

Article

Rock Mass Bolting in Coal Mine Workings for Floor Heave Control

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Abstract

This article addresses the problem of floor heave in preparatory mine workings of the Karaganda Coal Basin and proposes technological solutions based on rock mass anchoring systems. Long-term field observations, supported by analytical interpretation and numerical modeling, made it possible to identify the principal factors controlling deformation processes around mine roadways, including floor rock fracturing, stress redistribution during longwall mining, and the interaction between adjacent workings. Special attention is given to the performance of different types of anchors installed in the floor and to their optimal arrangement depending on the width and geometry of the excavation cross-section. Empirical relationships are proposed for estimating the magnitude of floor heave and determining the required anchor length, and their applicability is supported by numerical simulations performed using RS2 software v. 11.027. Based on the results obtained, technological schemes aimed at reducing the intensity of floor heave and improving the stability of the surrounding rock mass were developed for mining in inclined and gently dipping coal seams. Field experiments conducted in the Kazakhstanskaya, Saranskaya, and Abayskaya mines of the Karaganda Coal Basin confirmed the effectiveness of steel–polymer anchors in stabilizing excavation contours.

Keywords: mine workings; floor heave; support schemes; geomechanical processes; floor anchoring; rock pressure manifestations



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1. Introduction

Despite the significant progress achieved in underground coal mining, the problem of floor heave in preparatory workings and gate roads adjacent to longwall panels remains one of the most persistent geomechanical challenges. Floor heave leads to a progressive loss of excavation stability, deformation or failure of support systems, reduction in the effective cross-sectional area, and an increased risk of accidents, ultimately complicating mining operations and substantially increasing maintenance costs. The multifactorial nature of floor heave—associated with stress redistribution, lithological heterogeneity, water saturation, and mining-induced disturbances—together with the limited efficiency of conventional support systems, necessitates the development of advanced and adaptable reinforcement technologies capable of ensuring the long-term stability of mine workings under complex geological and mining conditions.

Recent studies by domestic and foreign researchers confirm the relevance and severity of this issue. For instance, Huang et al. [1] developed mechanical models of floor heave in preparatory workings of thin coal seams, demonstrating its considerable impact on excavation stability. Fan et al. [2] extended these approaches by developing predictive models of floor heave under high-stress conditions. While these studies provide valuable insight into deformation mechanisms, they are largely based on idealized assumptions and are often not supported by systematic in situ verification under real mining conditions.

In terms of control methods, including driving headings in hard rock under high rock mass pressure, Li et al. [3] proposed a floor slotting and pressure-relief technology that localizes and reduces heave intensity. Moreover, Xue et al. [4] highlighted that dynamic ground pressure and stress redistribution play a key role in excavation instability, emphasizing the need for systematic monitoring and adaptive reinforcement. However, unloading-based methods are often characterized by a localized and time-limited effect and may complicate roadway maintenance and operational logistics, particularly in deep or water-bearing conditions.

For not very high rock mass pressure, Wei et al. [5] demonstrated the effectiveness of rock bolting and injection reinforcement in mitigating asymmetric deformations caused by roof collapse. Shang et al. [6] emphasized the prospects of integrated technologies combining unloading measures with bolt-injection reinforcement for combating heave in deep mine workings. Li et al. [7] analyzed cases in deep workings and confirmed the high efficiency of combined reinforcement technologies. In weak rocks in coal mines, Sakhno et al. [8] analyzed heading drivage through coal pillars and demonstrated the effectiveness of anti-shear pile application in limiting rib and floor deformations. Nevertheless, most of these studies focus primarily on roof and sidewall stability, while the floor is often treated as a secondary element of the support system.

In Central Asia, particularly within the Karaganda Coal Basin, the floor heaving problem is especially acute, as intensive coal extraction has revealed the inadequacy of conventional support systems. The main methodological framework of floor heave studies relies on standards such as GOST 28985-91 [9], which allow the determination of deformation characteristics of rocks under uniaxial compression, thus providing reliable criteria for assessing stability. However, recent geomechanical studies underline that such approaches are insufficient for capturing heterogeneous deformation processes developing in the floor, especially under conditions of water saturation and mining-induced stress redistribution [10]. As underground conditions can change considerably, Akhmatnurov et al. [11] proposed optimized support schemes that limit floor uplift based on deformation monitoring data.

Therefore, He et al. [12] demonstrated the application of advanced numerical modeling and field-testing methods to assess the effectiveness of bolt-frame and injection reinforcement systems in weak, water-bearing rocks. Małkowski et al. [13] simulated heaving processes under both dry and water-saturated conditions, highlighting the decisive influence of groundwater on the final magnitude of floor uplift and progressive rock degradation. Zhu et al. [14] and Yang et al. [15] confirmed that the most unfavorable stability conditions occur in weakly cemented or water-saturated rocks, where traditional support systems fail to prevent intensive floor heave.

Despite these advances, a clear research gap remains. The majority of published studies focus either on numerical simulations without comprehensive field validation or on isolated reinforcement techniques without long-term monitoring of their effectiveness, particularly with respect to floor anchoring as an integral element of the support system.

Peng et al. [16] confirmed that in complex mining and geological conditions, only comprehensive measures—combining unloading, strengthening, and continuous monitoring—can ensure long-term control of floor heave.

Therefore, this study aims to address the identified research gap by developing and validating integrated technological solutions for floor heave control in preparatory mine workings of the Karaganda Coal Basin. The research combines in situ observations, deformation monitoring, laboratory testing, and numerical modeling to assess the effectiveness of various anchoring systems installed in the floor and sidewalls of mine roadways. The scientific novelty of this work lies in the field-based substantiation of floor anchoring technologies, the establishment of empirical relationships for estimating floor heave magnitude and anchor length, and the demonstration of their applicability under real mining and geological conditions.

Recent studies have increasingly focused on rock fracture and damage mechanisms under multi-field thermo–hydro–mechanical–chemical (THMC) coupling, demonstrating that the interaction of thermal shocks, fluid effects, mechanical loading, and physico-chemical processes plays a decisive role in crack initiation, propagation, and long-term mechanical degradation of rock masses [17,18]. Experimental investigations have shown that cyclic thermal shocks combined with liquid-nitrogen cooling significantly accelerate fracture evolution in granite, leading to pronounced reductions in strength and fracture toughness and being accompanied by characteristic acoustic-emission responses [19]. These results indicate that multi-field coupling effects should be considered when developing theoretical backgrounds and engineering approaches for assessing rock mass stability under complex loading and environmental conditions.

2. Phenomena of Floor Heave and the Main Mining-Engineering Factors Influencing Its Manifestation

After the excavation is driven, the rock mass in close proximity to the mined-out space becomes weakened due to the formation of fracturing and the development of plastic deformations, which leads to the creation of zones of inelastic behavior around the contour. Within these zones, stress redistribution occurs, and the area of stress concentration shifts deeper into the rock mass. This process is associated with the exposure of the roof, and the extent of the damaged zone increases as the width of the excavation grows [20]. The rocks located several dozen centimeters away from the excavation boundary are under triaxial compression, whereas the rocks immediately adjacent to the excavation contour along its perimeter experience a biaxial stress state. The absence of confinement in the direction perpendicular to the excavation walls causes lateral displacement, the magnitude of which increases as the strength of the side rocks decreases [21]. With an increase in the excavation size, the shear (tangential) component of the stress tensor grows, leading to higher boundary displacements [22].

Glass fiber-reinforced polymer (GFRP) anchors are an effective solution for reinforcing the floor of mine roadways, especially under conditions of floor heave and water-saturated rocks, and when subsequent floor trimming is required. Their corrosion resistance, explosion safety, and ability to perform under significant deformations make them preferable to conventional steel anchors in such conditions. At the same time, steel anchors retain advantages in zones subjected to high loads, where increased stiffness and load-bearing capacity of the support system are required. Table 1 presents a comparative assessment of these anchoring systems.

Table 1. Comparative characteristics of fiberglass and metal anchors.

Parameter	Fiberglass Anchors (GFRP)	Metal Anchors (Steel)
Material	Fiberglass on a polymer matrix (epoxy or polyester)	Carbon or alloy steel
Tensile strength	600–1200 MPa (high specific rate)	400–700 MPa
Modulus of elasticity	Low–medium (35–60 GPa)	High (190–210 GPa)
Working with pattern Deformations	Works well in conditions of large deformations, reduces stress concentration	Hard work, stress concentration in the contour zone is possible
Load-bearing capacity (fully fixed anchor)	120–180 kN (comparable to steel anchors)	150–250 kN (depending on diameter and steel)
Corrosion resistance	High, non-corrosive	Low–medium, requires protection in aggressive and flooded conditions
Behavior in water-saturated rocks	Stable, no reduction in bearing capacity	Corrosion and loss of cross-section are possible
Weight of the anchor	Low (2–3 times lighter than steel)	High
Manufacturability of installation	Easy installation, less feed and locking forces	Requires more force, especially with long anchors
Can be cut in soil sampling	Yes (can be cut or destroyed without sparking)	No (requires metal cutting, increased danger)
Explosion safety	High (no sparks at breakage)	Lower (risk of sparking)
Electrical conductivity	Dielectric	Electrically conductive material
Compatible with polymer resins	Very high	High
Rational application area	Soil of workings, heaving zones, watered and weak rocks, areas with the need for subsequent pruning	Roof, sides of workings, areas of high loads and harsh conditions
Key benefits	Corrosion resistance, safety, deformation adaptability	High rigidity and load-bearing capacity
Main limitations	Less stiffness, limited impact performance	Corrosion, difficulty of dismantling, danger during pruning

Observations of roadway behavior indicate that the main mining–technical factor influencing their stability is the depth of mining operations [23], so the first natural reason of the floor heave is the general rock mass stress. The second is the compressive strength of rocks in sidewalls and the floor. If sidewalls are stiff and strong, they act as “loading pistons” and extrude floor rocks inside the roadway. The high strength of the sidewall rocks makes them stable and only slightly fractured, and the stress concentration is located several dozen centimeters from the excavation contour. Therefore, two cases can be considered: (1) when the compressive strength of the sidewall rocks is greater than the vertical stresses in the rock mass σ_z (Figure 1) or (2) when this strength is lower than the stresses occurring in the rock mass (Figure 2).

In the first case, if the compressive strength of the sidewall rocks is high, they press against the underlying rocks, which, even if strong, are subject to destruction, and the extent of the destruction is at the edge of the vertical stress (purple arrows) concentration (Figure 1a). If these rocks are weak, the floor is intensively damaged, especially in the case of a thin, stratified immediate floor, and the extent of the destruction is greater than the extent of the stress concentration zone in the sidewalls. Moreover, the next bottom layer may crack, hence the intensity of uplift is usually high (Figure 1b).

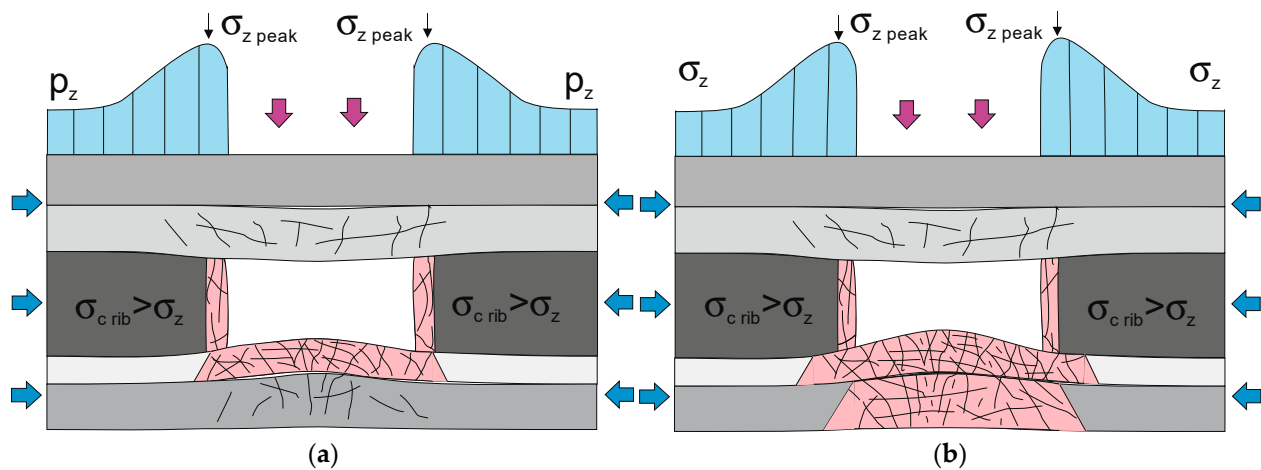


Figure 1. Stress distribution and floor heaving around mine working if sidewall compressive strength is higher than rock mass stress; (a) hard floor rocks, (b) weak floor rocks.

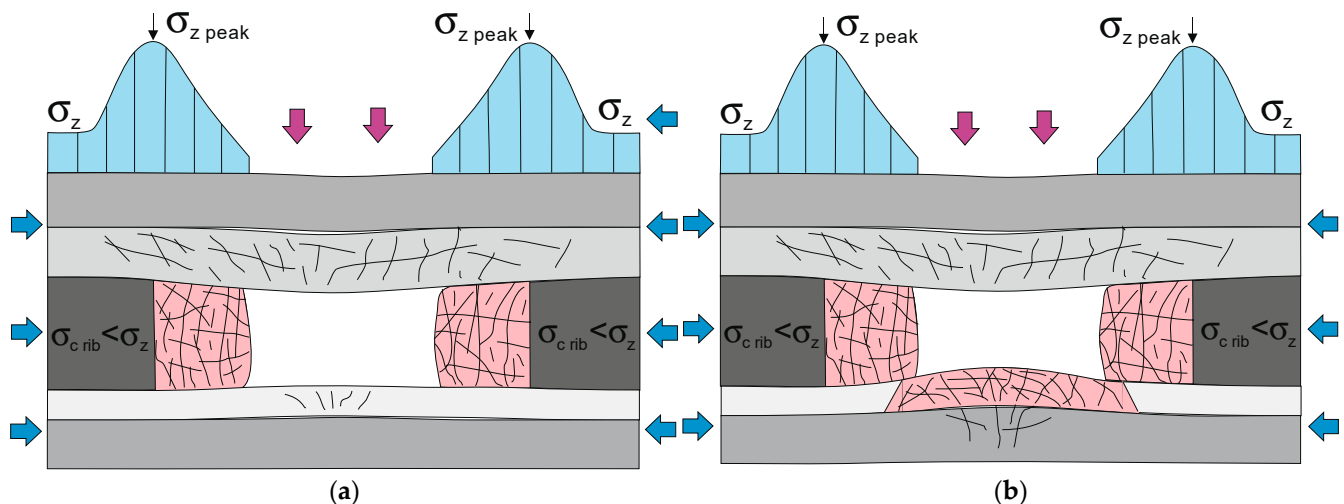


Figure 2. Stress distribution and floor heaving around mine working if sidewall compressive strength is lower than rock mass stress; (a) hard floor rocks, (b) weak floor rocks.

In the second case, if the compressive strength of the sidewall rocks is low, they are destroyed even up to several meters from the excavation. If the sidewall rocks are strong, the floor rocks may not be destroyed at all, and only slightly uplifted (Figure 2a). If the rocks are weak, the floor is destroyed, especially if it is layered and the immediate floor layer is thin, and shearing occurs close to the corners of the sidewalls (Figure 2b). In every case, the floor uplift effect can be more visible if there is high horizontal stress (blue arrows) or/and if rocks are waterlogged, which can markedly reduce the mechanical parameters of the rock mass, in particular clay rocks [24].

A preliminary assessment of the depth of the zone affected by floor heave, depending on the bulk density of floor and side rocks (t/m^3) and the excavation height (m), can be made using Tsimbarevich's nomogram (Figure 3) [25]. A general pattern has been established: an increase in the bulk density of roof rocks reduces the depth of the heaving zone, whereas an increase in the bulk density of side rocks and the height of the excavation leads to a greater depth of the heave-affected zone. The depth of the damaged zone may reach up to 1.8 m; however, the nomogram accounts for excavation heights only up to 4.0 m and does not consider mining depth. It should also be noted that the nomogram ignores rock mass heterogeneity, treating it as a uniform layer with averaged geomechanical

parameters, which is inadequate for deep coal mines where the side rocks often exhibit reduced strength.

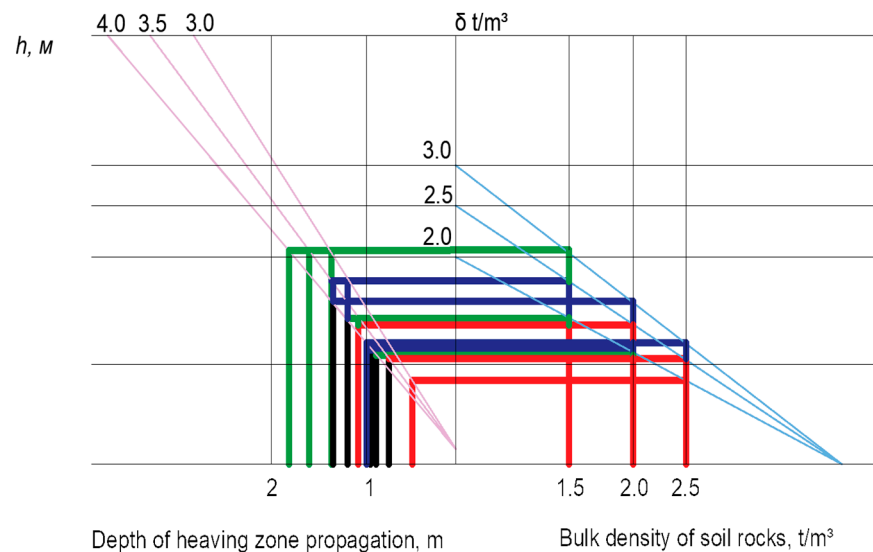


Figure 3. Nomogram for determining the depth of the floor heave zone in mine workings depending on influencing factors.

Under conditions of the Karaganda Coal Basin, floor heave is most intensively manifested in preparatory mine workings driven along coal seams. The thickness of the floor strata subject to heaving varies from 2 to 5 m, and the greater the thickness of weak layers, the more intense the heaving process becomes. However, two additional factors can significantly increase the stress state of the rock mass and, consequently, intensify floor heave. The first factor is the influence of the longwall face, which creates high abutment pressure; the second is the effect of a closely spaced (parallel) roadway.

At Karaganda coal mines, in roadways located within the influence zone of the longwall face typically at a distance of 80–100 m, the intensity of floor heave caused by abutment pressure increases and reaches its maximum as the face approaches. When the distance exceeds 100–120 m, the heave intensity decreases significantly and generally remains stable. Even though this effect is observed in other coal mines in the world, the distance from the longwall face where effects of abutment pressure are observed is different, and always specific for the local mining and geological conditions, but first of all, it depends on the mining depth. In Polish coal mines, at the depth of ca. 1000 m, it is ca. 110–120 m in front of the longwall face [26], while in Indian coal mines, depths of 400–500 m starts only at 70–80 m in front the longwall face [27]. In roadways driven in undisturbed areas, with neither mining nor geological disturbance of a rock mass occur, the greatest heave is observed immediately after drivage and gradually decreases over time (Figure 4).

Regarding a parallel roadway driven in close proximity to the first one, the determining parameter of heave magnitude is the size of the coal pillar left between them: the smaller the pillar, the higher the intensity of heave, especially if the pillar rocks are of high strength. Increasing the width of pillars and protecting preparatory roadways by leaving unmined coal (protective strip mining) reduces the intensity of heaving, the duration of which usually ranges from 1 to 3 months. This pattern is widely applied when driving wide roadways with single-sided or double-sided slots [28]. Hence, in cases of pronounced and intense floor heave manifestations, active preventive methods are required. The floor uplift observations in time in roadways driven in various mining conditions in Karaganda coal mines are shown in Figure 4.

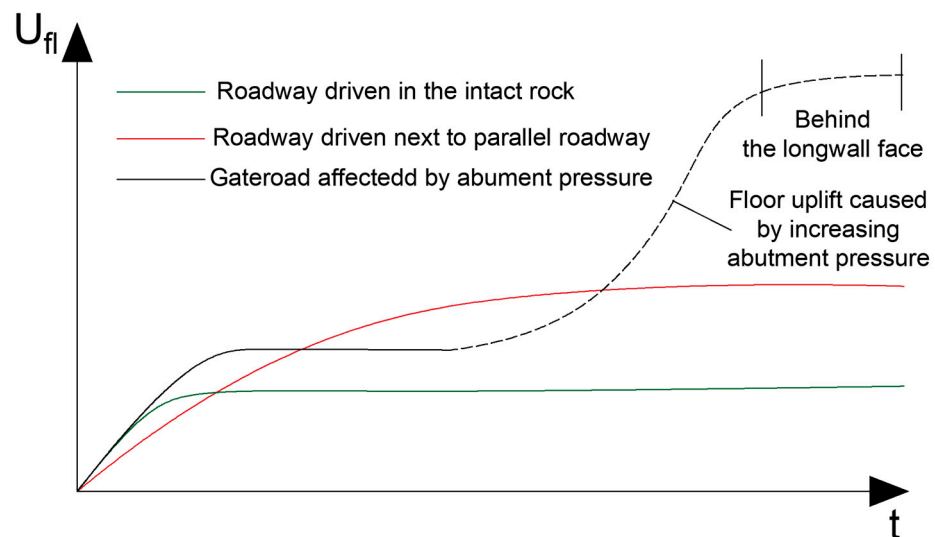


Figure 4. The floor uplift observations in time in roadways driven in various mining conditions.

In general, in preparatory workings of the Karaganda Coal Basin, where interbedded argillites and siltstones occur in the floor, three stages of deformation can be distinguished:

1. Increase in vertical stress at the edges of the sidewalls and elastic response of the floor and roof rocks—stratification along bedding without destruction of the detached rock;
2. Development of cracks in the sidewalls caused by vertical stress, displacement of the peak stress deeper into the sidewalls, initiation of floor heave, and shear at a certain depth—destruction of the stratified rock beneath the excavation into blocks with the formation of multi-hinged arches;
3. Fracturing of the sidewall rocks, further migration of the peak load deeper into the sidewalls, final uplift of the floor, and deeper shear beneath the sidewalls along the line of maximum stress—failure of the floor rocks beneath the excavation sides, resulting in their extrusion into the mined-out space.

The classification criteria for floor heave are rather subjective due to the multifactorial nature of the phenomenon and are usually based on the intensity of its manifestation over time, divided into the categories “slight,” “moderate,” and “severe”, where “slight” uplift is usually considered to unload the floor rocks (stage one) and perform “severe” uplift” (stage three). In the case of such a behavior of the floor, its strengthening during the first and second stage of uplift is technically possible. There is an intact rock during elastic displacements in stage one which allows either the rock mass to be strengthened or to destress; even dividing the floor rocks into blocks in stage two still gives the possibility of strengthening them.

To ensure a consistent linkage between the conceptual mechanical models of floor heave formation (Figures 1 and 2) and the field observations obtained from specific mine workings, the analyzed cases of the Karaganda Coal Basin were classified according to the dominant stress–strength interaction mechanisms.

The ventilation drift 334D6-1-v at the Kazakhstanskaya mine is characterized by high compressive strength and stiffness of the sidewall rocks combined with a weak, layered immediate floor composed predominantly of argillites. Under these conditions, the sidewalls remain mechanically stable and act as rigid stress concentrators, inducing extrusion and destruction of the weakened floor strata. This deformation pattern corresponds to the mechanical response shown in Figure 1b.

In the conveyor drift 52K7-z of the Saranskaya mine, both sidewall and floor rocks exhibit moderate to relatively high compressive strength, and the degree of sidewall fracturing

remains limited. The stress redistribution results in a minor floor uplift without extensive shear failure, which is consistent with the mechanical response illustrated in Figure 2a.

The conveyor drift 52K7-z of the Kostenko mine is driven at greater mining depths and under elevated rock mass stresses. In this case, the compressive strength of the sidewall rocks is comparable to or lower than the acting stresses, leading to their progressive destruction and to shear development in the floor near the sidewall corners. This behavior corresponds to the mechanism presented in Figure 2b.

The ventilation drift of the K10 seam at the Abayskaya mine is characterized by initially strong floor rocks that experienced a substantial reduction in strength due to water saturation. At the initial stage, the deformation behavior corresponds to the response shown in Figure 1a; however, with increasing water content and degradation of mechanical properties, the mechanism evolves toward a weakened-floor response analogous to Figure 1b.

Consequently, the four idealized models presented in Figures 1 and 2 represent generalized but practically realizable mechanical responses of the rock mass. Each model corresponds to a specific combination of sidewall strength, floor strength, and stress level, and collectively they provide a unified framework for interpreting the observed floor heave mechanisms and for selecting appropriate reinforcement strategies in the mine workings of the Karaganda Coal Basin.

3. Methods for Reducing Floor Heave in Mine Workings

Under varying geological and technical conditions and at different stages of a roadway's service life, the processes of floor heave develop differently, requiring the application of appropriate control methods. In general, two main approaches are distinguished: unloading of floor rocks and their reinforcement. As noted by Wang et al. [29], comprehensive measures to mitigate heave can be conventionally divided into three categories: avoidance–unloading–reinforcement. Since complete avoidance of floor heave is practically impossible, the latter two approaches are most commonly applied in practice.

Li et al. [7] and Fu et al. [30] demonstrated that during repeated mining of multiple coal seams, cyclic stress redistribution has a significant impact on floor deformation. Under such conditions, an effective unloading method can be floor slotting (deep grooves or cuts), which shifts the zone of maximum stresses deeper into the rock mass. This approach is especially applicable when layered surrounding rocks or mined-out spaces are present on one side of the roadway; however, in real underground conditions, it may complicate work organization.

One of the most widely used methods for reducing floor heave is localized blasting in the floor of the roadway. Guo and Xin [31] showed that blasting in the corner zones of a roadway disrupts the equilibrium of the support–fractured rock zone system, promotes stress redistribution, and decreases the magnitude of heave. Their studies confirmed that this method can reduce floor uplift by up to 30% and shift the peak horizontal stress deeper into the rock mass from 2.4 to 4.8 m. Additionally, the removal of heaved material after blasting was found to reduce the passive resistance of the floor by 50–60 kN per meter of roadway, which then requires compensating support loads by engineering means.

Modern compensation methods include rock bolting and injection reinforcement technologies. Chen et al. [32] emphasized that the use of high-strength steel and polymer bolts not only increases the bearing capacity of the roof and sidewalls but also reduces the specific pressure on the floor. In this case, the depth of maximum pressure shifts away from the excavation contour by a distance dependent on bolt length, inclination, and spacing. This factor is particularly critical under abutment pressure near the longwall face, where the risk of floor heave development is greatest.

In addition, Lai et al. [33] investigated the mechanisms of floor heave in roadways influenced by top coal caving and showed that asymmetric stress redistribution and progressive degradation of floor rocks play a decisive role in the development of large deformations. Their results confirm that conventional support systems are often insufficient under such conditions and require reinforcement schemes adapted to mining-induced stress evolution. Practical engineering solutions for preparatory mine workings supported by rock bolts were comprehensively summarized by Kamarov and Demin [34], who demonstrated that the effectiveness of floor and sidewall anchoring largely depends on the timely installation of support elements and their interaction with the surrounding rock mass during the early stages of deformation.

Furthermore, Yang et al. [35] analyzed floor heave mechanisms in soft rock roadways and emphasized the importance of combining unloading measures with reinforcement techniques to control long-term deformation processes. Their findings highlight that floor heave control should be treated as a coupled mechanical process involving stress redistribution, weakening of rock properties, and progressive failure of the surrounding mass.

4. Application of Anchoring Systems to Reduce Floor Heave in Mine Workings of Karaganda Coal Basin

The geological structure of the Karaganda Coal Basin is characterized by an alternation of coal seams with sedimentary sandy–clayey rocks. The structural setting is complicated by folding and faulting, which results in pronounced heterogeneity of the stress–strain state of the rock mass.

The roof of the mine workings is composed predominantly of mudstones and siltstones of low-to-medium strength, locally interbedded with fine-grained sandstones. Roof rocks are characterized by a layered texture, well-developed fracturing, and a pronounced tendency to bedding separation. The uniaxial compressive strength (UCS) of roof rocks typically ranges from 15 to 35 MPa for mudstones and 30 to 60 MPa for siltstones and sandstones.

The floor of the mine workings is mainly represented by sandstones and siltstones of medium-to-high strength; however, water-saturated and structurally disturbed zones are observed in several sections. Under such conditions, a reduction in bearing capacity and the development of floor heave are recorded. UCS values of floor rocks range from 40 to 80 MPa for dense sandstones and 25 to 45 MPa for siltstones, while in tectonically disturbed and highly moist zones, the strength characteristics decrease significantly.

The combination of a relatively weak, layered roof and a more rigid but locally weakened floor creates unfavorable conditions for the stability of mine workings and necessitates the application of reinforced support systems and floor anchoring.

Reinforcement of the rock mass around mine workings using bar and cable bolting systems must consider geological and technical conditions, as well as the interaction between bolts and the rock mass under given loads and deformation rates. The key factor is the redistribution of stresses in the roof and increasing frictional intensity in the floor when reinforcing the excavation contour with anchoring systems.

Numerical Modeling Approach

To substantiate the obtained results, numerical modeling of the stress–strain state of the rock mass surrounding the mine roadway was performed using the RS2 v. 11.027 finite element software). A two-dimensional plane strain formulation was adopted, which is appropriate for the analysis of long mine workings where deformations along the roadway axis are negligible compared to transverse displacements.

The numerical model represented a cross-section of the mine roadway together with the surrounding rock mass. The model boundaries were located at distances of at least five roadway widths from the excavation contour in all directions to minimize boundary effects. The excavation geometry (rectangular and arched cross-sections) corresponded to the actual roadway profiles used in underground conditions.

The rock mass behavior was described using an elastic–plastic constitutive model based on the Mohr–Coulomb failure criterion. This model was selected due to its widespread application in underground mining analyses and its ability to adequately represent plastic deformation and strength degradation of sedimentary rocks. The input parameters included density, Young’s modulus, Poisson’s ratio, cohesion, internal friction angle, and tensile strength. These parameters were determined from laboratory testing of rock samples in accordance with GOST 28985-91 [9], supplemented by in situ observations and published data. For water-saturated and structurally disturbed zones, reduced strength and stiffness parameters were applied to account for degradation effects.

The initial stress state was defined based on gravitational loading corresponding to the mining depth. Vertical stress was calculated from the weight of the overburden, while horizontal stresses were introduced using a lateral pressure coefficient consistent with geomechanical conditions of the Karaganda Coal Basin. The bottom boundary of the model was fixed in both vertical and horizontal directions, lateral boundaries were fixed horizontally and free vertically, and the upper boundary was left free.

Rock bolts and anchors were modeled using embedded structural elements with full bonding along their length, assuming complete load transfer between the anchor and the surrounding rock mass. The length, inclination, and spacing of anchors corresponded to the support schemes applied in field conditions. Excavation was simulated by removing the elements representing the roadway contour, followed by the installation of anchoring systems.

The modeling results were analyzed in terms of displacement fields, stress redistribution, and development of plastic deformation zones, with particular attention paid to vertical displacements in the floor and deformation of sidewalls. The numerical results showed good agreement with field monitoring data, with deviations not exceeding 15%, confirming the reliability of the adopted modeling approach.

Stress–strain state calculations using numerical modeling for various geological and technical conditions of coal seam extraction in the Karaganda Coal Basin have shown that the greatest reinforcing effect is achieved in rectangular cross-section workings with a combined support scheme for the roof and sidewalls. In this configuration, the installation of 2.4 m long bar anchors in the sidewalls and floor corners distributes the load in such a way that the upper side anchor (usually located deeper) lies in the high-stress zone beyond the excavation contour, within the rock mass. This arrangement shifts the peak of mining-induced pressure deeper into the rock mass—beyond the deformation (fractured) zone forming around the excavation. The lower side anchor creates an isolating (restrictive) zone that prevents the spread and extrusion of floor rocks into the mined-out space.

The numerical analysis of deformation distribution around the working (Figure 5) indicates that the reduction in deformation in the sidewalls and floor rocks is mainly ensured by side anchors rather than by anchors installed directly into the floor. Consequently, the acting deformations of the rock mass decrease, while the zone of maximum stresses shifts deeper into the intact rocks. This is of critical importance for ensuring the long-term stability of mine workings.

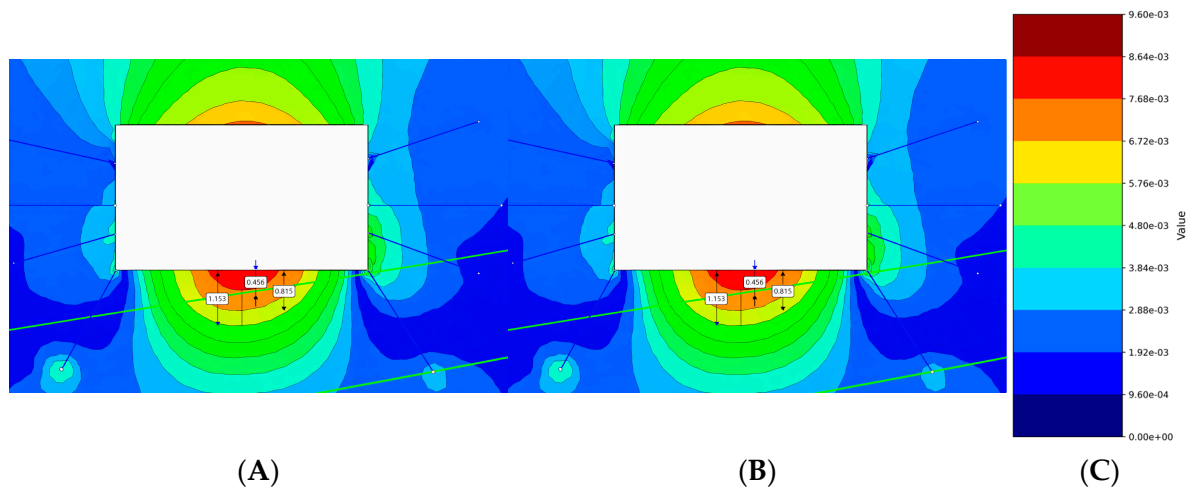


Figure 5. Distribution of deformations around the cross-section of a mine roadway reinforced with a 2.4 m long side and inclined floor bolts: (A) rectangular cross-section; (B) arched cross-section. (C) corresponding color scale. The color scale (generated in RS2, Rocscience Inc.) represents the calculated strain distribution within the numerical model and varies from 0.00×10^0 to 9.60×10^{-3} (dimensionless). The scale is divided into uniform intervals of 9.60×10^{-4} , where dark blue indicates minimum values and red corresponds to maximum values.

The research by Akhmaturov et al. [11] demonstrates that the use of modern reinforcement schemes, accounting for the stress–strain state of the rock mass (e.g., through numerical modeling) and verified by reliable laboratory testing of rock samples, significantly reduces floor heave intensity and enhances the operational stability of mine workings. Lai et al. [32] notes that the most effective method for reducing floor heave at depths greater than 700 m is the use of cable bolts, especially under asymmetric deformation conditions. Peng, L. [16] also recommends the use of cable bolts in areas affected by repeated extraction of multiple coal seams.

In the coal mines of the Karaganda Coal Basin, various technological schemes for reducing floor heave using rock bolts have been applied for many years, mainly in preparatory mine workings. On the basis of these experiences, the following general technology for floor heave reduction is recommended: during each excavation cycle, boreholes are drilled in rows in the roof and sidewalls. The length, diameter, and inclination angles of the boreholes are specified by the design support scheme and the dimensions of the holes in the steel straps. To unload the floor rocks, the boreholes for the floor bolts are reamed to a depth of 1.0–1.2 m, doubling the original diameter. This disrupts the integrity of the heaving layer and slows down (or prevents) the development of longitudinal and transverse bending of the layers. The enlargement of borehole diameter is considered in this study as a local technological unloading measure aimed at disrupting the integrity of the heaving layer and facilitating stress redistribution. Its effect was evaluated indirectly through field observations and deformation monitoring, while a detailed parametric numerical comparison of stress distributions for different borehole diameters was beyond the scope of the present work. As a result, the stratified floor rocks are relieved from horizontal stresses. To increase the load-bearing capacity and provide flexibility of the support elements, conical spacers are installed at the ends of the bolts. Considering the effect of abutment pressure ahead of the longwall face, the floor bolts are installed 1.5–2.0 times ahead of the abutment pressure zone.

The required thickness of the reinforced floor layer when using anchors installed with the formation of bearing blocks—serving as the foundation of the load-bearing arch of the roadway—can be estimated by Tsimbarevich’s method, presented in the form of a

nomogram in Figure 3. For example, under the conditions of the Yuzhnodonbasskaya No. 3 mine, in the section where the station of electric locomotives was located, steel cables were laid across the roadway and fixed in the floor. This created a distributed load of 30 kN per square meter and made it possible to reduce the magnitude of floor heave by 57%.

Based on the study of bedding separation processes, abutment pressure zones, and deformation zones, an empirical formula was developed to determine the required length of anchors installed in the floor of a mine roadway:

$$L = \frac{K_r \cdot B \cdot u_f}{\sigma_c}, m \quad (1)$$

where

K_r is the empirical coefficient (for the Karaganda coal basin equal to 6.75);

B is the width of the roadway in the clear, m;

u_f is the magnitude of floor heave, m;

σ_c is the compressive strength of floor rocks, MPa.

Equation (1) represents a dependence used to determine the required length of anchors installed in the floor of a mine roadway, taking into account the geometric parameters of the excavation and the intensity of floor heave. The anchor length L corresponds to the depth of anchorage beyond the zone of active floor deformation, within which shear and bending deformations of layered rocks develop as a result of stress redistribution induced by mining activities.

The empirical coefficient K_r is an integral parameter that accounts for the combined influence of geological and mining factors governing the development of floor heave. The value of K_r 6.75 was obtained through the analysis of field observations and instrumental monitoring data collected in preparatory roadways of the Karaganda Coal Basin. This coefficient reflects the effects of rock strength and deformability, degree of structural disturbance, water saturation, mining depth, and the redistribution of stresses in the zone influenced by longwall extraction.

On the basis of experience for mining and geological conditions of the Karaganda Coal Basin, the regularity of variation in the predicted floor heave magnitude (u_f) in relation to the ratio of bolt length (L) to roadway width (B) can be described by the following empirical equation:

$$u_f = \frac{1.22 - 1.5L}{B}, m \quad (2)$$

Equation (2) describes an empirical relationship between the predicted magnitude of floor heave u_f and the ratio of anchor length L to roadway width B . This relationship reflects the reinforcing effect of increasing anchorage depth, whereby the zone of maximum stresses and deformations is shifted into the deeper, more competent rock mass outside the excavation-disturbed zone, resulting in a reduction in floor rock extrusion into the roadway contour.

Equations (1) and (2) are applied within a unified iterative calculation framework, in which the required anchor length is first determined and then used to estimate the expected magnitude of floor heave for the selected reinforcement parameters. It should be emphasized that these relationships were developed and validated for the geological and mining conditions of the Karaganda Coal Basin. Their direct application to other mining regions requires prior recalibration of the coefficient K_r to account for local geomechanical, lithological, and hydrogeological conditions.

Equations (1) and (2) are not treated as independent relationships but are applied within a unified iterative calculation framework. Equation (1) is used at the design stage to determine the required parameters of the anchoring system, primarily the anchor length,

whereas Equation (2) serves to evaluate and refine the expected effect of the adopted solution in terms of the predicted magnitude of floor heave. The combined use of these equations ensures consistency between the calculated support parameters and the actual operating conditions of the mine roadway.

However, Małkowski [23] highlighted the necessity of integrating geomechanical calculations and monitoring systems in the design of floor reinforcement. He emphasized that only systematic monitoring of roadways allows for an adequate assessment of the effectiveness of implemented support schemes.

The convergence of the roadway sidewalls in Karaganda Coal Basin is determined using the mine convergometer KSh-1, which allows the measuring of the magnitude and evolution of horizontal displacements of the excavation contours during operation. The hallmarked rod is fixed into the rock mass like a steel anchor, but in the distance from the contour where no cracks are predicted, so usually 2–3 m inside a sidewall. The use of this instrument provided reliable data on sidewall deformations, which are necessary for assessing the effectiveness of the support system. The appearance and design of the KSh-1 mine convergometer is shown in Figure 6.



Figure 6. Mine convergometer KSh-1 used to measure sidewall convergence in underground mine workings.

To measure crack development and bedding separations within the rock mass, three-level tell-tales are used (Figure 7). Boreholes are drilled into the roof, and sometimes also in sidewalls, and the floor of the roadway. Three different colors show the safe range of roof/sidewall/floor displacements (green), elevated but still safe (yellow) or critical (red).

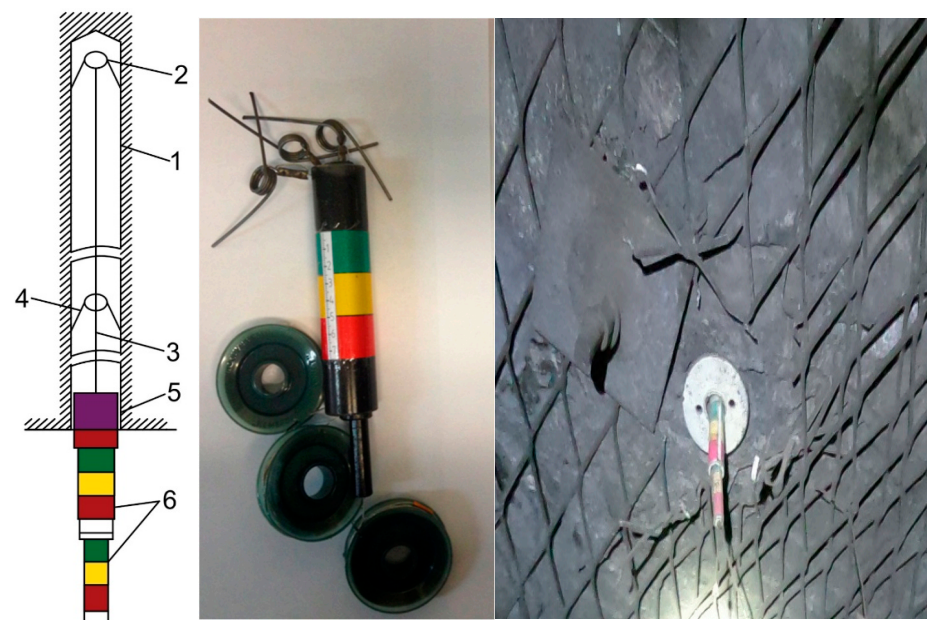


Figure 7. 1—borehole; 2—base anchor (reference benchmark); 3—steel wire; 4—lower anchor (control benchmark); 5—collar pipe; 6—indicator.

At eight mines of the Karaganda Coal Basin, pilot-industrial tests of systems for off-contour reinforcement of the near-contour floor rocks were conducted over three years to stabilize the immediate floor. Some of these projects are shown in the paper. The calculated

(using Equation (1)) and accepted length of bolts used as the reinforcement in chosen roadways is shown in Table 2.

Table 2. The anchor length determination for tested mine roadways.

Mine	Roadway (Gate)	Roadway Width B, m	Floor Rock UCS, MPa	Real Anchor Length L, m
Kazakhstanskaya	Ventilation drift 334D6-1-v	5.8	12	2.4
Saranskaya	Conveyor drift 52K7-z	5.0	25	1.6
Abayskaya	Ventilation drift (test section)	5.2	18	2.4
Kostenko	Conveyor drift 52K7-z	5.0	30–40	2.4

It is important to note that while the literature contains numerous publications devoted to the numerical analysis of anchored workings [6,8,16,25], there is a significant lack of data on practical technologies and monitoring results. The present study aims to bridge this gap.

5. Application of Floor Heave Control Technology

5.1. Ventilation Drift 334D6-1-v—Kazakhstanskaya Mine

Based on field observations, intensive floor heave was observed in the ventilation drift 334D6-1-v, with local uplift of the floor rocks reaching approximately 0.5 m, which necessitated the application of floor anchoring.

It was the first roadway where off-contour anchors were applied, and which allowed the development of guidelines for this technology of strengthening floors. The installation of perimeter floor anchors at the Kazakhstanskaya mine was carried out in the conveyor drift of the upper layer of the thick D6 seam, located on the eastern wing of the mine field, within the anticlinal and synclinal zones of the seam. The seam dip angles along the direction of drivage varied from -8° to $+3^\circ$, and near the face from -10° to -15° ; the average seam thickness in the test area was 5.4 m, and the depth of mining was approximately 520 m.

The D6 seam has a complex structure, consisting of seven to eight coal benches separated by thin interlayers of mudstone and carbonaceous shale that were 0.01–0.1 m thick. The coal benches are represented by semi-bright, banded coal with an uniaxial compressive strength of 8–15 MPa. In the lower part of the seam, approximately 0.7 m above the floor, lies a layer of crushed, earthy coal with high gas content, which was 0.1–0.25 m thick with a compressive strength of about 5 MPa. There was a mudstone layer in the floor with a thickness of approximately 4.0 m and an uniaxial compressive strength (UCS) of about 12 MPa. The immediate roof consisted predominantly of argillite (UCS $\approx$ 2.5–3.5 MPa) overlain by siltstone with UCS values of approximately 5.0–6.0 MPa.

The ventilation drift 334D6-1-v has a length of 1355 m, a width of 5.8 m, and a height of 3.8 m (cross-sectional area 18.3 m²) and was driven parallel to the conveyor drift 324D6-1-v. A coal pillar 2.0–2.5 m thick was left between them.

The technological scheme for the installation of perimeter floor anchors at the Kazakhstanskaya mine is shown in Figure 8. Anchors were installed on both sides of the roadway at a distance of 0.5 m from the frames of the arch support.

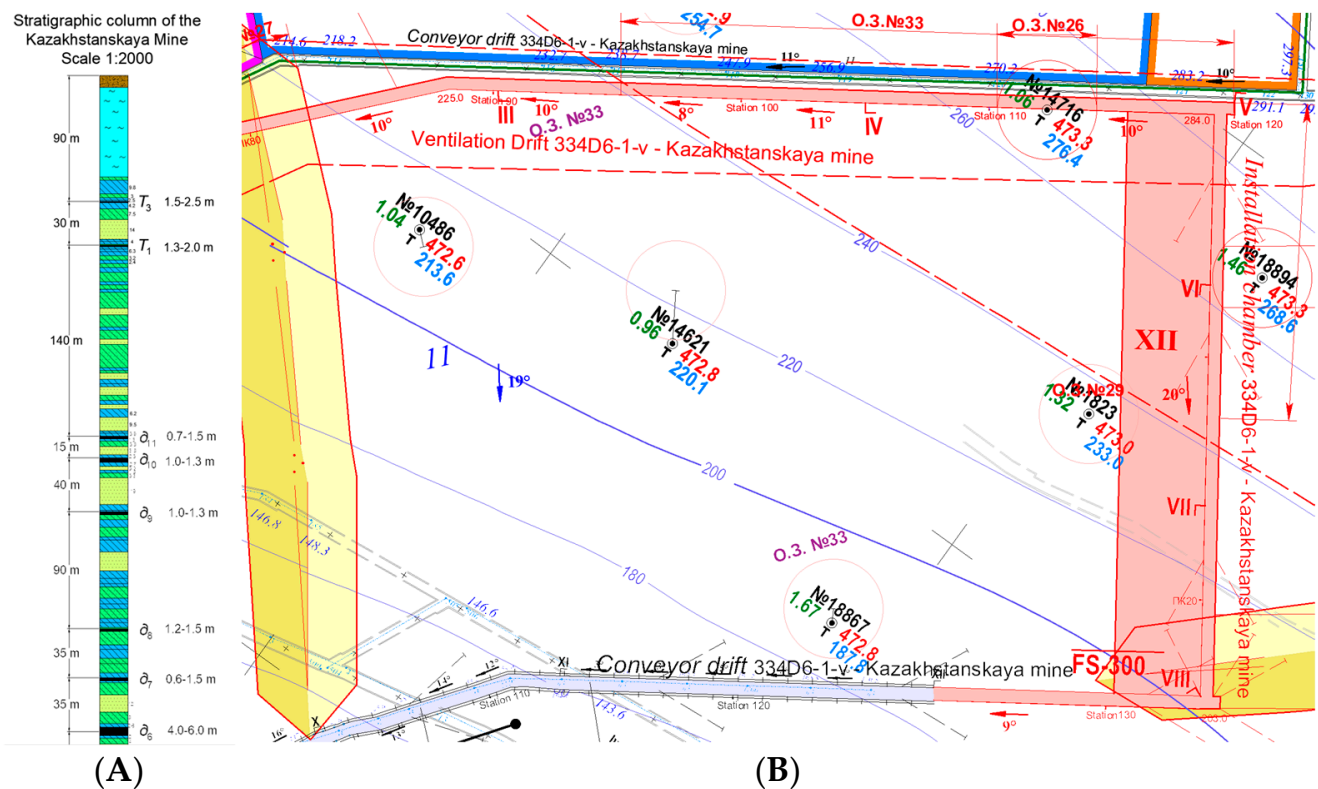


Figure 8. Ventilation drift 334D6-1-v at the Kazakhstanskaya mine: (A) stratigraphic column of the coal-bearing strata of the Kazakhstanskaya mine (scale 1:2000). (B) Plan of mine workings showing the location of the ventilation drift 334D6-1-v and the parallel conveyor drift 324D6-1-v, structural zones of the seam, stationing, and the test section where perimeter floor anchoring was applied.

For borehole drilling and anchor installation, a crew of three miners operated a Super-Turbo drilling rig (manufactured by Minova) with a working height of 1.0 m (available sizes: 1.0, 1.2, 1.4, and 1.6 m). Drill rods of 1.2 m (with a two-wing bit) and 1.8–2.2 m (with a three-wing bit) were used; later, only the latter two lengths were applied to accelerate operations. Boreholes were drilled at an angle of 45° outward beyond the roadway contour and at 45° along the drift axis. After drilling, the boreholes were cleaned of sludge before inserting resin cartridges. The average time required to drill and install one anchor was 8 min.

Anchors were positioned in a staggered arrangement, with a transverse offset of 0.1–0.2 m; penetration into the floor rocks was up to 0.5 m.

Threaded bar anchors, 2.4 m long, were installed using three resin cartridges: one fast-setting (0.35 m, setting time 15–20 s) and two slow-setting (0.6 m, setting time 3 min). Later, only two slow-setting cartridges were used. The spacing between pairs of perimeter floor anchors was 1.5 m along the roadway.

The results of field visual observations of roof and sidewall (lateral rock) displacements in the ventilation drift 334D6-1-v are presented in Figure 9. According to the observations, floor heave was recorded, with the main portion concentrated on the right side of the roadway (Figure 10A). The uplift of floor rocks reached ca. 50 cm. Additionally, deformations of arch support legs and failure of individual struts were observed (Figure 10B). The cause of floor heave was the presence of weak argillites in the immediate floor of the seam, with a thickness ranging from 1 to 16 m.

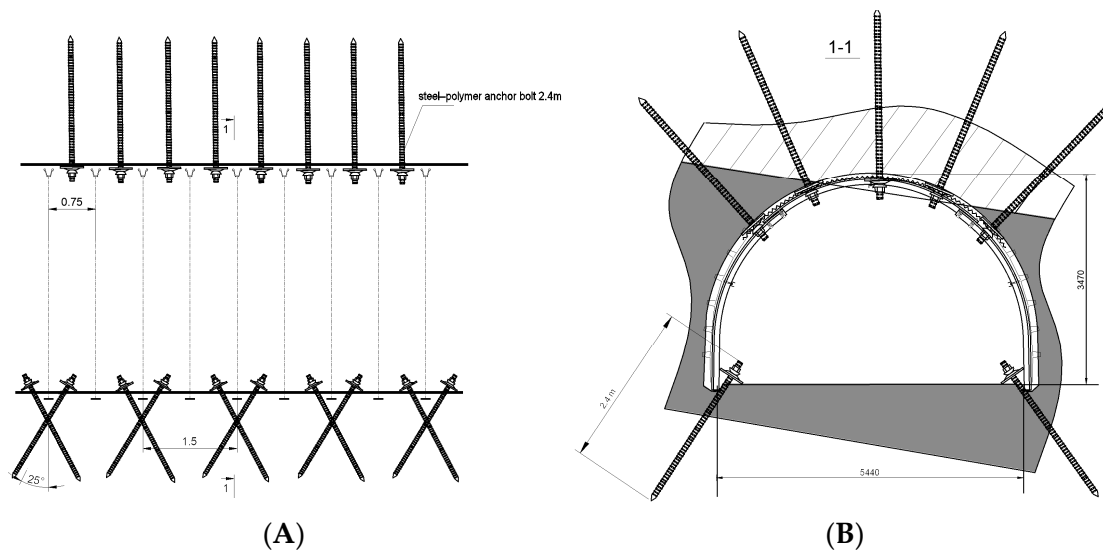


Figure 9. Technological scheme of contour anchoring at the Kazakhstanskaya mine: (A) longitudinal section; (B) cross-section.



Figure 10. Results of field visual observations in the ventilation drift 334D6-1-v of the Kazakhstanskaya mine: (A) floor heave concentrated on the right side; (B) deformation of arch support legs, extrusion of sidewall rocks, and failure of struts.

To measure deformations, crack development, and bedding separations within the rock mass, a total of six boreholes were drilled in the roadway cross-section: two in the roof, two in the sidewalls, and two in the floor. In each borehole, three-level tell-tales were installed to monitor relative displacements at different depths.

The boreholes were arranged symmetrically with respect to the roadway axis and located within the section where floor anchoring was applied after the installation of floor bolts. The layout of boreholes and the position of tell-tales relative to the roadway contour and support elements are shown in Figure 11. The following average bedding separations (delamination) of rocks were detected in the roof, sidewalls, and floor (Figure 11):

- In the roof: Up to 1.1 m and 0.2 m;
- In the left sidewall (along the seam rise): Up to 2.7 m and 1.9 m;
- In the right sidewall (along the seam dip): Up to 3.2 m and 1.4 m;
- In the floor: Up to 2.5 m and 1.2 m.

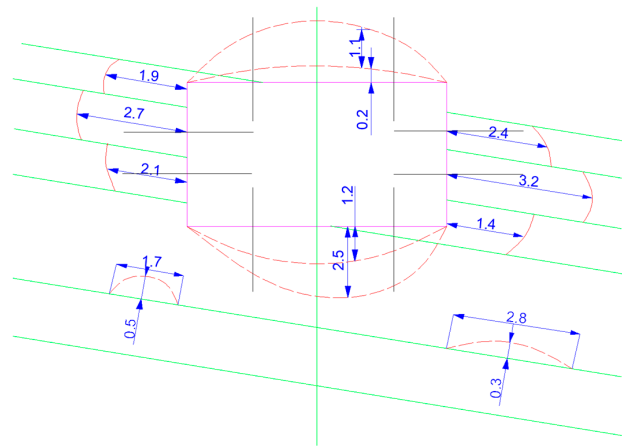


Figure 11. Results of field experiment—the range of rock separation around the ventilation drift 334D6-1-v after floor anchoring.

It can be noticed that rock separations were observed deeper in the right sidewall, that is, in the direction where the layers were inclined.

The analysis also showed that during the first month after the installation of monitoring stations, the magnitude of floor heave reached 18 mm. In the second month, it decreased to 11 mm, and in the third month, it decreased to 9 mm. The depth of disturbed floor rock was 2.5 m then.

5.2. Conveyor Drift 52K7-z—Saranskaya Mine

Reinforcement of the floor rocks was also carried out at the Saranskaya mine in the conveyor drift 52K7-z at a mining depth of approximately 600 m (Figure 12), where pairs of 1.6 m long anchors were installed in 42 mm diameter boreholes using six resin cartridges in each. The maximum coal seam thickness was 4.48 m and there were claystones and sandstones in the roof and claystones and sandy claystones in the floor.

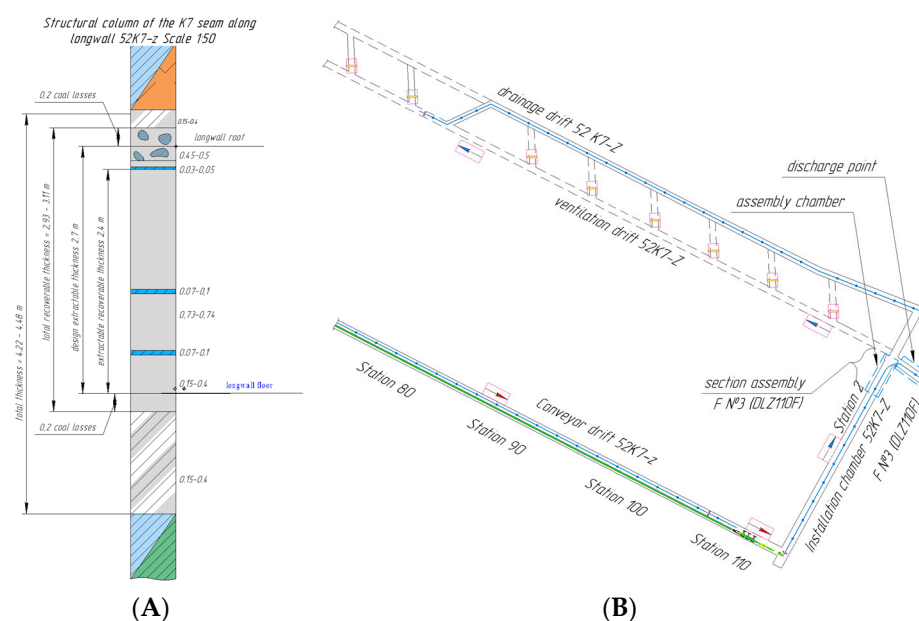


Figure 12. Conveyor drift 52K7-z at the Saranskaya mine: (A) structural column of the K7 coal seam along longwall 52K7-z (scale 1:50), showing the longwall roof and floor, coal losses, and extractable thickness. (B) Layout of mine workings, including the conveyor, ventilation, and drainage drifts of the K7 seam, installation and assembly chambers, discharge point, stationing, and locations of floor and sidewall rock bolting.

Steel anchors of 2.4 m in length were installed using resin cartridges at both corners at an inclination angle of 60–70° to improve roadway stability (Figure 13A). Due to the inclination of the rock layers, an additional 1.6 m long steel anchor was installed in the lower sidewall at a height of 1.5 m.

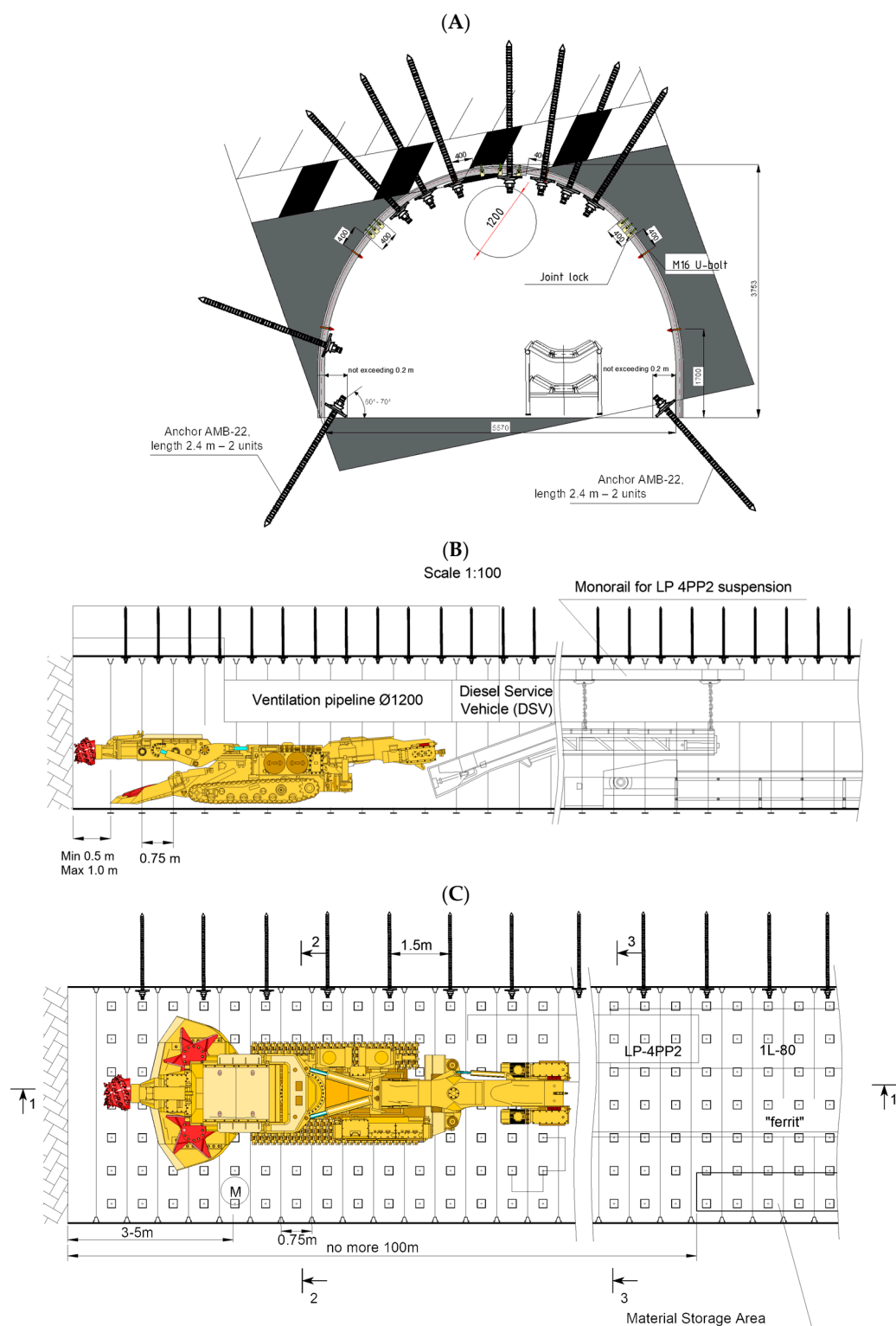


Figure 13. Floor reinforcement in the conveyor drift 52K7-z of the Saranskaya mine: (A) cross-section; (B) longitudinal section; (C) equipment layout. According to mine surveying observations, no floor heave was recorded in the experimental section.

The spacing of roof bolts was the same as that of the steel frames—0.75 m—in accordance with the developed principles for strengthening excavations with anchors (Figure 13B), while the additional sidewall anchor was installed at every second frame, i.e., at intervals of 1.5 m (Figure 13C).

At first, floor uplift on the drift section, where no floor anchoring was applied, was 50–60 cm. Therefore, a 100 m section was selected, where the above-described floor reinforcement technology was used. A total of 11 measurement stations were designed there to control the height of the roadway. Results of mine surveying observations of the height change in the experimental section of the ventilation drift 52K7-z are shown in Figure 14. It can be noticed that over a six-month period, the use of anchors in the floor of the excavation allowed for its height to be very well-maintained. The change in drift height after 6 months was a maximum of 20 cm, but the average was ca. 13 cm, so the roadway was very well-maintained due to floor anchoring.

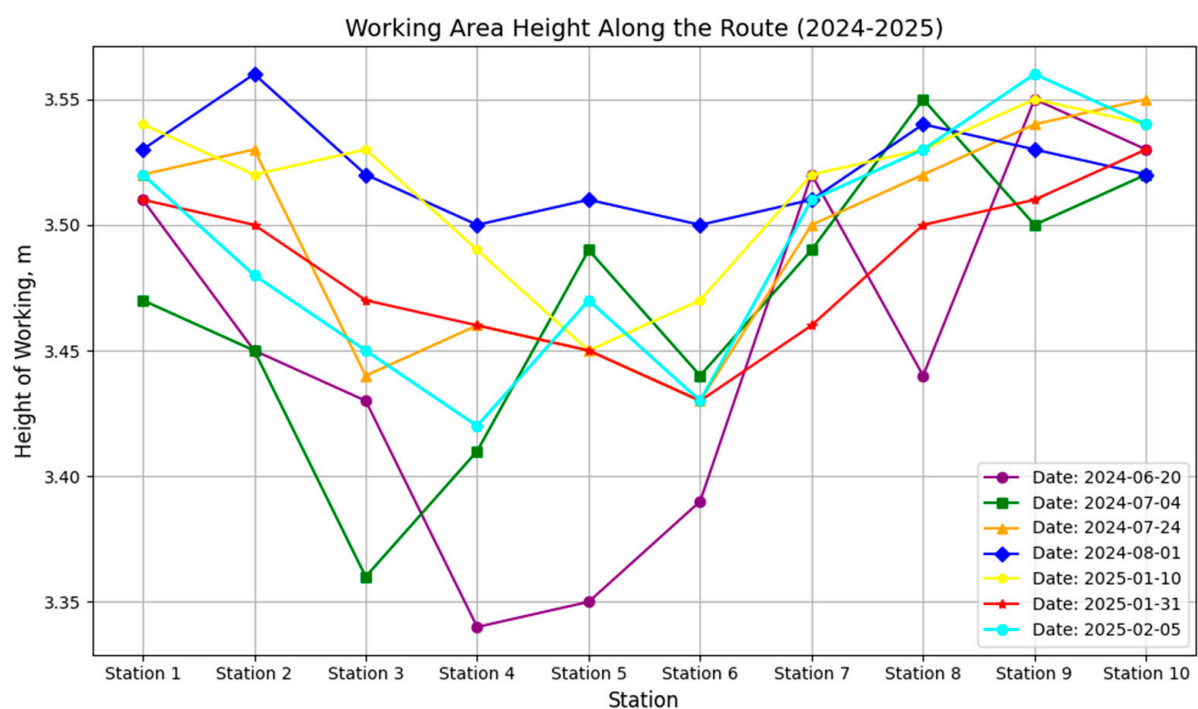


Figure 14. Dynamics of roadway height variation due to floor deformation in the experimental section of the ventilation drift 52K7-z at the Saranskaya mine, reinforced with steel bolts, according to observation results.

5.3. Conveyor Drift 341 K7-z—Kostenko Mine

The conveyor drift 341 K7-z will be driven along the K7 coal seam and through its roof rocks at a mining depth of approximately 650 m, with a total seam thickness ranging from 1.7 to 2.15 m and a seam dip angle of 3–7° (Figure 15). The seam is classified as hazardous in terms of gas and dust and is also prone to sudden coal and gas outbursts, although it is not susceptible to spontaneous combustion.

The immediate roof is composed of mudstones that are 0.2–0.5 m thick with a uniaxial compressive strength of about 2.2 MPa, and siltstones 3.5 m thick with a strength of approximately 3.4 MPa. The main roof, of medium stability, consists of sandstone up to 36.7 m thick, with a compressive strength of 5.4–5.5 MPa.

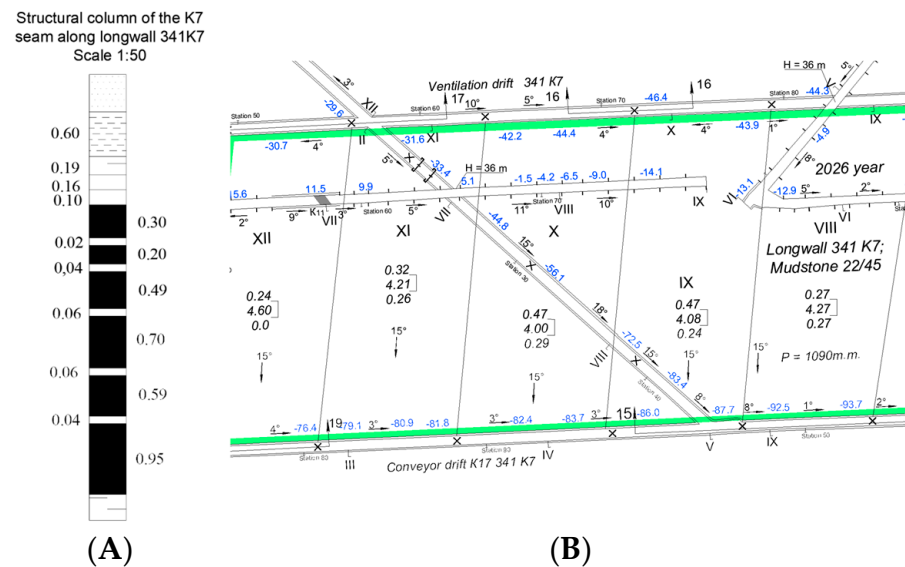


Figure 15. Longwall 341 K7: (A) structural column of the K7 coal seam along longwall 341 K7 (scale 1:50), showing the lithological composition and thickness of the seam and surrounding rocks. (B) Layout of mine workings, including the ventilation and conveyor drifts, stationing, geological structure of the panel, and the location of longwall 341 K7.

The floor of the K7 seam is composed of mudstones (0.35 m thick), sandstones (1.9 m), interbedded mudstones and siltstones (7.4 m), and sandstones up to 4 m thick. The compressive strength of mudstone is approximately 25 MPa, that of siltstone is 30–40 MPa, and that of sandstone is 50–60 MPa.

During the course of mining operations, several hazardous zones are expected to be intersected, which are presented in Table 3.

Table 3. Hazard factors for conveyor gate 52K7-z.

Name of the Mine Working	Hazardous Factors	Geodetic Tie-in	Possible Impacts
Conveyor drift 52K7-z	Hazardous zone caused by a geological disturbance, washout, $H = 0.4$ m.	Station 19–20	Weakening of the host rocks; replacement of part of the seam by rock.
	Intersection with small-amplitude geological disturbances (faults).	Station 25–40	Possible formation of domes, roof and sidewall collapses, and increased pressure on the support.

In the conveyor gate, 52K7-z off-contour floor anchors were installed in a roadway supported by an anchoring system under conditions of strong floor rocks. The anchors were embedded beyond the zone of active deformations, forming a load-bearing support block in the floor of the excavation. As a result of applying this technology, the magnitude of floor deformations was reduced by 0.7–0.8 m [34].

The highest efficiency of off-contour floor anchoring in this case was observed in anticlinal sections of the roadways, as well as in areas characterized by relatively high strength properties of the floor rocks. It was established that the main deformation processes in the sidewalls, roof, and floor of the roadways develop predominantly within the first month after drivage, which emphasizes the importance of early implementation of floor anchoring as part of the support system.

5.4. Ventilation Drift K10—Abayskaya Mine

Reinforcement works using contour anchoring were also carried out in the test section of the ventilation drift at the Abayskaya mine, located in the trough of a synclinal structure. There was 4-meter of claystone and thick bed of sandstone in the roof and mudstone in the floor (Figure 16). The main support there was steel yielding frames, KMP-A3.

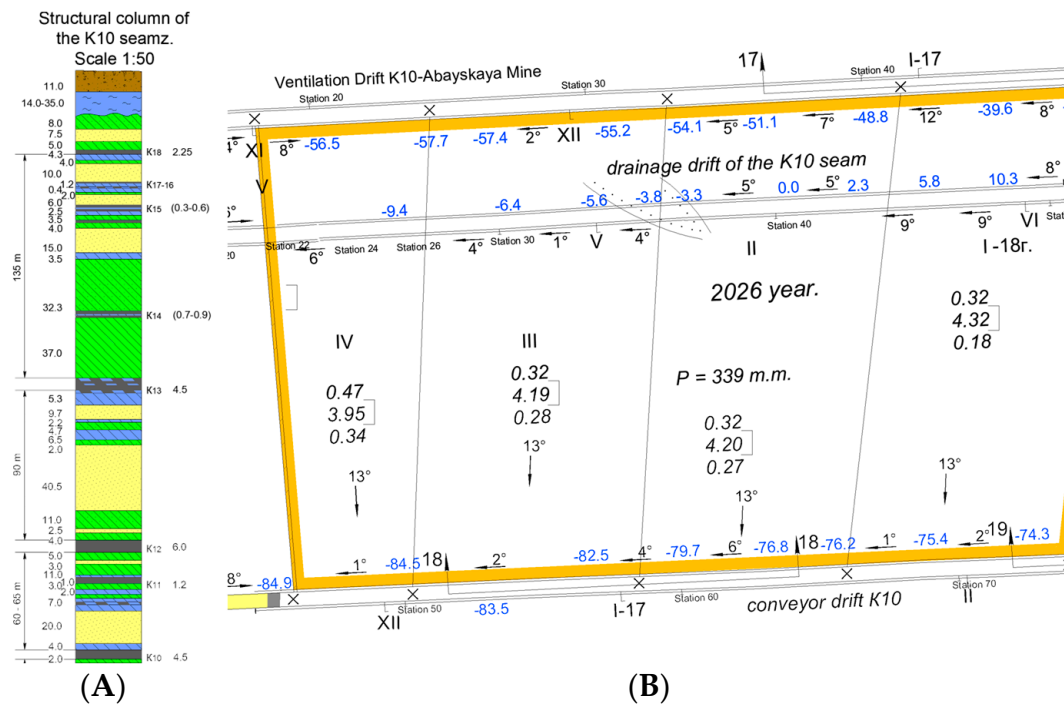


Figure 16. Ventilation drift K10 at the Abayskaya mine: (A) stratigraphic column of the coal-bearing strata; (B) layout of mine workings showing the ventilation and conveyor drifts, the drainage drift of the K10 seam, stationing, and the areas where rock bolting was applied.

After drivage but before the start of coal extraction and following the determination of the swelling layer thickness along the roadway sides (mudstone), inclined steel–polymer bolts are installed in the floor at angles of 30–45° to the bedding planes. Reinforcement begins with the installation of the first row of anchors, which includes six steel–polymer bolts in the roof (2.4 m long) and four glass-fiber-reinforced polymer (GFRP) bolts in the sidewalls (1.8 m long). As the excavation advances, a second row is installed, consisting of five steel–polymer bolts in the roof, each 2.4 m long. A metal mesh is pre-installed under the steel straps to serve as surface support (Figure 17). Since these bolts are fixed in boreholes using resin cartridges, a block-like reinforced mass forms around them, creating a load-bearing arch (reinforcement contour) that reduces compressive stresses from the roadway sides. The average mudstone compressive strength of about 30 MPa allowed for this behavior in this case.

Following the rupture of a water pipeline, a sharp increase in the water saturation of the floor rocks occurred, which led to a reduction in their strength characteristics and the development of intensive floor heave. According to field observations, intensive floor heave was recorded after water inflow into the floor rocks, with the uplift of floor rocks estimated to be in the range of approximately 0.3–0.4 m prior to the application of anchoring support.

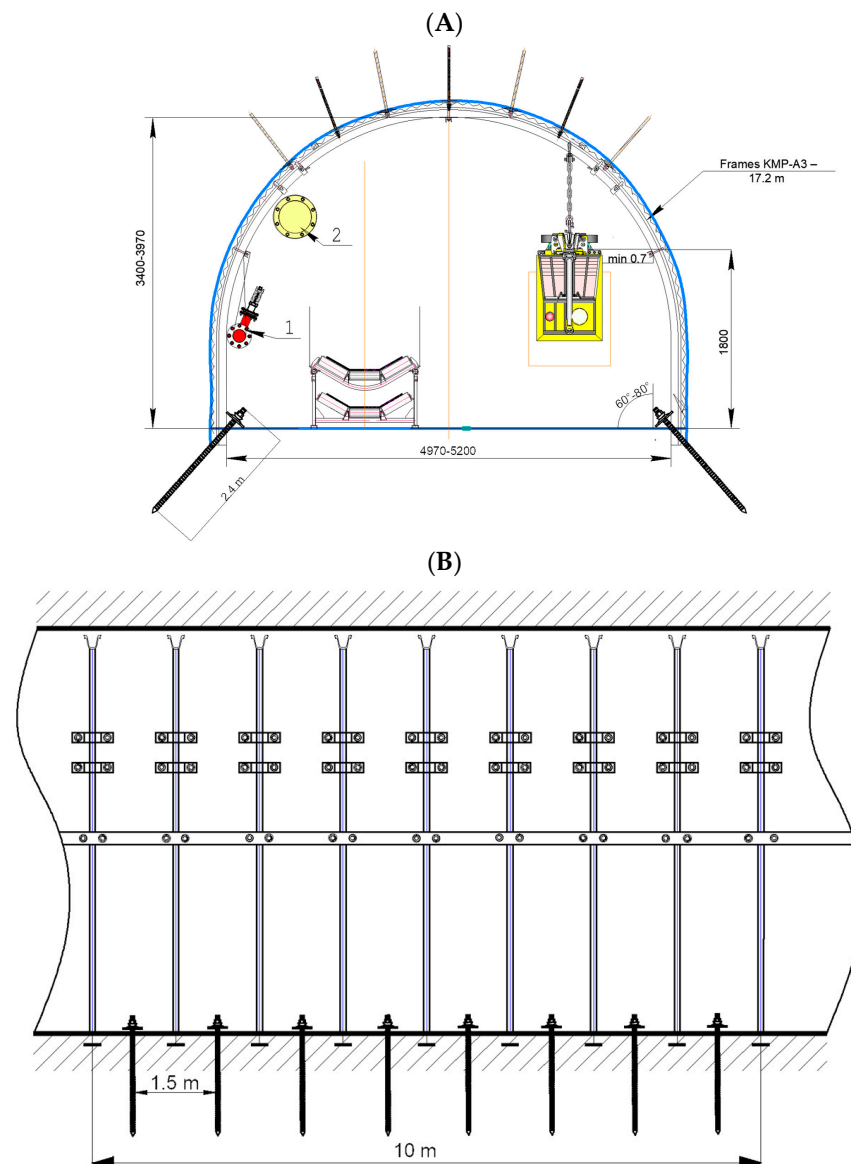


Figure 17. Testing of the anchoring technology aimed at reducing floor heave at the Abayskaya mine of the Karaganda Coal Basin: (A) cross-section; (B) longitudinal profile; 1—fire and irrigation pipeline; 2—longwall face degasification pipeline.

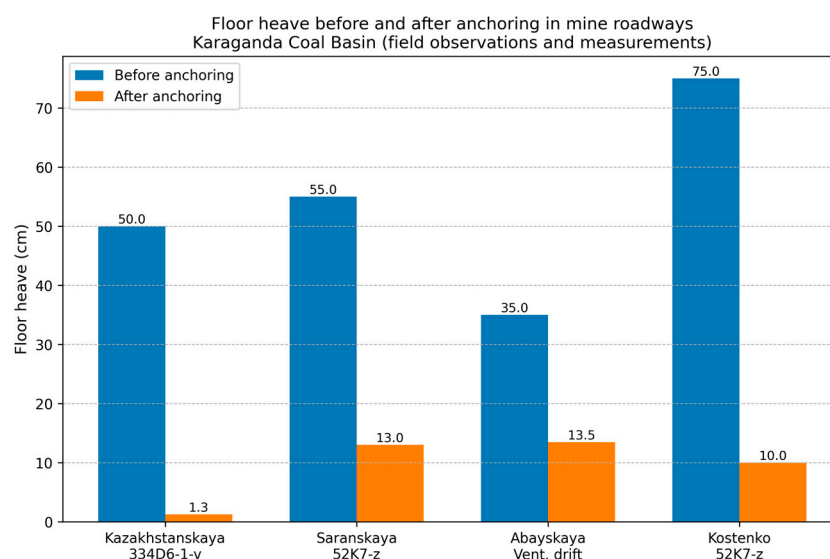
Despite the initially high strength of the floor rocks, significant deformations were observed across the entire width of the roadway, accompanied by cracking and the opening of discontinuities in the central part of the excavation. Under these conditions, a contour floor anchoring scheme was implemented, with anchor installation performed after stabilization of the water inflow. This approach made it possible to partially restrain the further development of deformations and to redistribute stresses within the rock mass.

To summarize the results of field observations and instrumental measurements, a quantitative comparison of floor heave before and after the application of anchoring support in mine roadways of the Karaganda Coal Basin is presented in Table 4. The table integrates data obtained from different mines and observation periods, clearly distinguishing between field-estimated values and instrumentally measured results, thereby providing a consistent basis for comparative analysis of anchoring effectiveness.

Table 4. Floor heave before and after the application of anchoring support in coal mine roadways of the Karaganda Coal Basin.

Mine	Roadway	Floor Heave Before Anchoring	Floor Heave After Anchoring	Observation Period	Data Type
Kazakhstanskaya	Ventilation drift 334D6-1-v	≈0.5 m (field-estimated)	12.7 ± 4.7 mm (measured)	3 months	Before—field observation; After—instrumental monitoring
Saranskaya	Conveyor drift 52K7-z	50–60 cm (measured)	13 ± 3.5 cm (avg), max 20 cm	6 months	Instrumental monitoring
Abayskaya	Ventilation drift	≈0.3–0.4 m (field-estimated)	≈12–15 cm (measured)	3–4 months	Before—field observation; After—instrumental monitoring
Kostenko	Conveyor drift 52K7-z	0.7–0.8 m (measured)	≤0.1 m (measured)	Long-term	Instrumental monitoring

The quantitative data summarized in Table 4 are further illustrated in Figure 18, which highlights the pronounced reduction in floor heave after the application of anchoring support across different mine roadways.

**Figure 18.** Comparison of floor heave before and after anchoring in mine roadways of the Karaganda Coal Basin, demonstrating a significant reduction in deformation after reinforcement.

The comparative analysis of field observations and instrumental measurements demonstrates that the application of anchoring support leads to a substantial reduction in floor heave in mine roadways of the Karaganda Coal Basin. In all analyzed cases, anchoring not only decreased the magnitude of floor deformation but also contributed to the stabilization of the rock mass behavior over time, confirming the effectiveness of the proposed reinforcement approach under different geological and mining conditions.

6. Discussion

The practical significance of this study lies in the development of integrated support schemes combining steel yielding frame support with rock bolting, including floor anchoring, aimed at reducing the phenomenon of floor heave. The research results obtained under the conditions of the Karaganda Coal Basin confirm that the mechanisms governing

the development of floor heave in underground mine workings are determined not only by the structural and strength properties of the surrounding rock mass, but also by the dynamics of stress redistribution as well as the influence of longwall mining and abutment pressure effects. The investigations have shown that changes in the rock mass structure, manifested by a reduction in uniaxial compressive strength (UCS) and deterioration of rock mass quality [26] caused by intensive mining, lead to the formation of local zones of intensive floor heave, which require targeted reinforcement of the excavation contour.

The identified regularities of changes in the stress–strain state of the coal–rock mass (displacements, stresses, zones of delamination and fracturing), depending on the main geological and mining–technical factors, make it possible to determine optimal parameters of floor support. It should be noted that the proposed empirical relationships are intended as an engineering-oriented predictive tool calibrated on in situ observations, and they do not explicitly account for time-dependent rheological effects or hydro–physico–chemical interactions of floor rocks, which were beyond the scope of the present study.

The implementation of new anchoring schemes, developed on the basis of in situ experience and geomechanical modeling results, ensures the formation of stable load-bearing blocks in the floor of mine workings and effectively limits the propagation of instability zones. It should be noted that similar mechanisms of floor stabilization through the formation of load-bearing structures and stress redistribution have also been demonstrated in studies concerning the control of floor heave in weak rocks and rocks prone to plasticization under the influence of water [7].

After eight pilot applications of floor anchors installed outside the excavation contour—both steel and polymer types—it was found that the use of floor anchoring produces a distinct positive effect in the form of a significant reduction in floor heave intensity and stabilization of deformation processes. This applies to the reinforcement of rocks surrounding roadways during the extraction of coal seams with small and moderate inclinations. Sections of workings reinforced with anchors were characterized by higher stability and a lower rate of deformation development, which contributed to extending the safe service life of the excavations. The results obtained confirm the effectiveness of floor anchors as one of the key methods for improving the stability of roadway-type mine workings, particularly when combined with other engineering techniques (e.g., steel beam systems—known as hoisting joists). It was also found that deformations and stresses in both the sidewalls and the floor of the excavation depend not only on floor anchors, but to a greater extent on the application of sidewall rock bolts. Significant positive experiences in the use of rock bolting systems for floor reinforcement have also been reported in coal mines in China [6,7,24,29,32], where convincing results were obtained. Nevertheless, the use of polymer anchors in the floor allows for their cutting or excavation when floor brushing is required, which makes this solution preferable for the Karaganda mines. At the same time, the load-bearing capacity of such fully grouted anchors amounts to 150–180 kN, placing them on a par with steel anchors.

At the same time, the conducted research program made it possible to determine the influence of contour and floor anchoring on the extent of roof delamination when the roadway is located within a zone of mining-induced pressure. Based on the results of investigations carried out in gate roadways adjacent to the longwall face, four characteristic deformation zones were identified, depending on the distance from the longwall face and the application of rock bolts (excluding steel yielding frame support):

- Zone 1: 100–200 m ahead of the longwall face;
- Zone 2: 100 m ahead of the face;
- Zone 3: 100 m behind the longwall face;
- Zone 4: 100–400 m behind the longwall face.

Additionally, the roadway width was taken into account in the study, and the results are presented in Figure 19. The figure shows the evolution of roof delamination for two roadway widths—4 m and 5 m—both without and with the application of rock bolts, indicated by the following colors:

- Red line: Width 5 m, without anchors;
- Blue line: Width 4 m, without anchors;
- Green line: Width 5 m, with anchors;
- Violet line: Width 4 m, with anchors.

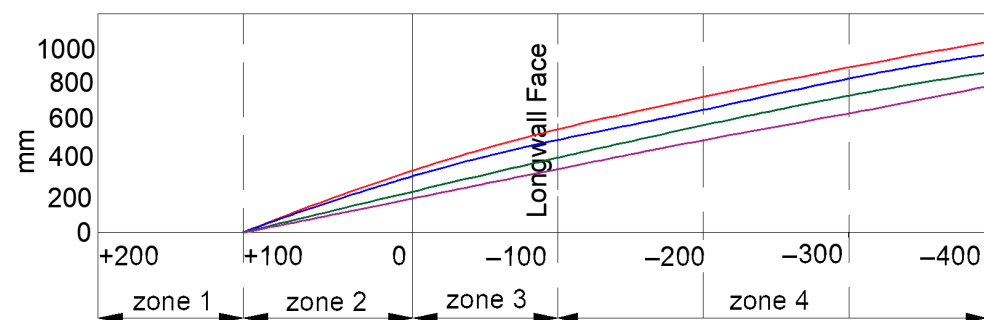


Figure 19. Technological parameters of bedding separations in the surrounding rocks (mm) as a function of distance to the longwall face.

The results indicate that an increase in roadway width leads to an increase in the magnitude of rock mass delamination; however, the application of anchors reduces it by approximately 40%. For example, in the case of a roadway with a width of 5 m, the absence of anchoring allows delamination to exceed 520 mm, whereas the use of anchors reduces it to approximately 330 mm. It should be emphasized that the observed reduction in roof delamination due to rock bolting is mechanically linked to the suppression of floor heave through the redistribution of stresses within the roadway support–rock mass system. The installation of roof and sidewall anchors increases the stiffness of the excavation contour and limits lateral convergence of the roadway. As a result, the magnitude of horizontal stresses transmitted from the sidewalls to the floor is reduced, while the zone of maximum stress concentration is shifted deeper into the surrounding rock mass. This effect decreases shear deformation and extrusion of weakened floor layers, thereby contributing to a reduction in floor heave even in cases where floor anchoring is applied locally. Consequently, sidewall and roof anchoring should be considered an integral element of the overall floor heave control mechanism under conditions of abutment pressure and asymmetric loading near the longwall face.

The investigations conducted in test workings of coal mines also allowed verification of the empirical relationship for estimating the magnitude of floor heave (Equation (2)). It was found that incorporating the roadway width, anchor length, and compressive strength of floor rocks into the formula under the conditions of the Karaganda mines provides a very good prediction of floor heave magnitude. A comparison of predicted and measured values shows satisfactory agreement, with deviations not exceeding 15%, which confirms the reliability and practical applicability of the proposed empirical model (Table 5). Furthermore, it should be noted that, in general, at an average depth of seam occurrence of approximately 600–700 m and in relatively weak floor rocks, the actual floor heave amounts to 10–20 cm. Despite the application of rock reinforcement anchors, this value is relatively low. Nevertheless, it can be concluded that for each coal basin, due to specific geological conditions, it is possible to develop an empirical formula that estimates the magnitude of floor heave with a high degree of accuracy.

Table 5. Comparison of measured and predicted floor heave values for tested mine roadways.

Mine	Roadway (Gate)	Roadway Width B, m	Floor Rock UCS, MPa	Anchor Length L, m	Measured Floor Heave u_f , Meas, m	Predicted Floor Heave, u_f , calc, m	Deviation, %
Kazakhstanskaya	Ventilation drift 334D6-1-v	5.8	12	2.4	0.18	0.16	−11
Saranskaya	Conveyor drift 52K7-z	5.0	25	1.6	0.13	0.14	+8
Abayskaya	Ventilation drift (test section)	5.2	18	2.4	0.11	0.12	+9
Kostenko	Conveyor drift 52K7-z	5.0	30–40	2.4	0.07	0.08	+14

The application of the proposed support solutions in the design and operation of mine workings with various profiles and functions confirms their practical applicability. The obtained results are consistent with current international research and confirm the effectiveness of an integrated approach combining numerical modeling, in situ monitoring, and adaptive implementation of support systems in accordance with local mining and geological conditions.

7. Conclusions

The conducted investigations demonstrate that floor heave in the Karaganda Coal Basin is governed by the combined influence of geological structure, stress–strain redistribution in the rock mass, and mining and technical conditions associated with longwall extraction. Floor instability is not limited to localized deformation but represents a progressive process that directly affects roadway stability, support performance, and overall mine safety.

For the first time in the Karaganda Coal Basin, a comprehensive approach integrating field observations, geodetic measurements, laboratory testing, and numerical modeling was implemented. This approach made it possible to quantify the extent of floor delamination and fracturing, identify the spatial limits of longwall-induced influence, and analyze the temporal evolution of deformation in gate roadways. Based on these results, engineering design parameters for floor reinforcement were refined and validated under real mining conditions.

The application of anchoring support systems demonstrated a pronounced quantitative effect. Field observations showed that the use of combined anchoring schemes reduced floor heave by 70–90% compared to unreinforced conditions. In absolute terms, floor uplift was reduced from 0.3–0.8 m to 0.01–0.15 m, depending on geological conditions and mining depth. In several test sections, the deformation process was fully stabilized, with no further progressive uplift observed during the monitoring period.

Compared with traditional frame and yielding support systems, anchoring support exhibits several key advantages. While conventional supports primarily react to ongoing deformation and require repeated floor cutting and rehabilitation, anchoring systems actively modify the stress–strain state of the rock mass by transferring loads into deeper and more competent strata. This results in improved roadway stability, reduced convergence rates, and a significant extension of the safe service life of mine workings. At the same time, anchoring support requires higher design accuracy and preliminary geomechanical assessment, which may limit its effectiveness if empirical parameters are applied without proper field validation.

Pilot industrial implementation confirmed that anchoring support can lead to measurable economic benefits. The reduction in floor heave intensity decreases the frequency of roadway rehabilitation operations, reduces material consumption for repeated support installation, and lowers labor and downtime costs. In practical terms, the observed sta-

bilization of roadways enables longer maintenance intervals and improves the overall efficiency of underground operations.

The results obtained highlight that empirical design relationships for floor anchoring must be verified through systematic field measurements and full-scale underground trials. Only such an approach ensures the reliable selection of anchor length and layout and prevents the incorrect application of design equations outside their validated geological domain.

Overall, the proposed anchoring solutions demonstrate clear prospects for large-scale engineering implementation in the Karaganda Coal Basin. When properly adapted to local geological conditions, these solutions may also be applied in other coal mining regions facing similar floor heave problems, contributing to improved industrial safety and sustainable underground mining practices.

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