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DETERMINING THE OPTIMAL METHOD OF EXCAVATION OF THE ZENICA TUNNEL AS A FUNCTION OF MINIMAL DAMAGE TO ROCK MASS OF POORER QUALITY OUTSIDE THE EXCAVATION PROFILE

Ekrem Bektašević[1], Reuf Kadrić[2], Kemal Gutić[3], Sead Kadrić[4]

ABSTRACT

Corridor Vc enhances the connection of Bosnia and Herzegovina with neighboring countries and improves the potential for economic development. The Corridor Vc motorway in Bosnia and Herzegovina stretches from the northern border with the Republic of Croatia from Svilaj to Čapljina on the southern border with the Republic of Croatia, in Bijača. The Zenica tunnel is part of the Corridor Vc motorway in the municipality of Zenica and is currently the longest excavated tunnel with a total length of 3,330 meters. Excavation works were performed by a combination of blasting and machine excavation. The excavation of the tunnel was performed in sedimentary formations (flysch-like Upper Vranduk series ^{2}JK). Excavation of tunnels by blasting with adjusted drilling and blasting parameters in rock mass of poorer quality achieves better results in relation to machine excavation.

Key words: tunnel, rock mass, machine excavation, drilling, blasting, overbreak

1. INTRODUCTION

The Zenica tunnel is part of the motorway on Corridor Vc, section Zenica Municipality Northern Administrative Boundary (Nemila) - Zenica North, subsection Ponirak - southern exit from the Zenica tunnel. The tunnel is designed with two tunnel tubes, each having two traffic lanes with a width of 3.50 m and a marginal strip with a width of 0.35 m.

The beginning of excavation of the left tunnel tube is from chainage km 0+153.62, and the end of the tunnel, or of the excavation, is at chainage km 3+435.614, and the length of the left tunnel tube is L=3,281.994 m'. The beginning of excavation of the right tunnel tube is from chainage km 0+155.76, and the end of the tunnel, or of the excavation, is at chainage km 3+485.61, and the length of the right tunnel tube is L=3,329.850 m'.

The excavation of the left tunnel tube in the length of 2,395.36 m' and the right tunnel tube in the length of 2,440.14 m' was carried out by the contractor Euro-asfalt, while the rest of the tunnel excavation was carried out by the Turkish company Cengiz. The maximum overburden of the Zenica tunnel is approximately 470 m'.

The left and right tunnel tubes are connected with ten cross passages for pedestrians and three passages for vehicles.

[1]"PPG" d.o.o. Sarajevo, Ph.D, bektasevic.ekrem@gmail.com
[2]JP Autoceste FBiH Mostar, B.Sc (Mining), reuf.kadric@gmail.com
3PdD, Faculty of Mining, Geology and Civil Engineering - University of Tuzla, kemal.gutic@untz.ba
[4]"FM INŽENJERING" d.o.o. Sarajevo, B.Sc (Mining), sead.kadric@gmail.com

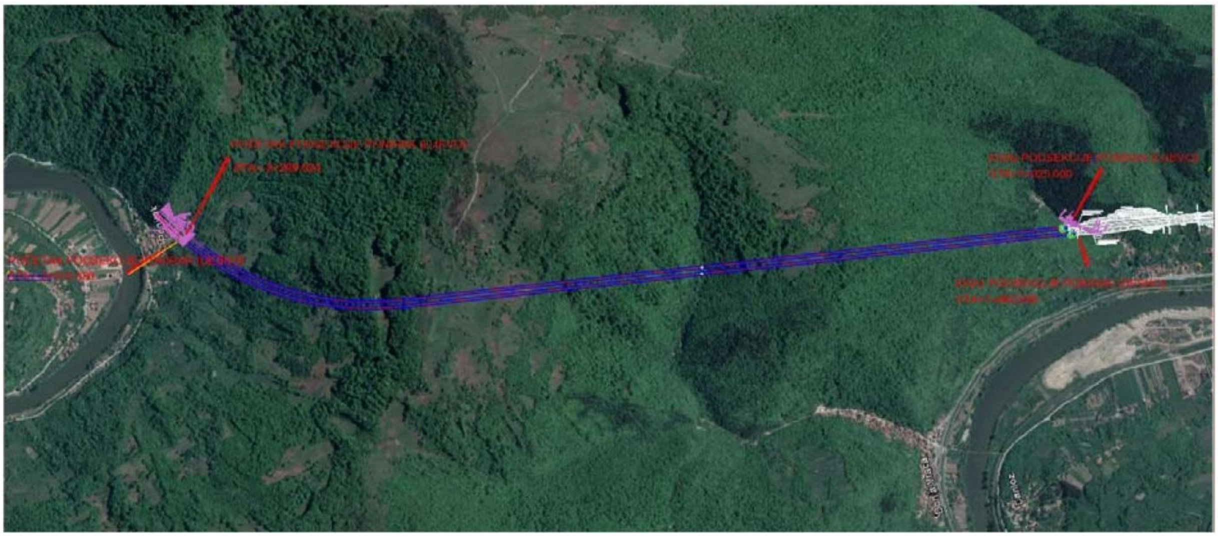


Figure 1. Geographical position of the Zenica tunnel on the Corridor Vc route

2. ENGINEERING GEOLOGICAL CHARACTERISTICS OF THE ROCK MASS IN THE ZENICA TUNNEL EXCAVATION ZONE

The lithological composition of the terrain is represented by Mesozoic formations (flysch-like Upper Vranduk series 2JK). Based on the mineral-petrographic analysis of rocks from the excavation, it was established that the following lithological members are present in the lithological composition: marls, clayey marls, tectonized hematite claystones and subordinately sandstones.

Based on the geomechanical RMR classification for the rock mass, geological mapping defined the following parameters from chainage 1+499.10 to 1+536.00 in the left tunnel tube:

- The lithological composition of the rock mass consists of claystones and marls. The percentage of claystone is about 60%, and marl about 40%. Claystone is characterized by a thinly stratified texture and poor physical and mechanical properties, while marl is characterized by a stratified texture and pelitic-clastic structure.

- Based on previous laboratory tests and field tests using a geological hammer, the uniaxial compressive strength of claystone is in the range from 25 to 50 MPa, and that of marl is in the range from 50 to 100 MPa. The mean value of compressive strength was determined based on the percentages of claystone and marl.

- The quality of the rock mass (RQD) measured at the face of the excavation is in the range from 25 to 50%, which corresponds to fractured rocks of poor quality.

- The discontinuity spacing is smaller, ranging from 6 to 20 cm, while the length of discontinuities is within the limits from 10 to 20 m.

- The discontinuity aperture is closed, and ranges from 1 to 5 mm, while larger ones of 5 mm are recorded. The discontinuities are smooth and slightly rough, filled with hard calcite and soft clayey infilling. Bedding discontinuity is dominant. The strata strike perpendicularly to the tunnel axis with a dip of 15° opposite to the direction of progress of tunnel excavation, which is a good bedding orientation in relation to the tunnel axis.

- Groundwater percolation is registered at the contact of claystone and marl in the form of wetting (less than 10 l/min).

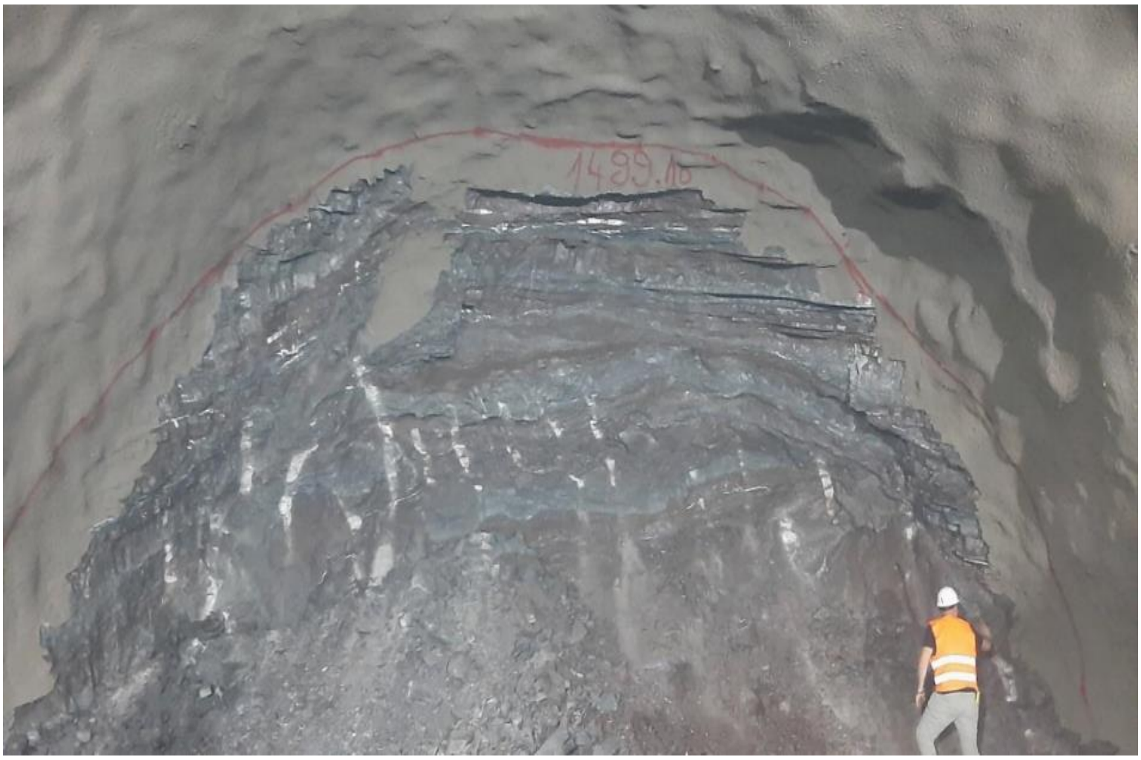
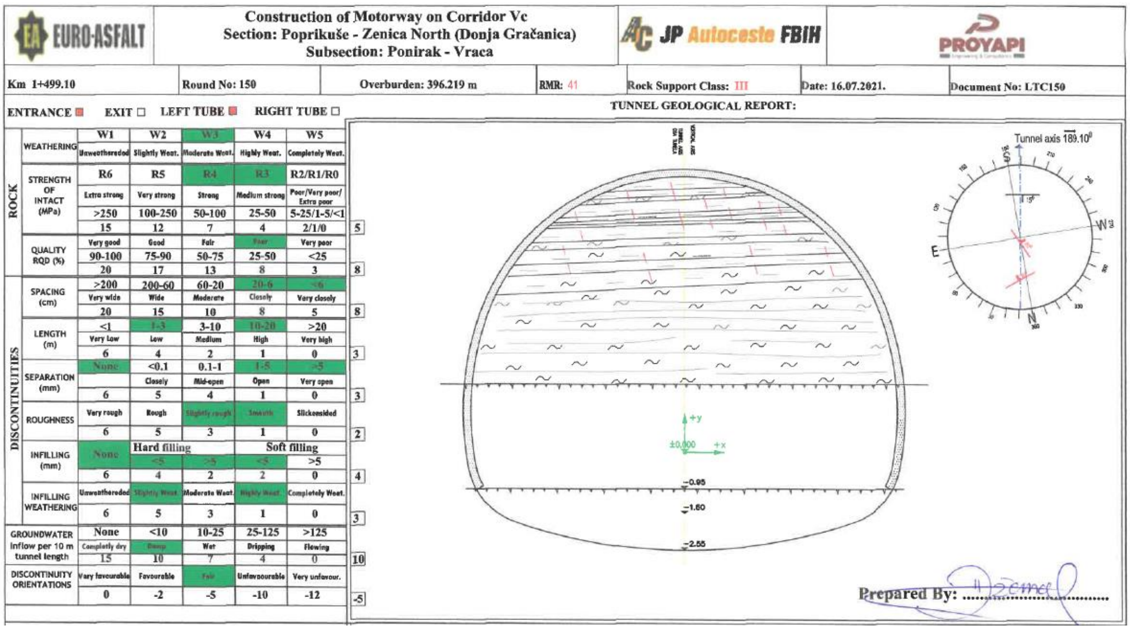


Figure 2. View of the work face, chainage 1+499.10 - left tunnel



EURO-ASFALT | Construction of Motorway on Corridor Vc
Section: Poprikuše - Zenica North (Donja Gračanica)
Subsection: Ponirak - Vraca | JP Autoceste FBIH | PROYAPI

Km 1+499.10 | Round No: 150 | Overburden: 396.219 m | RMR: 41 | Rock Support Class: III | Date: 16.07.2021. | Document No: LTC150

ENTRANCE ■ EXIT □ LEFT TUBE ■ RIGHT TUBE □

TUNNEL GEOLOGICAL REPORT:

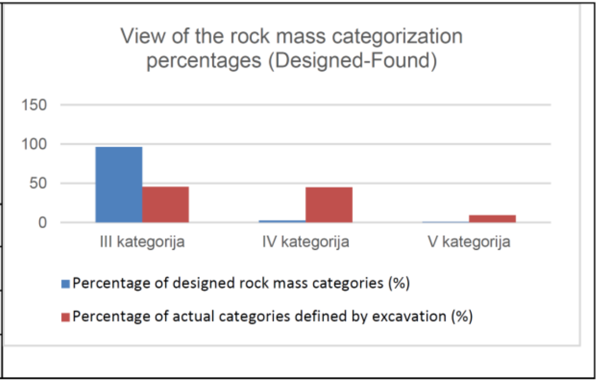
ROCK	WEATHERING	W1	W2	W3	W4	W5	
		Unweatheredod	Slightly Weat.	Moderate Weat.	Highly Weat.	Completely Weat.	
	STRENGTH OF INTACT (MPa)	R6	R5	R4	R3	R2/R1/R0	
		Extra strong	Very strong	Strong	Medium strong	Poor/Very poor/ Extra poor	
		>250	100-250	50-100	25-50	5-25/1-5/<1	
		15	12	7	4	2/1/0	5
	QUALITY RQD (%)	Very good	Good	Fair	Poor	Very poor	
		90-100	75-90	50-75	25-50	<25	
		20	17	13	8	3	8
DISCONTINUITIES	SPACING (cm)	>200	200-60	60-20	20-6	<6	
		Very wide	Wide	Moderate	Closely	Very closely	
		20	15	10	8	5	8
	LENGTH (m)	<1	1-3	3-10	10-20	>20	
		Very Low	Low	Medium	High	Very high	
		6	4	2	1	0	3
	SEPARATION (mm)	None	<0.1	0.1-1	1-5	>5	
			Closely	Mid-open	Open	Very open	
		6	5	4	1	0	3
	ROUGHNESS	Very rough	Rough	Slightly rough	Smooth	Slickensided	
		6	5	3	1	0	2
	INFILLING (mm)	None	Hard filling		Soft filling		
			<5	>5	<5	>5	
		6	4	2	2	0	4
	INFILLING WEATHERING	Unweatheredod	Slightly Weat.	Moderate Weat.	Highly Weat.	Completely Weat.	
		6	5	3	1	0	3
GROUNDWATER Inflow per 10 m tunnel length		None	<10	10-25	25-125	>125	
		Completly dry	Damp	Wet	Dripping	Flowing	
		15	10	7	4	0	10
DISCONTINUITY ORIENTATIONS		Very favourable	Favourable	Fair	Unfavourable	Very unfavour.	
		0	-2	-5	-10	-12	-5

Figure 3. Geological report, chainage 1+499.10 - left tunnel tube

The following table shows the percentage shares of the designed rock mass categories and those defined during excavation for both tunnel tubes.

Table 1. View of the rock mass categorization percentages (Designed-Found)

Rock mass categorization	Percentage of designed categories (%)	Percentage of actual categories defined by excavation (%)
Category III	96.43	45.72
Category IV	2.74	44.93
Category V	0.83	9.35
Total:	100.00	100.00



It is evident from the previous table that there are significant differences in the rock mass categorization percentages between the designed categories and the actual categories defined by excavation. Geological investigations were performed as a basis for development of the Main Design, but they were not sufficient for a more precise assessment of the geological structure in the Zenica tunnel excavation zone, which is why there were significant differences between the shares of the designed categories and the actual categories defined by excavation.

3. DETERMINING THE OPTIMAL METHOD FOR EXCAVATION OF THE ZENICA TUNNEL IN THE ROCK MASS WITH LOWER RMR VALUE

Excavation of the Zenica tunnel was carried out using the New Austrian Tunneling Method (NATM). Bearing in mind the frequent changes in engineering-geological properties of the rock mass in a part of the tunnel excavation, NATM made it possible to apply the multi-phase excavation, while simultaneously securing the excavation with primary support. The tunnel excavation was carried out by machines, by blasting or by combining blasting with machine excavation, all depending on the geological characteristics of the rock mass in the tunnel excavation zone.

Based on the conducted investigation works and available geotechnical data, it was estimated by the main geotechnical design that the excavation of the Zenica tunnel would be carried out in 96.43% of rock mass category III, and that only 3.57% would be category IV and V. Based on the assessment of rock mass categories by the supplementary mining project (SMP), all the technical and technological parameters of drilling and blasting for the rock mass with uniaxial compressive strength of 100 MPa were processed, and on the basis of the adopted strength, all the technical parameters for drilling and blasting were calculated, which are listed below:

q - specific consumption of explosives 1.60(kg/m^3);
Naux - number of auxiliary blast holes 69 (holes);
Nfloor - number of floor blast holes 12 (holes);
Ncon - number of contour blast holes 44 (holes);
N - number of all blast holes 125 (holes).

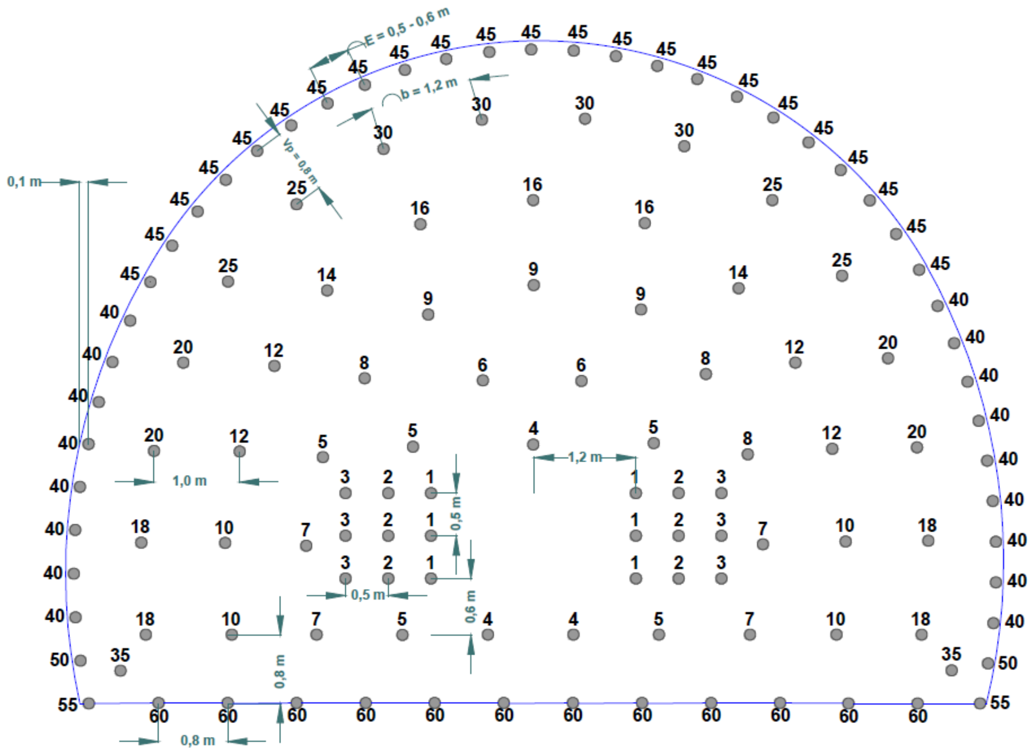


Figure 4. Layout scheme of blast holes on the cross section, as well as the layout of detonators by boreholes, for an advancement step of 3.0 m'

To excavate the tunnel, the contractor used the blasting method in both tunnel tubes with some corrections of technical parameters that they determined approximately without detailed analysis of engineering geological properties of the rock mass, and of the drilling and blasting parameters in the weaker rock mass of RMR (32-43) and as the result had irregular excavation profile shapes and significant overbreaks that exceeded more than 13 m^2 over the entire tunnel profile. One such typical profile of tunnel excavation by blasting is shown on the geodetic survey (Figure 5) of the tunnel face in LTT below, chainage 1+435.998.

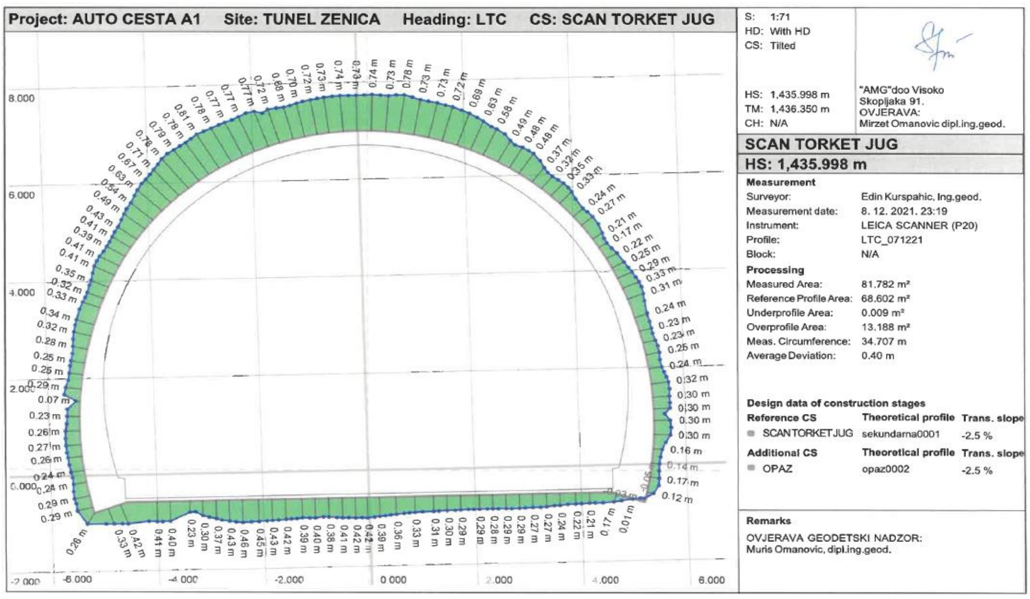


Figure 5. Geodetic survey after excavation by blasting in a weaker rock mass

After obtaining very poor results of the tunnel excavation by blasting, the contractor decided to perform the tunnel excavation in weaker rock masses (defined by RMR from 32 to 43) by machines. The following figure (Figure 6) shows the full cross section excavation by machine in LTT from chainage 1+518.00 to 1+520.00, while the figure (Figure 7) provides a geodetic survey of the face in LTT after the excavation by machine from chainage 1+518.00 in a length of 2.0 meters.

Figure 6. Machine excavation of full cross section in LTT at chainage 1+518.00 to 1+520

Figure 7. Geodetic survey of the face in LTT after machine excavation

As a result of this method of excavation, the contractor reduced the overbreak from 8-13 m^2 to the overbreak of 5-6 m^2, however, due to the structural geological properties of the rock mass, the hammer tunnel excavation still caused a large overbreak.
Any tunnel excavation that does not have an approximately regular shape, as well as an overbreak larger than 3 m^2, is not acceptable from an economic point of view, which is why the authors of this paper undertook a thorough analysis for determining all the necessary drilling and blasting parameters in a weaker rock mass.

Determining the coefficient of rock mass strength (f)
In order to calculate the approximate drilling and blasting parameters, it is necessary to determine the coefficient of rock mass strength (f), which is the basis for calculating the number of blast holes on the tunnel cross section (pcs), as well as the specific consumption of explosives (kg/m^3). The coefficient of rock mass strength is determined by Baron's equation:

$$f = \frac{\sigma_p}{300} + \sqrt{\frac{\sigma_p}{30}}$$

The determined RMR by the Contractor's geologists and Supervision [8] from chainage 1+499.10 to chainage 1+536.00 is 41, and based on this, the uniaxial compressive strength is adopted, taking into account all other geomechanical parameters for the calculation of 40 MPa so that we can calculate the coefficient of rock mass strength for the specified part of the tunnel.

$$f = \frac{\sigma_p}{300} + \sqrt{\frac{\sigma_p}{30}} = \frac{400}{300} + \sqrt{\frac{400}{30}} = 4{,}98 \approx 5$$

Determining the specific consumption of explosives in excavations with one free surface (q)
In the construction of road, railway and hydraulic tunnels (with larger cross-sections > 25.0 m^2), the specific consumption of explosives is determined based on the cross-sectional area of the room, the number of free excavation surfaces, the drilling diameter of blast holes, the characteristics of the explosives and the characteristics of the rock mass. To calculate the specific consumption of explosives, one can use one of the most common formulas for determining this blasting parameter, the F. Lares formula, which takes into account the type of rock mass (its physical and mechanical characteristics), but also the type of explosive used, which is:

$$q = q_1 \cdot v \cdot s \cdot \frac{e}{g} \cdot d \cdot k \left(\frac{kg}{m^3} \right)$$

Where is:

$$q_1 = \frac{\sigma_p}{2000}$$

σp- is the compressive strength of the rock mass in which blasting is carried out, 40MPa=400bar;
v - blast confinement coefficient (for one free surface - tunnel face) 2.5;
s - coefficient of rock mass structure complex for conditions of massive homogeneous structure 1.0;
g - explosive charge compactness coefficient (for plastic explosives) 1.0;
d - blast stemming coefficient (for holes that are not well plugged) 0.8;
k - correction coefficient (for the corresponding hole diameter to explosive diameter ratio) 1.0;

e - coefficient of relative strength of explosive and is calculated using the following formula:

$$e = \frac{A}{A_x}$$

The coefficient of rock strength for the given conditions is:

$$q_1 = \frac{\sigma_p}{2000} = \frac{400}{2000} = 0{,}20$$

The coefficient of relative strength of explosive is:

$$e = \frac{A}{A_x} = \frac{480}{390} = 1{,}23 \approx 1{,}25$$

Using the Lares formula and substituting the calculated coefficient values in it, we obtain the specific consumption of explosives:

$$q = q_1 \cdot v \cdot s \cdot \frac{e}{g} \cdot d \cdot k = 0{,}20 \cdot 2{,}5 \cdot 1{,}0 \cdot \frac{1{,}25}{1{,}0} \cdot 0{,}8 \cdot 1{,}0 = 0{,}50 \left(\frac{kg}{m^3} \right)$$

To ensure the positive effect of blasting, the specific consumption of explosives obtained by calculation should be increased by approximately 10%, so that the specific consumption of explosives of 0.55 (kg/m^3) is adopted for further calculation.

Determining the number of blast holes for excavation of full cross section (Nb)
There are a number of empirical formulas and expressions of individual authors (Protođakonov, Sieberg, Š. I. Ibrajev) for the approximate determination of the number of blast holes. The total required number of blast holes, in the presence of one free surface, depends on the type of explosive used, the size of the tunnel cross section and the coefficient of rock strength, and is divided into two different calculations:

$$N_b = N_p + N_k (buš.)$$

Where:

Np – is the number of auxiliary blast holes (holes);
Nk – number of contour blast holes (holes). (Note: buš. = borehole)

Determining the number of auxiliary blast holes (Np)
The number of auxiliary blast holes can be determined on the basis of several relations, where it is best to take one that is a function of the strength of the rock being blasted and the area of the cross section being excavated; one of such relations is as follows:

$$N_p = 0{,}27 \cdot F_i \cdot \sqrt{\frac{10 \cdot \sigma}{F_i}}$$

Where:

Fi - is the area of the cross-section being excavated (77 m^2 without shallow foundations);
σ - compressive strength of the rock massive in which blasting is carried out (40 MPa).

$$N_p = 0{,}27 \cdot F_i \cdot \sqrt{\frac{10 \cdot \sigma}{F_i}} = 0{,}27 \cdot 77 \cdot \sqrt{\frac{10 \cdot 40}{77}} = 47{,}39 \approx 48\,(bu\check{s}.)$$

Determining the number of contour blast holes (Nk)
In order to determine the number of contour blast holes, first of all we must determine the distance between the contour blast holes.
The distance between contour charges (in the tunnel crown and walls) can be expressed based on the following relation:

$$E = (12 \div 15) \cdot d\,(m')$$

Where:

d – is the blast hole diameter 0.045 m'.

$$E = (12 \div 15) \cdot d = (12 \div 15) \cdot 0{,}045 = 0{,}55 \div 0{,}65(m')$$

Considering that this is a case of weaker rock strength, the mean value is adopted: E=0.60 m′
The number of contour holes depends on the circumference of the room without the floor (Po) and the distance between the contour blast holes and is calculated according to the formula:

$$N_k = \frac{P_o}{E}\,(bu\check{s}.)$$

Both data are known, so that:

$$N_k = \frac{P_o}{E} = \frac{21}{0{,}6} = 35(bu\check{s}.)$$

So, when blasting in full cross section, the total number of blast holes is:

$$N_b = N_p + N_k = 48 + 35 = 83(bu\check{s}.)$$

Based on the calculated distance between contour blast holes, we determine the line of least resistance for contour blast holes (W), according to the following relation:

$$W = \frac{E}{m}\,(m')$$

Where:

E – is the distance between contour holes 0.60 m';
m – convergence coefficient of blast holes, which for weaker rocks is on average 0.8.

$$W = \frac{E}{m} = \frac{0{,}6}{0{,}8} = 0{,}75 \approx 0{,}80(m')$$

Determining the geometry of the blast field (a, b)

The distance of the contour blast holes is known, and so is the distance of the floor blast holes, while the spacing between the contour blast holes and the first row of separation -auxiliary blast holes is determined for harder rocks based on the following relation:

$$V_p = W + (0{,}2 \div 0{,}3)(m')$$

W - is the line of least resistance in our case is 0.8 m', so we obtain:

$$V_p = W + (0{,}2 \div 0{,}3) = 0{,}8 + (0{,}2 \div 0{,}3) = 1{,}0 \div 1{,}1(m')$$

The distance between auxiliary blast holes is equal to the line of least resistance of auxiliary mine holes and is calculated according to the formula:

$$b = k_z \cdot W$$

Where:

kz - is the blast hole convergence coefficient, for our case we adopt 0.95;

W - line of least resistance of auxiliary blast holes (m').

The line of least resistance for auxiliary blast holes can be determined based on the following formula:

$$W = d \cdot \sqrt{\frac{7{,}85 \cdot \rho \cdot k_p}{q \cdot k_z}}(m')$$

Where:

d - is the drilling diameter, in our case it is 45 mm, i.e. 0.45 dm;

ρ - bulk density of the explosive 1.45 kg/dm^3;

q - specific consumption of explosives 0.55 kg/m^3;

kz - hole convergence coefficient is 1.0;

kp - coefficient of filling of the blast hole cross section and is calculated using the following formula:

$$k_p = \frac{d_1^2}{d^2}$$

Where:

d1 - is the diameter of explosive cartridge 38 mm;

d - is the blast hole diameter 45 mm.

$$k_p = \frac{d_1^2}{d^2} = \frac{38^2}{45^2} = 0{,}72$$

So we can calculate the line of least resistance for auxiliary blast holes:

$$W = d \cdot \sqrt{\frac{7{,}85 \cdot \rho \cdot k_p}{q \cdot k_z}} = 0{,}45 \cdot \sqrt{\frac{7{,}85 \cdot 1{,}45 \cdot 0{,}72}{0{,}55 \cdot 1{,}0}} = 0{,}45 \cdot \sqrt{14{,}9} \approx 1{,}74(m')$$

This means that the distance between auxiliary blast holes is equal to:

$$b = k_z \cdot W = 0{,}95 \cdot 1{,}74 = 1{,}65(m')$$

We adopt a distance between auxiliary blast holes of 1.60 to 1.70 (m'). The quantity of explosive for one blasting is calculated according to the formula:

$$Q = q \cdot S \cdot l (kg)$$

Where:

q - is the specific consumption of explosives 0.55 kg/m^3;
S - area of the excavated cross section, in our case it is approximately 77 m^2;
I - length of the blast hole approximately 2.0 m'.

So we obtain the consumption of explosives per blasting for an advancement step of 2.0 m' in the amount of:

$$Q = q \cdot S \cdot l = 0{,}55 \cdot 77 \cdot 2{,}0 = 84{,}70 (kg)$$

On the basis of all the defined and calculated parameters, the layout scheme of blast holes on the tunnel face is made, as well as the layout of series of non-electric detonators by holes, which is shown below.

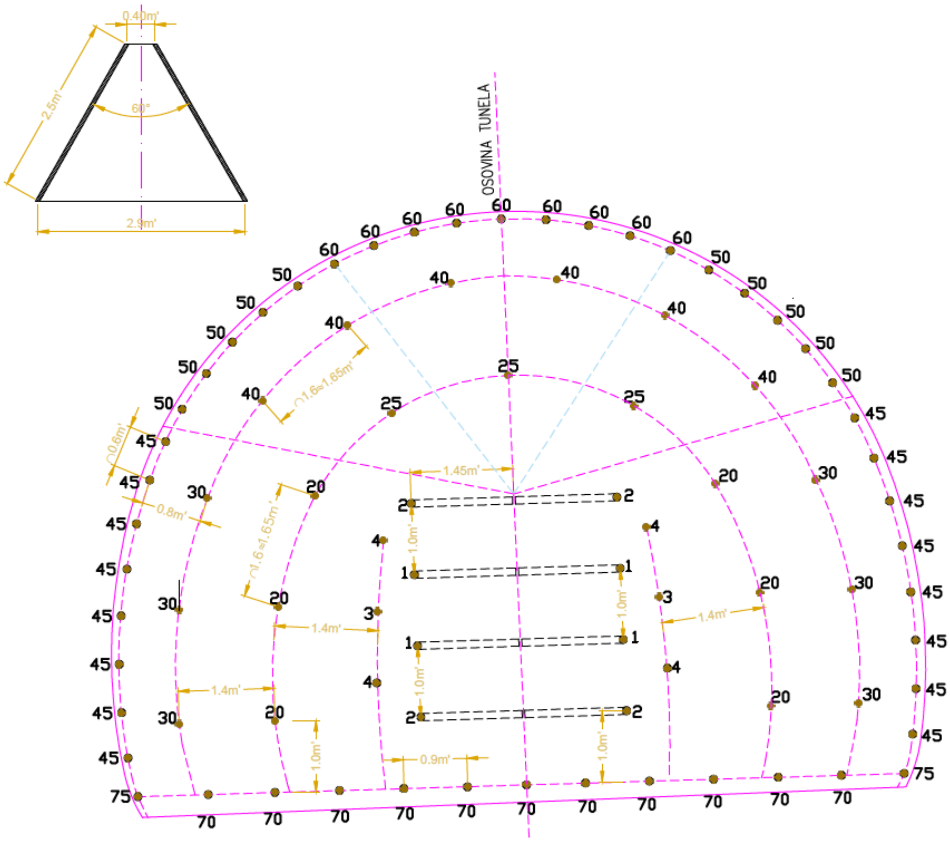


Figure 8. New layout scheme of blast holes and detonators on the tunnel face

The layout of the series of non-electric detonators is given on the sketch with numbers 1-75. The connection of non-electric detonators into a grid is carried out using a C-10 or C-12 detonating fuse for each detonator separately. The connection is performed by connecting the C-10 (C-12) detonating fuse to each conductor line of the non-electric detonator individually and attaching it to the detonator line with a special plastic clip that is manufactured at the factory and delivered on each of the detonator conductors.

The detonating fuse is taken out of the blast field to a shorter distance of 2 to 3 m' from the work face, where it is attached with insulating tape to the electric detonator, which is further connected to the main line of the blast field grid, which leads to the blasting machine.

Table 2. Overview of quantities, types of explosives and non-electric detonators by individual boreholes for one step of advancement, with the depth of blast holes of 2.0 m'

OVERVIEW OF THE QUANTITIES OF EXPLOSIVES BY BLAST HOLES													
Series of non-electric detonators used	1	2	3	4	20	25	30	40	45	50	60	70	75
Number of holes of the series	4	4	2	4	6	3	6	6	16	10	9	11	2
Total number of holes	83.00												
Σ expl. per hole (kg/hole)	1.61	1.60	1.60	1.60	1.60	1.44	1.60	1.43	0.256	0.256	0.256	1.60	1.80
Σ expl. for one blasting (kg)	84.70												

The basic technical characteristics of the initiation means and explosives used during the excavation of the left tunnel tube of the Zenica tunnel in the part considered in this paper are given below.

Table 3. Overview of non-electric detonators used

The serial non-electric detonators used marked on the layout scheme of blast holes	1	2	3	4	20	25	30	40	45	50	60	70	75
Delay (ms)	100	200	300	400	2000	2500	3000	4000	4500	5000	6000	7000	7500

Table 4. Overview of basic characteristics of explosives used

Characteristics of explosives used	Plastic explosive	Contour explosive
Density (kg/dm³)	1.20	1.10
Explosion energy (KJ/kg)	4850.00	4850.00
Gas volume (l/kg)	921.00	921.00
Cartridge diameter (mm)	38.00	27.00
Cartridge length (mm)	400.00	230.00
Cartridge weight (kg)	0.475	0.128

By defining the scheme of blast holes (Figure 8) with the layout of detonators by holes on the cross section for excavation of the Zenica tunnel in the weaker rock mass RMR (33+43), all the obtained drilling and blasting parameters at chainage 1+522.08 were applied, which is shown in figures below. The photo (Figure 9) shows the face of the LTT at chainage 1+522.08 after drilling all blast holes according to the defined scheme (Figure 8). After all blast holes at the specified chainage were drilled, filling and blasting were carried out while adhering to all defined technical parameters (Figure 8 and Tables 2 and 3). Upon completion of blasting and ventilation of the site, loading and removal of blasted material was carried out, as well as processing of the excavation profile (Figure 10) and geodetic verification of the excavation line in LTT at chainage 1+524.088 (Figure 11).

Figure 9. LTT face cross section at chainage 1+522.08 after completion of drilling of all blast holes according to the defined scheme

Figure 10. LTT face cross section at chainage 1+524.088 after blasting and treatment of the cross section

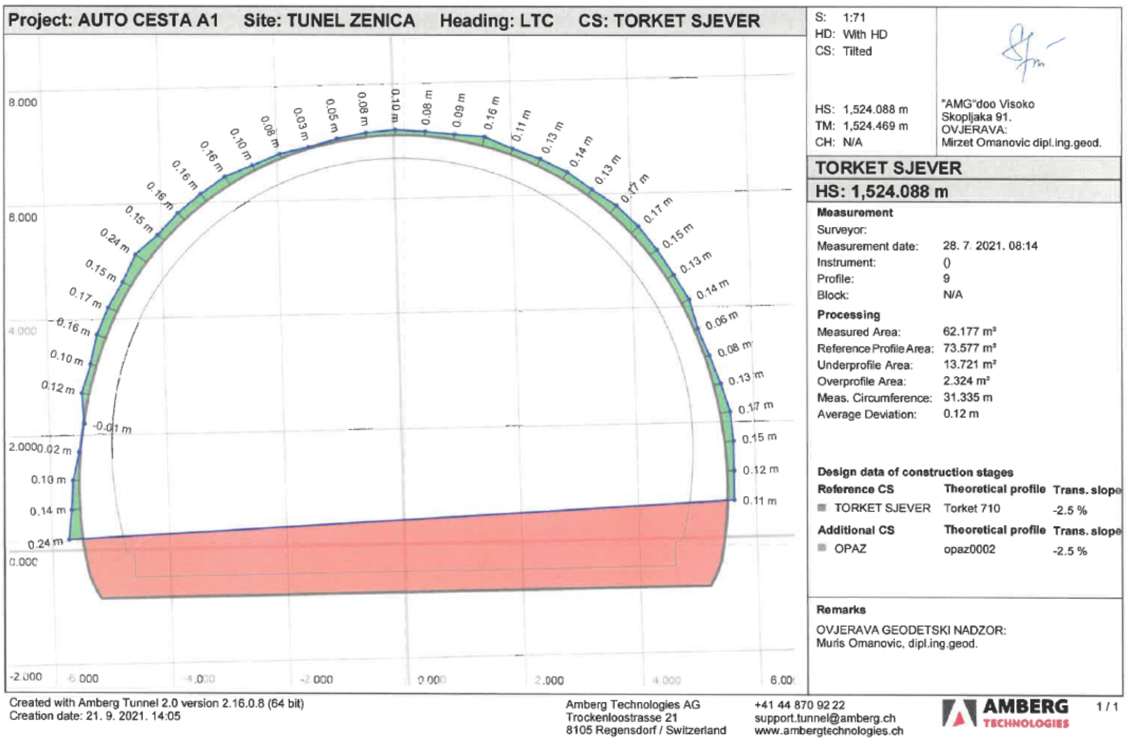


Figure 11. Geodetic survey of the face in LTT at chainage 1+524.088 after the completion of the excavation by blasting according to the defined scheme

CONCLUSION

The behavior of the underground excavation contour is primarily controlled by the structural geological characteristics of the rock mass. In carbonate rocks, the general stability of the excavation contour is reduced to the occurrence of local instabilities in the form of falling out of blocks due to the loss of shear strength of discontinuities. The depth of the zone damaged by blasting has the greatest influence on the initiation of local instability. If excessive, it can lead to progressive local failure, i.e. endangering the stability of the entire underground opening. In order to confirm or correct the drilling and blasting parameters defined by the supplementary mining project, it is customary to perform a test blasting in each category of rock mass that is planned for blasting in the project.

The optimal method of tunnel excavation, which ensured an approximately correct shape of the excavation, and a smaller tunnel overbreak, was selected by meticulously analyzing all available technical and technological parameters in the Zenica tunnel.

When excavating certain sections of the Zenica tunnel by machine, the shape of the excavation profile as well as the overbreak area largely depended on the structural geological characteristics of the rock mass and the skills of the worker operating the excavation machine, which directly reflected on the economic viability of the tunnel excavation.

Also, the excavation of certain sections of the tunnel by the method of drilling and blasting in a weaker rock mass due to the non-adjustment of all drilling and blasting parameters to the geological conditions in which the tunnel was excavated resulted in a negative economic viability in relation to the tunnel excavation in the same conditions using the method of mechanical excavation.

By choosing the optimal excavation method, the damage zone, which is inevitable during blasting excavation, was minimized in the sense that the strength and stiffness of the rock mass around the excavation contour were minimally reduced, which resulted in the preservation of the bearing capacity of the rock mass as the most important support element. Such an approach reduced the quantities of support set elements and the construction costs of the Zenica tunnel.

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