolve aluminum and other metals from soils in amounts becoming toxic to plants and aquatic organisms. Acid rain dissolves cement and minerals used at building [2].

Terrestrial eutrophication ( $\text{NO}_x$ ,  $\text{NH}_3$ ) can be defined as the state of increased nutrient availability in soil as a result of input of plant nutrients. The balance between nutrients available in the soil and natural vegetation has been disturbed in large areas in Europe mostly due to atmospheric input of nitrogen as a result of human activities. Excess inputs of nitrogen may lead to an undesirable shift in plant composition in natural or semi-natural ecosystems or loss of biodiversity. In addition, a high nitrogen load may affect groundwater making it unsuitable as a resource for drinking water [2].

Ecotoxicity (e.g. dioxins, PAH, PCB, metals, oil, cyanides and phenols in water) includes various chronic and acute effects on the natural organisms. In this work, the assessment of ecotoxicity is based on the EDIP 2003 methodology [13] in which the impact category is divided into two sub-categories – chronic aquatic and terrestrial ecotoxicity. In aquatic ecotoxicity, airborne and waterborne emissions are taken into account, whereas emissions into the air (as fall-out) are taken into account in terrestrial ecotoxicity. In the determination of characterisation factors, the simple exposure model in which the environmental conditions of Northern and Eastern Europe are roughly taken into account predicts the concentrations of substances in the environment. In the method, the result is obtained by multiplying the emissions (kg) by the corresponding characterisation factors ( $\text{m}^3/\text{kg}$ ) [14].

The selected impact categories correspond to the division recommended by the international LCA community [14] except for the group 'other impacts'. Other impacts consist of the local impacts caused by such factors as land use, solid wastes and water pumping from mines.

The characterisation factors change the values of interventions into the commensurable unit within the impact category so that the values of different interventions can be added together. The unit varies impact by impact depending on the chosen indicator for measuring the effects. For acidification and terrestrial eutrophication, the latest country-specific characterisation factors were used instead of site-generic characterisation factors [9]. This is due to the fact that the location of the emission source can cause different responses in the surrounding ecosystems in the context of these impact categories, depending, e.g., on local atmospheric conditions and the sensitivity of the ecosystems [10]. In the context of these three impact categories, the emissions of life cycle stages were calculated using characterisation factors specific to Estonia. It is assumed that all important interventions occur in Estonia. Thus, for acidification and terrestrial eutrophication Estonia-specific characterisation factors were used (Table 1).

**Table 1. Characterisation factors and reference value for acidification and terrestrial eutrophication [11]**

Impact category	Emission	Unit	Factors for Estonia
Acidification	SO <sub>2</sub>	eq/kg	0.369
	NO <sub>2</sub>		0.194
	NH <sub>3</sub>		0.405
Terrestrial eutrophication	NO <sub>2</sub>		1.483
	NH <sub>3</sub>		4.418

Different acidifying emissions were aggregated by characterisation factors, which are derived from the results of European air quality and transport model [11] and critical loads determined over Europe [12]. The critical load means that harmful effects will occur if the deposition of acidifying emissions exceeds a certain limit.

## Results and discussion

Data offered by *Eesti Põlevkivi* were the basis to study emissions from mines into water and air according to impact categories like acidification, terrestrial eutrophication and ecotoxicity. Annual outlet of mining water and air has served as the measure of emission. For an opencast, gaseous emission into the air was calculated considering diesel combustion of working machines (the processes of stripping, cutting, drilling, loading, transportation, recultivation). The supply unit of blasting operation – the production of ammonium

nitrate from ammonia and nitric acid  $\text{NH}_4\text{NO}_3$  – was used to calculate the emission from the explosion process. Some machines work only on electricity and do not apply diesel combustion, therefore emissions attributed to power generation were calculated.

Emissions from mining, transportation of oil shale to customers and auxiliary material production accompanied with production of oil shale are presented in Table 2.

Table 2. Emissions per one tonne of extracted oil shale (Eesti Põlevkivi 2005)

Activity	SO <sub>2</sub>	NO <sub>x</sub>	NH <sub>3</sub>
Viru mine	3.00E-02	8.00E-04	
Estonia mine	1.00E-02	4.00E-04	
Aidu opencast	4.00E-02	4.00E-02	
Narva opencast	6.00E-02	3.00E-02	
Mining total	1.00E-01	6.00E-02	
Auxiliary material	4.00E-02	6.00E-02	1.00E-02
Transportation	1.00E-02	2.00E-01	

### Acidification

At assessment, the effects of emissions causing acidification were expressed with the help of an indicator called ‘accumulated exceedance’. It describes the effects as acidification equivalency (eq) where eq corresponds to 1 mol proton ( $\text{H}^+$ ) released. [12].

In the Viru mine, acidification is three times higher than that in the Estonia mine. Opencasts emit more NO<sub>x</sub> resulting in greater acidification. (Fig. 1a).

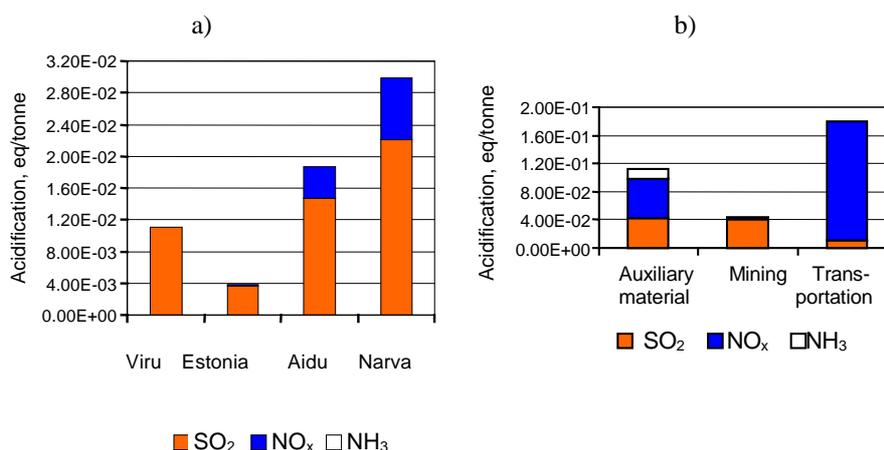


Fig. 1. Contribution of different emissions to the acidification process. (NH<sub>3</sub> is “out of scale” because of its low amount).

The result showed that mining causes less acidifying emissions than the production of auxiliary materials (including production of diesel oil, explosives, steel etc.) and transportation of oil shale to customers. Also, as for acidification, the main contributor in the mining process is  $\text{SO}_2$ , while at transportation the level of  $\text{SO}_2$  is not so high as that of  $\text{NO}_x$  (Fig. 1b).

### Terrestrial eutrophication

Terrestrial eutrophication means a state of increased nutrient availability in soil as a result of input of plant nutrients. The characterisation method is based on the same approach as used in the case of acidification [14]. In the context of terrestrial eutrophication, an indicator describes the accumulated exceedance of critical loads of eutrophication. It measures the effects as eutrophication equivalency (eq) that corresponds to 1 mol nitrogen.

The Narva opencast is the leader of  $\text{NO}_x$  production in comparison with the mines Aidu and Estonia. However, the role of the Viru mine in terrestrial eutrophication is insignificant (Fig. 2a).

Transportation of oil shale to customers causes more impact on terrestrial eutrophication than production of auxiliary materials and mining (Fig. 2b).

New machinery and modern technology in the Estonia mine should guarantee greater extraction of oil shale than in the Viru mine using old machinery. On the other hand, old machinery works only on electricity and does not emit from diesel combustion, therefore its emissions are calculated considering power generation. In opencasts, surface miner Wirtgen 2500SM used for selective mining allows to exclude the drilling-blasting process. Necessary data for evaluation of emissions can be calculated from the specification of the corresponding equipment.

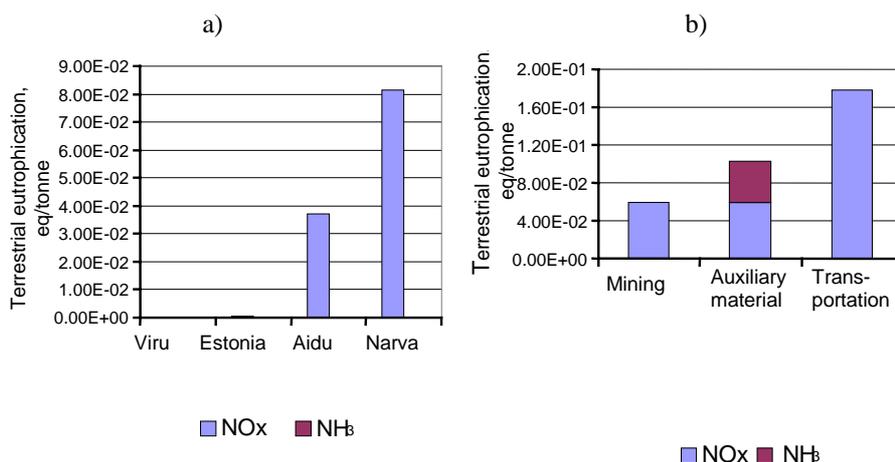


Fig. 2. Contribution of different emissions to the terrestrial eutrophication processes. ( $\text{NH}_3$  is "out of scale" because of its low amount).

### Ecotoxicity

The measure of ecotoxicity is obtained by multiplying the emissions (kg) by the corresponding characterization factors ( $\text{m}^3/\text{kg}$ ).

In comparison with auxiliary material production (Fig. 3), the role of excavation processes is less.

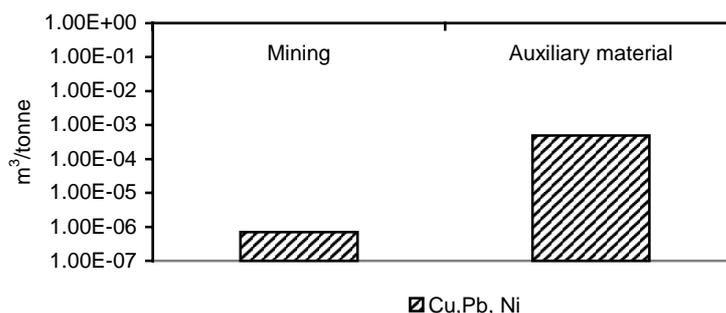


Fig. 3. Contribution of oil shale mining and production of auxiliary materials to chronic terrestrial ecotoxicity.

### Ground surface subsidence and land use

The processes in overburden rocks and pillars have caused unfavorable environmental side-effects accompanied by significant subsidence of the ground surface. Ground surface subsidence can cause soil erosion and flooding, swamp formation, agricultural damage, deforestation, changes in landscape, decrease in groundwater level and make the formation unstable. Nowadays underground oil shale excavation is made by room-and-pillar method with blasting. The commercial oil shale bed and immediate roof consist of oil shale and limestone seams. The main roof consists of carbonate rocks of various thicknesses. The characteristics of various oil shale and limestone seams are quite different. The strength of the rocks increases in the southward direction. For this reason, stability of pillars is difficult to prognose. Ground surface subsidence results in pillar collapse. Depth of a subsidence depends on the thickness of the extracted seam. The first spontaneous collapse of pillars and surface subsidence in an Estonian oil shale mine took place in 1964 [15].

Up to the present, over 70 spontaneous collapses of chamber blocks in Estonia, Viru, Ahtme and Tammiku mines have been recorded [16]. Figure 4 shows that most of collapses occurred during 30 months of the exploitation, the number decreasing to the point of 60 months. Collapses of chamber blocks after the period of 60 months occurred in locations of complicated geological conditions and below the area of rock dump formation under loading.

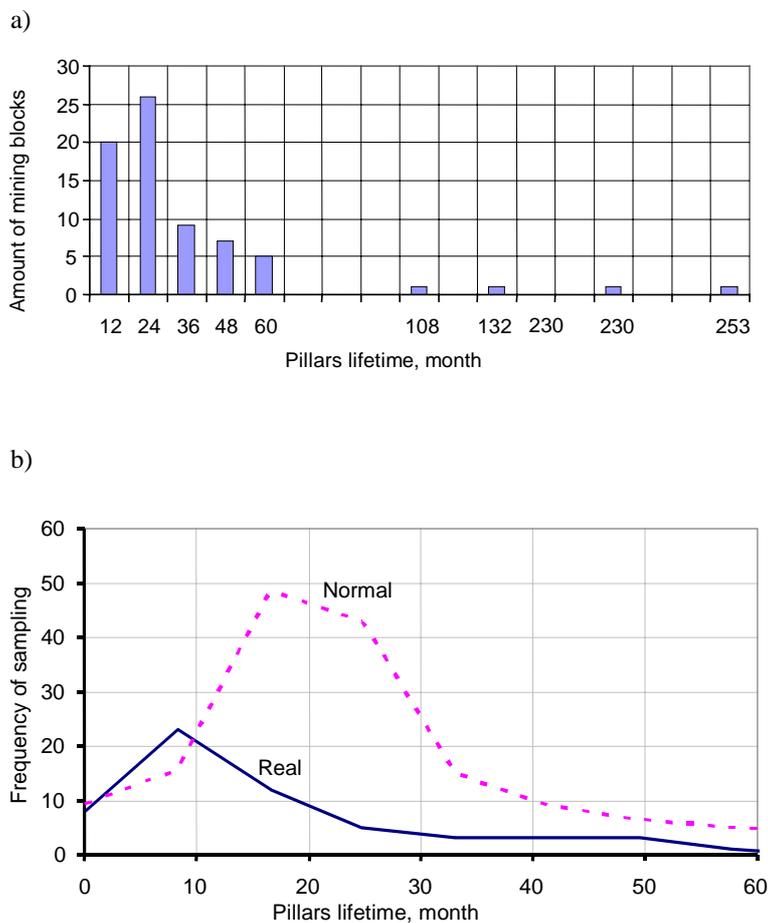


Fig. 4. a) The amount of collapsed chamber blocks during the time period under investigation. b) Logarithmic normal distribution of pillars' lifetime.

Statistical data and analysis of pillars' lifetime [16] enabled to express pillars collapsed during this period in the logarithmic normal distribution scale. Logarithmic normal distribution, as a rule, well approximates such random variables  $X$  which are formed as a result of multiplication of a big number of independent or poorly dependent non-negative random variables; the dispersion of each is small in comparison with the dispersion of their sum. Summation of data enables to assume that chamber block collapses occurring during 60 months were caused by diminishing cross-sectional area of pillars and increasing chamber volume (Fig. 4b).

The total volume of waste from oil shale mining is 180 million tonnes, and it covers 188 ha from mines and 150 ha from opencasts forming a cone-shaped dump. The volumes of waste used as landfills were 70 million tonnes in the Estonia mine and 35 million tonnes in the Viru mine. From the area as

large as 220 km<sup>2</sup>, oil shale has been mined by underground method. 61% of the total amount of oil shale has been extracted from underground mines. A half tonne of waste is generated per tonne of extracted oil shale in the process of oil shale enrichment. Wastes destroy agricultural land cover and ecosystems. In the case of surface mining, a very successful recultivation method allows to restore ecosystem and landscape.

The mining causes many local environmental impacts affecting land use and ground surface subsidence which cannot be assessed by the life cycle impact assessment methodology.

### Mine water pollution

Annual water outlet of the mining enterprises of *Eesti Põlevkivi Ltd.* amounts to 200–240 million m<sup>3</sup> (Fig. 5) [17]. On this reason, the depth of the sinkhole reaches 70 m, with impact radius 5–10 km. Disturbances in the natural regime of groundwater bring about changes in its chemical composition. The pumped-out water does not meet the requirements to waters discharged into the environment. In the course of mining, water becomes enriched with saline compounds, containing, for instance, up to 500 mg/l of sulphates instead of natural 20 mg/l. The content of minerals has increased to 1 g/l (natural 0.3–0.4 g/l). Several breakdowns of mines have caused pollution with phenols, up to hundreds times exceeding their permissible limit (0.001 mg/l) [18].

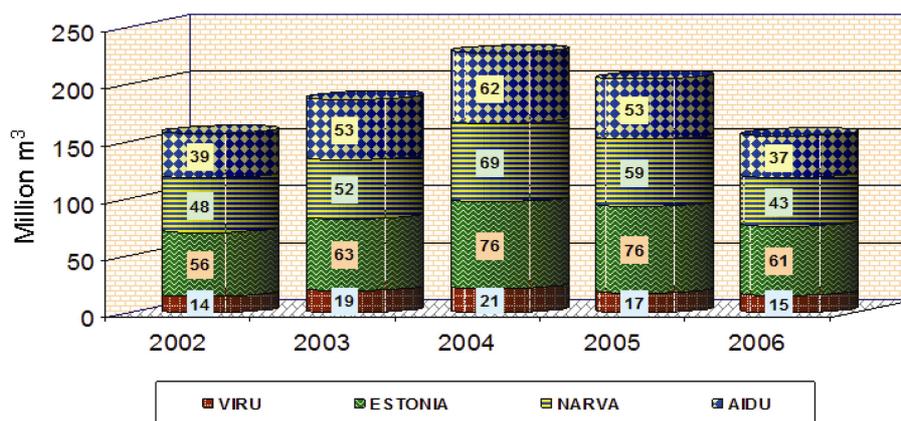


Fig. 5. Mining water outflow.

## Conclusions

As shown by investigation, mining processes cause smaller impacts on acidification, terrestrial eutrophication and ecotoxicity than production of auxiliary materials and transportation of oil shale to customers.

Statistical analysis of pillars' lifetime shows that pillars collapsed during the period under investigation express logarithmic normal distribution. Summation of data enables to assume that chamber block collapses occurred during 60 months because of diminished cross-sectional area of pillars and increased chamber volume. Correct choice of dimensions of pillars and chamber will guarantee stability of the ground surface. To prevent subsidence of ground surface and restriction of land use, it is recommended to use the backfilling method, which allows to restore the ecosystem and landscape.

Mining waters exert minor impact on the composition of natural waters: the amount of heavy metal compounds in mining waters is lower than their natural level in North-East Estonia.

The method of technological and environmental assessment of the impact of a combination of different mining processes gives an opportunity to find a better way for planning new mines in accordance with environment protection measures for conditions of the Estonia oil shale deposit.

## Acknowledgements

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## **Risk Assessment Of Surface Miner for Estonian Oil-Shale Mining Industry**

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**Keywords: high-selective mining, risk estimation**

The paper deals with risk assessment of a high-selective oil-shale mining technology using surface miner Wirtgen 2500SM. This study addresses risk associated with productivity and cutting quality on example of Estonian oil shale deposit in areas with complicated layering conditions. The risk assessment method allows choosing relevant technology with friendly environment and economic value. For risk estimation the event tree is used. The results of the risk assessment are of practical interest for different purposes.

### **1 Introduction**

About 98% of electric power and a large share of thermal power were produced from Estonian oil shale. Mining sector faces challenges to increase the output of mines and to minimize the environmental impact of mining at the same time. Continuous mining and milling techniques for the hard rock industry are up to now limited through the hardness of rock material. The application limits for the future technique will be placed above the limits of bucket wheel excavating systems with a diggability of normal up to 10 MPa of uniaxial compressive strength (UCS). This can be expanded with special designed excavators for frozen hard coal or soft limestone (Wilke at al. 1993). Horizontal and vertical ripping techniques are currently used for materials up to 50 MPa UCS, sometimes combined with in-pit crushing systems.

Surface mining is carried out in open casts with maximum overburden thickness of 30 m. Draglines with 90 m boom length and 15 m<sup>3</sup> bucket size are used for overburden removal. Hard overburden consists of limestone layers and is blasted before excavation. Oil shale layers are blasted as well or broken by ripping (semi-selective mining). Disadvantage of ripping is excessive crushing of oil shale by bulldozer crawlers. Excavated rock is transported with 32-42 or 60 tonnes trucks (Belaz and Euclid) to the processing or crushing plant depending on opencast.

Aim of the research and in-situ SM testing is to introduce continuous mining technology on example of Estonian oil shale deposit in areas with arduous layering conditions. The results of in-situ testing can be used to improve existing situation in mining fields with complicated geological conditions and in densely populated regions.

Continuous surface miners can find their natural applications in projects where drilling and blasting is prohibited or where selective mining of mineral seams, partings and overburden is required. Besides they offer further advantages less mineral loss and dilution, improved mineral recovery especially in areas sensitive to blasting, less stress and strain on trucks due to minimum impact of the excavated material, primary crushing and fragmentation of mineral rock, reduced capacity requirements for preparation plants.

The high-selective oil-shale mining technology introduced by surface miner (SM) Wirtgen SM2500 and the first 9 months of testing results at "Narva" open-pit in Estonia. The technology allows to decrease oil-shale losses from 10-15% up to 5-7% on in-situ conditions. Mining process of the surface miner has a lower disturbing impact, which is topical in open casts and quarries especially in densely populated areas. The low level of dust and noise emissions and also very low vibration are arguments to mine oil shale with surface miner instead of drilling-blasting operations. (Nikitin at al., 2007)

The most perspective advantage of SM is high-selective mining. Surface miner can cut limestone and oil-shale seams separately and more exactly than rippers (2-7 cm) with deviations about one centimeter. It is estimated that due to precise cutting enables surface miner to increase the output of oil shale up to 1 tone per square meter. It means, that oil-shale losses on the case of SM technology can be decreased from conventional 12 up to 5 percent

## 2 Risk analysis of surface miner Wirtgen 2500M technology

Continuous surface miner, which are designed to cut softer rock materials like sandstone, clay, bauxite, hard coal, phosphate, gypsum and marl are operating between 10 MPa and 70 MPa compressive strength. Nowadays, road cutting machines are working materials up to 100-110 MPa compressive strength. The very recent developments show that there is a need for investigations to enlarge the mentioned application limits.

The Wirtgen 2500SM design with a mid-located cutting drum (diameter 1.4m, cutting width 2.5m) was expected to be more promising for hard rock (80-110 MPa) applications than the front-end designs. Here, the whole weight of the machine (100 t) is available for the penetration process and only a smaller torque resulting from the cutting process (cutting depth up to 0.6m) has to be counterbalanced. Besides, the surface miner with middle drum concept moves during the winning process. Due to this great moved mass, much more dynamic mass forces are possible than during the movement of the small mass of the cutting organ mounted on a boom.

Modifications and development work for the tested SM focused mainly on the corresponding cutting drums (number of cutting lines) and specifications of the cutting tools, different loading technologies (windrowing or direct truck loading) also (Figure 1 a, b).

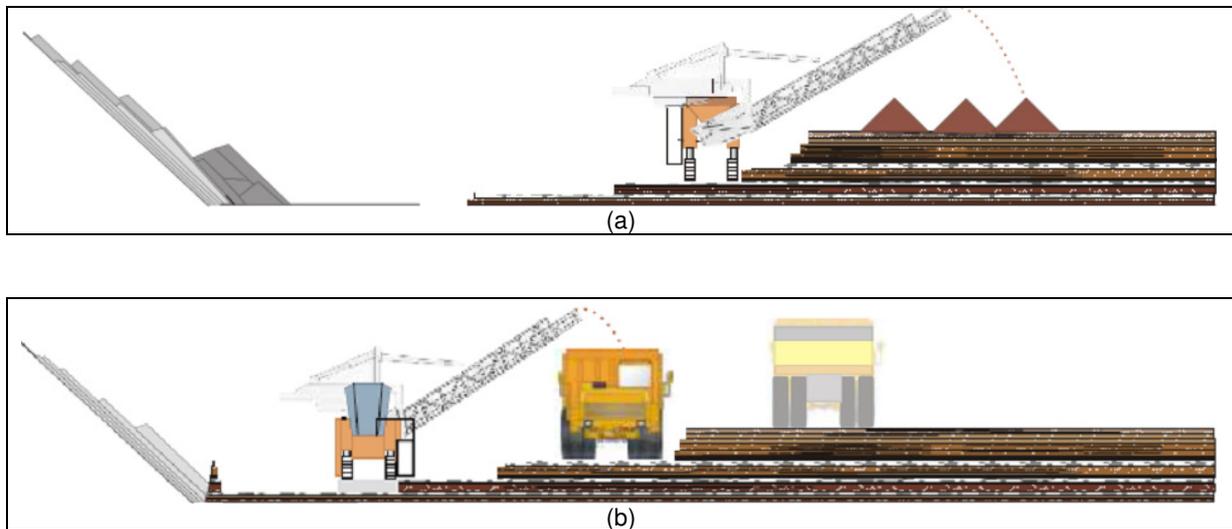


Figure 1. Different loading technologies: windrowing (a) and direct truck loading (b)

### 2.1 Rock Breakability Results

To be able to transfer the achieved results to other EU rock mines, it is necessary to identify the SM and cutting rock parameters responsible for the breakability factor of a deposit. The development of such a generalised classification system is therefore an important objective of the project as well.

Applying statistical distribution according to Weibull, the function of size distribution of oil-shale particles may be assumed as follows:

$$W = 1 - \exp[-(d/d_0)^m] \quad (1)$$

where  $d_0 = d_{0.63}$  is diameter of screen opening to pass 63.2% of broken oil shale;  $m$  is breakability factor.

The results from sieving analysis made for limestone and oil-shale layers show that for "Narva" open pit test site conditions breakability factor  $m = 1.1$ . Hence, the share of oil shale  $\delta$  passing the  $25 \times 25$ -mm screen in the total mine-run shale equals to

$$\delta_{-25} = d_{-25} = 1 - \exp[-(25/d_{0.63})^{1.1}] \quad (2)$$

where,  $d_{0.63} = 20.0 + 2.16S'$  for SM up-cutting direction (see Figure );  $S'$  is cross-section of cut,  $\text{cm}^2$ .

In 1968 E. Reinsalu had proposed an approximate relationship between energy consumption by different methods of breaking and average size of mined oil-shale particles, which was later completed with the present investigation data (Figure 2).

Concentratibility and trade oil shale grade depend on sizing extracted oil shale, which, in its turn, is closely

related to energy consumption and the selected method of oil shale breaking. Equation (3) and Figures 3 a, 3 b demonstrates the correlation of the distribution law parameters and specific energy consumption with the parameters of oil-shale and limestone particles sizing. The tested SM sizing parameters are inside the areas with number 7 and 8 ( Figure 2 a).

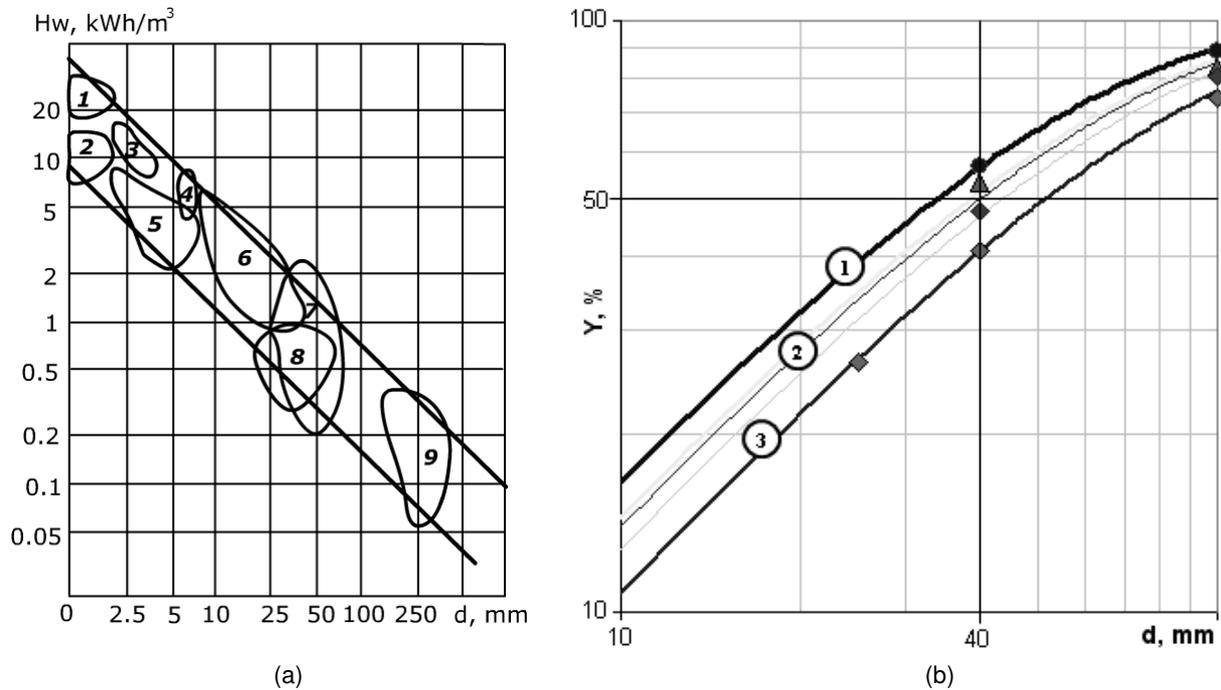


Figure 2. Effect of method of breakage on specific energy consumption (A.) and the resulting average oil shale and limestone sizing (B.)

Where, 1 – drilling in limestone; 2 – drilling in oil shale; 3 – cutting machine in limestone; 4 – cutting machine in oil shale; 5 – cutting of layer B with shearer loader UKR-1; 6 – cutting with shearer; 7 – cutting with DKS (a mean for measuring cuttability) in limestone; 8 – cutting with DKS in oil shale; 9 – breaking with ripper (surface mining); Wirtgen 2500SM sizing data (up-cutting direction): ①– cutting in oil-shale complex EF (0.43m); ②–cutting in limestone seams A/B (0.18m) and C/D (0.25m); ③– cutting in oil-shale complex CB (0.36m).

### 3 Risk estimation of surface miner testing results

The Wirtgen 2500SM was delivered to AS Eesti Põlevkivi at the end of 2006. The testing of SM was beginning at “Narva” oil-shale open cast. The SM testing was held from 01.01.2007 to 30.09.2007 and was estimated by four testing phase (Figure 3). The machine was operated in two or three-shift systems.

During the first testing phase (I) 145 total operating hours from 200 available (9.4 m/min) and during the second testing phase (II) 151 from 208 available shift-hours (9.0 m/min) the SM with direct truck loading was tested. But the real cutting time was 35 and 41% from available shift-hours for the each period correspondingly. During the third testing phase (III), 4130 total operating hours from 5416 available shift-hours the SM with about 26% of windrowing. For the fourth testing phase (IV) 111 total operating hours from 112 available shift-hours the SM windrowing achieved 100%. The average cutting speed during the real cutting time was 11.5 m/min. For the while period real cutting time is about 46%, where on average during the shift-time 33% SM operated on oil-shale layers and 13% on limestone (layers C/D and B/A). The Figure 3 illustrates the shift-hours distribution graphics for the testing phases.

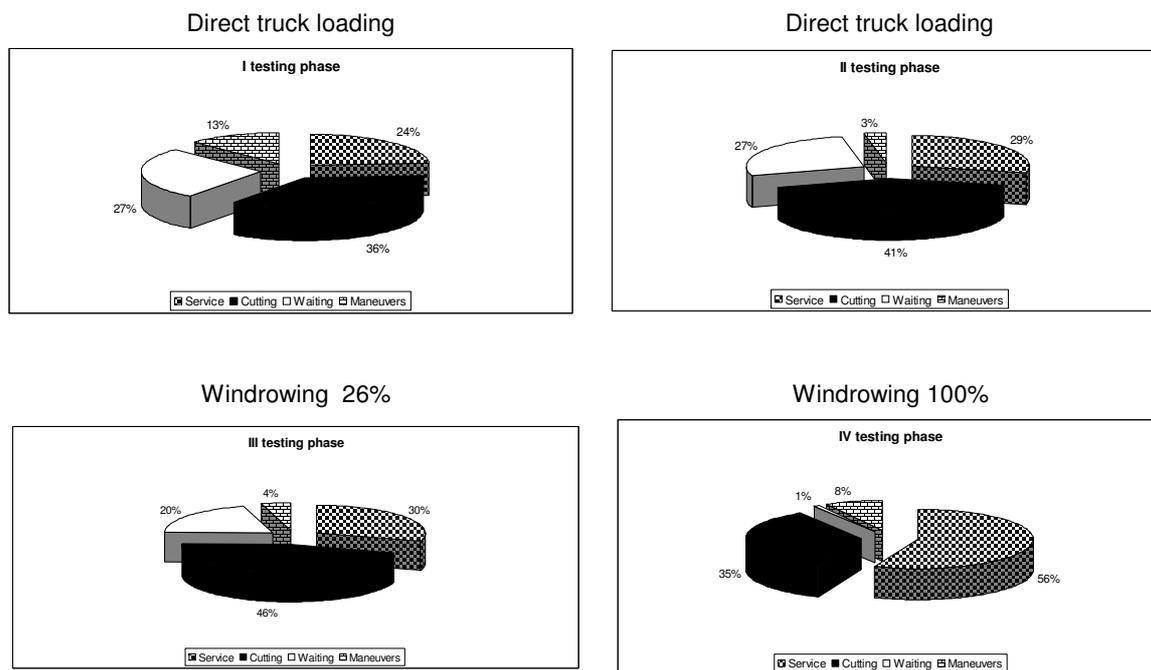


Figure 3. The shift-hours distribution graphics for the while testing period.

During the testing phase registered “waiting” is about 27%. Obviously, the main reason is direct truck loading method. Analysis has shown that by direct truck loading method, truck-waiting downtime decrease real cutting time by 1.0-1.5 hour per shift and average cutting speed by 20-25%. The percent of “waiting” include about 7% of time loses for trucks exchanging, about 6% for SM upper conveyor manoeuvres, then about 6% for spade-work (SM controlling before and after the shift) and up to 10% time loses due to the ground water problems.

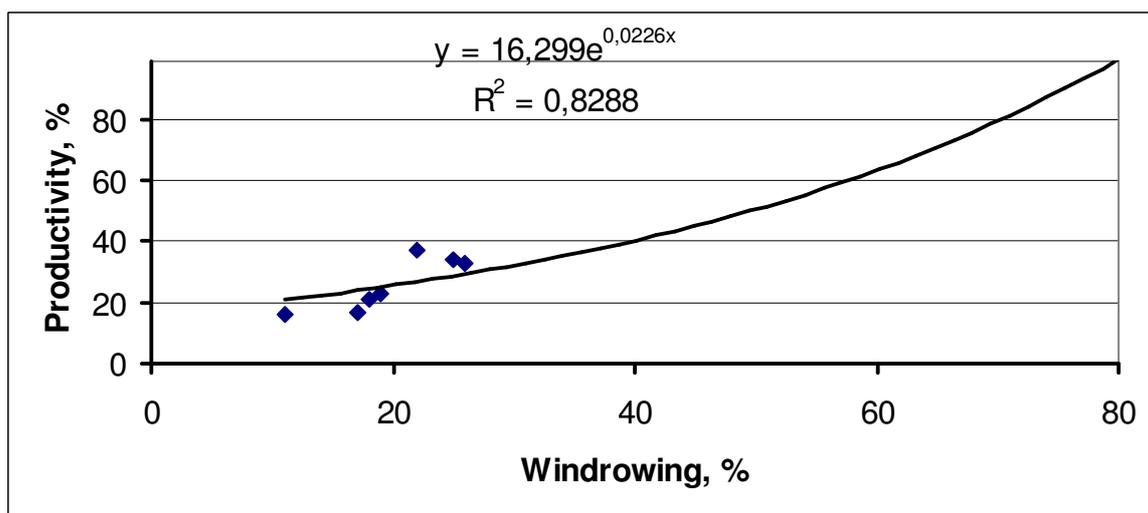


Figure 4. Dependence of SM productivity on windrowing percentages

As you can see from the graph (Figure 4), there is a great SM productivity potential when windrowing percent is growing. The additional LHD-machine operating and SM depreciation costs greater oil-shale excavation rate is coating. As a result the oil shale operating cost can be reduced up to 10-15%.

Main aspects influencing the efficiency of the combine work concern the duration of the processes. Cutting

different layers, track dumper loading (waiting), manoeuvres and maintenance processes are the most important factors. Investigations have shown that duration of the processes influence on productivity. The main quantitative approach used in risk estimation is the event tree method (Calow, P. 1998). This method was selected as the most appropriate one for the risk estimation of the SM. In the first stage of the project time factor was taken into consideration. For probability determination the empirical approach was used (Williams at al. 2004). The event tree indicating the probabilities of the SM processes and spent time. It is possible to select different variants and to determine the probability of one.

The event tree allows determining time deviations from average value (Figure 5). Four different testing phases (I-IV) of the SM were observed. For determination suitable variant greatest negative numbers were chosen in comparison analysis with maximal possible productivity received during the tests. Application of the fault tree is presented in Table 1. Selected variant of the tests give different value of the probabilities and deviations from the average value. For determination the higher productivity is necessary to give attention on processes with positive value and improve it quality (Figure 5).

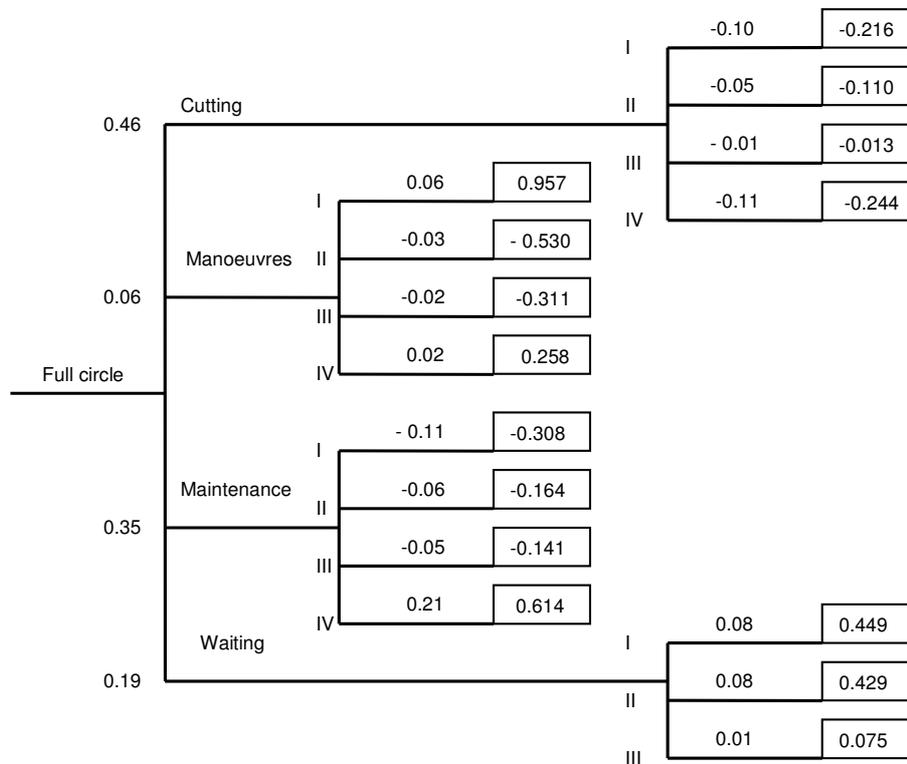


Figure 5. Event tree of different processes

In case of excluding complicated geological condition higher productivity can achieve owing to the windrowing method.

Table 1. Time deviations from the average value

Testing phases	I	II	III	IV
Maintenance	-0.308	-0.164	-0.141	0.614
Cutting	-0.216	-0.110	-0.013	-0.244
Waiting	0.449	0.429	0.075	-
Maneuvers	0.957	-0.530	-0.311	0.258

#### 4 Risk evaluation

The thickest and harder limestone seam "C/D" (60-80 MPa) has sufficient quality to produce aggregate for road



building and concrete. Separately extracted limestone (C/D and A/B) can be left directly in mine, which reduces haul costs and increase run-out oil shale heating value without additional processing.

The oil yield increase by 30%, up to 1 barrel per tone during the oil shale retorting, because of better quality. The same principle is valid for oil shale burning in power plants because of less limestone containing in oil shale. Its results higher efficiency of boilers, because up to 30% of energy is wasted for limestone decompose during the burning process. Positive effect would result in lower carbon dioxide and ash emissions (Adamson et al. 2006).

Another perspective of surface miner would apparent in places with relative small overburden thickness (less than 10 m) and near the towns where the removal of hard overburden with SM should be considered as well instead of overburden blasting. On these cases the SM would "cut" considerably operating costs of stripping and possibility mine out reserves near the densely populated areas.

Another problem is the oil-shale bed geological characteristics. Estonian oil-shale bed consists from oil-shale and limestone seams with different thickness and compressive strength. Oil shale is relatively soft rock with UCS 15-40 MPa but limestone is 40-80 MPa. There are also places near the karsts zones with 100-120 MPa compressive strength. During the cutting process the loads in cutting tools vary greatly due to the differences in rock physical and mechanical parameters, which lead increased loading of the cutting drum.

The applicants have recently encountered many situations where manufacturers cutting drum/head designs could be significantly improved upon, as they were not tailored to the actual geotechnical conditions predominant at the mine. However, without more user-friendly tools, the opportunity to make such improvements in practice has been limited. Improved designs have the potential to increase cutting speed and efficiency, reduce pick replacement costs, reduce machine down time through gearbox failure and pick changing, improve machine reliability by reducing excessive vibration during cutting, improve loading efficiency and reduce fine oil shale and dust production. Research program to develop design of cutting tools/drums to minimise cutting tools consumption and machine down time on the basis of testing data will be develop. New design of cutting drums will lead to improved tool cutting (pick) loading efficiency with less fine rock and dust production. The result of this work will be taken into account for the next SM design.

Development of mining machinery and mining technology by the way of selective mining will improve environmental situation in Europe and Baltic Sea region. Effect can be achieved in decreasing CO<sub>2</sub> emission, ash and water pollution.

Selective mining enhances the quality of oil shale. Through the cutting quality the mineral resource utilisation is more effective and environmental impact is lower. The disturbing impact of drilling-blasting operations in quarries and open casts next to densely populated areas causes vibration, dust and noise emissions which are arguments to stop operations where blasting is used. Surface miner high-selective technology has perspectives due to reduced dust and noise, non-existent vibration and dust emission levels also.

By extending the applicability of the surface miner/road cutting technology from soft material into semi-hard and hard rocks with UCS of up to 110-120 MPa, an economically and environmentally acceptable alternative to drilling and blasting could be available. By taking into account the rock-mechanical and mine planning aspects of the test application, an evaluation of the overall economical feasibility and the transfer of the results to other hard rock mines can be ensured.

## 5 Conclusions

Event tree allow determining suitable variant of different processes for continuous surface miner. For determination suitable variant greatest negative numbers were chosen in comparison analysis with maximal possible productivity received during the tests. Surface miner higher productivity in testing phase (IV) was achieved on account of 100 % windrowing method. The high cutting performance can be explainable absence of waiting time. This information allows finding adequate decision to improve quality of the processes and avoid negative influence.

Results obtained by this project can be using in different industrial sectors. The main applications will be found in the surface mining and road construction sectors. New usage could be in zones where rock soils will transformed into zones with agricultural capacities.

There is a couple of direct and indirect effects which reduce oil shale cost prise on 20% due to less mineral losses, loading method (windrowing) can optimize fuel consumption when high-selective mining technology with surface miner is applied. The result of this work will be taken into account for the next surface miner design.

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